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Mine Power Systems

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PREFACE

The application of electricity to the mining industry is a distinctive area of both mining engineering and electrical engineering. The difficult environment, the dynamic power loads, the cyclic and mobile operation and stringent safety requirements that characterize mining, all place unique demands on the mine power system. No other industry makes such extensive use of portable extensible equipment or has such complex grounding problems. Mine power systems can range from relatively simple installations for small surface mines to complex underground systems where the harsh environment of dust, humidity, and cramped spaces stretches the ingenuity and creativity of the engineer to provide reliable service.

At the present time there is no up-to-date engineering text available that deals specifically with mine power systems. This has created extensive difficulties for educators, industry engineers, and regulatory agency personnel. The need for a suitable reference for students in mining engineering provided the main impetus for this book, since the technician-level material that was in existence proved unsuitable for teaching young engineers who have little practical experience.

The objective in preparing this manuscript was to assemble a single engineering reference on mine electrical power systems that is as comprehensive as possible. Earlier drafts of this material have been used successfully to instruct university students in courses ranging from basic electrical engineering through power-system design. It is felt, however, that the usefulness of this material extends beyond that of a student text. While not intended to replace other electrical or mining references, this publication is also an indexed, reasonably comprehensive reference handbook for industry engineers and training personnel, and a source of material for electrical engineers who wish to expand their education into industrial power-system applications. Obviously, there will be some omissions; to include all aspects of mine electrical systems in one volume would approach an impossibility, but an attempt has been made to collect together the most significant information, thereby providing the tools needed to continue a knowledgeable involvement in mine electricity.

This reference work is divided into three general content areas. Chapters 1 through 5 contain information considered elementary, chapters 6 through 11 deal with power-system components, and chapters 12 through 17 contain specifics on mine power systems. A person familiar with electrical principals can use the earlier chapters as review material, but all chapters contain material relevant to mining and discuss the necessary combinations of equipment and components that should be contained in the mine power system. Emphasis throughout is placed on coal mining systems, although much of the material pertains to all mining operations. Both surface and underground power systems are discussed, the latter in more detail since these are the more complex systems and encounter the most problems.

This publication is a thoroughly upgraded and extensively revised edition of Bureau of Mines Open File Reports 178(1)-82 and 178(2)-82, prepared under Bureau contract J0155009 by The Pennsylvania State University. It contains new chapters, new illustrations, and example problems that were not included in the original report.

The assembly of this material has been a major undertaking. Many industry, academic, and Government agency personnel helped to review and critique practically every stage of draft preparation. The original report version was made available to students taking the mine power-systems courses at The Pennsylvania State University, and their involvement was critical input to manuscript preparation.

The author is grateful to all the companies and individuals who contributed or cooperated in this effort; so much information could not have been gathered without their help. A special thanks is owed to the late Robert Stefanko. He originally perceived the need for this text and provided guidance and encouragement throughout the project that produced the original report version. Others deserving special mention are A. M. Christman, R. H. King, J. A. Kohler, G. W. Luxbacher, T. Novak, J. N. Tomlinson, F. C. Trutt and D. J. Tylavsky. Each contributed directly to the text while on the faculty or staff at The Pennsylvania State University; acknowledgements for their contributions are made in the individual chapters.

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MINE POWER SYSTEMS

By Lloyd A. Morley[1]

ABSTRACT

This Bureau of Mines publication presents a comprehensive review of mine electrical power-system theory and practice. It discusses fundamental theory and the vital aspects to be considered in planning and designing mine electrical power systems. The report is divided into three major sections. The first presents the history of electricity in mining and the fundamentals of electrical phenomena and components. The second focuses on power-system components: motors, grounding systems, cables, and protective equipment and devices. The final section includes mine power-center equipment, switchhouses and substations, batteries, and mine maintenance.

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CHAPTER 1.—ELECTRICAL POWER IN MINING

Probably no other mining area has grown so rapidly yet been as little understood by the average mine worker or operator as the mine electrical power system. Traditionally, the field has held little interest for the mining engineer, who has tended to avoid it, or for the electrical engineer, who has given it scant attention. But today's mine power system is both complex and subject to numerous legal constraints, and it is no longer possible to treat it with the indifference of the past.

Underground mining machines are among the most compact and rugged equipment over designed, and individual units can have up to 1,000 total horsepower. Mining equipment is usually mobile and self-propelled; most is powered electrically through portable cables and, for safety, must be part of an elaborate grounding system. The machines and power-distribution equipment are seldom stationary, must be adapted to continuous cyclic operation, and must resist daunting levels of dust and vibration.

Surface mining can involve the largest earth-moving equipment built, where one piece can have 12,000 or more connected horsepower—the largest today is over 30,000 hp. The electrical loads created by this machinery are cyclic and extremely dynamic: the largest excavator, for example, can require electrical loads that range from 200% motoring to 100% generating every 50 to 60 s, under the most exacting physical conditions. In the ever-moving mining operation where distribution of power must be constantly extended and relocated, subjected to abuse by machine and worker alike, the potential for safety hazards is always present.

Engineering and maintaining such an electrical system is demanding and challenging. It requires a specialist with knowledge of both mining and electrical engineering. Yet conversely, the effective management of a mine requires that anyone responsible for production and safety also be conversant with the mine electrical system. Management should understand the advantages and disadvantages of one system over another, for if the power system is poorly designed, not only will safety be compromised but the mine operator will pay for the resulting conditions with high power bills, high-cost maintenance, and loss of production.

Too often, a new mine is designed to use the type of power system employed in the preceding mine, without a comprehensive power study to determine the system needs and examine the alternatives available. Problems arise in existing mines when new mining equipment has been adopted without due regard for its impact on the operating power system; these problems haunt the mine electrical engineer who must frequently cope with a system that is a mongrel, bred from diverse inheritances from the past combined with recent changes and additions. New laws, standards, and safety requirements must frequently be accommodated by power systems not originally designed to meet their specifications; new and unfamiliar equipment must be grafted to the existing network, and the result can be a hybrid of considerable complexity. This text has been produced to assist the power engineer and the student in understanding these complexities and the principles that lie behind them.

The material presented here is structured so an individual unfamiliar with electrical engineering can first develop the necessary fundamentals before embarking into mine electrical design. A basic physics and calculus knowledge is necessary to understand the content completely. The goal has been to assemble the most significant information required for comprehension of mine power systems so that the reader may then progress to more specialized topics. But first, a brief review of the development of electrical usage in mines is given, in order that the reasons for some of the peculiarities of mine power systems can be appreciated.

MINE ELECTRICAL HISTORY

Electricity was first introduced into coal mines shortly before the beginning of the 20th century in the form of direct current (dc) for rail haulage. This form of current was used because at that time most systems were powered by dc generators. It had a number of advantages for haulage; the most outstanding was that the dc series-wound motor had (and has) excellent traction characteristics. Speed control was a simple matter of placing a resistance in series with the motor armature or field circuits.

Batteries served as the first power source, and hence the vehicle was extremely mobile even though constrained on rails. However, keeping the batteries charged was bothersome, so trolley wires were soon introduced in several mines. Allowing the trolley wire to act as one conductor and the rail as the other provided the simplest form of power distribution yet known to the mining industry. Available haulage machinery of that period was low in horsepower and the mines were relatively small so the increased resistance that reduced voltage and power supplied to the motors was still acceptable. Thus, the dc system at a voltage of 250 or 550 V became firmly entrenched in coal mines.

Underground Mine History

Underground, the first electrically driven coal mining machine, the coal cutter, was installed in the early 1920's. Although dc offered no special advantage, it was readily available; hence, the machine was equipped with a dc motor and added to the system. The cutter was followed almost immediately by the loader, and it too was driven by dc motors. If there was rail haulage in the mine, trailing cables supplied power from the trolley wire and the rail to the machines.

The next significant increase in power consumption came with the introduction of the shuttle car, almost 20 yr after the coal cutter. Actually, when the shuttle car was first invented in 1937, it was battery powered. The addition of an automatic reeling device to handle a trailing cable came later, in an attempt to overcome battery deficiencies. These trailing cables were also connected to the haulage power system, and this equipment, when combined with the cutters and loaders, placed additional stress on the dc distribution system.

At that time, the horsepower required to operate each piece of electrical mining equipment was quite small and no individual machine used a large amount of current. However, when all machines were combined, significant power was required, and because all the conductors offered resistance, voltage drops and transmission losses in the distribution system were extensive. Alternating current (ac) would have been more practical because it could have been distributed easily at a higher voltage, thereby reducing current, voltage drops, and transmission losses. But many States had stringent limitations on maximum voltage levels, usually around 300 V, and with this restriction ac had no advantage over dc. Hence, dc continued to be used to operate the successful combination of cutters, loaders, and shuttle cars.

Development in ac-to-dc conversion equipment played an important role in underground coal mine power utilization throughout this period. Motor-generators or synchronous converters were originally employed for conversion purposes, but in addition to being heavy and bulky, they could not be operated effectively in a gassy and dusty atmosphere, and maintenance requirements were substantial. As a result, most conversion installations were placed on the surface with borehole connections to the underground mine. This was acceptable as most mines were then relatively shallow.

In the 1930's, the same decade that saw the inception of the shuttle car, mercury-arc-ignition rectifiers began to be employed to provide dc underground. The arc tubes allowed more efficient use of electricity in deeper and larger mines than had previously been possible. As the tubes had no moving parts, maintenance was lower, efficiency was higher, and portability was improved. These rectifiers were usually centrally located in the mines because a liquid heat exchanger made them heavy and bulky. In this way, distribution to the mine rectifier was ac, but distribution throughout most of the mine electrical system was still dc. At about the same time, some mines found that haulage of materials by conveyor could be more efficient than haulage by rail. The conveyors were also powered by dc motors, and stress continued to be added to the electrical system.

In the late 1940's, when continuous mining machines first began to be used extensively, dc was again expected to provide the power. However, the continuous miners normally needed more energy input than the sum of the various conventional mining equipment they replaced, and because the required horsepower created high current demand, dc was found to be entirely unsatisfactory in most cases. The attendant current demand created enormous voltage drops in the distribution system. As a possible solution, the dc supply system was separated from the haulage system, but even this was unable to improve voltage regulation. During peak operation periods, voltages at the machines were so far below the values called for that even moderate efficiency was impossible. The increasingly large cable sizes required to supply the needed power created difficult cable-handling problems. The use of three-phase ac distribution and motors was an obvious answer, but for at least a decade some mining companies were reluctant to make the change. In many instances this was because the laws in some States limited maximum voltages in the mine. Lawmakers were convinced that high voltages were synonymous with high safety risks. Some State laws were not updated until the mid 1960's.

When higher voltages were finally permitted, the desirable economics of ac employment could be realized and there was a swift transformation from dc to ac for both distribution and high-horsepower loads in underground coal mines. Unfortunately, many mine electrical systems were at least partially modified without concern for the compatiblity of these changes with the remainder of the system, and various safety and production problems arose.

As a result of conversions, mine power systems generally had two voltage levels, one for distribution and one for utilization. The simplified mine electrical arrangement shown in figure 1.1 illustrates the results. Here, the substation transforms the utility voltage down to distribution levels, which are most often at high voltage greater then 1,000 V. Power at this voltage is transmitted or distributed through conductors from the substation to the power center; hence, this system is called the distribution system. The power center or load center, actually a portable substation, transforms the voltage to utilization levels, which are typically at low voltage of 660 V or less, or medium voltage of 661 to 1,000 V. At this level, or face voltage, power is normally delivered to the equipment. Despite this reference to voltage levels, it should be noted that distribution and utilization describe functions of a power system segment, not specific voltage ranges.

Originally, primary ac distribution was made at 2,300 or 4,160 V. In most mines, these levels were later increased to 7,200 V. Some operations recently increased the voltage to 12,470 or 13,200 V for both longwall and continuous-mining applications. Each new distribution voltage, it may be noted, is a factor of $\sqrt{3}$ times the previous value ($\sqrt{3}$(2,300) = 4,160). The principal reason for increasing the voltage was that, for the same load, current would be correspondingly smaller, and lower distribution losses would result even though the same conductor sizes were used.

From the beginning, 440 Vac was the most popular voltage for utilization, despite the fact that when the continous miner proved so successful its horsepower was progressively increased, following the sometimes misguided notion that a directly proportional increase in coal production would follow. As with dc, the additional horsepower resulted in an increase in trailing-cable sizes, until the weight of the cables was almost more than personnel could handle. To compensate, the most common move was to raise the rated motor voltages to 550 Vac. More recently, manufacturers have produced machines with 950-V (550 $\sqrt{3}$) motors to further overcome the trailing-cable problems.

While these changes were being made to ac for machine operation and distribution, the use of dc for haulage continued to be advantageous. In the early 1960's, silicon diode rectifiers with large current capabilities became available. Simple, efficient, and small, these rectifiers were ideally suited for use underground and made ac distribution possible for the entire electrical system except rail haulage. Through the use of rectifiers, the benefits of dc for traction and of ac for distribution and utilization on high power loads could be realized. For example, while continuous miners normally used ac, part of the supply at the power center was rectified to dc, primarily for powering the shuttle cars.

These underground electrical systems appeared to be simple, and as a result they did not become the focus of attention for some time. Systems were frequently designed and maintained by a "seat-of-the-pants" approach, to the point that ac distribution and equipment were installed

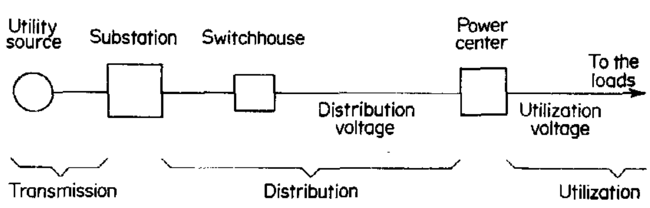

Figure 1.1.—Simple mine electrical system arrangement.

employing dc concepts. However, ac systems are more complicated than dc systems and call for meticulous planning; if wrong decisions are made, the results can be extremely costly in terms of safety, production, and economics. A great deal of effort is needed to maintain an electrical power supply within the requirements of the individual pieces of mining equipment, and mixing ac and dc in the same mine has added greatly to the problems.

This brief review of the development of electrical systems in underground coal mines has shown that the mines went from minor electrical usage with the introduction of rail haulage to almost total dependency on electricity in a period of 50 yr. In the same period, surface coal mining underwent changes that were as substantial if less numerous. They were centered around the enormous growth in equipment size.

Surface Mine History

The first mechanization of strip mining occurred in 1877 with the application of an Otis-type steam shovel in a Pittsburg, KS, mine *(5)*.[1] This early attempt was somewhat unsuccessful, but it served as an important step in the evolution of strip-mining machinery. Several successful attempts to use steam shovels and draglines were made in the next 30 yr, and these proved that the surface mining of coal was completely practical. In time, the advantages of electricity over steam became more apparent, and the first significant introduction of electric-powered shovels was made in the early 1910's.

Whereas dc series motors were universally employed in underground rail haulage, the first large motors used in surface mining were dc shunt wound because of their constant-speed characteristics. These motors almost directly replaced the single-speed steam engine found on practically all shovels prior to that time and allowed an immediate reduction in work force requirements. Before long another important advance in shovel design occurred: the application of separate steam engines to power the shovel motions of hoist, crowd, and swing. This change gave increased flexibility through the individual control of each operation. In a short time, the two major shovel manufacturers of that era, Marion and Bucyrus, began to produce both steam and electric multimotor shovels with similar characteristics *(5)*. Since series-wound dc motors had speed-torque relationships similar to those of steam engines when they were used for this type of duty, they were employed to drive each shovel motion.

The initial distribution for electric shovels was dc because of the nature of the power generation, but technological advances soon made ac power systems superior, and ac motors were tried with some success. However, by 1927, ac-dc motor-generator (m-g) sets and the invention of the Ward-Leonard control concept caused these efforts to be abandoned. The new control system enabled the motor characteristics to be modified as desired within the motor and generator commutator limits, and as a result, separately excited dc motors became more attractive than series-wound motors. The m-g sets functioned as on-board power-conversion units, thereby establishing the use of ac distribution in surface mines.

Motor-generator sets driven by synchronous or induction ac motors, Ward-Leonard control, and separately excited dc motors established the standard, and even now the combination is used on most mining excavators, especially the larger varieties *(2)*. On smaller machines, some single ac electric-motor drives with either mechanical-friction or eddy-current clutch systems have evolved, but these are often driven by diesel engines.

Present excavating equipment is generally classified into three size groups, although actual capacity ranges are normally not assigned. Small shovels are used primarily in general excavation, while the intermediate types work at bench mining and coal production, and large shovels handle overburden stripping. Draglines of all sizes are used only for stripping. Small and intermediate equipment originally ran on rails, but crawler mountings that give improved mobility made their first appearance in 1925 *(5)*. Today, small and intermediate-sized shovels and draglines are mounted on two crawlers, while large shovels have eight crawlers *(2)*. Large draglines and some intermediate sizes are usually walking types that feature a circular base or tub that provides low ground-bearing pressure and a walking device for mobility.

The design of surface mine drilling equipment paralleled excavator development. Initially, most drills were pneumatic-percussion types, but because electricity was readily available in mines, some machines were designed with internal motor-driven compressors. By the early 1950's, large rotary drilling equipment was necessary to satisfy the blasting requirements of thick, hard overburden *(5)*. This drilling equipment was again electrically powered and was very successful.

The most outstanding change that has taken place in electrically powered surface mining equipment has been in connected horsepower. For example, a 25-yd^3 dragline or stripping shovel that had a maximum total load of around 2,000 hp was considered enormous in the late 1940's *(5, 9)*. By 1955, 50- to 70-yd^3 excavators were being manufactured with maximum horsepower at 4,650 hp. Five years later, shovels had reached a 140-yd^3 capacity wth 12,000 hp of main drive motors *(7)*. In 1976, the largest excavator in service had 20,000 hp in m-g set drive motors *(4)*.

Distribution and utilization voltages also increased to keep pace with the peak load demands of this machinery. Sometimes the mine distribution and machine voltages for these excavators remained the same. Until the mid-1950's, 4,160 and 2,300 V were the usual mine levels *(9)*. Then, with the advent of larger concentrated loads, 7,200 V was considered advisable *(10)*. However, this level was found to be unsatisfactory for the newly introduced machines with a capacity larger than 100 yd^3, and so 13,800-V mine and excavator voltage became a standard. With machines having greater then 200-yd^3 capacity, 23,000-V utilization was established *(4)*, but even with these substantial increases in distribution, some loads up to 1,000 hp continued to be driven at 480 V *(10)*. Production shovels with loads up to 18 yd^3 commonly stayed at 4,160 and 7,200 V, while in general, 4,160 V became standardized for machinery with 1,500 hp or less. As a result, more than one voltage level could be required at a mine when excavators of different sizes were employed.

MINE POWER EQUIPMENT

A few pieces of power equipment have already been mentioned but only to the extent necessary to describe the concepts of distribution and utilization. The evolution of

[1]Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

mine systems has resulted in major items of power apparatus, each serving a specific function (*1,9*). In general, they can be listed as

- Generation,
- Main substations,
- Portable and unit substations,
- Switchhouses,
- Distribution transformers and power (or load) centers, and
- Distribution (conductors and connectors).

The following paragraphs explain these components only in sufficient detail for their inclusion in system arrangements to be understood. More detailed descriptions of substations, switchhouses, and power centers are presented in chapters 12 and 13, while chapter 8 is devoted to distribution. Power generation is beyond the scope of this text, but Ehrhorn and Young (*13*) provide a thorough discussion of generation related to mining.

Substations

It is common mining practice to purchase all or most power from utility companies if it is available. As utility voltages usually range from 24 to 138 kV, a main (primary) substation is required to transform the incoming levels down to a primary distribution voltage for the mine. In addition to having the transformer, substations contain a complex of switches, protection apparatus, and grounding devices, all having a function in safety. Main substations are often permanent installations. The nature of the mining operation and its power needs dictate how many main substations are required and where they should be placed. They may be owned by the utility or the mining company; the decision of ownership is commonly dependent on economics. However, if the total connected load is greater than 1,000 hp, mine ownership is often more favorable (*13*).

Portable and unit substations are similar in operation to main substations except they serve to transform the primary distribution voltage to a lower distribution level. The term "unit" means that the substation and power equipment are designed and built as a package. In a typical strip-mining deployment, a large dragline may require 24 kV while the production shovels and other mining equipment need 4,160 V.

Switchhouses

Switchhouses are portable equipment that protect the distribution circuits. Their internal components are chiefly protection devices, with circuit deenergization performed by disconnect switches, oil circuit breakers, or vacuum circuit breakers. The switchhouse may contain more than one complete set of devices, for instance, a double switchhouse, which can independently protect two outgoing circuits. This category encompasses disconnect switches, which are power equipment containing only manual devices, with the prime function of allowing mine power to be removed from the main supply.

Power Centers

At the outermost distribution points there are power centers and distribution transformers, which transform and convert the distribution voltage to utilization levels. Included here are ac to dc conversion equipment or rectifiers, which convert the distribution voltage to dc for use on rail trolley and other systems. The power center, also termed a *load center,* usually implies an internal bus, which is defined shortly, in the section, "Radial System." In essence, these are all portable substations, and as with switchhouses, each outgoing circuit has its own set of internal protection components. However, an individual unit may supply from 1 to as many as 20 machines. Power centers can be considered the heart of an underground mining section power system. In surface mines, power centers supply power to low-voltage auxiliary machinery and loads; there may be no need for this equipment with the primary mining machinery.

Distribution Equipment

This category of major power equipment is often referred to as the weakest link in the mine power systems. It encompasses all the overhead powerlines, cables, cable couplers, and trolley lines used to carry power and grounding between the power equipment and eventually to the loads. The conductors are usually called *feeders* when they are part of distribution; at utilization, when connected to portable mining machines, they are called *trailing cables.*

BASIC DISTRIBUTION ARRANGEMENTS

The basic distribution arrangements available for industrial applications are radial, primary selective, primary loop, secondary selective, and secondary-spot network (*6*). Radial systems are the most popular arrangements in mining, though other configurations can be found where special circumstances call for greater system reliability (*3*). Surface mines have, of course, greater flexibility than underground mines and employ a wider range of distribution arrangements. Secondary-spot networks, which are the most popular system for large facilities in other industries, are uncommon but could be applied to preparation and milling plants. The following descriptions of the main distribution patterns are based on the Institute of Electrical and Electronics Engineers (IEEE) definitions (*6*). This institute, the leading national professional electrical organization, sets standards and recommended practices that are internationally renowned for their correctness.

Radial System

Figure 1.2 shows radial distribution in its simplest form. Here, a single power source and substation supply all equipment. The single vertical line represents one connection point for all feeders, or all connecting lines, and is termed a *bus.* Voltage along the bus is considered to be constant. Radial systems are the least expensive to install as there is no duplication of equipment, and they can be expanded easily by extending the primary feeders. A prime disadvantage is tied to their simplicity; should a primary component fail or need service, the entire system is down.

An expanded radial system, the load-center radial, is illustrated in figure 1.3. As in figure 1.1, two or more voltage levels are established, but the feeders form a treelike structure spreading out from the source. This system has the advantages of the simple system and several others too. If the load centers or distribution transformers are placed as close as practical to the actual loads, most distribution

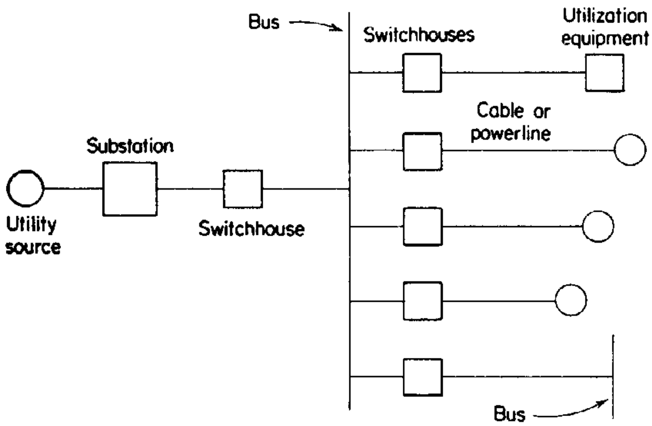

Figure 1.2.—Simple radial distribution system.

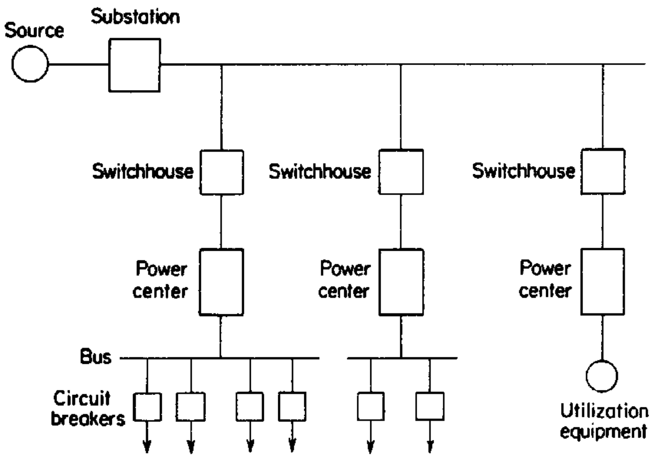

Figure 1.3.—Power-center type of radial distribution.

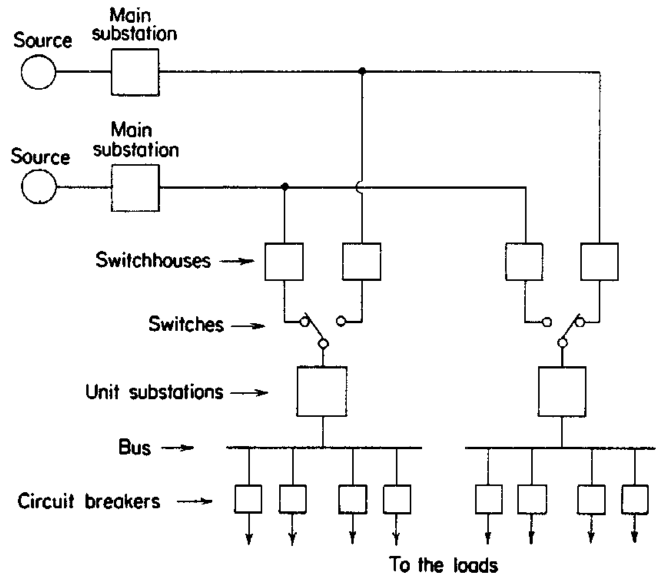

Figure 1.4.—Primary-selective distribution system.

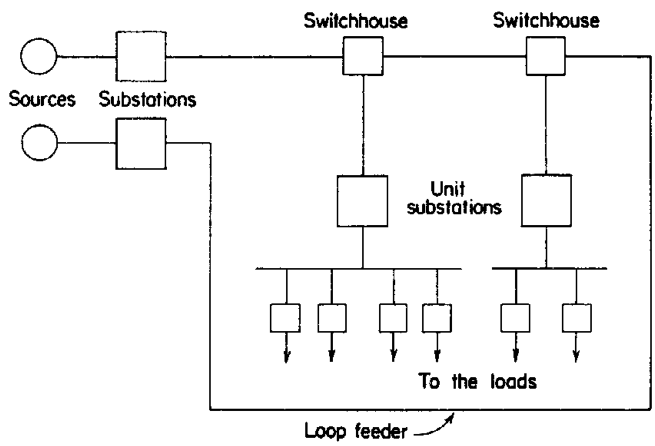

Figure 1.5.—Primary-loop distribution.

will be at the higher voltage. This allows decreased conductor investment, lower electrical losses, and better voltage regulation.

Primary-Selective System

The primary-selective system (fig. 1.4) adds downtime protection through continuity of service. Each substation can receive power by switching from either of two separate primary feeders. Each feeder should have the ability to carry about 80% of the load, so that one feeder can accept a temporary overload (*10*) and provide continued operation if one source should fail. During normal service, each feeder should handle one-half of the load. The system is simple and reliable but costs are somewhat higher than for the radial system because of the duplication of primary equipment.

Primary-Loop System

Though found in some mines, the primary-loop system (fig. 1.5) is not considered good practice. It offers the advantages and disadvantages of the primary-selective system and the cost can be slightly less, but this configuration can result in dangerous conditions when a primary feeder fails. For instance, a failed portion can be energized at either side, creating an extreme hazard to maintenance personnel.

Secondary-Selective System

In a secondary-selective system, a pair of substation secondaries are connected through a normally open tie circuit breaker as shown in figure 1.6. The arrangement allows greater reliability and flexibility than do the preceding techniques. Normally, the distribution is radial from either substation. If a primary feeder or substation fails, the bad circuit can be removed from service and the tie breaker closed either manually or automatically. Maintenance and repair of either primary circuit is possible without creating a power outage, by shedding nonessential loads for the period of reduced-capacity operation. Other methods that can be used to provide continuity of service include oversizing both substations so that one can carry the total load, providing forced-air cooling to the substation in service for the emergency period, or using the temporary overload capacity of the substation and accepting the loss of component life (*6*). Economics often justify this double-ended arrangement if substation requirements are above 5,000 kVA. Note that the substation capacity or ability to transform power is rated in kilovoltamperes.

Secondary-Spot Network

In the secondary-spot network, two or more distribution transformers are supplied by separate primary-distribution feeders as illustrated in figure 1.7. The secondaries are tied together through special circuit breakers, called network protectors, to a secondary bus. Radial secondary feeders are tapped to the bus and feed the loads. This arrangement creates the most reliable distribution system available for industrial plants. If a failure occurs in one distribution transformer or primary circuit, perhaps by acting as a load to the bus, its network protector can quickly sense the reverse power flow and immediately open the circuit. Total power interruption can occur only with simultaneous mishaps in all primary circuits or a secondary-bus failure. However, this type of system is expensive, and the reliability gain is not warranted for the majority of mining applications.

It may be obvious that these basic distribution techniques can be combined into hybrid systems. When this is done, there can be confusion about what is primary or secondary. Ordinarily, the subsystems are defined by the specification of the substations. This will be demonstrated in the next two sections.

UTILITY COMPANY POWER

As utility companies are the principal power source for mines, an understanding of utility system power transmission and distribution is important. Often this system greatly affects the power available to the mine, including voltage regulation, system capacity during power failures in the mine, and overvoltage occurrences.

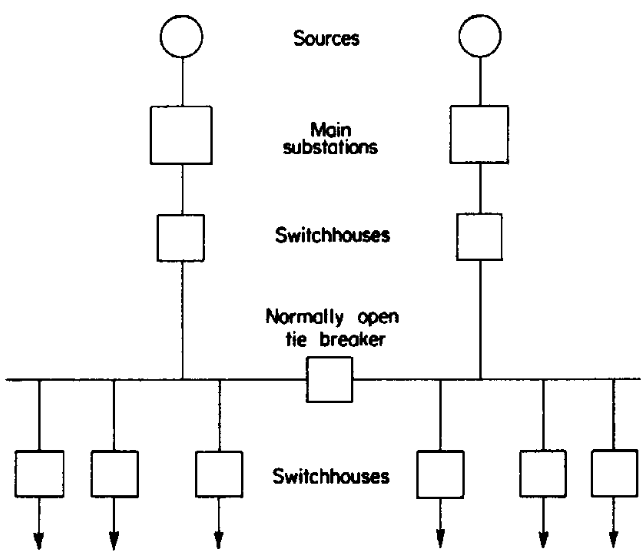

Figure 1.6.—Secondary-selective system.

In a nearby substation, power from a generating station is transformed up to a transmission voltage, commonly 69,000 V or more (*6*). This power is carried on transmission lines to major load areas, either supplying large industrial users directly or powering the utility's own distribution substations. Distribution substations step the voltage down, this time to a primary-distribution level ranging from 4,160 to 34,500 V, but most often at 12,470 or 13,200 V (*6*). These stages are illustrated in figure 1.8.

The utility service, therefore, can be any of the following standard values, in kilovolts: 138, 115, 69, 46, 34.5, 23, 13.8, 12.47, 6.9, 4,800, 4.16, and 2.4 (*13*). Generally, the delivered voltage ranges from 23 to 138 kV, but other values such as 480, 2,300, and 7,200 V are also found. What is available to the mine depends on whether the possible connection is to the power company transmission system, a primary-distribution system, or a distribution transformer.

It is the responsibility of the mining company to select that voltage best suited to its needs. Primarily, the choice depends on the amount of power purchased. It is not safe to assume that the power company has the capability to serve a large mine complex from existing primary-distribution lines or even from the transmission system. The problem stems from the fluctuating nature of mine loads. For example, large excavators in surface mines can require high peak power for a short time, followed by regenerative peak power, cycling within the span of 45 s. The fluctuating load may create voltage and frequency variations beyond the limit set for other utility customers. Accordingly,

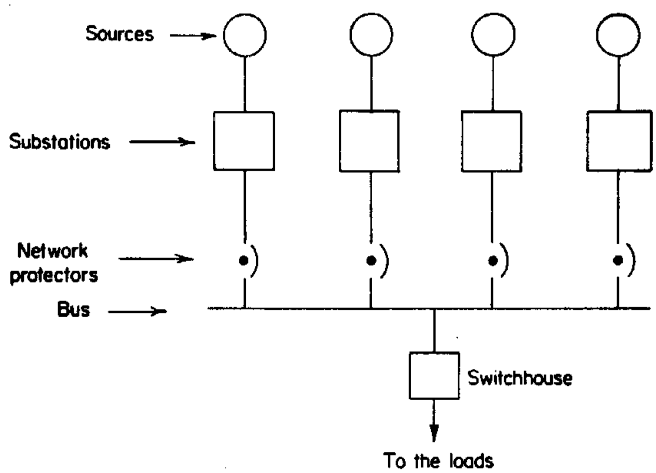

Figure 1.7.—Secondary-spot network technique.

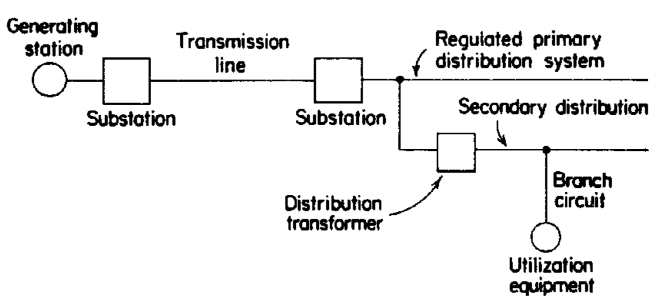

Figure 1.8.—Representative utility transmission and distribution.

most large draglines and shovels require power from 69- to 138-kV transmission systems to get adequate operational capacity, and the construction of several miles of transmission equipment can result in a sizable cost for the mine budget.

Regardless of where the main mine substation is tied into the utility complex and who owns that equipment, its outgoing circuits will here be termed the *mine primary-distribution system* or just *distribution*. The incoming power will most often be referred to as a *transmission system*.

The following sections identify the main types of mines in the United States, classify the major equipment employed, and describe the power-distribution arrangements that are found in them. Of necessity this can be only a very brief overview, but it is designed to indicate the problems and complexities that can arise in mine power distribution and utilization. Individual topics mentioned here are expanded in detail in later chapters.

SURFACE MINING

Surface mining methods are selected over underground methods when the overburden, the earth above the coal seam, can be removed economically to expose the coal. Productivity, safety, and economics usually favor surface mining of seams less than 150 ft deep. Surface coal mining consists of four basic operations: overburden removal to expose the coal, coal loading, haulage, and reclamation. The mining method is generally classified according to such physical characteristics as topography or land contour, overburden thickness, coal thickness, number of coal seams, type of overburden, fragmentation characteristics of the overburden, climate, and hydrology. The mining method is also affected by Federal and State requirements. The mining method selected must protect the health and safety of the workers and minimize environmental disturbance and be designed for the specific set of prevailing physical conditions. The major surface mining methods for coal are contour mining and area mining.

Contour mining methods are commonly used in rolling or mountainous terrain; they are called contour mining because overburden removal progresses around the hillside at the coal seam horizon such that the pit resembles a contour line. There are many varieties of contour mining, but in all methods overburden is fragmented by drilling and blasting, and removed to expose the coal seam. The overburden may be removed by small diesel or electric draglines, or by diesel-powered front-end loaders and trucks. In soft overburden and for topsoil removal, scrapers and bulldozers may be used.

Area mining is the predominant stripping method in more level terrain. As its name implies, area mining can cover an extensive region, using various box-cut or strip pit and benching techniques. It may be used to mine both thick and thin seams, or multiple seams; where these seams are dipping, area mining is modified to approximate the open pit methods common in metal mining. In all cases, overburden handling and reclamation are an integral part of the process. Equipment varies, depending on the scale of the operation, from small draglines and dozers to massive equipment that has more than 30,000 connected horsepower.

In general, the magnitude of electrical distribution and utilization is greater in area mining than in contour mining. Combination of equipment employed in large multiseam operations may include tandem draglines, dragline and shovel, pan scrapers with attendant dozers, and dragline and bucket-wheel excavators. Bucket-wheel excavators can be very effective where overburden is soft and does not require drilling and blasting. Front-end loaders, electric and diesel shovels, ripping dozers, and tracked highlifts can all be combined with truck haulage for coal removal.

POWER SYSTEMS IN SURFACE MINES

Mine power systems can be divided into three categories, depending upon the purpose of a specific portion:

1. Subtransmission,
2. Primary distribution (or distribution), and
3. Secondary distribution (or utilization).

Often, if a subtransmission system is needed, it will have the same general arrangement in any mine. At distribution and utilization, power-system installations can vary greatly, but in some mines distribution and utilization can be the same system. Electrical installations in surface coal mines are regulated under 30 CFR 77 (*14*).

Main Substations and Subtransmission

Main substations may range from 500-kVA capacity, supplying 480 V for only pumps and conveyors, to 50,000 kVA, servicing a large strip-mining operation and preparation plant (*10*). The substation location is usually an economic compromise between the cost of running transmission lines and power losses in primary distribution. From the main substation, power is distributed to the various centers of load in the operation. However, individual loads or complexes, such as preparation plants and other surface facilities, may have large power requirements or be so isolated that primary-distribution operation is not practical. In these cases, or for safety reasons, incoming utility transmission should be extended close to the load. The extension is designated a *subtransmission system*, and the conductors are usually suspended as overhead lines (*13*).

As shown in figure 1.9, subtransmission commonly requires a primary switchyard of high-voltage switching apparatus for power tapping. Branch circuits are fed through

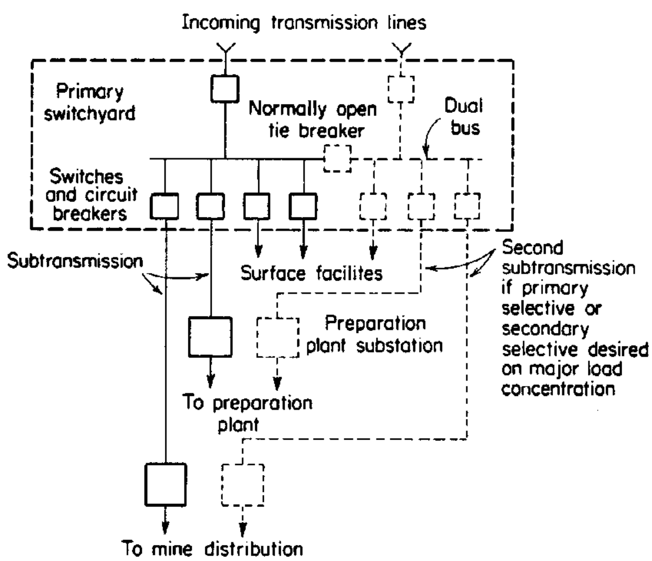

Figure 1.9.—Subtransmission for surface mine.

circuit breakers to protect the subtransmission line and the utility's system. Dual-bus configurations are employed if primary-selective or secondary-selective distribution is desired on major load concentrations to provide high reliability. This additional subtransmission circuitry is illustrated in figure 1.9 by dashed lines.

Subtransmission circuits, primary switchyards, and main substations are almost always permanent installations located in areas unaffected by the mining operation. The main substation is where the grounding system for the mine is established. This ground is carried along the powerlines through overhead conductors or in cables and is connected to the frames of all mobile mining equipment.

Surface Mine Distribution

Mine power distribution, in its simplest radial form, has already been shown to consist of a substation, distribution, and a power center feeding the mining equipment. The arrangement is very common in small surface operations where the distribution voltage is commonly 4,160 V but can be 2,300 V in older equipment. In the smallest mines, power is purchased at low-voltage utilization (often 480 V) and fed to a distribution box to which motors and equipment are connected. At times, simple radial distribution is employed in large surface mines where only one machine must be served or an extensive primary-distribution network cannot be established, as in some contour operations.

The great majority of strip mines employ radial distribution, but secondary-selective and primary-loop designs can also be found. Simplified examples of the three systems are provided in figure 1.10 to 1.12. In all configurations, a portion of the primary distribution is established at a base line or bus. The base line is usually located on the highwall, paralleling the pit for the entire length of the cut. Its location is typically maintained 1,500 ft ahead of the pit, and it is moved as the pit advances (*3*). Distribution continues from the base line to the mining equipment, with the connections maintained at regular intervals. As the machines move along the pit, the base-line connections are changed to another convenient location.

The base line can consist of overhead polelines or a cable-switchhouse configuration, figures 1.13 and 1.14 (*3*). It can be seen that cable distribution brings power into the pit area, where shielded trailing cables connect to the machines. The overhead poleline plus cable arrangement is common in older mining operations, especially when utilization is at 7,200 V or less (*3*). Typical spacing between poles, or line span, is 200 ft. Drop points are noted in figure 1.13 by triangles. These are terminations between the overhead conductors and the cables, mounted about 8 ft above the ground on poles spaced at regular intervals of around 1,000 to 1,500 ft.

Cables connected to the drop points deliver power to skid-mounted switchhouses located on the highwall or in the pit. The switchhouses may contain manual disconnect switches, which are commonly termed *switch skids* or *disconnect skids*, automatic circuit-protection devices or breaker skids, or a combination of both. The skids can either be boat design with flat bottoms or have fabricated runners, depending upon the allowable bearing pressure of the mine terrain. Couplers or plug-receptacle pairs are commonly used for both feeder and trailing-cable connections. Disconnect and circuit-protection functions are required for each distribution load, and double switchhouses (two-breaker skids) are frequently employed for two loads. Unit substations often contain internal circuit protection on the incoming side, and thus do not require a breaker skid.

Trailing cables are usually 1,000 ft in length, although lengths to 2,000 ft can be found. When longer cables are necessary to reach a breaker skid, in-line coupling systems can be used, and these are commonly mounted on small skids for easy movement. Trailing-cable handling for stripping equipment is often assisted by cable reels mounted on skids or self-propelled carriers. Large excavators can require the self-propelled variety.

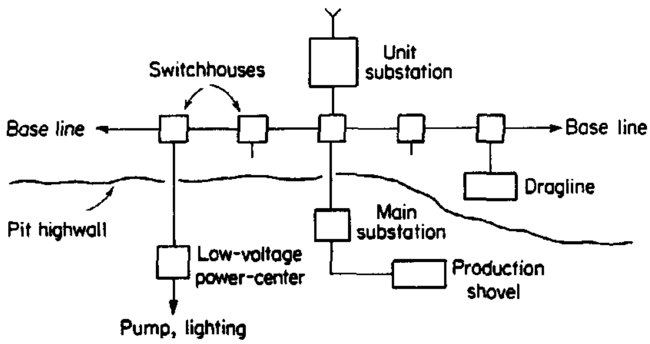

Figure 1.10.—Radial strip mine distribution system.

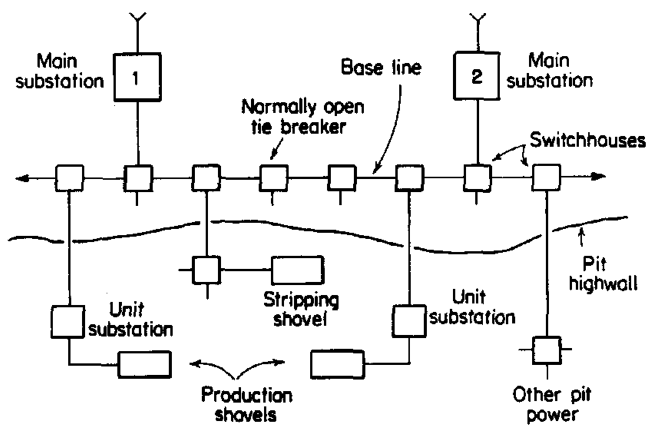

Figure 1.11.—Secondary-selective distribution in strip mining.

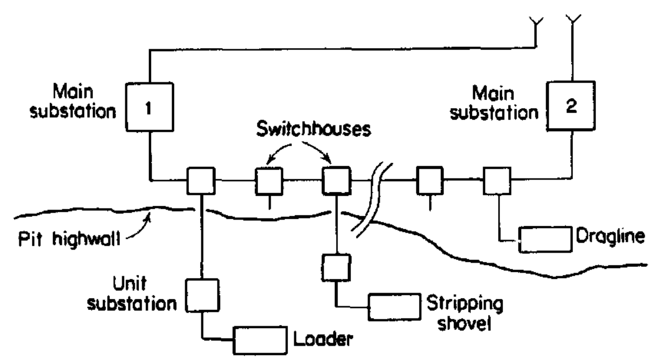

Figure 1.12.—Primary-loop design for strip mining.

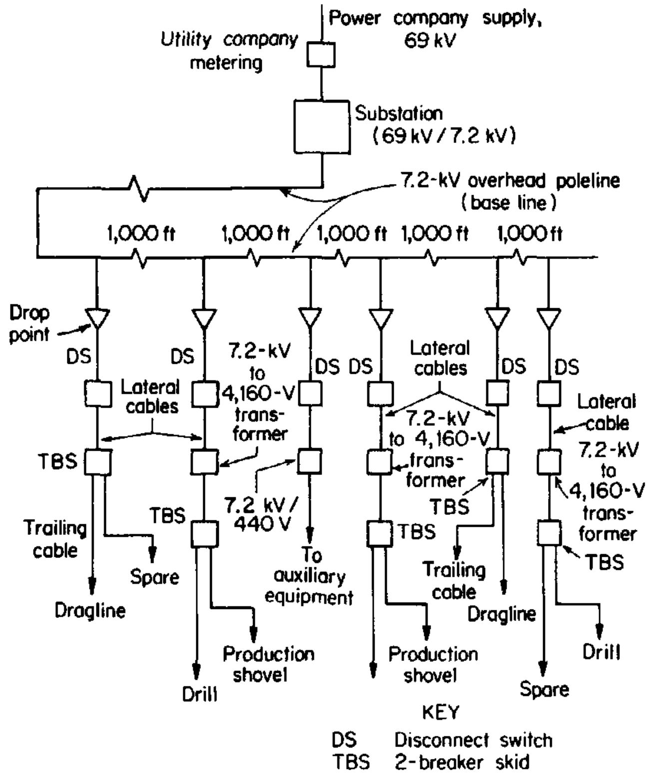

Figure 1.13.—Radial distribution for strip mine with overhead-poleline base line.

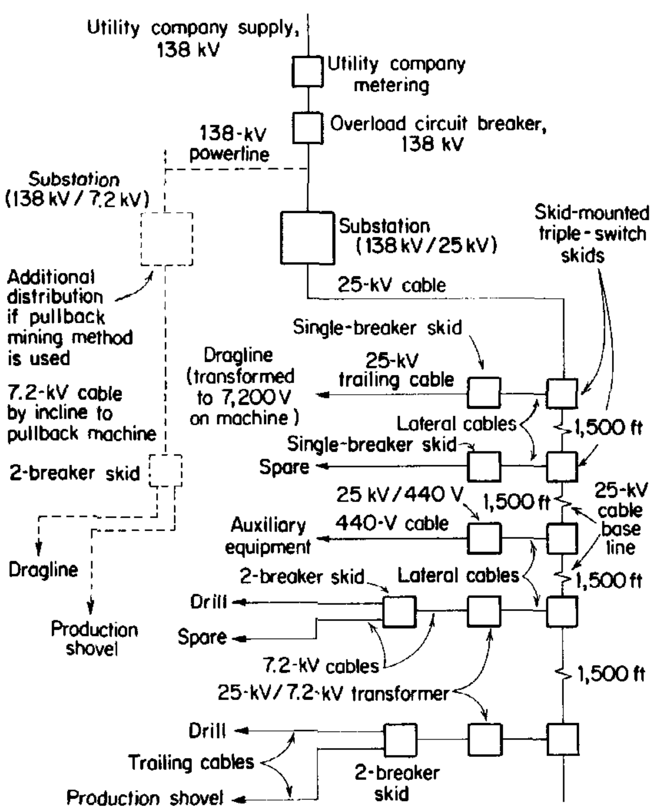

Figure 1.14.—Radial distribution for strip mine with all-cable distribution.

The layout for an all-cable mine distribution, figure 1.14, is very similar to that just described. In this case, however, the base line is assembled using cable-interconnected switchhouses. As noted in the illustration, the common approach is to use disconnect skids with three internal switches in the base line and to have separate breaker skids in line with the cables feeding the machinery. Another approach is to combine the single-breaker skids into the baseline switchhouses.

When a secondary-selective configuration is used, as shown in figure 1.11, a normally open tie circuit breaker is placed in the base line in a location approximately equidistant from the main substations. In some operations, the two substations and the tie circuit breaker may be in the same location, with two feeders running from the substation area to the base line. More than two main substations may be established in very large operations.

Primary-loop systems have occasionally been used in strip mining. It can be noted from figure 1.12 that the substations actually operate in parallel, considering the base line to be a bus. Here the substations can be smaller than those needed for a radial system. Notwithstanding, certain precautions should be taken with this configuration (*9*). For example, the substations must be identical if they are to share the load, but as an unbalanced load distribution is probable on any system, it is likely that the two substations will not be equally loaded. Regardless, because of the safety hazards, primary-loop distribution is considered unsatisfactory and is not recommended.

Distribution voltage for the surface mine may be 7.2, 13, or 23 kV, and to a lesser extent 4.16 kV. Regardless of the level, drills and production shovels usually operate at 7,200 or 4,160 V. Therefore, when higher distribution levels are needed, portable unit substations are commonly used in the pit. One instance would be when the load created by a large machine is several times that for auxiliary machines. Another method is to establish two base lines on the highwall for two distribution voltages, as shown in figure 1.15. Here, a large unit substation interconnects the two base lines. Even in this situation, as can be seen in the preceding illustrations, low-voltage unit substations

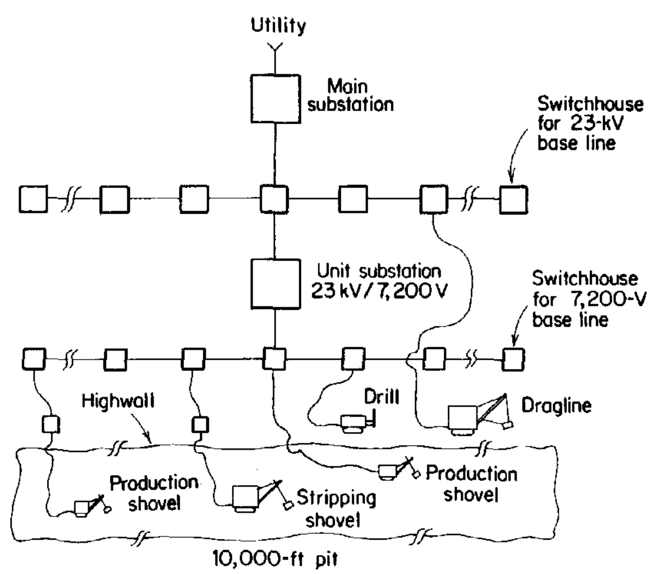

Figure 1.15.—Surface mine distribution system using two base lines.

or power centers are often required for 480-V auxiliary equipment.

The primary purpose of any primary-distribution scheme in a surface mine is to provide a flexible, easily moved or modified power source for the highly mobile mining equipment. System designs must also be considered as an integral part of the total mine operation. Those described have these objectives in mind. As will be seen, the distribution system in any surface or underground mine that serves portable equipment is subject to damage from the mining machinery itself, and as a result, the system must be designed with optimum flexibility and consideration for personnel safety.

Open pit power systems are quite similar to those in stripping mines but with one main exception: primary distribution typically establishes a ring bus or main that partially or completely encloses the pit. Radial ties to the bus complete the circuit to switchhouses located in the pit, and portable equipment again uses shielded trailing cables. An example is shown in figure 1.16. Distribution voltage is normally 4.16 kV, but 7.2 or 6.9 and 13.8 kV are sometimes used. Unit substations are employed if equipment voltages are lower. Primary distribution is almost invariably through overhead lines.

UNDERGROUND COAL MINING

Figure 1.17 is a plan view of a typical U.S. underground coal mine. A system of main entries, each 16 to 20 ft in width, is developed from the coal seam access point to the production areas, which are called panels or sections. Pillars of coal are left during mining to support the overburden above the entries. Crosscuts are mined between the entries. The main entries may remain standing for several years while coal is being extracted from the panels. Haulage of personnel, supplies, and coal, together with provision of ventilating air and dust-suppression water, and electrical distribution are necessary functions of the main entries throughout the life of the mine.

The mining method is defined by the configuration of the open workings and by the classification of equipment used. The important underground coal mining methods are room and pillar, which may be either conventional or continuous, and longwall. To the miner, the type of mining machinery used is implied by each category. The room-and-pillar method remains dominant in the United States, although there has been a recent substantial increase in longwall mining. The choice of a specific mining method is frequently dictated by such natural conditions of the mine as the characteristics of the overburden, roof, and floor, plus the seam dip, water, methane, and seam height (*11*). Essentially, the method and equipment selected are based on the combination that will provide the safest and most profitable extraction within the given set of geologic conditions, while complying with State and Federal health, safety, and environmental regulations.

Room-and-Pillar Mining

Room-and-pillar mining is named for the regular pattern of openings made in the coal seam and was the earliest form of underground coal workings.

Conventional Mining

The conventional mining method represents a direct evolutionary link with the early mining techniques. It is based on the original loading machine, which came into use

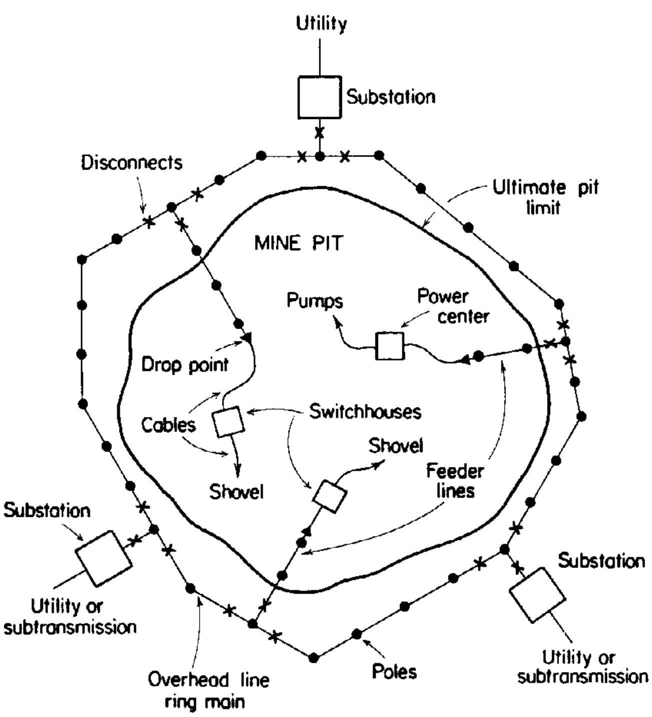

Figure 1.16.—Open pit power system.

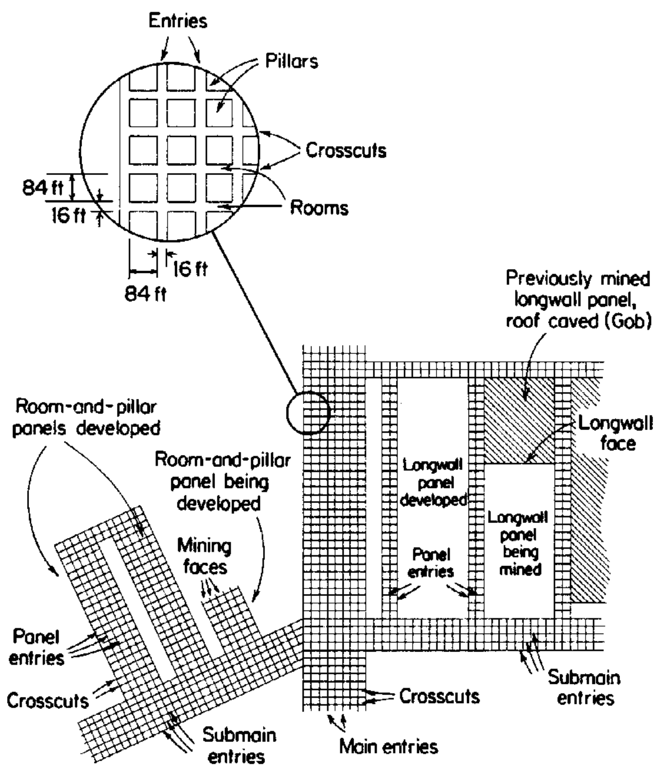

Figure 1.17.—Layout of underground coal mine.

in the early 1920's. Modern conventional mining consists of six distinct operations:

1. Undercutting the coal face,
2. Drilling holes in the face for blasting,
3. Blasting,
4. Loading the broken coal onto a face haulage system,
5. Hauling the coal from the face area to a subsequent haulage system, and
6. Providing roof support.

In order, these steps comprise a mining cycle; after roof support is completed, work begins again at step 1. Ventilation, although essential, is not included as a separate step in the cycle as it must be provided continuously. Other safety procedures include careful examination of the face and roof after blasting and before each job begins at the face.

Mobile self-propelled mining equipment performs most of the operations in conventional mining. The cutting machine, basically an oversized chain saw, is employed to cut a slot at floor level, called the kerf, which allows coal expansion during blasting. A self-propelled face drill follows the cutter and makes several holes in the face with its carbide-tipped auger-type drill bits. Blasting is carried out either by chemical explosives approved as permissible by the U.S. Mine Safety and Health Administration (MSHA) or to a lesser extent by high-pressure air. Permissible explosives will not ignite methane and coal dust when used correctly.

A crawler-mounted loading machine loads the broken coal onto the face haulage vehicle, typically a shuttle car. The car is equippped with a chain conveyor that moves the coal from the load end to the discharge end and subsequently unloads it from the vehicle. Shuttle cars almost invariably work in pairs and move the coal to rail cars or a conveyor belt, which makes up the next stage of materials handling in the mine. The roof bolter, sometimes called a roof drill, is a rubber-tired vehicle that secures the roof by first drilling vertical holes and then emplacing roof bolts, which secure the roof either by clamping thin roof layers together to form a thick beam, or by hanging weak strata to a more competent upper layer. Drilling is usually accomplished by rotary action with auger-type bits. The resulting dust is collected through the bit and hollow drill rod by vacuum.

With few exceptions, all these machines are electrically driven, powered via trailing cables from the mine power system. Since the mining equipment is continually moved among several faces in a coordinated plan designed for maximum production efficiency, the handling of trailing cables is a significant part of the mining cycle. The result of coal removal is a system of open rooms divided by coal pillars that support the roof as mining advances toward the property boundaries. When the equipment approaches the property limit, the operation is turned around and retreat mining takes place. If surface subsidence is permitted, the pillars are removed in an organized extraction plan and the roof is allowed to cave. The broken material that then fills the mined void is known as gob.

Continuous Mining

The heart of the continuous coal mining method is the continuous mining machine, which replaces the conventional room-and-pillar unit operations of cutting, drilling, blasting, and loading. The mining functions of haulage and roof support remain, although some continuous miners also perform roof bolting. The term "continuous" is actually a misnomer because of legal constraints that mandate interruptions in the mining process for safety checks and ventilation requirements.

The most common form of face haulage in continuous mining is again the shuttle car. One of the main problems associated with continuous mining is the intermittent nature of the shuttle car haulage system, which causes frequent delays at the face. As a result, various types of continuous haulage systems have been developed to alleviate this problem. Mobile chain and belt conveyors are the most popular of these systems, and these are applicable to mining low coal. Continuous haulage systems have not been without problems, and some designs have been hampered by poor reliability and lack of maneuverability. Hydraulic systems have shown great promise; they operate by pulverizing and slurrying the coal immediately behind the miner, then pumping the slurry to the surface.

Longwall Mining

Longwall mining is the most popular underground coal mining technique in Europe, and it is growing rapidly in the United States. In contrast to room-and-pillar mining, longwall is capital intensive rather than labor intensive. Longwalls are usually 300 to 600 ft wide, and the direction of mining with respect to the main entries classifies them as either advancing or retreating longwalls. The latter is the most frequent in the United States.

A typical retreating longwall is shown in figure 1.18. The section of coal to be mined, the longwall panel, is first delineated by two room-and-pillar entries or headings driven perpendicular to the main entry. These two headings, the headgate and the tailgate, handle haulage equipment and ventilation. The longwall panel is then mined back and forth, retreating toward the main entry. The roof is allowed to cave immediately as the longwall equipment moves, as is shown by the gob area on the diagram.

The longwall equipment consists of an interconnected system of cutting machine, roof support equipment, and

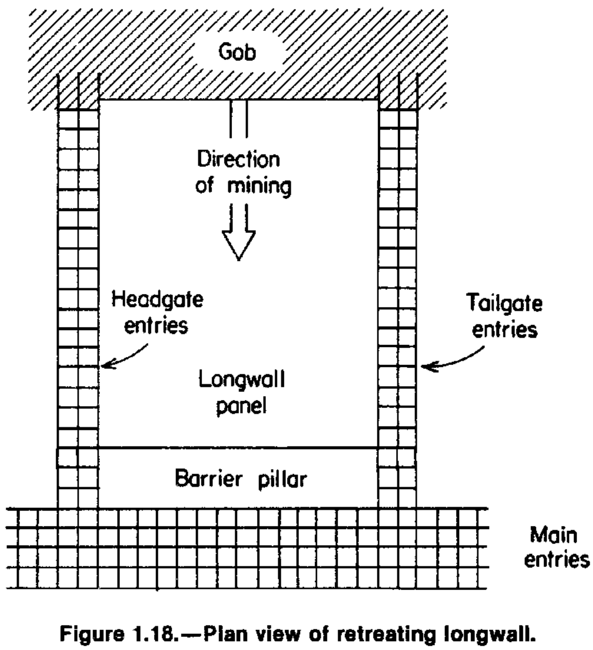

Figure 1.18.—Plan view of retreating longwall.

conveyor haulage. The cutting machine moves along the face on a conveyor that also carries away the mined coal. Behind the face conveyor, and connected to it, is the roof support equipment, which supports a protective metal canopy or shield that extends over the face area. These roof support units provide both the protection and the forward mobility of the system.

The typical face conveyor is a flexible armored-chain conveyor powered by motors at the headgate and tailgate. Mined material moves toward the headgate, where it discharges to the panel belt via an elevated intermediate haulage unit, the stage loader.

Shortwall mining is a less common mining method; it is very similar to longwall mining except that the shortwall panel is normally 150 to 200 ft wide. From the standpoint of equipment, shortwall can be considered as a compromise between room and pillar and longwall in that the extractive and face haulage systems are identical to those in continuous mining, while the roof support equipment is similar to that used in longwall mining.

POWER SYSTEMS IN UNDERGROUND MINES

Regulations

Underground mine power systems have different characteristics from those for surface mines, and these two basic mining operations are regulated by separate codes and standards. For instance, although 30 CFR 77 covers electrical installations of surface coal mines and surface facilities of underground coal mines, Part 75 regulates the underground installations and Part 18 specifies standards for electrically powered face machinery (*14*). Part 77 illustrates an overlap between surface and underground legal demands, which is logical because the surface electrical counterparts of both mine types are similar; examples include substations and subtransmission. Figure 1.19 can be compared with figure 1.9 to see the similarity between surface mine and underground mine subtransmission. As a general situation, however, the mine distribution system is related to the mining method. Hence, underground mine systems become different from surface mines at the point where the circuits leave the substation and go underground.

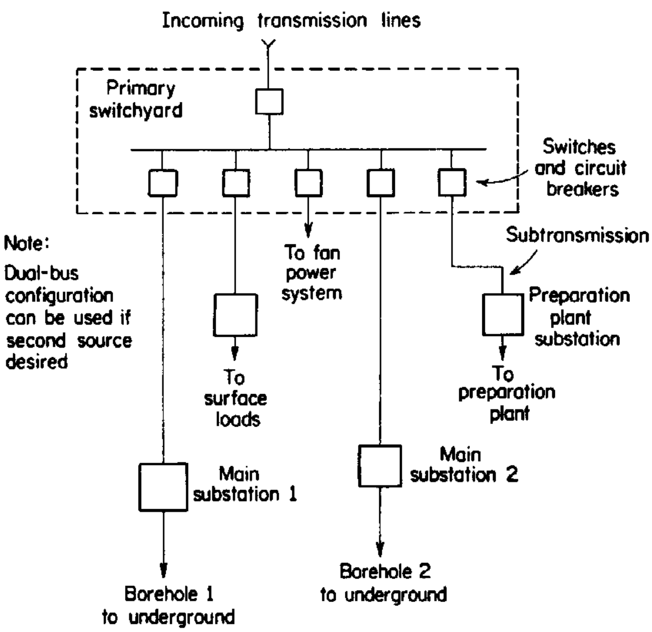

Figure 1.19.—Subtransmission for underground mine.

Underground Mine Distribution

As shown in figures 1.20 and 1.21, underground mine power systems are somewhat more complicated than those for surface applications. Because of the nature of the mine and its service requirements, distribution must almost always be radial (fig. 1.20); the freedom in routing distribution enjoyed by surface mines is not available underground. For increased reliability, secondary-selective main substations are employed (fig. 1.21). The secondary-selective operation is defined by the use of two substations and mine feeders with a normally open tie breaker. Primary-distribution voltage is most commonly 7,200 V; however, older 4,160-V systems can still be found, and 12,470 V has increased in popularity in recent years, especially for longwall operations. The grounding system for the underground mine distribution must be separated from that used for surface equipment.

Power and mine grounding are fed underground in insulated cables, either through a shaft or borehole or a fresh-air entry. The cables terminate in disconnect switches within 500 ft of the point of power entry into the coal seam. These switches allow total removal of underground power in an emergency. From the disconnects, which may be a part of a switchhouse, the power is distributed through cables to power centers or rectifiers located as close to the machinery as practical. All the cables on high-voltage circuits, usually involving only distribution, must have shielding around each power conductor.

The prime load concentrations in underground mining are created by the mining sections. Distribution terminates at the section power center, which is a transformer combined with a utilization bus and protective circuitry. From this, several face machines are powered through couplers and trailing cables. Power-system segments for typical continuous and longwall operations are given in figures 1.22, 1.23, and 1.24. Rated machine voltage for most installations is 550 Vac, but 250-Vdc and 440-Vac equipment is used extensively, and 950 Vac has become quite popular for high-horsepower continuous miners and longwall shearing machines. In the longwall system, power is fed through controls to the various motors. On conventional or continuous equipment, the utilization approximates the radial system shown in figure 1.22.

If belt haulage is used, distribution transformers are located close to all major conveyor belt drives and are referred to as belt transformers. After transformation, power is supplied through starter circuitry to the drive motors. With rail haulage, distribution terminates at rectifiers that contain a transformer and rectifier combination. The rectifiers are located in an entry or crosscut just off the railway. As shown in figure 1.25, dc power is then supplied through circuit breakers to an overhead conductor or trolley wire and the rail, with additional rectifiers located at regular intervals from 2,000 to 5,000 ft along the rail system. For further protection, the trolley wire is divided into electrically isolated segments. The typical rectifier supplies the ends of two segments of trolley wire and each feeder has its own protective circuitry to detect malfunctions. Each trolley-wire segment is called a dead block. This loop-feeding arrangement is continued throughout most of the haulage system except for the most inby segment, which is dead-ended. In some mines, dc face equipment and small dc motors are powered from the trolley system through a fused connection (or nip) to the trolley conductor and rail. The dc distribution can also serve large motors directly through switchgear; however, this practice is rare in underground coal.

All power equipment used underground must be rugged, portable, self-contained, and specifically designed for installation and operation in limited spaces. In addition, all equipment and the cables connecting them must be protected against any failures that could cause electrical hazard to personnel. This is primarily provided by protective relaying built into each system part, with redundancy to maximize safety (*4*).

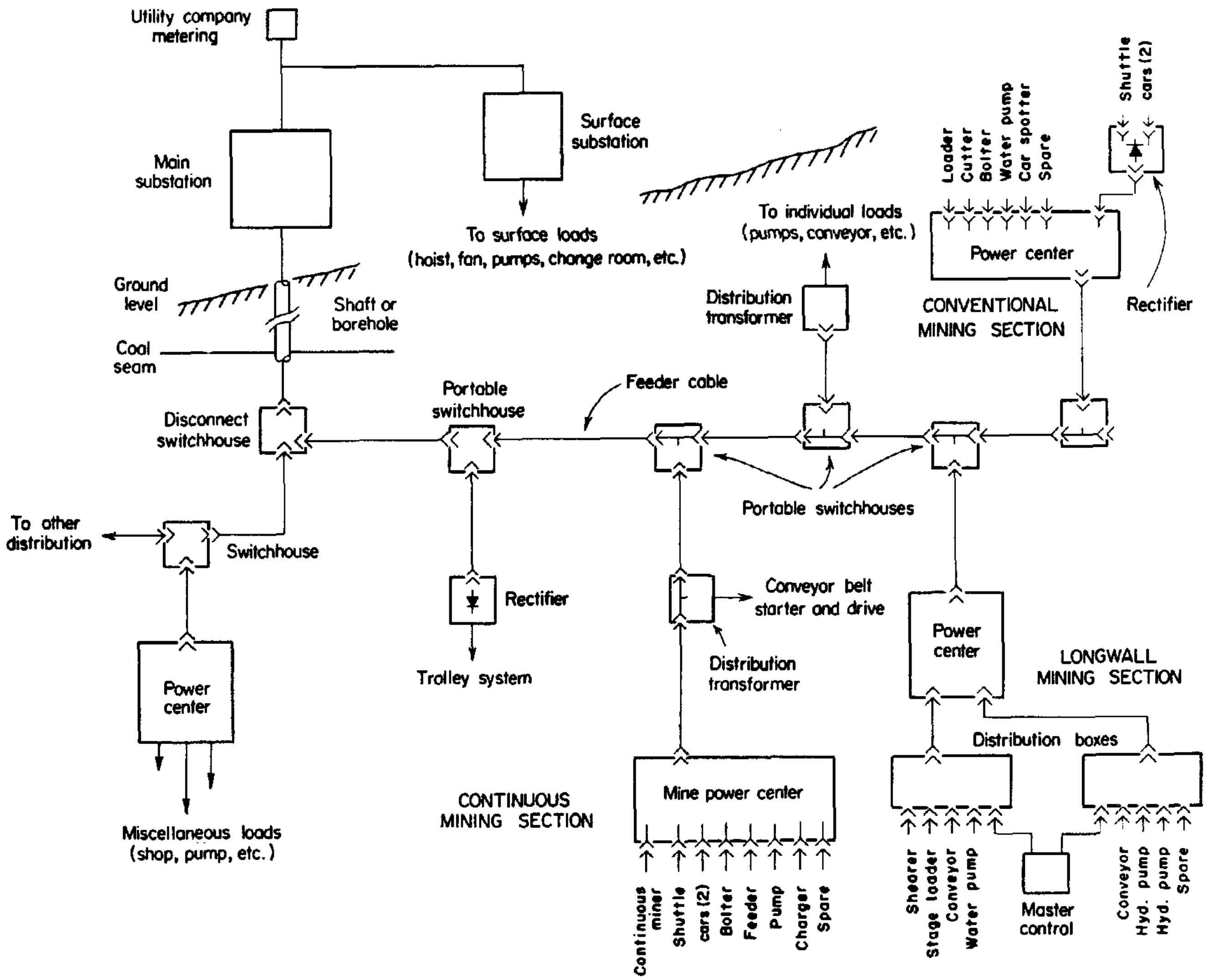

Figure 1.20.—Radially distributed underground power system.

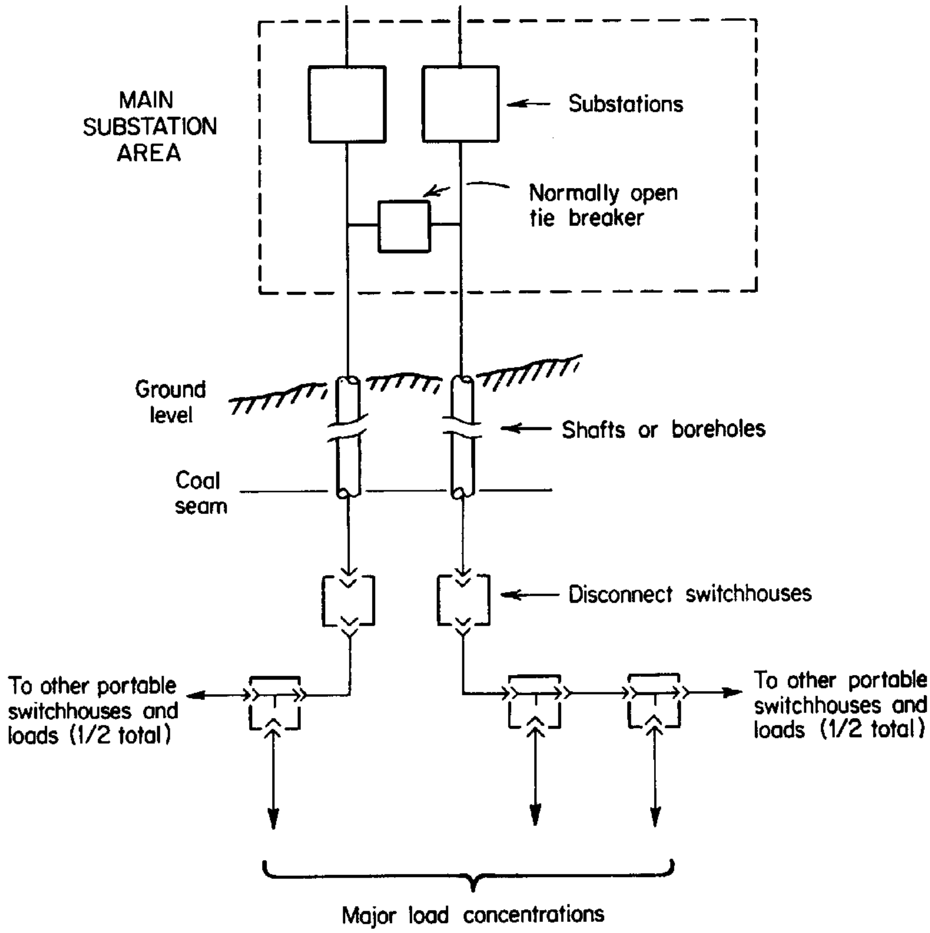

Figure 1.21.—Secondary-selective distribution in underground mines.

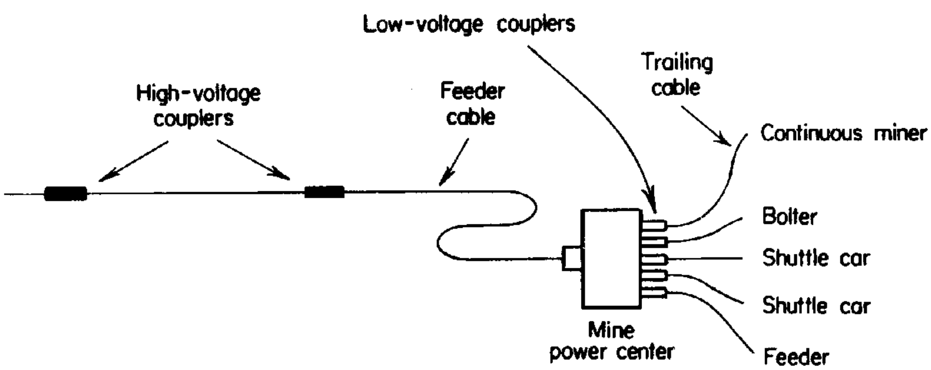

Figure 1.22.—Utilization in continuous mining section.

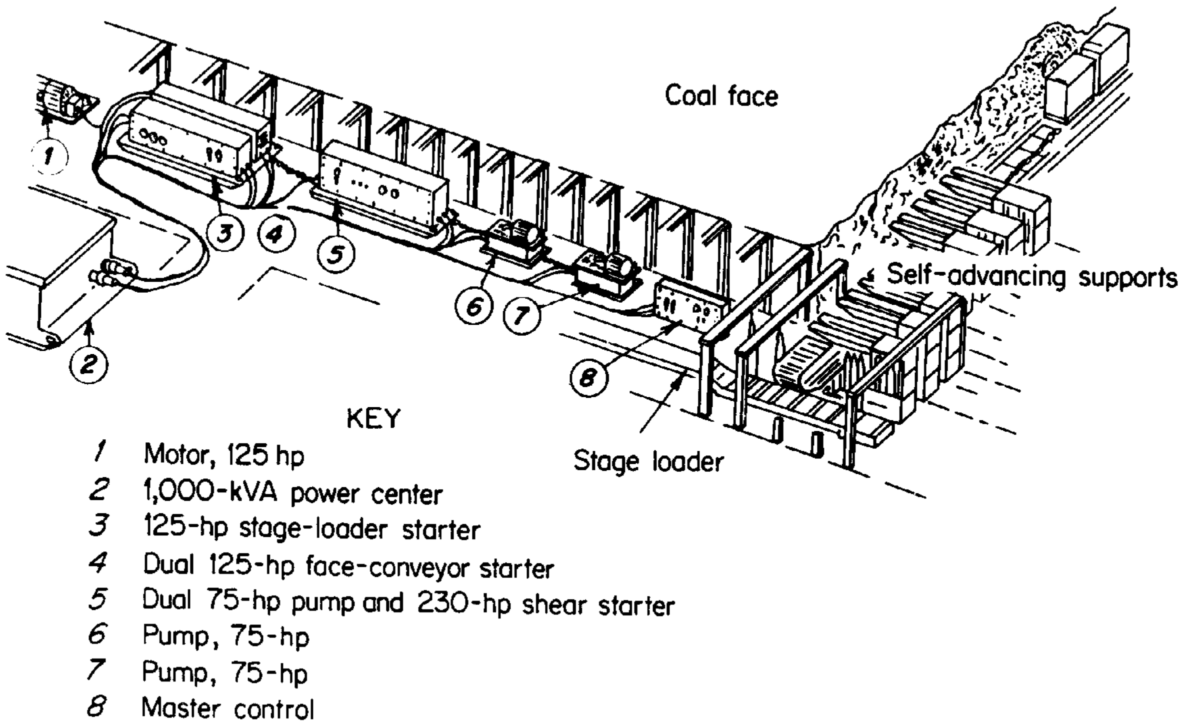

Figure 1.23. – Power-system segment with longwall equipment.

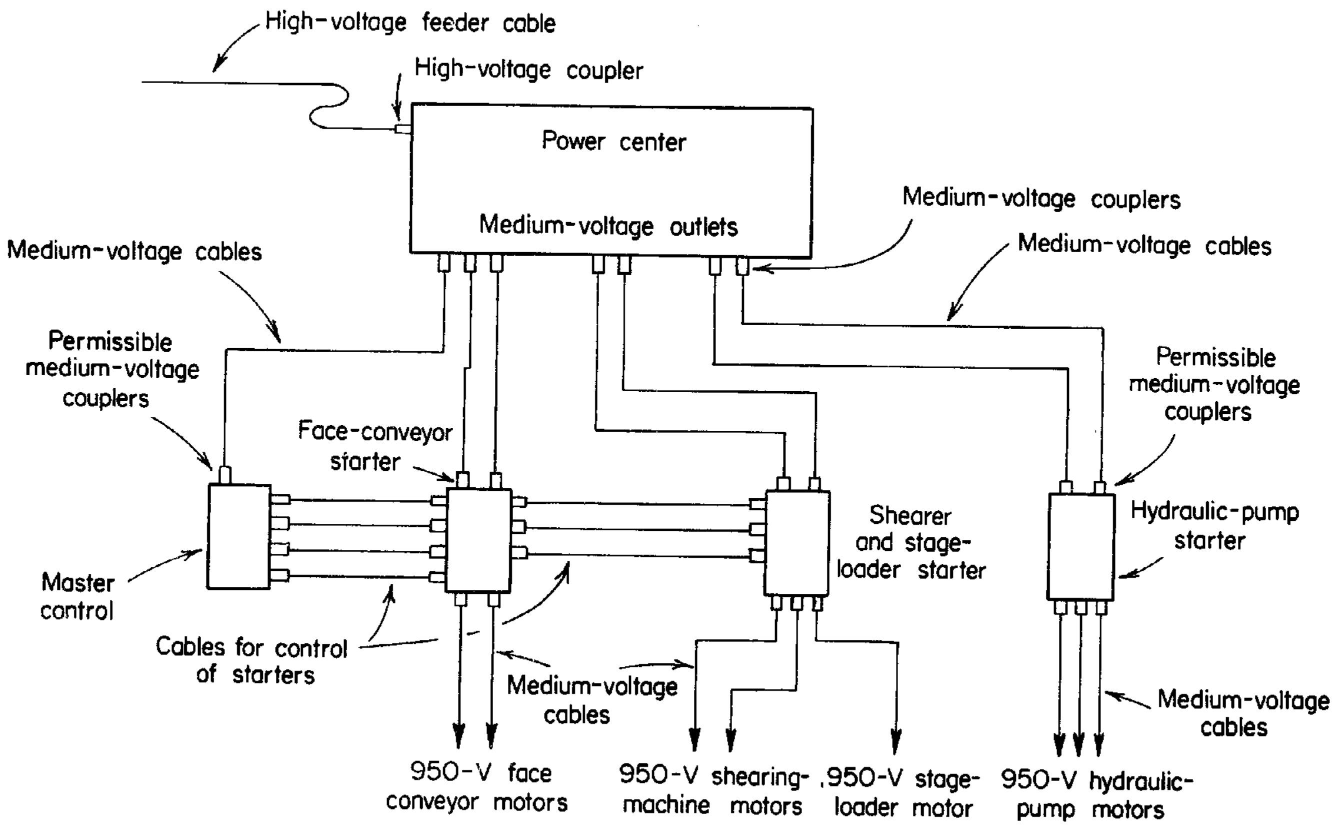

Figure 1.24. – Diagram of electrical-system segment for longwall.

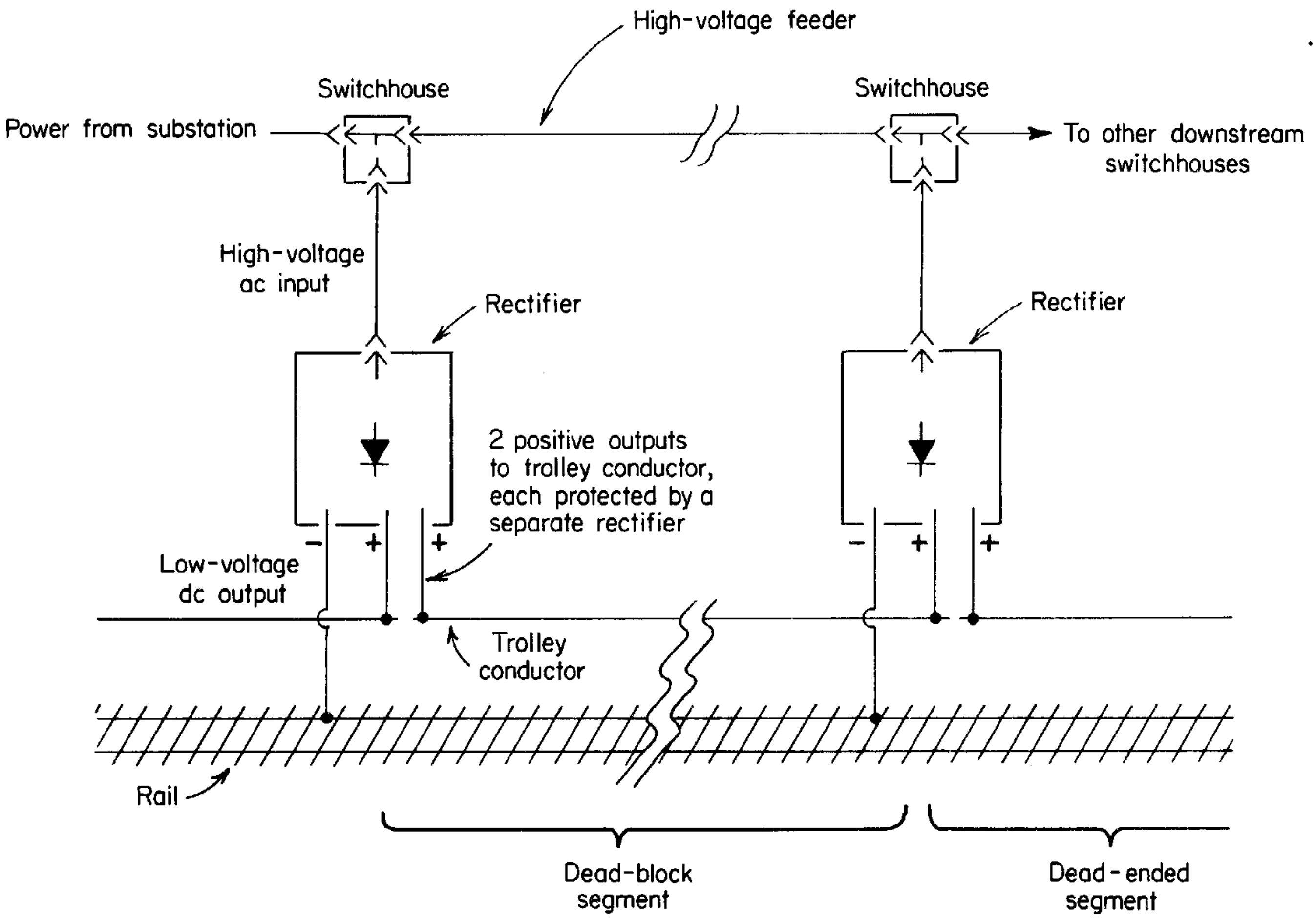

Figure 1.25.—Parallel-feed haulage system.

SURFACE FACILITY POWER REQUIREMENTS

The surface activities of any mine, which may include shops, changing rooms, offices, ventilation fans, hoisting equipment, preparation plants, and so forth, can have large power requirements. For safety, these facilities should at least have an isolated power source and at times a separate substation.

In preparation plants, the distribution arrangements are almost always expanded radial or secondary selective (*8*). Representative system layouts are shown in figures 1.26 and 1.27. In both, distribution is at 2.4 to 13.8 kV, with 4,160 V the most common level. Power is distributed at one of these levels to centers of electric load. This power may be used directly for high-voltage motors, but usually the voltage is stepped down to supply groups of motors or single high-horsepower motors. The power centers must be in an elevated location or totally enclosed. The rooms used for these and other electrical components may also be pressurized to exclude coal dust.

The most popular voltage for preparation plant utilization is 480 V. This voltage is used to drive all motors throughout the plant except those with high-horsepower demands, such as centrifugal dryers and large fan drives, where 2,300 or 4,160 V is commonly employed. These higher voltages may also be preferred for any motor that requires continuous service or independence from the power-center loads. Note that 240-V motors are unsatisfactory for typical preparation plant demands. Most modern preparation plants use group motor control instead of individually housed control units, since this method facilitates maintenance and enables the interlocking of the various motor functions required for semiautomatic facilities. All manual controls, indicating lights, and so on are grouped in one central operating panel to allow easy access and visual indication of plant operation. The panel is often called a motor control center (MCC), as shown in figure 1.27.

BASIC DESIGN CONSIDERATIONS

The goal of the power engineer is to provide an efficient, reliable electrical system at maximum safety and for the lowest possible cost. The types of information made available to the power engineer include the expected size of the mine, the anticipated potential expansion, the types of equipment to be used, the haulage methods to be employed, and whether or not power is available from a utility company. The amount of capital assigned for the electrical system will also be designated.

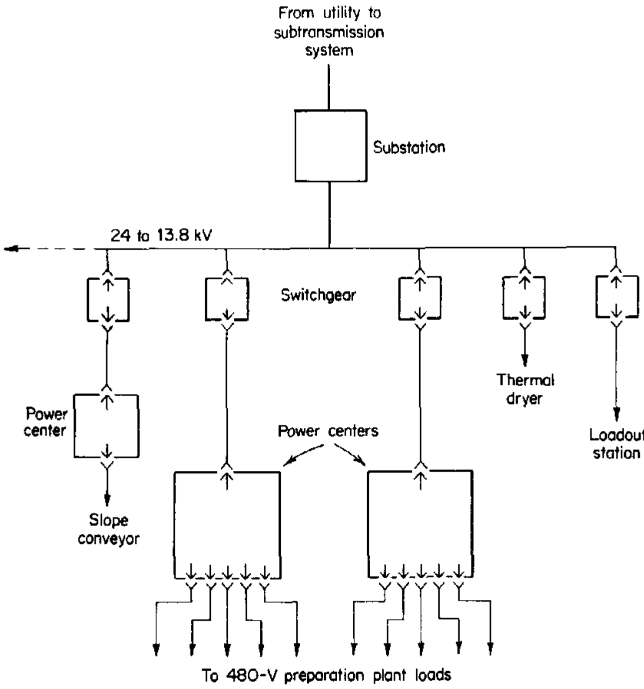

Figure 1.26.—Representative expanded radial distribution for preparation plant.

The designed system must meet certain minimum criteria. IEEE (*12*) has defined these basic criteria for industrial electrical systems that must be applied to mines:

- Safety to personnel and property,
- Reliability of operation,
- Simplicity,
- Maintainability,
- Adequate interrupting ability,
- Current-limiting capacity,
- Selective-system operation,
- Voltage regulation,
- Potential for expansion, and
- First cost.

Of these, safety, reliability, and simplicity are closely related. All are dependent on good preventive maintenance. In the cramped uncompromising environment of an underground mine, these are of vital concern. Since continuous operation is the aim of every mine operator, planned maintenance should be held to a minimum. Most routine maintenance should be capable of being performed by unskilled personnel, since it will be done by the miners themselves. Training for these tasks must be provided.

Adequate interrupting capacity, current-limiting capability, and selective-system operation are projected at safety through reliability. The first two areas ensure protection during a disturbance. Current limiting, when applied to grounding, is perhaps the most significant personnel safety feature of mine electrical systems. Selective-system operation is a design concept that minimizes the effect of system

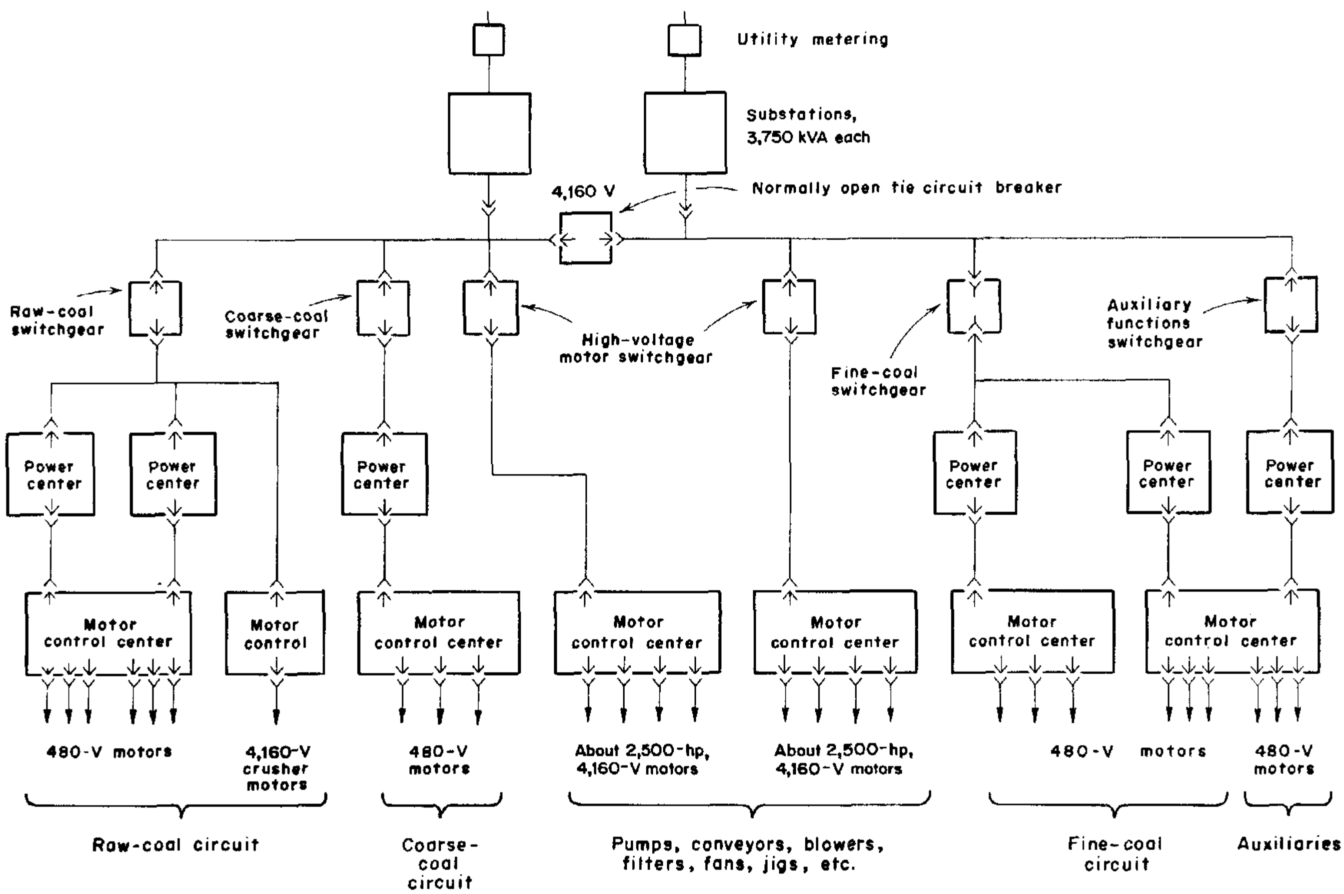

Figure 1.27.—Representative secondary-selective distribution for preparation plant.

disturbances. Voltage regulation is a limiting factor in system design, particularly underground, and is often the main constraint to system expansion. It should be anticipated that when the size of the mine is increased, this might involve augmenting the power-system supply through additional power sources.

While first cost is important, it should never be the determining factor, since high-cost equipment, projected at maximizing safety and reliability, can easily offset the increased first cost through the reduction in operating costs. At times, this fact appears to elude some company purchasing agents.

Using the data available, the task of the power engineer is to select one combination of power equipment over another, provide power or circuit diagrams, estimate the equipment, operating and maintenace costs, set the specifications for the system, and receive and assess the proposals from suppliers. For success, the engineer requires a firm knowledge of mine power systems, but this understanding cannot be based on a "standard mine electrical system" because such a standard does not exist: no two mines are exactly alike. The engineer must resort to the fundamental concepts, an awareness of what has worked in the past, and a clear understanding of the legal constraints. This information is provided in the subsequent chapters.

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CHAPTER 2.—ELECTRICAL FUNDAMENTALS I

The technique used to solve problems in complex electronic circuits or mine power systems is called circuit analysis. It involves calculating such circuit properties as currents, voltages, resistances, inductances, and impedances. Circuit analysis serves as the knowledge base on which an understanding of mine electrical systems can be built.

This chapter will diverge from classical circuit-analysis presentations by not covering transient effects in circuits. From experience, studying currents and voltages existing in a circuit immediately after a change in circuit configuration can be confusing and clouds understanding of the most used segments of circuit analysis. Therefore, although some necessary statements will be made, the subject of transients is delayed until chapter 11, where it can be combined with practical examples.

This chapter commences by introducing electrical phenomena and continues through to a presentation of steady-state ac circuit analysis. Chapter 3, "Electrical Fundamentals II," continues the coverage of basic electrical subjects and starts with the basics of electrical power consumption.

Numerous excellent circuit-analysis textbooks have been produced over the years. Many can be employed effectively to cover the subject, and some of these are provided in the bibliography at the end of this book. Because practically all fundamental electrical relationships are considered common knowledge, the concepts introduced in this chapter will seldom be referenced other than by giving credit to the discoverer.

BASIC ELECTRICAL PHENOMENA

The nature of electricity is not yet fully understood, but it is well known as a form of energy that can be conveniently converted into and utilized as light, heat, and mechanical power. Like all science, knowledge about electricity has been developed from observation and experimentation. The generalization of this experimental evidence combined with information about the nature and behavior of electrons and electron flow forms the basis of electron theory.

The atoms of each element consist of a dense nucleus around which electrons travel in well-defined orbits or shells. The subatomic particles, the building blocks out of which atoms are constructed, are of three different kinds: the negatively charged electron, the positively charged proton, and the neutral neutron. The negative charge of the electron, e^-, is of the same magnitude as the positive charge of the proton, e^+. No charges of smaller magnitude have yet been concretely observed. Thus the charge of a proton or an electron is taken as the ultimate natural unit of charge. It is these two particles that are of principal interest in electricity.

Coulomb's Law

The force, F, between two charges, q and q′, varies directly as the magnitude of each charge and inversely as the square of the distance (r) between them. This relationship, known as Coulomb's law, is represented mathematically by

$$F = k\frac{qq'}{r^2}, \tag{2.1}$$

where k = proportionality constant that depends on units used for force, charge, and distance.

If force is in newtons, charge in coulombs, and distance in meters, then

$$k = 9 \times 10^{+9}\ \mathrm{N{\cdot}m^2/C^2}.$$

The unit of charge, the coulomb (C), can be defined as the quantity of charge that, when placed 1 m from an equal and similar charge, repels it with a force of $9 \times 10^{+9}$ newtons (N). The charge carried by an electron or by a proton is $e = 1.602 \times 10^{-19}$ C.

Voltage and Current

A proton in the nucleus of an atom can hold only one electron in orbit around it. When an atom contains fewer than the normal number of electrons that the protons can attract, the atom has an excess of positive charge and is said to be *positively charged.* Atoms with an excess of electrons are said to be *negatively charged.* The net amount of these charges is termed *potential* or *electromotive force* (emf) and is measured in volts. The separation of opposite charges of electricity may be forced by physical motion or may be initiated or complemented by thermal, chemical, or magnetic causes or even by radiation.

The potential difference or voltage existing between two points can be measured by the work necessary to transfer a unit charge from one point to the other. The volt is the potential between two points when 1 joule (J) of work is required to transfer 1 coulomb (C) of charge. In other words,

$$1\ \mathrm{V} = 1\ \mathrm{J/C}.$$

In some metals or conductors, electrons in the outermost orbit of the atoms are rather loosely bound to their respective nuclei. These are called conduction electrons, since they can leave the atom upon the application of a small force and become free to move from one atom to another within the material. In some materials, however, all the electrons are tightly bound to their respective atoms. These are called insulators, and in these materials it is exceedingly difficult, if not impossible, to free any electrons. Conductors and insulators are the principal materials used in electrical systems.

The application of a voltage across a conductor causes the free electrons within the conductor to move. Electrical current is defined as the motion of electrical charge. If the charge in the conductor is being moved at the uniform rate of 1 coulomb per second (C/S), then the constant current existing in that conductor is 1 ampere (A), the unit of electrical current. The amount of current in a conductor can also be measured as the rate of change of the charge flow. Such changing current at any point in time is called instantaneous current or

$$i = (\text{the rate of change of charge}) = \frac{dq}{dt}, \tag{2.2}$$

where i = instantaneous current, A,
q = flow of charge, C,
t = time, s.

When electricity was first discovered, it was erroneously thought that it was the flow of positive charges. Since the laws of attraction and repulsion were known, the movement was assumed to be from positive to negative. This theory was accepted until the discovery of the radio tube, when it was recognized that the flow was movement of electrons from negative to positive. However, the concept of positive-charge flow was firmly entrenched and has remained standard in the United States, and so it will be used here.

SYSTEM OF UNITS

Most material contained in this text is given in the International System of Units (SI); exceptions are calculations that are more conveniently expressed in terms of the English or American engineering systems. A listing of the basic symbols, units, and abbreviations that are used is given in table 2.1. The decimal system is used to relate larger and smaller units to basic units, and standard prefixes are given to signify the various powers of 10; for example:

pico-	(p-, 10^{-12})
nano-	(n-, 10^{-9})
micro-	(μ-, 10^{-6})
milli-	(m-, 10^{-3})
kilo-	(k-, 10^{3})
mega-	(M-, 10^{6})
giga-	(G-, 10^{12})

Voltage, current, and power variables are represented by the letter symbols V, I, and P in both uppercase and lowercase letters. Uppercase letters represent voltage, current, and power when the variable is constant, as in dc circuits. In ac circuit work, uppercase V and I represent effective values and uppercase P represents average power. Lowercase v, i, and p depict voltage, current, and power when these quantities are varying with time.

Where needed, double-subscript notation is used to describe current and voltage. V_{AB} represents the voltage of point A with respect to point B. I_{CD} represents the current flowing through a circuit element from C to D. Note that in the circuit shown in figure 2.1, the voltage V_{AB} causes the current I_{AB} to flow. These meet with the standard for electrical current, which is positive-charge flow from positive to negative.

EXPERIMENTAL LAWS AND PARAMETERS

It is remarkable that the entire theory of electrical circuits is based on only six fundamental concepts. One is Ohm's law, two are named for Kirchhoff, two relate to inductance and capacitance, and one has to do with power. To understand any electrical system, comprehension of these relationships is mandatory.

Ohm's Law

Georg Simon Ohm (1789-1854) discovered that the electrical current through most conductors is proportional to the voltage (potential) applied across the conductors. This phenomenon is known as Ohm's law and is expressed mathematically as

$$v = Ri, \tag{2.3}$$

where v = applied potential, V,
i = current through the conductor, A,
R = proportionality constant known as resistance of conductor, Ω.

Table 2.1—SI symbols and units

Quantity	*Variable symbol*[1]	*SI unit*	*Unit symbol*	*Identical unit*
Charge	Q	coulomb	C	A·s
Current	I	ampere	A	
Voltage	V,E...U	volt	V	W/A
Electromotive force	V	...do	V	
Potential difference	V,Φ	...do	V	
Resistance	R	ohm	Ω	V/A
Conductance	G	siemens	S	A/V
Reactance	X	ohm	Ω	V/A
Susceptance	B	siemens	S	A/V
Impedance	Z	ohm	Ω	V/A
Admittance	Y	siemens	S	A/V
Capacitance	C	farad	F	C/V
Inductance	L	henry	H	Wb/A
Energy, work	W	joule	J	N·m
Power (active)	P	watt	W	J/s
Power—apparent	S...P_s	voltampere	VA	
Power—reactive	Q...P_q	var	var	
Resistivity	ρ	ohm-meter	Ω-m	
Conductivity	γ,σ	siemens per meter	S/m	
Electric flux	ψ	coulomb	C	
Electric flux density, displacement	D	coulomb per square meter	C/m^2	
Electric field strength	E	volt per meter	V/m	
Permittivity	ϵ	farad per meter	F/m	
Relative permittivity	ϵ_r,κ	(numeric)		
Magnetic flux	Φ	weber	Wb	V·s
Magnetomotive force	F...*F*	ampere (amp turn)		
Reluctance	R...*R*	ampere per weber reciprocal henry	A/Wb H^{-1}	
Permeance	P...*P*	weber per ampere henry	Wb/A H	
Magnetic flux density	B	tesla	T	Wb/m^2
Magnetic field strength	H	ampere per meter	A/m	
Permeability (absolute)	μ	henry per meter	H/m	
Relative permeability	μ_r	(numeric)		

[1] V,E indicates alternative symbols;...U indicates reserve symbols.

No restriction is placed on the form of v and i. In dc circuits they are constant with respect to time, and in ac circuits they are sinusoidal.

For metals and most other conductors, R is constant. In other words, its value is not dependent on the amount of current, i. In some materials, especially in crystalline materials called semiconductors, R is not constant, and this characteristic is useful in diodes, amplifiers, surge arresters, and other devices.

Further experiments by Ohm indicated that the resistance of a piece of metal depends on its size and shape. However, the resistivity, ϱ, of the metal depends only on its composition and physical state. This is an inherent property that opposes current through the conductor just as the frictional resistance of a pipe opposes the flow of water through it. Resistivity is defined as the resistance of a unit cube of homogeneous material; hence, resistivity can be thought of as a property of the material at a point. Its value remains the same at all points in a homogeneous conductor, but if the material is not homogeneous, its resistivity can vary from point to point. The value may also vary greatly for different conductors. The concept of resistivity is often used in the grounding and distribution aspects of mine electrical systems.

Using the definition, practical resistivity units would be ohm-centimeter (Ω-cm) and ohm-inch (Ω-in). However, resistivity is usually expressed in ohm-meter (Ω-m) (SI) and ohm-circular-mill-foot (English). The ohm-meter is the resistance of a material 1 mm² in cross section with 1 m length. Likewise, the ohm-circular-mill-foot (usually abbreviated to Ω-cmil-ft) refers to the conductor resistance for a volume 0.001 in (1 mil) in diameter and 1 ft long. For calculating the resistance in this latter case, the cross-sectional area of the conductor is measured in circular-mills, which can be found from

$$A = d^2, \tag{2.4}$$

where A = cross-sectional area of circular conductor, cmil,
and d = conductor diameter, 10^{-3} in.

Resistivity values of some common conductors are given in table 2.2.

Table 2.2—Resistivity of some common materials at 20° C

Material	Temperature coefficient (α)	Resistivity (ρ)	
		10^{-8} Ω-m	Ω-cmil-ft
Aluminum, commercial	0.0039	2.824	17.1
Copper, annealed	.00393	1.724	10.5
Iron, annealed	.005	9.50	57.4
Lead	.0034	21.83	132.31
Nichrome	.04	100	606.1
Silver	.0038	1.63	9.85
Steel, mild	.002	11.91	72.17
Tin	.0042	11.50	69.7
Tungsten	.0045	5.50	33.2

The resistance of any specific conductor can be calculated from the material resistivity using the formula

$$R = \varrho \frac{\ell}{A} \tag{2.5}$$

where R = resistance, Ω,
ℓ = conductor length,
A = conductor cross-sectional area,
and ϱ = material resistivity.

If ℓ is in meters and A is in meters squared, then ϱ must be given in units of ohm-meters.

Electrical resistivity does not remain constant if the temperature is permitted to change. For most materials, the resistance increases as the temperature increases; carbon is an exception to this rule (negative temperature coefficient, 0.005). If the temperature coefficient is known, the resistance of a given conductor at a given temperature is

$$R = R_o\,[1 + \alpha\,(t - t_o)], \tag{2.6}$$

where R = resistance at temperature t, Ω,
R_o = resistance at reference temperature t_o, usually 20° C, Ω,
α = temperature coefficient, Ω/°C,
t = given conductor temperature, °C,
t_o = reference temperature, °C.

At very low temperatures (about −200° C for copper) or as the melting point is reached, the temperature coefficient is no longer constant and changes with temperature. As a result, equation 2.6 is not valid for very high or low temperatures.

The symbol illustrated in figure 2.1 portrays a resistor in a circuit, and often its resistance is stated. Again by definition,

$$R = \frac{v}{i}. \tag{2.7}$$

Sometimes, the element's conductance, G, is referenced and is defined as the reciprocal of resistance:

$$G = \frac{1}{R} = \frac{i}{v}. \tag{2.8}$$

In circuit analysis, it is occasionally more convenient to use conductance than resistance. Later, the explanation of this symbol will be generalized.

Kirchhoff's Voltage Law

In the simple series circuit shown in figure 2.2, three resistors are connected in tandem to form one single closed loop. Kirchhoff has shown that when several elements are

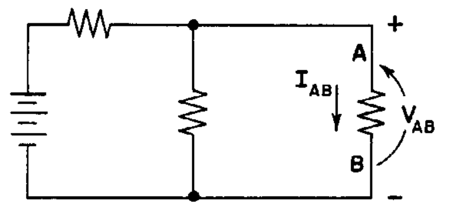


Figure 2.1.—Circuit element illustrating voltage polarity and current flow direction.

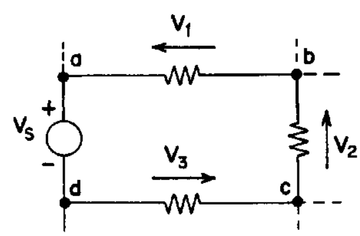


Figure 2.2.—Simple series circuit.

connected in series, the current in the circuit will adjust itself until the sum of voltage drops in the circuit is equal to the sum of voltage sources in the circuit. This can be restated as the "sum of all voltages around any closed circuit is zero," which is called Kirchhoff's voltage law. For the circuit shown in figure 2.2,

$$v_{ab} + v_{bc} + v_{cd} + v_{da} = 0 \tag{2.9}$$

or

$$v_{ab} + v_{bc} + v_{cd} - v_{ad} = 0 \tag{2.10}$$

or

$$v_1 + v_2 + v_3 - v_s = 0. \tag{2.11}$$

Obviously, some of these potential differences could be negative and some positive. This circuit shows only resistances and a voltage source, but the network could contain other kinds of elements and might be as complicated as desired. However, Kirchhoff found that the sum of the voltages around any closed loop in a circuit, such as a-b-c-d, is always zero.

The symbol shown in figure 2.2 beside v_s represents an ideal voltage source. Such a source maintains a given voltage across its output (terminals) regardless of the load, but actual voltage sources cannot supply an infinite current if the terminals are short-circuited; that is, they are tied together so the resistance approaches zero. Therefore, actual sources are usually considered to be ideal voltage sources with an internal resistance connected in series with the source and the output terminals. The assumption is illustrated in figure 2.3.

EXAMPLE 2.1

Find the current I flowing in the single-loop circuit in figure 2.4.

SOLUTION. Adhering to the assigned clockwise direction for current, Kirchhoff's voltage law produces the following equation:

$$-50 + V_1 + 100 + V_2 = 0,$$

where V_1 and V_2 are the voltages across the $1-\Omega$ and $2-\Omega$ resistances. From Ohm's law,

$$V_1 = 1I,$$
$$V_2 = 2I.$$

Inserting these expressions into the voltage law equation produces

$$-50 + 1I + 100 + 2I = 0$$

or

$$3I = -50,$$
$$I = -16.7 \text{ A}.$$

The negative sign states that the actual current flow is in the opposite direction from that shown in figure 2.4.

It should be noted that when writing the voltage-law equation, voltages that oppose the assigned current flow are considered positive, otherwise negative. Therefore, the 100-V source is positive, and the 50-V source is negative. The positive signs for V_1 and V_2 assumed opposition by the convention shown in figure 2.1.

Kirchhoff's Current Law

The other law attributed to Kirchhoff specifies that "the sum of all electrical currents flowing toward a junction is zero." In figure 2.5, five wires are soldered together at a common terminal and the current in each wire is measured. If current flowing toward the junction is called positive (the direction shown in the figure) and the current outwards is negative (against the arrows), then the sum of the five currents is zero:

$$i_1 + i_2 + i_3 + i_4 + i_5 = 0. \tag{2.12}$$

As was the case for equation 2.9, this equation implies that some currents must be positive, some negative.

If two or more loads are connected between two common points or junctions, these elements are said to be in parallel, as shown in figure 2.6*A*. The same is true for figure 2.6*B*, and moreover, the two circuits illustrated in figure 2.6 are identical, just drawn differently. It is important to

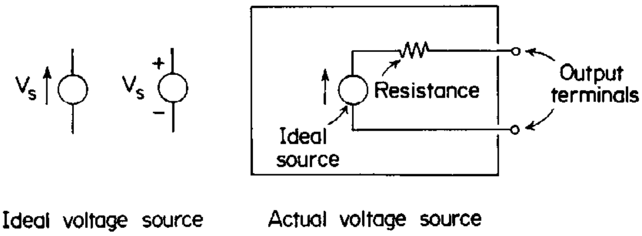


Figure 2.3.—Ideal and actual voltage sources.

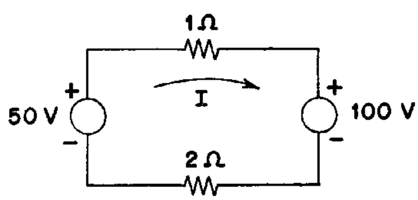


Figure 2.4.—Circuit for example 2.1.

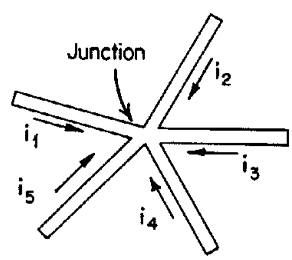


Figure 2.5.—Demonstration of Kirchhoff's current law.

note that the lines in these and all circuit diagrams usually show no resistance. Each line is only a connection between elements or between an element and a junction. The similarity in the diagrams can be shown using Kirchhoff's current law. In both, there are only two independent junctions, a and b, and for either point,

$$i_1 = i_2 + i_3 + i_4. \tag{2.13}$$

The circuit symbol next to i_1 in figure 2.6 represents an ideal current source, and a similar situation exists for all practical current sources as was mentioned for practical voltage sources. However, the internal resistance is effectively connected in parallel across the ideal current source. Both ideal and actual current sources are shown in figure 2.7.

EXAMPLE 2.2

Verify that Kirchhoff's current law holds for junction x in figure 2.8.

SOLUTION. The three resistances in figure 2.8 are in parallel, and the 100 V produced by the voltage source exists across each. Therefore, by Ohm's law, the current through each resistance is

$$I_{25} = \frac{100}{25} = 4 \text{ A},$$

$$I_{50} = \frac{100}{50} = 2 \text{ A},$$

$$I_{100} = \frac{100}{100} = 1 \text{ A}.$$

Kirchhoff's current law states that for junction x,

$$I_{25} + I_{50} + I_{100} = 7 \text{ A}.$$

Accordingly,

$$4 + 2 + 1 + = 7 \text{ A}.$$

Series Circuits

To restate the earlier definition of a series circuit, elements are said to be connected *in series* if the same current passes through them. Such is the situation for the four resistors shown in figure 2.9. It would be convenient to find a resistance, R, that could replace all series resistors. This equivalent resistance can be found by returning to the Ohm and Kirchhoff voltage laws. By Kirchoff's law,

$$v = v_1 + v_2 + v_3 + v_4,$$

but by Ohm's law,

$$v_1 = iR_1,\ v_2 = iR_2,\ v_3 = iR_3, \text{ - - - - -}.$$

Therefore,

$$v = iR_1 + iR_2 + iR_3 + iR_4.$$

For the circuit in figure 2.9, if the voltage, v, produces the same current, i, through the circuit, then

$$v = iR,$$

but

$$v = i\,(R_1 + R_2 + R_3 + R_4),$$

$$iR = i\,(R_1 + R_2 + R_3 + R_4),$$

or

$$R = R_1 + R_2 + R_3 + R_4. \tag{2.14}$$

Here R is said to be the equivalent resistance for the previous series circuit. In other words, R is the series resistance of that circuit. The same logic applies to all electrical elements in series.

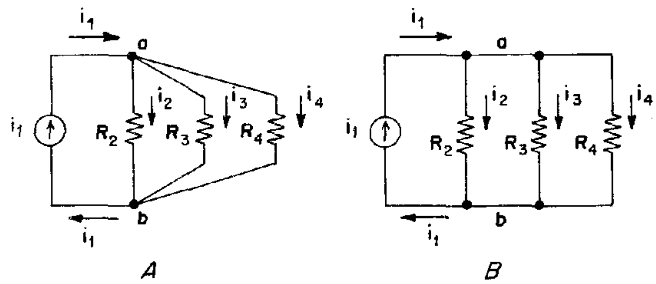


Figure 2.6.—Simple parallel circuits. (Italic letters are cited in text.)

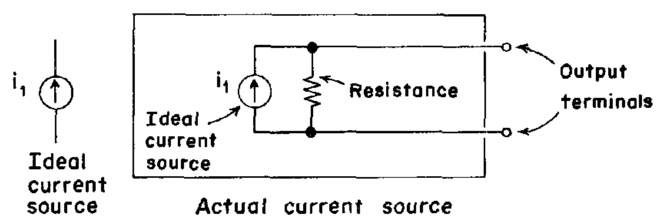


Figure 2.7.—Ideal and actual current sources.

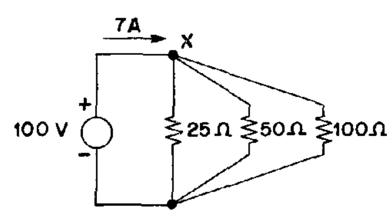


Figure 2.8.—Parallel circuit for example 2.2.

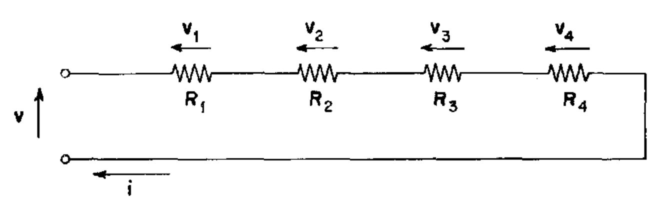


Figure 2.9.—Simple series circuit and equivalent.

It is often useful to find the voltage drop across just one element in a series circuit. To arrive at an expression, again refer to figure 2.9. For the current through the circuit, it is obvious that

$$i = i_1 = i_2 = i_3 = i_4,$$

but $$i_1 = \frac{v_1}{R_1},\ i_2 = \frac{v_2}{R_2}, \ldots .$$

Therefore, $$i = \frac{v_1}{R_1} = \frac{v_2}{R_2} = \frac{v_3}{R_3} = \frac{v_4}{R_4}.$$

As before, consider R, the equivalent circuit resistance, and

$$i = \frac{v}{R}.$$

Therefore, $$\frac{v}{R} = \frac{v_1}{R_1},\ \frac{v}{R} = \frac{v_2}{R_2}, \ldots\ldots$$

or $$v_1 = \frac{R_1}{R} v,\ v_2 = \frac{R_2}{R} v, \ldots\ldots . \qquad (2.15)$$

In other words, the voltage drop across any one element is equal to the total circuit voltage times the ratio of the element's resistance to the total circuit resistance.

Parallel Circuits

Following the discussion of series circuits, it would be useful to have similar equivalence, voltage, and current relationships for parallel circuits. For the circuit shown in figure 2.10, the voltage is the same across each resistor and is a corollary to current through series elements. Using the same basic procedures as for series circuits, it can be shown that

$$G = G_1 + G_2 + G_3 + G_4 + \ldots\ldots \qquad (2.16a)$$

and also

$$\frac{1}{R} = \frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3} + \frac{1}{R_4} + \ldots\ldots \qquad (2.16b)$$

Restated, the total conductance of parallel-connected resistors is equal to the sum of all individual conductances. Likewise, the reciprocal of the total resistance of parallel-conducted resistors is equal to the sum of the reciprocals of the individual resistances. A special case that is very often found occurs when two resistances are in parallel. If these resistances are R_1 and R_2, then

$$R = \frac{R_1 R_2}{R_1 + R_2}. \qquad (2.17)$$

If current distribution through parallel circuits is of interest rather than voltage distribution, Kirchhoff's current law and Ohm's law can be employed to show that

$$i_1 = \frac{G_1}{G} i \text{ or } i_1 = \frac{R}{R_1} i, \qquad (2.18)$$

and so on for the balance of currents.

The preceding paragraphs have been used to show the immediate application of Ohm's law plus Kirchhoff's voltage and current laws to circuits that have more than one element. The results are extremely valuable in circuit analysis and are used extensively to solve circuit problems. It is important to note now that these concepts are also valid when circuits contain components other than resistance. Later, after the balance of fundamental laws and parameters have been covered, more applications of these laws will be shown.

EXAMPLE 2.3

A series-parallel circuit is shown in figure 2.11. Find the equivalent resistance.

SOLUTION. The objective is to find an equivalent resistance between terminals a and b. The process is to combine resistances in series or in parallel until the equivalent resistance is obtained. The 2-Ω and 4-Ω resistances between point 1 and terminal b are in series, and from equation 2.14,

$$2 + 4 = 6\ \Omega.$$

If a 6-Ω resistance replaces these two series resistances, it can be seen that two 6-Ω resistances are in parallel between point 1 and terminal b. Applying equation 2.17,

$$\frac{(6)(6)}{6 + 6} = 3\ \Omega.$$

which means that a 3-Ω resistance can replace the two 6-Ω parallel resistances. Therefore, the 3-Ω resistance between point 2 and point 1 is in series with the equivalent of 3 Ω between point 1 and terminal b, and again

$$3 + 3 = 6\ \Omega.$$

Now between point 2 and terminal b, there are the equivalent of two 6-Ω resistances in parallel and

$$\frac{(6)(6)}{6 + 6} = 3\ \Omega.$$

Consequently between terminal a and terminal b, a 7-Ω resistance is in series with an equivalent 3-Ω resistance, and the equivalent resistance of the entire circuit is

$$R = 7 + 3 = 10\ \Omega.$$

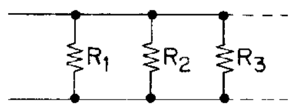


Figure 2.10.—Simple parallel circuit.

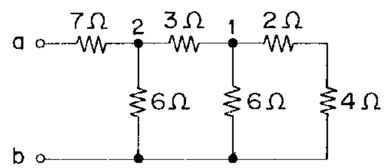


Figure 2.11.—Series-parallel circuit for example 2.3.

EXAMPLE 2.4

Find the equivalent resistance of the circuit illustrated in figure 2.12.

SOLUTION. Point b and point b′ can be seen in the center of the circuit, but these are electrically just one point, because b and b′ are only separated by a line that does not contain an electrical element. Thus, the 15-Ω and 30-Ω resistances between a and b are in parallel, as are the two 40-Ω resistances between b and c.

From equation 2.17,

$$\frac{(15)(30)}{15 + 30} = 10\ \Omega$$

and

$$\frac{(40)(40)}{40 + 40} = 20\ \Omega\ .$$

Therefore, the resistance of the circuit between terminal a and terminal d can be reduced to three series resistances, and the equivalent resistance is

$$R = 10 + 20 + 10 = 40\ \Omega.$$

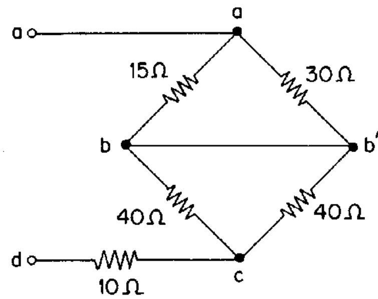


Figure 2.12.—Series-parallel circuit for example 2.4.

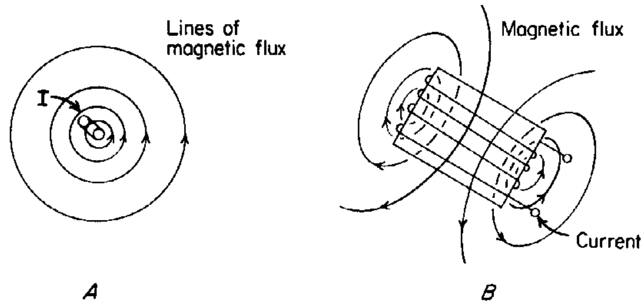


Figure 2.13.—Magnetic flux in a straight conductor (*A*) and in a long coil (*B*).

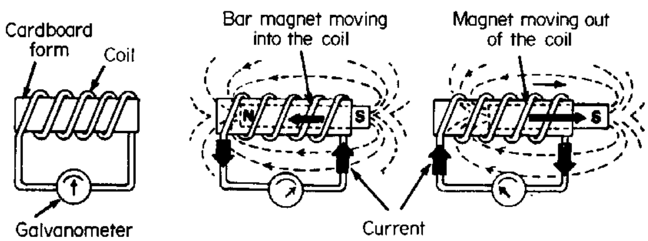


Figure 2.14.—Demonstration of induced current.

The Magnetic Field

A. M. Ampere was the first scientist to establish that the conductor through which electric current is passing is enclosed in a magnetic field. The relationship is depicted in figure 2.13*A*. After Ampere's discovery, many experimenters tried to reverse the process and create electric current from a magnetic field. Finally, in 1831, Michael Faraday discovered that as a magnet is inserted into a coil of wire, an impulse of electrical current will flow through the wire. When the magnet remains stationary within the coil, no current is produced. When the magnet is withdrawn, a current impulse is again observed, but this time it flows in the opposite direction. The process is demonstrated in figure 2.14. Faraday visualized the effect as a result of magnetic flux lines cutting or moving through the conductor. Whenever relative motion occurs, an emf is produced in the conductor. This disclosure laid the foundation for electromechanical conversion, that is, the conversion from mechanical energy to electrical energy and vice versa, as found in generators and motors.

The magnetic field mentioned here is a condition of space. The direction of a magnetic field flux line is the direction of force on a magnetic pole, and the flux line density is in proportion to the magnitude of force on the pole. Each line represents a certain quantity of magnetic flux, measured in webers. It is a magnetic field characteristic that every flux line is a closed curve, forming the concentric-circle pattern shown in figure 2.13*A*. These conditions of the magnetic field are employed to develop relationships in magnetic devices, which are covered in upcoming sections.

When a wire is wound into a coil, an interesting action occurs: as the magnetic flux builds up around one wire, it tends to cut through adjacent turns of wire. In this way a voltage is induced into the coil windings. The concept is shown as dashed flux lines around one winding of the coil in figure 2.13*B*.

Inductance

Joseph Henry found that electricity flowing in a circuit has a property analogous to mechanical momentum; that is, current is difficult to start but once started it tends to continue. This is the case for any element from a simple conductor to the most complex. Faraday explained the phenomenon by visualizing the magnetic field in space around the conductor. In terms of the coil in figure 2.14, the voltage induced in the other windings is proportional to the rate at which the magnetic flux lines are cutting through the coil. Yet the magnetic flux is also proportional to the current in the coil. The induced voltage is such that at every instant, it opposes any change in the circuit current. For this reason, the induced voltage is called a counterelectromotive force, cemf. This interrelationship is so important that it has the status of a physical law and is known as Lenz's law after the scientist who first defined it.

The property that prevents any change of current in the coil is called self-inductance; hence the coil is known as an inductor. The greater the induced voltage, the greater is the opposition to the change in current flow. Therefore, the cemf produced by a specific change of current is a measure of circuit inductance. Expressed as a formula,

$$v = L(\text{rate of change of current})$$

or

$$v = L(\frac{di}{dt}), \tag{2.19}$$

where v = voltage across coil, V,
L = proportionality constant known as inductance, H,
i = current through coil, A.

As noted, inductance is given the symbol L and is measured in units called henries in honor of Joseph Henry. A circuit has an inductance if 1 H when a current change of 1 A/S causes a cemf of 1 V to be induced in the coil. The expression "di/dt" represents the rate of change of current, i, in the coil.

When two separate coils are placed near each other, as shown in figure 2.15, the magnetic field from one coil can cut through the windings of the second coil. It follows that a change in the current in coil 1 can produce an induced voltage in coil 2. This current-voltage relationship is expressed as

$$v_2 = L_{21}\ (\text{rate of change in } i_1)$$

or

$$v_2 = L_{21}\ (\frac{di_1}{dt}). \tag{2.20}$$

Similarly, if the current in coil 2 is changing, it induces a voltage in coil 1:

$$v_1 = L_{12}(\text{rate of change in } i_2)$$

or

$$v_1 = L_{12}\ (\frac{di_2}{dt}). \tag{2.21}$$

L_{12} and L_{21} are called mutual inductances and are again expressed in henries. The mutual inductances increase if the coils are brought closer together and decrease as the coils are moved further apart. Two magnetically coupled coils are usually called a transformer. Although not by all means obvious, the two mutual inductances of a pair of magnetically coupled circuits are equal, or

$$L_{12} = L_{21}. \tag{2.22}$$

The self-inductance of an actual coil is a function of both the coil configuration and the total number of turns. Further, because the magnetic flux may induce currents in adjacent conductors, the environment in which the coils are placed may also have an effect. Numerous inductance equations are available in handbooks and other reference books, each valid for a given coil configuration; consequently, only a few that give approximate inductance values are provided here to demonstrate the parameters that affect inductance. The two symbols used to indicate inductance are shown in figure 2.16; the symbol on the right is that commonly found in power-circuit diagrams.

For a long coil as shown in figure 2.16, the inductance is

$$L = \frac{\mu N^2 A}{\ell}, \tag{2.23}$$

where L = self-inductance, H,
μ = permeability, H/m
(for air, $4\pi \cdot 10^{-7} = 12.566 \cdot 10^{-7}$),
N = turns of coil,
A = coil cross-sectional area, m^2,
and ℓ = coil length, m.

The coil cross section need not be circular. The formula is only approximate because it assumes that all flux lines link all turns of the coil, which cannot occur at the coil ends. However, the formula gives good results for long coils and does reveal the following important relationships.

- Coil inductance is proportional to the square of the number of turns.
- Inductance is proportional to the core permeability.
- Inductance is proportional to the cross-sectional area of the core.
- Inductance decreases as the length increases.

For a shorter single-layer circular solenoid (coil), the inductance is approximately

$$L = \frac{\mu N^2 A}{\ell + 0.45d}, \tag{2.24}$$

where d = coil diameter, wire center to center, m.

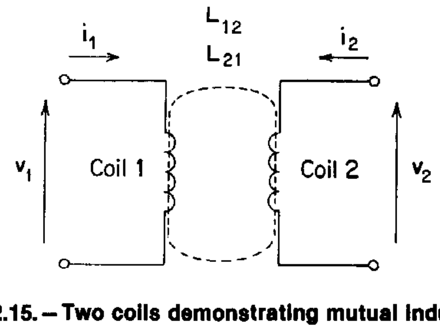


Figure 2.15. — Two coils demonstrating mutual inductance.

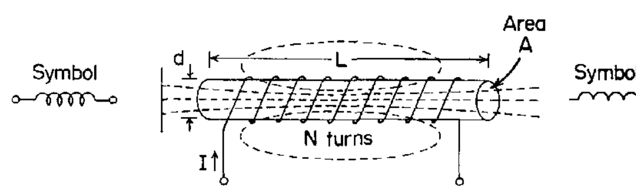


Figure 2.16. — Long-coil inductance and inductor symbols.

For the toroidal coil of rectangular cross section in figure 2.17,

$$L = \frac{\mu N^2 h}{2\pi} \ln\left(\frac{d_2}{d_1}\right), \tag{2.25}$$

where d_1, d_2 = inner and outer diameters as shown, m,
and h = thickness, m.

Note that ln indicates the natural logarithm, that is, the logarithm to the base e.

Capacitance

When two conducting surfaces are separated by a dielectric or insulating material, an effect known as capacitance is observed. If two electrical conductors are at different potential, there is some storage of charge upon them. A capacitor is a device included in a circuit for the purpose of storing or exchanging this electrical charge. Further, when capacitance is present, the charge observed to flow into the capacitor is proportional to the voltage applied. This can be expressed as:

$$q = Cv, \tag{2.26}$$

where q = stored charge in capacitor, C,
v = applied voltage, V,
and C = proportionality constant called capacitance, F.

To analyze circuits, a relationship between the voltage applied and the current flowing into and from the capacitor is more useful. Current is the rate at which charge flows ($i = dq/dt$). It therefore holds that for a given capacitance,

$$i = C(\text{rate of change of } v)$$
$$= C\left(\frac{dv}{dt}\right), \tag{2.27}$$

where i = current flowing into the capacitor, A,
and v = voltage across capacitor, V.

This is very similar to equation 2.19 and, using the discussion in that section, capacitance can be defined as that electrical circuit property which tends to oppose any change in voltage. The capacitance of a capacitor depends on the size of the conductors or plates, their proximity, and the nature of the material between them. For most dielectric materials, C is constant.

Equations 2.26 and 2.27 have algebraic signs consistent with the arrows in figure 2.18. The symbol shown is for capacitance; note that a positive terminal voltage produces positive current and hence positive charge.

If the voltage across a capacitor is desired, equation 2.27 can be integrated, resulting in

$$v = \frac{1}{C}\int_0^t i\,dt + V_0. \tag{2.28}$$

This equation represents the change in voltage across the capacitor from some arbitrary reference time, called $t = 0$, to a later time, t. V_0 is the potential across the capacitor at time $t = 0$. The expression

$$\text{“}\frac{1}{C}\int_0^t i\,dt\text{”}$$

is the voltage change across the capacitance from time $t = 0$ to time $t = t$. From the formula, if the voltage across the capacitor remains constant, as in dc circuits, no current will flow into or out of the capacitor.

Electric Field

An electric field exists anywhere in the neighborhood of an electrical charge, for example, between the plates of a capacitor. The direction of this field is by definition the direction of the force on a positively charged exploring particle (a particle free to move within the electric field). The strength of the field, E, is proportional to the magnitude of the force. If the charge of the exploring particle is q, then the force is

$$F = qE, \tag{2.29}$$

where F = force on particle, N,
and E = strength of electric field, V/m.

Electric-field flux lines are visualized as issuing from positive electric charge and terminating on negative charge as shown in figure 2.19.

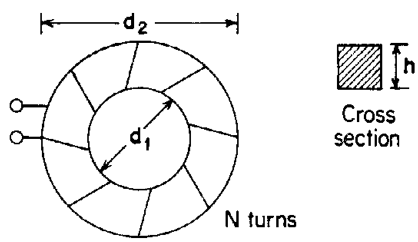


Figure 2.17.—Toroidal coil.

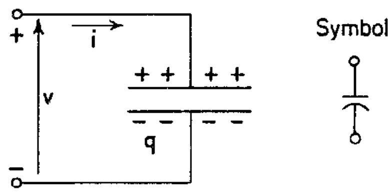


Figure 2.18.—Charge, voltage, and current relationships of capacitor.

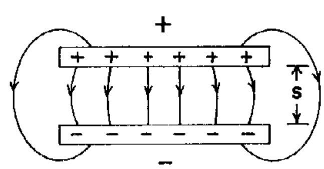


Figure 2.19.—Electric lines of force between two parallel charged plates.

Voltage or potential difference is by definition the integral of electric-field strength or

$$v = \int E \cdot ds. \tag{2.30}$$

A simple application of this concept can be demonstrated from figure 2.19. If the electric field between the two parallel plates is constant, the voltage between the plates is

$$v = E \cdot s. \tag{2.31}$$

Assume that a positively charged particle, q, is released from the positive plate in figure 2.19, the particle being within the electric field and free to move. If it moves, work is performed on it by the electric field. The amount of work can be found by employing the mechanical formula

$$w = F \cdot s, \tag{2.32}$$

where w = work done, J,
F = force on particle, N,
and s = distance particle moves, m.

Since $F = qE,$ (2.29)
work is $w = qE \cdot s,$ (2.33)
but $v = E \cdot s,$ (2.31)
so $w = qv.$ (2.34)

Therefore, when electricity moves from one potential to another, the work done is equal to the product of the amount of electricity and the potential difference. In the next section, this concept is applied to a common electrical component.

Instantaneous Power

Consider the resistor shown in figure 2.20. A charge, dq, is free to move in the resistor from the point s to s + ds. It moves the distance, ds, in time, dt, and is impelled by the electric field in the region, E.

The electric field exerts a force on the charge, dq, while it moves through ds, or

$$F = dq \cdot E. \tag{2.35}$$

The work done in this section of the resistor during time dt can be expressed as

$$dw = F \cdot ds = dqE \cdot ds. \tag{2.36a}$$

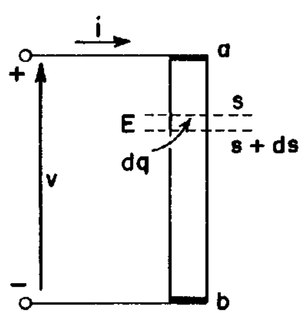


Figure 2.20.—Resistor used to demonstrate instantaneous power.

Power is work per unit time (in other words, the rate of doing work), or for this section of the resistor,

$$dp = \frac{dw}{dt} = \frac{dqE \cdot ds}{dt}, \tag{2.36b}$$

where p = power, W.

However, the current through the resistor is the rate at which charge flows, i = dq/dt; therefore,

$$dp = iE \cdot ds. \tag{2.37}$$

Current is constant throughout the resistor and is not a function of distance, s. The potential difference across the region, ds, is

$$v = \int E \cdot ds, \tag{2.30}$$

and the power across the whole resistor, from a to b, is then

$$p = \int_a^b iE \cdot ds = i \int_a^b E \cdot ds = iv. \tag{2.38}$$

Formula 2.38 represents only the instantaneous power consumed by the resistor, or the power occurring at only one instant in time. This is an extremely important formula as it forms the basis for most power relationships.

Idealization and Concentration

The foregoing has established the elementary laws and parameters that can be applied to investigate electrical circuits. Practical circuits found throughout a mine, or in fact anywhere else, are composed of wires, coils, and electrical devices of varying complexity. Before these fundamentals can be employed, it is necessary to translate the practical world into an ideal and simple world. The translation is called idealization and is in essence the construction of a model. Here, electrical effects that create insignificant results are eliminated. For instance, two adjacent conductors in a coil always exhibit capacitance but the capacitance might be so small that the stored charge is negligible. Yet for many situations, the resistance and inductance must remain.

For every conductor or component in a circuit, resistance, capacitance, and inductance are distributed throughout the entire length or breadth of the portion. It would be much simpler to apply the preceding relationships if these circuit parameters were combined or concentrated into separate circuit elements. For most circuit analysis needs, fortunately, these can indeed be consolidated.

The fundamental aspects of idealization and concentration are illustrated in figure 2.21*A*, which shows a voltage generator connected to a coil of wire and a resistor in series. Figure 2.21*B* gives the translation. The distributed resistance and inductance of the coil have been combined into R_L and L, and the coil's capacitance has been ignored. The resistance of the resistor and its lead wires has been concentrated into one value, R. Finally, the voltage generator is represented by an ideal voltage source and an internal series resistance. Note that the lines shown in figure 2.21*B* serve only to connect components and exhibit no electrical properties or effects. Another example can be expressed from figure 2.22*A*. Here, a load center is shown connected to a shuttle car through a trailing cable. Again, figure 2.22*B* gives the translation. The distributed resistance, inductance, and capacitance of the trailing cable have been

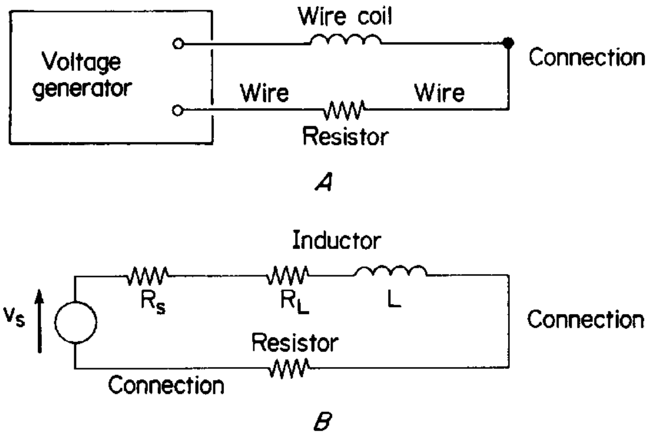


Figure 2.21.—Simple example of idealization and concentration.

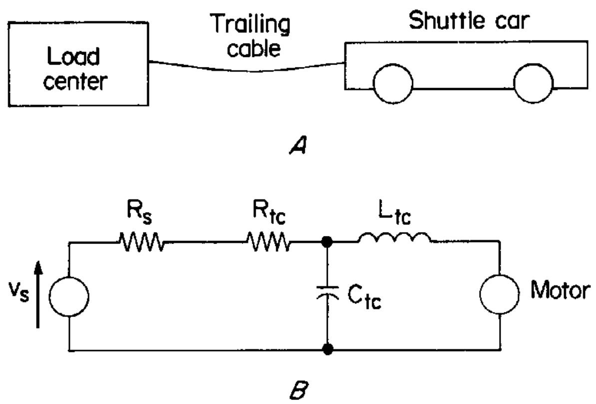


Figure 2.22.—Modeling of load center, trailing cable, and shuttle car.

represented by the combined R_{tc}, L_{tc}, and C_{tc}. Such models can be developed for all portions of a mine distribution system. In this case, although the cable capacitance is rather small, it is shown here to emphasize that it may not always be negligible. The shuttle-car motor is depicted by the symbol shown. The symbol is obviously the same as that for a source, for reasons given later.

After constructing the circuit representation, or schematic, as it is most often called, the relationships covered previously in this chapter can be used to solve for currents and voltages within the circuit. A differential equation will usually result, but if the circuit contains only resistance, the only necessary expression is Ohm's law. This will be the case for most dc circuit analyses.

DIRECT CURRENT CIRCUITS

Electrical current consists of the motion of electrical charges in a definite direction. The direction and magnitude of current can vary with time, and accordingly, all currents can be classified into one of three basic types:

Direct current (dc),
Alternating current (ac) or sinusoidal current, and
Time-varying current.

Direct current is a steady, continuous, unidirectional flow of electricity. In other words, voltage (V) and current (I) have uniform values. Beyond this, the term "dc" is also applied to ordinary or practical currents that are approximately steady.

The following section explores dc circuit analysis, an important topic because of its extensive use for mine haulage and for driving electronic components. The study of dc analysis at this time allows the fundamental electrical laws and parameters to be applied and extended without having the effort clouded by complex current relationships.

Direct Current and Circuit Elements

Figure 2.23 gives the basic elements of resistance, inductance, and capacitance, each having a voltage and current as shown. A powerful simplification of complex circuits can be understood by examining the effect of dc on these elements. As before, the voltage-current relationship for the resistor is Ohm's law:

$$V_R = IR. \tag{2.39}$$

For the inductor, the voltage across the element is

$$V_L = L\frac{dI}{dt},$$

but, because I does not change,

$$\frac{dI}{dt} = 0$$

and

$$V_L = 0. \tag{2.40}$$

Likewise, for the capacitor, the current through the element is

$$I_C = \frac{CdV}{dt},$$

but again,

$$\frac{dV}{dt} = 0,$$

$$I_C = 0. \tag{2.41}$$

Therefore, inductance and capacitance phenomena are not present under pure dc. In other words, the capacitor appears as an open circuit, while an inductor resembles a conductor showing only resistance. An example of this simplification is available in figure 2.24. The circuit on the left shows all circuit elements, but under dc the effective circuit is given on the right. The result is a simple series-resistance arrangement, and the only voltage-current relationship necessary for the analysis is again Ohm's law.

Series and Parallel Resistance

The expressions used to find the equivalent resistance of parallel or series resistances are as before:

for series, $R_{eq} = R_1 + R_2 + R_3 + \ldots . + R_n$;

for parallel, $\dfrac{1}{R_{eq}} = \dfrac{1}{R_1} + \dfrac{1}{R_2} + \dfrac{1}{R_3} + \ldots . + \dfrac{1}{R_n}$;

using Ohm's law, $V = IR_{eq} = I(R_L + R_1)$

or $I = \dfrac{V}{R_{eq}} = \dfrac{V}{R_L + R_1}$.

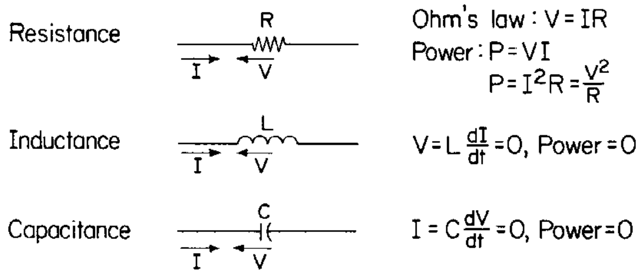


Figure 2.23. – Basic elements of resistance, inductance, and capacitance.

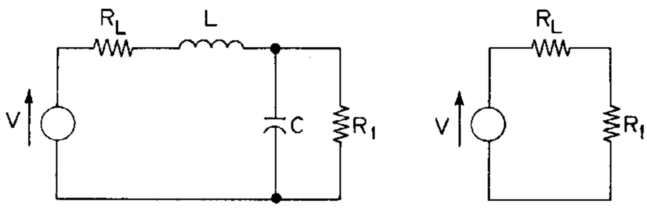


Figure 2.24. – Simplification of dc circuit.

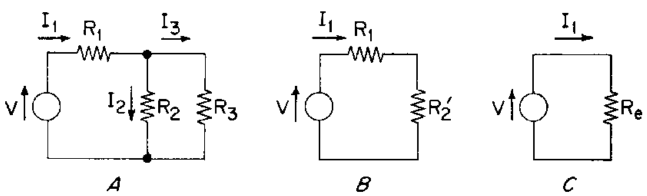


Figure 2.25. – Simple circuit reduction.

Hence, the current through or the voltage across the circuit can be found. By employing the previously given formulas, the voltage-current relationships for each circuit part can be determined.

This concept can be elaborated considering figure 2.25, which shows a series-parallel circuit where an element of the circuit may be in parallel or series with other elements. Arrangements such as these can be solved by observing which individual elements are in series or parallel, then making the appropriate combinations. The objective is to gradually reduce the circuit to an equivalent series arrangement, which can then be replaced by a single equivalent resistance. Simple illustrations of this process have been shown in examples 2.3 and 2.4. Accordingly, the circuit in figure 2.25*A* can be changed to figure 2.25*B* by the parallel combination

$$\frac{1}{R_2'} = \frac{1}{R_2} + \frac{1}{R_3} \quad \text{or} \quad R_2' = \frac{R_2R_3}{R_2 + R_3}\,.$$

The circuit in figure 2.25*B* is then reduced to the circuit in 2.25*C* for the series combination

$$R_{eq} = R_1 + R_2' = R_1 + \frac{R_2R_3}{R_2 + R_3}\,.$$

Afterwards, if V is known, $I_1 = \dfrac{V}{R_{eq}}$.

Further, the current and voltage distributions can be determined by

$$I_2 = I_1\left(\frac{R_2'}{R_2}\right) \quad \text{and} \quad I_3 = I_1\left(\frac{R_2'}{R_3}\right),$$

$$V_1 = \left(\frac{R_1}{R_{eq}}\right)V \quad \text{and} \quad V_2 = \left(\frac{R_2'}{R_{eq}}\right)V,$$

$$\text{or} \quad V_1 = R_1I_1 \quad \text{and} \quad V_2 = R_2I_1.$$

In this way, all voltages and currents in figure 2.25 can be found. In summary, the main process used is the substitution of a single resistance for several series-parallel resistances. In concept, the same terminal resistance (R_{eq}) implies equivalence and results in identical current and voltage delivered from the source. This process of solution is formally known as circuit reduction.

The power consumed by all or part of the circuit can be found by applying equation 2.38:

$$P = VI. \tag{2.42}$$

Noting Ohm's law, V = IR, two other convenient power expressions are

$$P = (IR)I = I^2R \tag{2.43}$$

and

$$P = V\left(\frac{V}{R}\right) = \frac{V^2}{R}. \tag{2.44}$$

These three expressions can be used to find the power loss, expressed as I^2R loss, due to conductor resistance before the current is delivered to a load, as well as the power used by that load.

EXAMPLE 2.5

For the circuit shown in figure 2.26, determine the current I flowing through the 30-Ω resistance, the power supplied to the circuit by the voltage source, and the power consumed by the 15-Ω resistance.

SOLUTION. The 25-Ω, 15-Ω, and 10-Ω series resistances are in parallel with the 50-Ω resistance, and

$$25 + 15 + 10 = 50\ \Omega,$$

$$\frac{(50)(50)}{50 + 50} = 25\ \Omega.$$

The equivalent resistance seen by the 50-V ideal source is the sum of three series resistances:

$$R_{eq} = 30 + 25 + 45 = 100\ \Omega.$$

The current delivered by the source is then

$$I_1 = \frac{V}{R_{eq}} = \frac{50}{100} = 0.5\ \text{A},$$

and the power supplied to the circuit by the source is

$$P_1 = VI_1 = 50(0.5) = 25\ \text{W}.$$

The current through the 15-Ω resistance can be found by using current division:

$$I_2 = I_1\left(\frac{50}{50 + 25 + 15 + 10}\right) = 0.5\left(\frac{50}{100}\right) = 0.25\ \text{A}.$$

Therefore, the power consumed by the 15-Ω resistance is

$$P = I_2^2R = (0.25)^2 15 = 0.94\ \text{W}.$$

EXAMPLE 2.6

Find the current between points a and b, I_{ab}, in the circuit of figure 2.27.

SOLUTION. The circuit is very similar to figure 2.12 as used in example 2.4. Point a and point b are at the same potential, so the right-hand side of the circuit is essentially two parallel arrangements of 15-Ω and 30-Ω resistances. The two parallel arrangements are in series. As

$$\frac{(15)(30)}{15 + 30} = 10\ \Omega,$$

the equivalent resistance seen by the 30-V source is

$$R_{eq} = 10 + 10 + 10 = 30\ \Omega,$$

and total circuit current is

$$I_1 = \frac{V}{I} = \frac{30}{30} = 1\ \text{A}.$$

Because of effective parallelism, I_a, I_b, I_c, and I_d can be found by

$$I_a = I_1\left(\frac{30}{15 + 30}\right) = 1\left(\frac{30}{45}\right) = 0.67\ \text{A},$$

$$I_b = 1\left(\frac{15}{15 + 30}\right) = 0.33\ \text{A},$$

$$I_c = 1\left(\frac{15}{15 + 30}\right) = 0.33\ \text{A},$$

$$I_d = 1\left(\frac{30}{15 + 30}\right) = 0.67\ \text{A}.$$

Even though the potential at point a is the same as the potential at point b, the line connecting a to b has the ability to carry current. By Kirchhoff's current law for point a,

$$I_a = I_{ab} + I_c,$$

and for point b,

$$I_{ab} + I_b = I_d.$$

By either relationship,

$$I_{ab} = 0.67 - 0.33 = 0.33\ \text{A}.$$

EXAMPLE 2.7

The circuit in figure 2.28 is a series-parallel arrangement of conductances. Find the voltage V.

SOLUTION. As total conductance of parallel-connected conductances is the sum of the individual conductances, the combination of the two elements between points a and b is

$$G_{ab} = 2 + 2 = 4.$$

This combined conductance is in series with the 1-S conductance so that total conductance of the circuit portion from point a through point b to point c is

$$\frac{1}{G_{abc}} = \frac{1}{G_{ab}} + \frac{1}{G_{bc}}$$

or

$$G_{abc} = \frac{G_{ab}G_{bc}}{G_{ab}+G_{bc}} = \frac{(4)(1)}{4+1} = 0.8\ \text{S}\ .$$

G_{abc} is in parallel with the 2-S conductance between points a and c, and the equivalent conductance seen by the ideal by the ideal current source is

$$G_{eq} = 2.0 + 0.8 = 2.8\ \text{S}.$$

Therefore, the voltage shown in figure 2.28 is

$$V = \frac{I}{G_{eq}} = \frac{28}{2.8} = 10\ \text{V}.$$

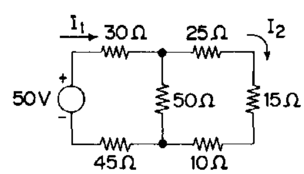


Figure 2.26. – Circuit for example 2.5.

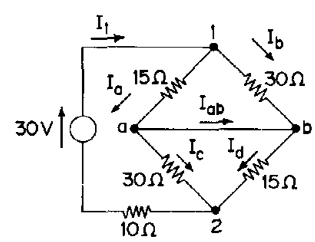


Figure 2.27. – Circuit for example 2.6.

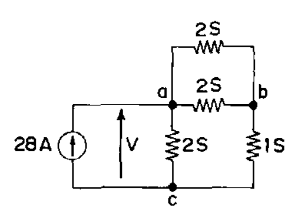


Figure 2.28. – Series-parallel conductances for example 2.7.

EXAMPLE 2.8

For the circuit shown in figure 2.29, find the total circuit current, I_1, with the components as shown, with the 5-Ω resistor short-circuited, and with the 5-Ω resistor open-circuited.

SOLUTION. For the circuit as shown, the two 10-Ω resistors between points c and d are in parallel and

$$R_{cd} = \frac{(10)(10)}{10 + 10} = 5\Omega.$$

This resistance is in series with the 5-Ω resistance and these three elements are in parallel with both the 15-Ω and 10-Ω resistances between points b and d. Thus,

$$R_{bcd} = 5 + 5 = 10\ \Omega$$

and

$$\frac{1}{R_{bd}} = \frac{1}{15} + \frac{1}{10} + \frac{1}{R_{bcd}},$$

or

$$\frac{1}{R_{bd}} = \frac{1}{15} + \frac{1}{10} + \frac{1}{10},$$

$$R_{bd} = 3.75\ \Omega.$$

R_{bd} is in series with the 7-Ω resistance and

$$R_{abd} = 7 + 3.75 = 10.75\ \Omega,$$

and R_{abd} is in parallel with the 10-Ω resistance between points and a and d; both are across the 120-V source. Therefore,

$$R_{eq} = \frac{(10.75)(10)}{10 + 10.75} = 5.18\ \Omega,$$

and total circuit current is

$$I_1 = \frac{V}{R_{eq}} = \frac{120}{5.18} = 23.2\ \text{A}.$$

When the 5-Ω resistance in figure 2.29 is short-circuited or replaced with zero resistance, points b and c are electrically the same. Four resistances are now in parallel between points b and d; three 10-Ω and the 15-Ω resistance. Following the same procedure as before, the equivalent resistance becomes

$$R_{eq}' = 4.93\ \Omega,$$

and total circuit current is

$$I_1' = \frac{120}{4.93} = 24.3\ \text{A}.$$

For the case of an open-circuited 5-Ω resistance, the resistance between b and c is assumed to be infinite, and the two 10-Ω resistances between points c and d are disconnected from the circuit. The equivalent resistance is now

$$R_{eq}'' = 5.65\ \Omega,$$

and the total circuit current is

$$I_1'' = \frac{120}{5.65} = 21.2\ \text{A}.$$

This example illustrates an important concept. When an element in a circuit is short-circuited, the equivalent resistance of the circuit will decrease, and total circuit current will increase. Conversely, with an open-circuited element, the equivalent resistance of the entire circuit will increase while total circuit current decreases.

Wye-Delta Transformations

Any of the circuits now covered can be reduced to a two-terminal network, as seen in figure 2.30*A*. The circuit receives power from an external source and can contain resistance, inductance, and capacitance. Such networks are called passive. For dc, only resistance is of interest, and it can be found from the terminal voltage and current by

$$R = \frac{V}{I}.$$

Numerous circuits can be represented by a two-terminal arrangement. Other circuits, including several in mine power systems, cannot be represented in this way, but many of these can be resolved into the three-terminal network given in figure 2.30*B* Even though with three terminals there now exist three voltages and three currents, the concept of circuit equivalence still holds; that is, voltages and currents are identical and the circuits are equivalent.

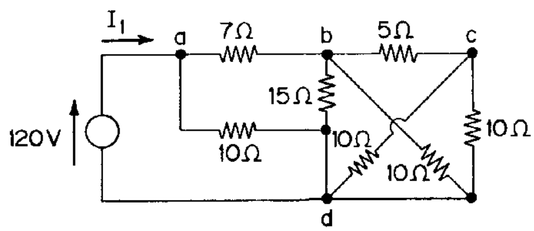


Figure 2.29.—Series-parallel circuit for example 2.8.

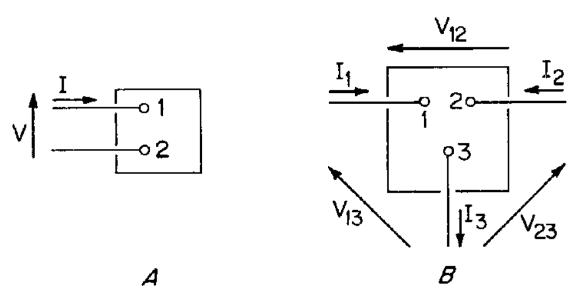


Figure 2.30.—Two-terminal (*A*) and three-terminal (*B*) networks.

For three-terminal networks, there are two basic circuit configurations: the wye (y) and the delta (Δ) (fig. 2.31). The wye is sometimes called a star, but the term *y* is standard. It is sometimes advantageous to replace or substitute the three wye-connected resistances with another set that is delta-connected, or vice versa.

By using equivalence of input currents and voltages for wye and delta circuits, delta-wye (or delta-to-wye) and wye-delta transformations can be derived. Thus for equivalence of the circuits in figure 2.31,

$$R_{ab} = \frac{R_a R_b + R_b R_c + R_c R_a}{R_c}, \tag{2.45}$$

$$R_{bc} = \frac{R_a R_b + R_b R_c + R_c R_a}{R_a}, \tag{2.46}$$

and

$$R_{ca} = \frac{R_a R_b + R_b R_c + R_c R_a}{R_b}. \tag{2.47}$$

In other words, the delta is equivalent to the wye if the resistances of the delta are related to the wye by equations 2.45, 2.46, and 2.47. Accordingly, with three terminals a, b, c, containing wye-connected R_a, R_b, R_c, the circuit performance is unaffected by replacing them with a delta-connected R_{ab}, R_{bc}, R_{ca}. Likewise, for equivalence of delta-to-wye sets,

$$R_a = \frac{R_{ab} R_{bc}}{R_{ab} + R_{bc} + R_{ca}}, \tag{2.48}$$

$$R_b = \frac{R_{ab} R_{bc}}{R_{ab} + R_{bc} + R_{ca}}, \tag{2.49}$$

and

$$R_c = \frac{R_{ac} R_{bc}}{R_{ab} + R_{bc} + R_{ac}}. \tag{2.50}$$

These transformations are useful in allowing three-terminal circuit reduction because they allow substitution when a network does not contain either series or parallel elements. The circuit or circuit portion may not outwardly appear as three-terminal, and common examples, π and T, are given in figure 2.32. These are actually delta and wye circuits drawn in a slightly different fashion. It will be shown in chapter 4 that delta and wye circuits are the two most important configurations for power systems. These transformations will be called upon again at that point.

It has already been shown that when circuit elements are neither all in series nor all in parallel, but have some other series-and-parallel arrangement, the elements can be handled in groups to reduce the circuit to an equivalent resistance. This important kind of circuit analysis has been called circuit reduction. Now that delta-wye and wye-delta transformations have been introduced, the substitution process can be employed to solve networks that contain elements neither in series nor parallel. A prime instance is the common bridge circuit shown in figure 2.33. The bridge is one of the most used configurations in electrical instrumentation. The objective here is to find all available currents and voltage drops in the network, and an overall solution approach is illustrated in the following example.

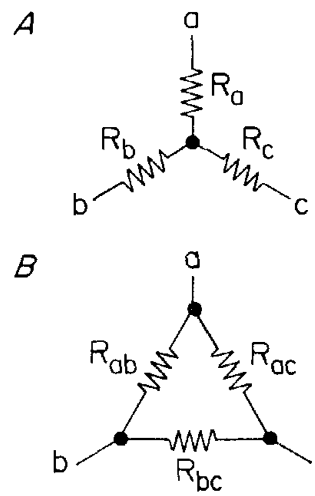


Figure 2.31. – Wye (*A*) and delta (*B*) circuit configurations.

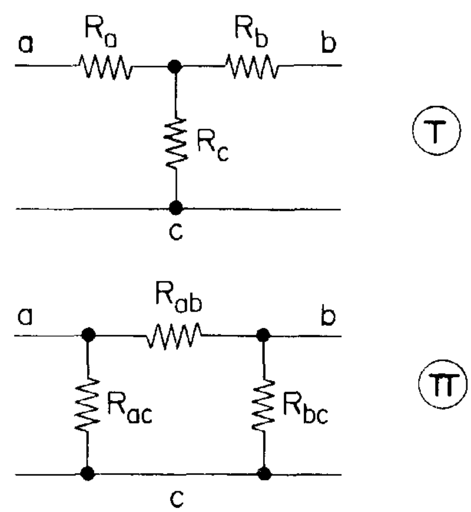


Figure 2.32. – "T" and "π" circuit configurations.

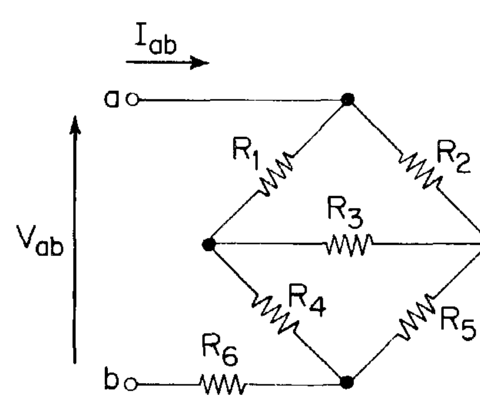


Figure 2.33. – Common bridge circuit.

EXAMPLE 2.9

Consider that the resistances shown in figure 2.33 are as follows:

$R_1 = 5\ \Omega \qquad R_2 = 10\ \Omega \qquad R_3 = 15\ \Omega$
$R_4 = 20\ \Omega \qquad R_5 = 25\ \Omega \qquad R_6 = 0.4\ \Omega$

Find the equivalent resistance of the circuit between points a and b.

SOLUTION. The original circuit has been redrawn in figure 2.34*A*, in which a delta configuration is clearly defined by points a, c, d. The first step for circuit reduction is to convert the delta to a wye. From equations 2.48, 2.49, and 2.50,

$$R_a = \frac{R_{ad}R_{ca}}{R_{ad} + R_{cd} + R_{ca}} = \frac{(10)(5)}{10 + 15 + 5} = 1.67\ \Omega,$$

and

$$R_c = \frac{(5)(15)}{30} = 2.5\ \Omega,$$

$$R_d = \frac{(15)(10)}{30} = 5\ \Omega.$$

This conversion results in the simple series-parallel circuit in figure 2.34*B*. Combining the series elements and parallel branches in the center of the circuit further reduces the circuit to that shown in figure 2.34*C*:

$$R_c + R_4 = 2.5 + 20 = 22.5\ \Omega,$$

$$R_d + R_5 = 5 + 25 = 30\ \Omega,$$

$$R_x = \frac{(22.5)(30)}{22.5 + 30} = 12.9\ \Omega.$$

The equivalent resistance of the total circuit is then simply

$$R_{eq} = R_a + R_x + R_6$$

$$= 1.67 + 12.9 + 0.4 = 15\ \Omega.$$

The total circuit current can now be found using Ohm's law; for instance, if

$$V_{ab} = 30\ \text{V},$$

then $$I_{ab} = \frac{V_{ab}}{R_{eq}} = \frac{30}{15} = 2\ \text{A}.$$

Finally, Kirchhoff's current and voltage laws and the voltage and current distribution formulas can be employed to find currents through and the voltage drops across each circuit element. For example, if I_4 is the current through R_4, then

$$I_4 = I_{ab}\left(\frac{R_d + R_5}{R_c + R_4 + R_d + R_5}\right)$$

$$= 2\left(\frac{30}{22.5 + 30}\right) = 1.14\ \text{A}.$$

It should be noted that the current through R_c is also I_4, but R_c does not exist in the original circuit of figure 2.33. Thus, a problem exists in finding the currents through R_1, R_2, and R_3. One solution would be to solve for the three potentials among points a, c, d and use Ohm's law to find the three currents in the assigned delta connection.

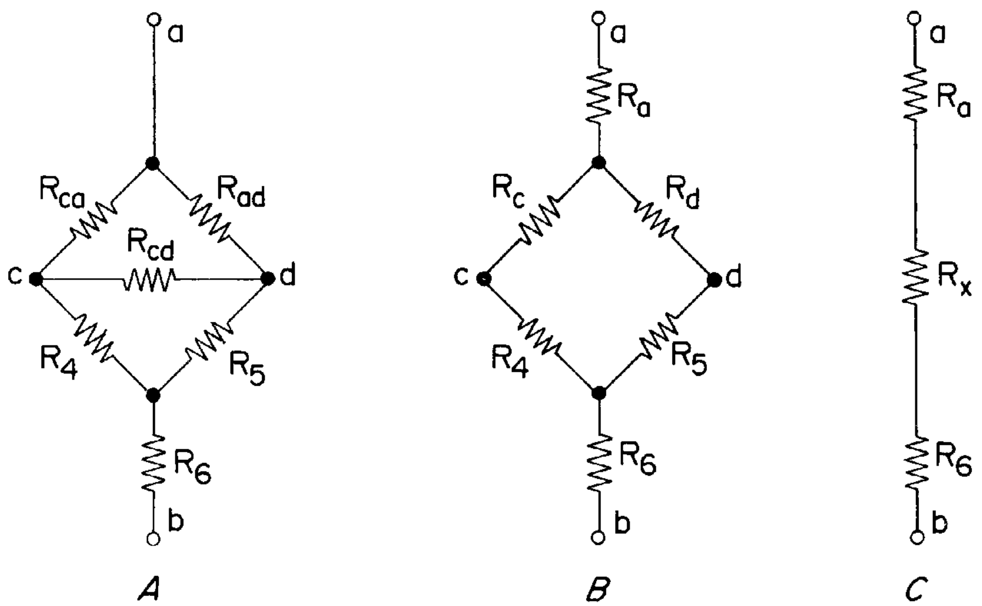


Figure 2.34.—Circuit reduction of bridge circuit.

EXAMPLE 2.10

Consider that the resistances shown in figure 2.33 are

$R_1 = 15\ \Omega$ $\quad R_2 = 15\ \Omega$ $\quad R_3 = 15\ \Omega$
$R_4 = 20\ \Omega$ $\quad R_5 = 25\ \Omega$ $\quad R_6 = 10\ \Omega$

Find the equivalent resistance of the circuit between points a and b.

SOLUTION. Three identical resistances form a delta configuration in the circuit, or

$$R_1 = R_2 = R_3 = 15\ \Omega.$$

Following the same processes as in example 2.9 for figure 2.34*B*,

$$R_a = \frac{(15)(15)}{15 + 15 + 15} = 5\ \Omega,$$

$$R_b = 5\ \Omega$$

and

$$R_c = 5\ \Omega.$$

Now, the center resistance of figure 2.34*C* is

$$R_x = \frac{(5 + 20)(5 + 25)}{5 + 20 + 5 + 25} = 13.64\ \Omega,$$

and the equivalent resistance of the circuit is

$$R_{eq} = 5 + 13.64 + 10 = 28.6\ \Omega.$$

It should be noted that an important situation is established where all resistances in a delta or a wye configuration are equal. If R_Δ is each resistance in the delta and R_y is that in the wye, then from equations 2.48, 2.49, or 2.50,

$$R_y = \frac{R_\Delta R_\Delta}{R_\Delta + R_\Delta + R_\Delta}$$

or

$$R_y = \frac{R_\Delta}{3}.$$

The majority of delta or wye configurations used in power systems consist of identical elements in each leg.

Much of circuit analysis can be handled by circuit reduction, but as circuits become more complex this process becomes cumbersome. Nevertheless, circuit reduction should always be used when it produces results more easily than other methods. There are solution approaches that are more systematic, and the next two sections discuss two of these.

Circuit and Loop Equations

Before more general solution methods can be identified, the meanings of some words need to be clarified. A *node* is the position or point in a circuit where two or more elements are connected. When three or more elements extend from a node, the node is called a *junction*. A *branch* is a circuit portion existing between two junctions and may contain one element or several in a series. A *loop* is a single closed path for current. Figure 2.35 illustrates all these circuit parts.

The following technique, loop analysis, is based entirely on Ohm's law and Kirchhoff's voltage law. The analysis principle produces n simultaneous equations requiring the solution of n unknowns, and the unknowns are currents.

In loop analysis it is only necessary to determine as many different currents as there are independent loops; that is, the equations are constructed by defining independent-loop currents. For example, in figure 2.36, the current I_1, flowing out of source V_1 and through R_a, will be around loop 1. The current flowing from source V_2 through R_c will be around loop 2. Although not essential, these directions follow the general convention of assigning all reference loops clockwise. It is sometimes more desirable to use other directions, for instance with currents flowing out of a source positive terminal, but it is imperative that the use of currents within a specific loop remains consistent after the loop is assigned.

Notice in figure 2.36 that both I_1 and I_2 flow through R_b. Depending on the loop direction, that is, the direction defined by I_1 or I_2, the total current through R_b is either $I_1 - I_2$ or $I_2 - I_1$. Thus if I_1 and I_2 can be found, the current through each circuit element can be determined.

The first task in loop analysis is to use Kirchhoff's voltage law to write equations about each current loop, stating that the sum of voltages about each loop equals zero. For loop 1,

$$0 = -V_1 + R_a I_1 + R_b(I_1 - I_2). \tag{2.51}$$

Notice that $R_a I_1$ equals the voltage drop across R_a, and $R_b(I_1 - I_2)$ equals that across R_b. In the latter case, the voltage can be taken as $R_b I_1 - R_b I_2$, considering that the voltage produced by I_2 opposes that produced by I_1. Likewise, for loop 2,

$$0 = -V_2 + R_c I_2 + R_b(I_2 - I_1). \tag{2.52}$$

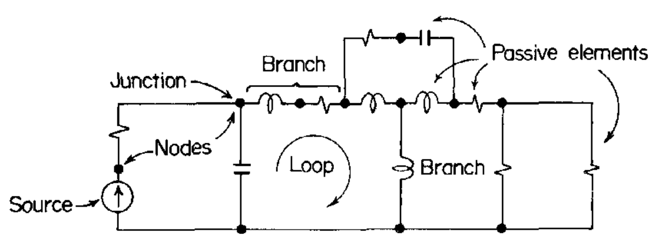


Figure 2.35. – Parts of circuit.

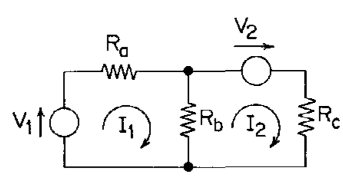


Figure 2.36. – Circuit demonstrating two independent loops.

By rearranging equations 2.51 and 2.52,

$$(R_a + R_b)I_1 - R_bI_2 = V_1 \tag{2.53}$$

and

$$-R_bI_1 + (R_b + R_c)I_2 = V_2, \tag{2.54}$$

which are two simultaneous equations with two unknowns, I_1 and I_2. These can be solved easily by simultaneous methods.

It should be noted that one additional loop equation could be written, that for the loop containing both V_1 and V_2. However, this will not provide another independent equation. Information concerning the maximum number of independent equations available will follow shortly.

EXAMPLE 2.11

Find the current through the 1.5-Ω resistor in figure 2.37 using loop analysis.

SOLUTION. Two loops are defined in the figure where the current through the 1.5-Ω resistor is $I_1 + I_2$. Applying Kirchhoff's voltage law to loop 1,

$$0.5I_1 + 1.5(I_1 + I_2) + 1.0I_1 = 250$$

and for loop 2,

$$0.5I_2 + 1.5(I_1 + I_2) + 1.0I_2 = 300.$$

By simplifying these equations,

$$3I_1 + 1.5I_2 = 250,$$
$$1.5I_1 + 3I_2 = 300.$$

Simultaneous solution of these results is

$$I_1 = 44.4 \text{ A}, \qquad I_2 = 77.8 \text{ A},$$

and the current through the 1.5-Ω resistor is

$$I_1 + I_2 = 122.2 \text{ A}.$$

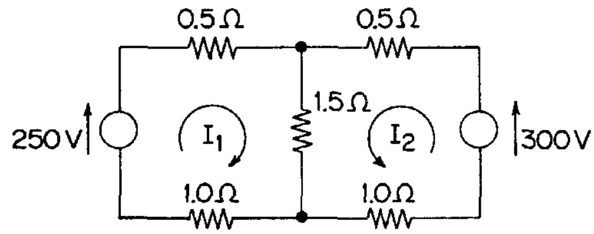


Figure 2.37.—Two-loop circuit for example 2.11.

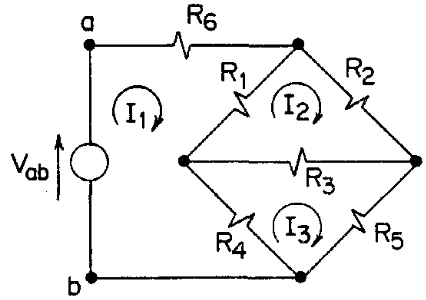


Figure 2.38.—Bridge circuit demonstrating loop analysis.

To further enforce the concept of loop analysis, again consider the common bridge circuit, which is redrawn in figure 2.38 to include current loops. Three loop equations can be written because there are three possible independent loops. For loop 1,

$$R_1(I_1 - I_2) + R_4(I_1 - I_3) + R_6I_1 - V_{ab} = 0; \tag{2.55}$$

for loop 2,

$$R_1(I_2 - I_1) + R_2I_2 + R_3(I_2 - I_3) = 0; \tag{2.56}$$

for loop 3,

$$R_4(I_3 - I_1) + R_3(I_3 - I_2) + R_5I_3 = 0. \tag{2.57}$$

Again, rearranging,

$$(R_1 + R_4 + R_6)\, I_1 - R_1I_2 - R_4I_3 = V_{ab}, \tag{2.58}$$

$$-R_1I_1 + (R_1 + R_2 + R_3)\, I_2 - R_3I_3 = 0; \tag{2.59}$$

$$-R_4I_1 - R_3I_2 + (R_3 + R_4 + R_5)I_3 = 0; \tag{2.60}$$

which are three simultaneous equations with three unknowns, I_1, I_2, and I_3. The proper combination of these currents will yield the current through each branch of the circuit. The process was again to employ Kirchhoff's voltage law for the purpose of finding the unknown currents.

Other loops about the bridge could be assigned and will produce the same valid results. Generally, the particular choice of loops can enhance a desired result. For instance, if only the current through R_3 of figure 2.38 is desired, establishing one loop current through that resistor would create a more direct solution.

EXAMPLE 2.12

Using loop equations, solve for each branch current in the circuit shown in figure 2.39.

SOLUTION. Applying Kirchhoff's voltage law to loops 1 and 2 respectively,

$$2(I_1 - I_2) + 5(I_1 - I_3) + 2I_1 = 56,$$
$$2(I_2 - I_1) + 10I_2 + 1(I_2 + I_3) = 0.$$

As the assigned current for loop 3 passes through an ideal current source,

$$I_3 = 6\ \text{A}.$$

Therefore, the equations for loops 1 and 2 become

$$2(I_1 - I_2) + 5(I_1 + 6) + 2I_1 = 56,$$
$$2(I_2 - I_1) + 10I_2 + 1(I_2 + 6) = 0,$$

or

$$9I_1 - 2I_2 = 26,$$
$$-2I_1 + 13I_2 = -6.$$

Solution of the last two simultaneous equations yields

$$I_1 = 2.9\ \text{A}, \qquad I_2 \cong 0\ \text{A}.$$

Each branch current can now be resolved from the loop currents. For the branch containing the 2-Ω resistor and the 56-V source,

$$I = I_1 = 2.9\ \text{A};$$

for the other 2-Ω resistor,

$$I = I_1 - I_2 \cong 2.9\ \text{A},$$

and the resistors in the other branches,

$$I_{10\Omega} = I_2 \cong 0\ \text{A},$$
$$I_{1\Omega} = I_3 - I_2 \cong 6\ \text{A},$$
$$I_{4\Omega} = I_3 = 6\ \text{A},$$
$$I_{5\Omega} = I_1 + I_3 = 8.9\ \text{A}.$$

A loop equation could have been written for loop 3, but it can only state that the voltage drops across the 1-Ω, 5-Ω, and 4-Ω resistors in that loop are equal to the voltage across the 6-A ideal current source, which is unknown. Such an equation would only complicate the solution to the problem.

As circuits become more complex and the number of possible loops increases, a method for determining the number of required equations is useful. By counting the number of branches and junctions in the circuit, the following expression provides the necessary number of loop currents:

$$\text{number of equations} = \text{branches} - (\text{junctions} - 1).$$

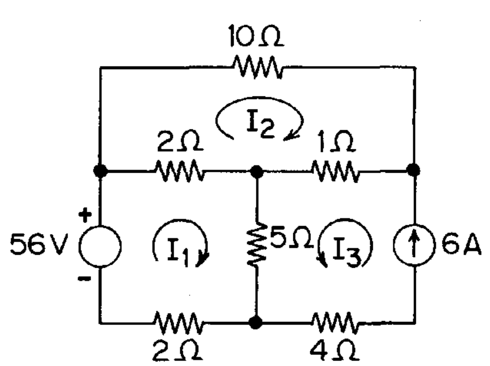


Figure 2.39. – Three-loop circuit for example 2.12.

For figure 2.38, there are six branches and four junctions; therefore, the number of equations needed equals $6 - (4 - 1) = 3$.

Node Equations

In the preceding analysis, Kirchhoff's voltage law established the method of loop equations. Kirchhoff's current law did not receive any attention, yet it was satisfied. This can be demonstrated with figure 2.38 by taking any junction and summing the currents through it. Considering that the currents through R_1 and R_2 flow from junction a′,

$$I_1 - I_{R1} - I_{R2} = 0$$

or

$$I_1 - (I_1 - I_2) - I_2 = 0;$$

hence,

$$I_1 - I_1 + I_2 - I_2 = 0.$$

Kirchhoff's current law is used directly in node analysis, and the unknowns are voltages across branches. The technique by which these voltages are referenced or measured provides a simplifying procedure for a circuit being analyzed. Each junction or principal node in a circuit is assigned a number or letter. Voltages can then be measured from each junction to one specific junction, called the reference node. In essence, the reference node is dependent on all other nodes in the circuit. Node analysis consists of finding the voltages from each junction to the reference node.

The procedure can be demonstrated easily with the simple two-junction circuit shown in figure 2.40, in which I_a, R_b, and R_c are known. The existing junctions are A and O, and O is taken as the reference. The voltage from A to O is then V_{AO}, and Kirchhoff's current law can be used to write an equation for junction A:

$$I_a - I_b - I_c = 0. \qquad (2.61a)$$

By Ohm's law,

$$V_{AO} = I_b R_b$$
$$= I_c R_c.$$

Therefore,

$$I_a - \frac{V_{AO}}{R_b} - \frac{V_{AO}}{R_c} = 0; \qquad (2.61b)$$

since $1/R = G$,

$$I_a - V_{AO}G_b - V_{AO}G_c = 0. \qquad (2.62)$$

Equation 2.62 can be further solved for unknown, V_{AO}.

During the process, an equation was written for each junction, excluding the reference node. The number of required equations for node analysis is therefore always one less than the number of junctions in a circuit.

To illustrate node analysis further, consider the three-junction circuit in figure 2.41. If junction 0 is taken as the reference node, V_{AO} and V_{BO} are the unknown voltages. The reference node, which establishes a reference potential across the bottom of the circuit, is normally assumed at zero potential. Accordingly, the double-subscripted voltages are unnecessary and unknown values can be simply called V_A and V_B. Further, as zero potential is often referenced to earth or ground, a most convenient reference, figure 2.41 can be redrawn as shown in figure 2.42. These circuit elements are still connected to a reference node through the ground symbols, as shown. Hence, each of the circuit elements is said to be grounded.

Now, applying Kirchhoff's current law to junctions A and B,

$$I_a - I_c - I_b = 0, \tag{2.63a}$$

$$-I_e - I_d + I_c = 0. \tag{2.63b}$$

By Ohm's law, $I_b = V_A G_b$, $I_d = V_B G_d$,

and $$I_c = (V_A - V_B)G_c. \tag{2.64}$$

The last expression is evident because, by Kirchhoff's voltage law, the voltage across G_c is the potential at junction A minus that at junction B. Therefore,

$$I_a - (V_A - V_B)G_c - V_A G_b = 0, \tag{2.65}$$

$$-I_e - V_B G_d + (V_A - V_B)G_C = 0, \tag{2.66}$$

or $$(G_b + G_c)V_A - G_c V_B = I_a, \tag{2.67}$$

$$-G_c V_A + (G_c + G_d)V_B = -I_e, \tag{2.68}$$

which are two simultaneous equations with unknowns, V_A and V_B.

The same procedure can be applied to circuits with more nodes. The foregoing examples have shown only current sources that are known, but node analysis can also be used with voltage sources or known voltages. Such is the case with figure 2.43, where the current through G_a is

$$I_a = (V_A - V_B)G_a;$$

likewise, $$I_c = (V_c - V_B)G_c.$$

The analysis procedure can then continue as before. Mixed voltage and current sources can be handled in much the same manner, realizing that the current source establishes the current through the branch in which it is contained.

With both loop and node analysis available, a decision must be made as to which technique best suits the solution of a circuit. Simply, the one to select is that providing the fewest equations to resolve. Since circuit reduction may still lead to the most efficient procedure for some circuits, it should always be considered.

EXAMPLE 2.13

Use node analysis to find the voltage across the 0.5-Ω resistance in figure 2.44.

SOLUTION. The circuit contains three junctions. If the junction at the bottom of the circuit is taken as the reference node, A and B can be considered as the independent junctions. Here, Kirchhoff's current law yields

$$I_A + I_{AB} = 1{,}500 \text{ A},$$
$$I_B - I_{AB} = 1{,}000 \text{ A}.$$

The unknown voltages for node analysis are V_A and V_B, existing between each independent junction and the reference node, where

$$V_A = I_A(1),$$
$$V_B = I_B(2).$$

It can also be noted that

$$V_A - V_B = I_{AB}(0.5).$$

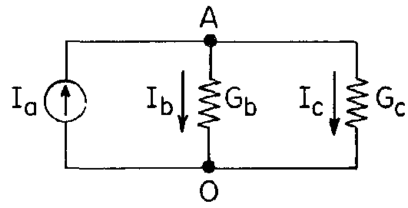


Figure 2.40.—Simple two-node circuit.

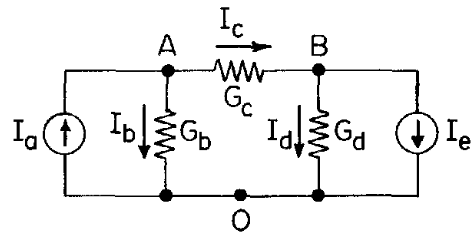


Figure 2.41.—Three-junction circuit.

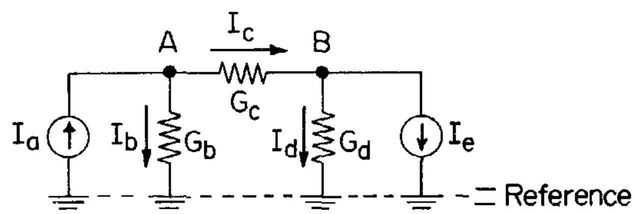


Figure 2.42.—Three-junction circuit with grounds.

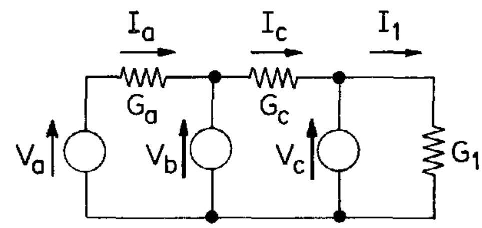


Figure 2.43.—Voltage-source circuit demonstrating node analysis.

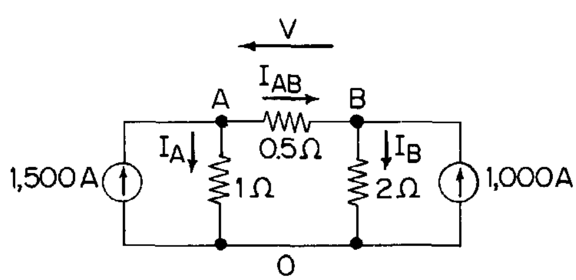


Figure 2.44.—Circuit for examples 2.13, 2.15, and 2.16.

Substituting these Ohm's law relationships into the current-law equations produces

$$1V_A + 2(V_A - V_B) = 1{,}500,$$
$$0.5V_B - 2(V_A - V_B) = 1{,}000.$$

Rearranging, $3V_A - 2V_B = 1{,}500,$
$$-2V_A + 2.5V_B = 1{,}000.$$

Solving these two simultaneous equations gives

$$V_A = 1{,}644 \text{ V}, \quad V_B = 1{,}716 \text{ V}.$$

The voltage across the 0.5-Ωresistance is then

$$V_A - V_B = -72 \text{ V},$$

which means that the actual voltage polarity is the reverse of that used in the solution and shown in the figure.

EXAMPLE 2.14

Find the voltage, V_1, across the 1-Ω resistor in figure 2.45 using node analysis.

SOLUTION. The circuit contains a mixture of current and voltage sources. This presents a difficulty for applying node analysis, as the currents associated with the voltage sources are not known. However, as the objective of node analysis is to find unknown voltages, the difficulty can be eliminated by avoiding the voltage sources in the solution. This can be done by assigning nodes on both sides of each ideal voltage source, treating both nodes and the voltage source together, and applying Kirchhoff's current law to both nodes simultaneously. For instance in figure 2.45, nodes 1 and 2 are on both sides of the 6-V source, and nodes 3 and 4 are associated with the 12-V source. Each voltage source can be considered a short circuit joining its associated nodes, and current flow into the combined source and two nodes equals current leaving the combination. The node-source combinations are often termed *supernodes* and are signified in figure 2.45 by the enclosed dashed lines. Each supernode reduces the number of nonreference nodes by one, thus greatly simplifying the application of node analysis.

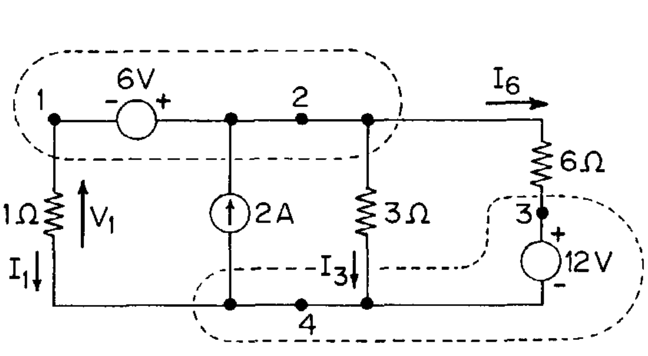


Figure 2.45.—Circuit for example 2.14.

Using this concept for the supernode containing the 6-V source, Kirchhoff's current law gives

$$I_1 + I_3 + I_6 = 2 \text{ A}.$$

Notice that Kirchhoff's current law for the 12-V supernodes produces the same equation. Assigning junction 4 as the reference node, the voltages of the circuit associated with the nonreference nodes 1, 2, and 3 are V_1, V_2, and V_3, where

$$I_1 = \frac{V_1}{1}$$

$$I_3 = \frac{V_2}{3},$$

$$I_6 = \frac{V_2 - V_3}{6}$$

$$V_2 - V_1 = V,$$

and $$V_3 = 12 \text{ V}.$$

Rewriting the current-law equation

$$\frac{V_1}{1} - 2 + \frac{V_2}{3} + \frac{V_2 - V_3}{6} = 0$$

or $$\frac{V_2 - 6}{1} - 2 + \frac{V_2}{3} + \frac{V_2 - 12}{6} = 0$$

or $$V_2 = 4 \text{ V}.$$

Since $$V_2 - V_1 = 6 \text{ V},$$

the voltage across the 1-Ω resistance is

$$V_1 = 4 - 6 = -2 \text{ V},$$

which states that the voltage is in the opposite direction to that shown in figure 2.45.

Network Theorems

Practically any circuit can be analyzed using either circuit reduction, loop equations, or node equations. There are also several theorems that allow the simplification of particular circuits so that these three methods can be applied more easily. The most commonly used theorems are

- Substitution,
- Superposition
- Reciprocity
- Source transformation,
- Maximum power transfer,
- Thevinin's, and
- Norton's.

Substitution has already been used extensively and simply states that equivalent circuits produce equivalent results. The remaining theorems are discussed here.

Superposition

The superposition theorem relates that for a linear, bilateral network with two or more electromotive sources (voltage or current), the response in any element of the circuit is equal to the sum of responses obtained by each source acting separately, with all other sources set equal to zero. Although the word "bilateral" is new, it does not create problems in dc analysis because passive circuits under dc are always bilateral. This concept will be discussed in more detail later.

The meaning of superposition can be illustrated using figure 2.46*A*, a network with two voltage sources. The theorem relates that

1. If one source is set equal to zero (removing it from the circuit) and the currents produced by the other source are found,
2. Then if the second source is set equal to zero and currents caused by the first source are found,
3. By summing both findings, the results are the currents with both sources operating.

In other words, by letting $V_2 = 0$, as in figure 2.46*B*, through circuit reduction,

$$R_{e1} = R_1 + \frac{R_2R_3}{R_2 + R_3}$$

$$I_{1[1]} = \frac{V_1}{R_{e1}},$$

$$I_{3[1]} = (\frac{R_2}{R_2 + R_3})I_{1[1]},$$

$$I_{2[1]} = -(\frac{R_3}{R_2 + R_3})I_{1[1]} .$$

The second part of the double subscripts is used only to signify that the currents are caused by source 1. The negative sign in the last expression is caused by the current direction assumed in the illustration. The next step is letting $V_1 = 0$, thus restoring V_2 (fig. 2.46*C*),

$$R_{e2} = R_2 + \frac{R_1R_3}{R_1 + R_3},$$

$$I_{2[2]} = \frac{V_2}{R_{e2}},$$

$$I_{3[2]} = (\frac{R_1}{R_1 + R_3})I_{2[2]},$$

$$I_{1[2]} = (\frac{R_3}{R_1 + R_3})I_{2[2]} .$$

Finally, the sums of steps 1 and 2 yield

$$I_1 = I_{1[1]} + I_{1[2]}, \quad (2.69)$$

$$I_2 = I_{2[1]} + I_{2[2]} , \quad (2.70)$$

$$I_3 = I_{3[1]} + I_{3[2]} , \quad (2.71)$$

which are the currents with both sources in operation as in figure 2.34*A*. The process is adaptable (and perhaps more useful) for circuits having more than two voltage or current sources. As with current sources in node analysis, the unknowns in each step are voltages. Nevertheless, superposition allows many sources to be considered separately, and it is of great benefit in the analysis of circuits.

EXAMPLE 2.15

Use the superposition theorem to find the voltage across the 0.5-Ω resistance in figure 2.44. Note that this is the same circuit used for example 2.13.

SOLUTION. Following the first step of the superposition theorem, the 1,000-A current source on the right side of the circuit will be turned off. The circuit is now operating as shown in figure 2.47*A*. Only $I_{AB[1]}$ need be known to solve the problem. Using current division for the parallel branches,

$$I_{AB[1]} = 1{,}500(\frac{1}{3.5}) = 429 \text{ A}.$$

Figure 2.47*B* shows the second step in the problem solution, where the 1,500-A source is turned off. Now the current through the 0.5-Ω resistor is

$$I_{AB[2]} = 1{,}000(\frac{2}{3.5}) = -571 \text{ A}.$$

Summation of these two findings produces the current from A to B with both sources operating.

$$I_{AB} = I_{AB[1]} + I_{AB[2]},$$

$$I_{AB} = 429 - 571 = 143 \text{ A}.$$

Thus,

$$V_{AB} = (-143)(0.5) = -72 \text{ V}.$$

It is obvious that this technique produces the answer faster than the process given in example 2.13. However, node analysis may give a more efficient solution with other problems.

Reciprocity

The reciprocity theorem states that in a linear passive circuit, if a single source in one branch produces a given result in a second branch, the identical source in the second branch will produce the same result in the first branch.

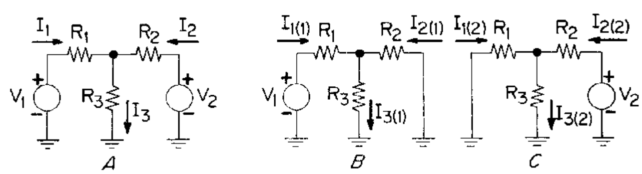


Figure 2.46.—Circuit for demonstrating superposition theorem.

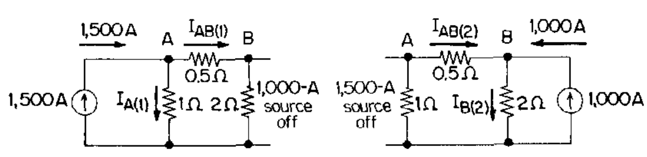


Figure 2.47.—Circuit in figure 2.44 with sources turned off.

This reciprocal action is demonstrated in figure 2.48. In figure 2.48*A*, if V_1 produces I_1 in the branch that goes through R_5, moving V_1 to the R_5 branch will produce I_2 in the original location of V_1 (fig. 2.48*B*). The currents I_1 and I_2 will be equal. The dual form of reciprocity has a similar function in relating a current source to the voltage produced. The great advantage of this theorem is that a source may be moved to another location that is more convenient to analyze.

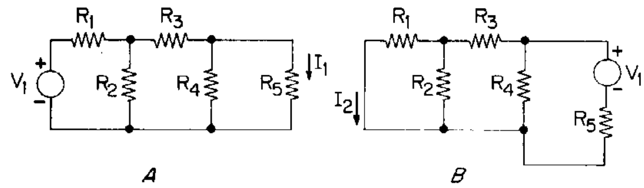


Figure 2.48. – Demonstration of reciprocity theorem.

Source Transformation and Maximum Power Transfer

Before defining the theorems associated with source transformation and maximum power transfer, it is advisable to expand the topics of ideal and practical sources. An ideal voltage source has been defined as a device whose terminal voltage is independent of the current that passes through it. Although no such device exists in the practical world, it is convenient to assume a resistance in series with an ideal source as a datum, against which the performance of an actual voltage source can be measured. This is shown in figure 2.49 where the performance of a 12-V automotive storage battery is plotted against an ideal voltage source. The internal resistance, R_v, compensates the output voltage, V_L, for varying load currents, I_L. These currents are obtained by changing the load, R_L. It will be found that with small current the practical source approximates the ideal one. But under heavy duty where there are high current and low load resistance, the output voltage drops substantially. Using the Ohm and Kirchhoff voltage laws,

$$V_L = (\frac{V_s}{R_v + R_L})R_L, \quad (2.72)$$

$$I_L = \frac{V_s}{R_v + R_L}, \quad (2.73)$$

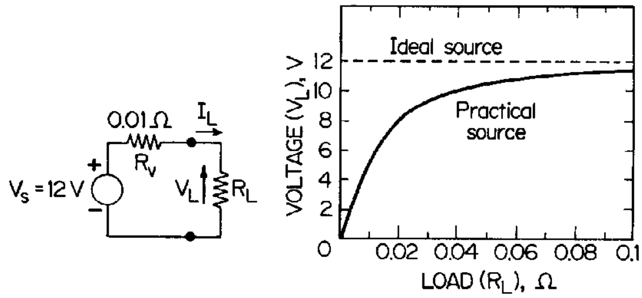


Figure 2.49. – Practical voltage-source model.

V_s equals the voltage of the ideal source, which can be found by measuring the terminal voltage with no load resistance. The internal resistance, R_v, can then be determined by applying a known R_L and measuring V_L.

Similarly, figure 2.50 models a practical current source where R_i is the internal shunt resistance. The graph illustrates the effect of this resistance: as the load resistance increases, terminal current decreases. Using Kirchhoff's current law, it can be shown that

$$V_L = (\frac{R_i R_L}{R_i + R_L})I_s, \quad (2.74)$$

and

$$I_L = (\frac{R_L}{R_i + R_L})I_s\,. \quad (2.75)$$

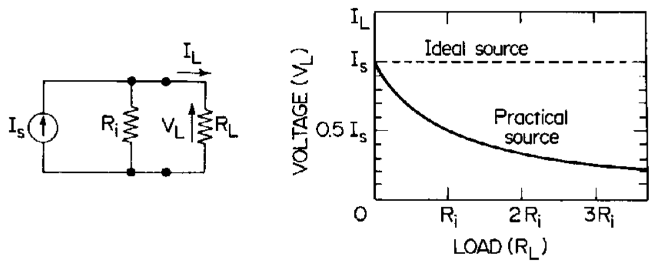


Figure 2.50. – Practical current-source model.

The output of the ideal current source, I_s, can be found by short-circuiting the output terminals and measuring the resulting current. Then R_i can be calculated by measuring V_L and I_L with a known load, R_L. Actually, shorting the terminals of a source is usually unwise because it could damage the real-world source, not to mention being an unsafe practice.

I_s can also be determined through source transformation, which uses the fact that two sources are equivalent if each produces identical terminal voltage and current in any load. Therefore, for equivalence of practical voltage and practical current sources, equations 2.72 and 2.73 must equal 2.74 and 2.75, respectively. It is obvious that both sets are interrelated. In other words, for load current,

$$I_L = \frac{V_s}{R_i + R_L} = \frac{R_i I_s}{R_i + R_L}. \quad (2.76)$$

If equation 2.76 is valid for any load, R_L, it must hold that

$$R_i = R_s, \quad (2.77)$$

$$V_s = R_s I_s, \quad (2.78)$$

where R_s = the internal resistance for either equivalent practical source,
V_s = output voltage of ideal voltage source,
and I_s = output current of ideal current source.

This relationship is shown in figure 2.51. The two circuits shown will be named shortly. Source transformation states that if one source is known, it can be replaced with the other. Note however that even if two practical sources are equivalent, the power that the two internal ideal sources supply and the internal resistances absorb may be quite different. Notwithstanding, this substitution is helpful in writing network equations because constant-current sources are more convenient for node equations, and constant-voltage sources are best for loop equations. In addition, the exchange of particular sources may permit direct circuit reduction.

EXAMPLE 2.16

Solve the problem in example 2.13 using only source transformation.

SOLUTION. Two practical current sources exist in figure 2.44 between junctions A and O and between junctions B and O. Applying equation 2.78 for the left-hand source,

$$I_{sA} = 1{,}500 \text{ A},$$
$$R_{sA} = 1\ \Omega,$$
$$V_{sA} = 1{,}500(1) = 1{,}500 \text{ V};$$

and for the right-hand source,

$$I_{sB} = 1{,}000 \text{ A},$$
$$R_{sB} = 2\ \Omega,$$
$$V_{sB} = (1{,}000)(2) = 2{,}000 \text{ V}.$$

R_{sA} V_{sA} and R_{sB} V_{sB} describe two practical voltage sources that can replace the current sources between junctions A and O and junctions B and O, respectively. Figure 2.52 shows the results of this transformation, where the circuit becomes a simple loop. The current from A to B is now

$$I_{AB} = \frac{1{,}500 - 2{,}000}{3.5} = -143 \text{ A},$$

and the voltage between is

$$V_{AB} = (-143)(0.5) = -72 \text{ V}.$$

Source transformation also produced results quicker than node analysis, but again, this might not occur with other circuit configurations.

In the above solution, practical current sources were replaced by practical voltage sources. By comparing figure 2.44 with figure 2.52, it can be seen that points A, B, and O exist in both. Caution should always be taken to ensure that a desired node is not lost after the transformation.

Since load resistance can vary from zero to infinity, some value of resistance must exist that will receive the maximum power available from a particular source. It can be proven, using the concepts just presented, that an independent voltage source in series with a resistance, R_s, or an independent current source in parallel with a resistance, R_s, delivers maximum power to a load resistance, R_L, when $R_L = R_s$. This is called the maximum power transfer theorem.

Thevenin's and Norton's Theorems

These theorems are closely related to source transformation. They can be illustrated by considering the active network (one that delivers power) with two output terminals shown in figure 2.53. Here, the internal configuration is unimportant, but the elements must be linear. The sources can be either ideal voltage or ideal current.

Thevenin's theorem states that if an active network (fig. 2.53*A*) is attached to any external network (fig. 2.53*B*), it will behave as if it were simply a single ideal voltage source, V_o, in series with a single resistance, R_o (fig. 2.53*C*). In other words, the active circuit will appear as a practical voltage source. Values for V_o and R_o can be found as follows. When all internal sources are operating normally and no loads are connected, the open-circuit voltage across the output terminals equals V_o. With all the ideal sources turned off, a resistance, R_o, can be measured at the terminals. This is because when an ideal current source is turned off, it appears as an open circuit (an infinite resistance). An ideal voltage source that is not operating acts as a short circuit, thus having zero resistance.

This theorem is important because it means that any linear circuit where the internal components are unknown can be considered as a constant-voltage source in series with a resistance. Any circuit reduced to this form is called a Thevenin circuit.

Norton's theorem is the corollary to Thevenin's theorem. Norton relates that if such an active network is attached to any external network, it will behave as a single ideal current source, I_o, in parallel with a single resistance, R_o.

The values for V_o and R_o can be determined by considering the same linear active network, this time as shown in figure 2.54*A*, with internal sources operating normally. The

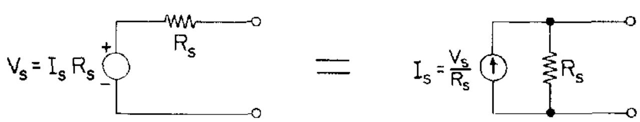


Figure 2.51.—Source transformation.

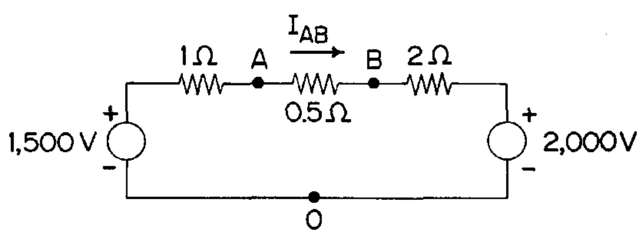


Figure 2.52.—Circuit in figure 2.44 with current sources transformed to voltage sources.

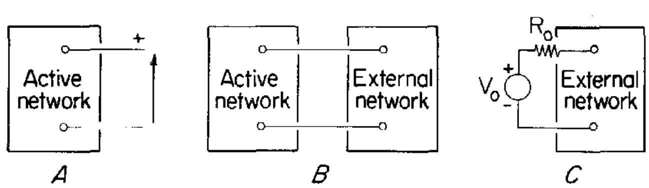


Figure 2.53.—Thévenin's theorem.

output terminals are short-circuited, and a terminal current is measured to give the value for I_o. R_o is found in exactly the same way as in Thevenin's theorem. The combination of these elements gives the practical current source shown in figure 2.54*C*, which is also known as a Norton circuit.

The Thevenin and Norton circuits are obviously related by source transformation so that if one is known, the other can be constructed. The equations relating the two are shown in figure 2.55. These theorems are usually employed when a series of calculations involves changing one part of a network while keeping another part constant. This manipulation helps to simplify complex computations such as power-system short-circuit currents.

EXAMPLE 2.17

Find the Thevenin and Norton equivalents for the circuit shown in figure 2.56.

SOLUTION. Applying Thevenin's theorem, the equivalent resistance of the circuit between a and b with the internal source off is R_o. When the 50-V source is off, it acts as a short circuit, shorting out the 50-Ω resistance in parallel with it. Thus,

$$R = R_{ab} = 2 + \frac{(10)(10)}{10 + 10} = 7\ \Omega.$$

The voltage across a and b with the internal source operating is V_o. Using circuit reduction, the equivalent resistance as seen by the 50-V source with no load across the terminals a and b is

$$R_{eq} = \frac{50(10 + 10)}{50 + 10 + 10} = 14.3\ \Omega.$$

(Note that this resistance is not R_o.) The current delivered by the source is

$$I_1 = \frac{50}{14.3} = 3.5\ \text{A},$$

and from current division,

$$I_2 = (3.5)\,\frac{50}{70} = 2.5\ \text{A}.$$

As no current is flowing between terminals a and b, V_2 shown in figure 2.56 is equal to V_{ab}, which is equal to V_o. Thus,

$$V_o = V_{ab} = V_2 = (2.5)(10) = 25\ \text{V}.$$

V_o and R_o describe the Thevenin equivalent, and I_o and R_o represent the Norton equivalent where

$$I_o = \frac{V_o}{R_o} = \frac{25}{7} = 3.6\ \text{A}.$$

ALTERNATIVE SOLUTION. The definition for R_o in Norton's theorem is the same as in Thevenin's, again,

$$R_o = R_{ab} = 7\ \Omega.$$

However, Norton states that if the terminals a and b are short-circuited, the current through that short circuit is I_o. The short circuit is noted by the dashed line in figure 2.56. Using circuit reduction, the 2-Ω resistance connected to terminal a is in parallel with the 10-Ω resistance connected to terminal b, or

$$\frac{(10)(2)}{10 + 2} = 1.67\ \Omega.$$

The equivalent resistance as seen by the 50-V source is

$$R_{eq} = \frac{(50)(10 + 1.67)}{50 + 10 + 1.67} = 9.46\ \Omega\,,$$

and the current from the source is

$$I_1 = \frac{50}{9.46} = 5.28\ \text{A}.$$

From current division,

$$I_2 = (5.28)\left(\frac{50}{50 + 10 + 1.67}\right) = 4.28\ \text{A},$$

and the current through the shorted terminals is

$$I_o = I_{ab} = (4.28)\left(\frac{10}{10 + 2}\right) = 3.6\ \text{A}.$$

R_o and I_o again describe the Norton equivalent.

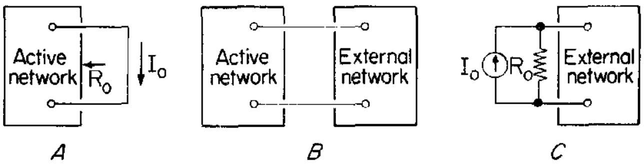


Figure 2.54. – Norton's theorem.

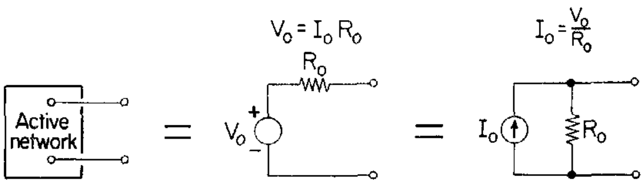


Figure 2.55. – Comparison of Thévenin's and Norton's circuits.

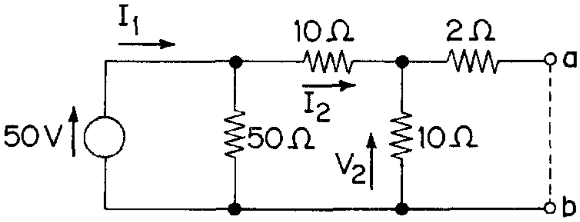


Figure 2.56. – Circuit for example 2.17.

EXAMPLE 2.18

Determine the Thevenin's and Norton's equivalents for the circuit in figure 2.57.

SOLUTION. In the branch containing the 900-V source, the two 5-Ω resistances are in series. If these are combined into one 10-Ω resistance, it should be quite obvious that two practical voltage sources exist between junction 1 and the junction connected to terminal b. Source transformation can be employed to solve the problem. The resistance and magnitude of the ideal current source of the Norton equivalent to the 900-V and 10-Ω source are

$$R_1 = 10\ \Omega,$$

$$I_1 = \frac{900}{10} = 90\ \text{A}.$$

For the Norton equivalent of the 2,250-V and 15-Ω source,

$$R_2 = 15\ \Omega,$$

$$I_2 = \frac{2{,}250}{15} = 150\ \text{A}.$$

Figure 2.58*A* shows the voltage sources transformed to practical current sources. Notice that junction 1 and the junction associated with terminal b still exist. Between these two terminals, the 90- and 150-A sources are operating in parallel, and the 10- and 15-Ω resistances are connected in parallel. Combining these ideal current sources and resistance results in the circuit of figure 2.58*B*. Again, notice that the aforementioned junctions are retained. Converting the 60-A and 6-Ω current source to its Thevenin equivalent produces the circuit in figure 2.58*C*. The 6- and 4-Ω resistances in series with the 360-V are combined in figure 2.58*D*. The 360-V source and 10-Ω resistance form a practical voltage source between terminals a and b, and this is converted to its Norton equivalent in figure 2.58*E*. Here, simple combination of the two parallel 10-Ω resistances yields one answer to the original problem and is shown in figure 2.58*F*. The remaining answer, the Thevenin equivalent, is in figure 2.58*G*, obtained by source transformation of figure 2.58*F*.

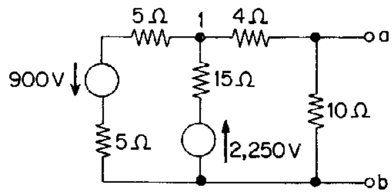


Figure 2.57. – Active circuit for example 2.18.

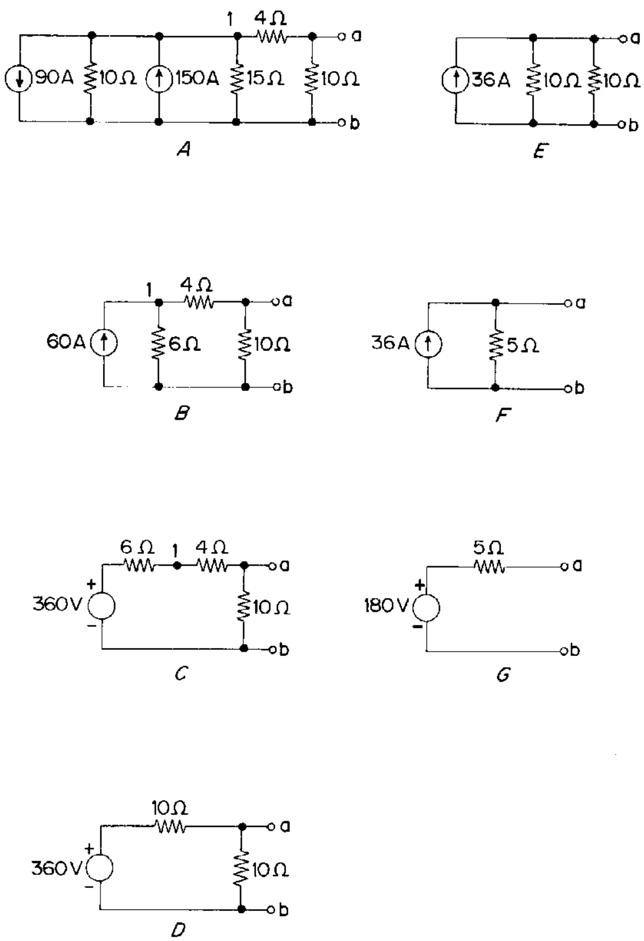


Figure 2.58. – Circuits illustrating solution steps to example 2.18.

To summarize the preceding sections, the fundamental laws and parameters were first applied to circuits under the influence of dc. Expanding upon these laws, several circuit-analysis techniques and theorems were covered. Because only dc was considered, resistance was the only circuit element of interest. As will be shown shortly, most of this theory is also valid for circuits acting under current forms other than dc, where inductance and capacitance may also enter into the picture.

TIME-VARYING VOLTAGES AND CURRENTS

As the name implies, the magnitude of time-varying voltages and currents may not be constant with time. Consequently, the instantaneous values of the voltage and current waveforms, v and i, must be considered. Both v and i are functions of time, as they were when originally introduced in this chapter, and they can assume any form from constant to the most complex. Figure 2.59 presents just a minor sampling of time-varying waveforms to illustrate their general characteristics.

As with dc, the method for analyzing circuits that have time-varying current and voltage is first to form a model of the circuit, then to apply the fundamental laws and relationships. Unlike dc circuits, a differential equation usually results. To demonstrate the effect of time-varying and current on circuit elements, this section will first consider a special waveform, steady alternating current (ac).

An example of a steady-state ac waveform is provided in figure 2.60. The repetitive nature of this sinusoidal function can be expressed mathematically as

$$i = I_m \cos(\omega t) \qquad (2.79)$$

where i = current at any time, t,
I_m = crest or maximum value of current, a constant
ω = radian frequency, rad/s.

The term *sinusoid* or *sine wave* is used collectively to include cosinusoidal or cosine-wave expressions. The above equation could also use a sine function, but the cosine is employed for convenience when referring to current.

It can be seen in figure 2.60 and equation 2.79 that the instantaneous value of current repeats itself every 2π rad or 360°; that is, the waveform goes through one complete cycle every 2π rad. The number of cycles per second is $\omega/2\pi$ which is defined as the frequency, f, of the waveform or

$$f = \frac{\omega}{2\pi} \qquad (2.80)$$

or

$$\omega = 2\pi f. \qquad (2.81)$$

The units of frequency are hertz (Hz). One hertz is equal to 1 cycle-per-second (cps), an expression whose use is now obsolete. The common power frequency in the United States is 60Hz, for which ω = 377 rad/s, or just simply ω = 377.

A more general form of ac is

$$i = I_m \cos(\omega t + \theta), \qquad (2.82)$$

where θ = phase angle.

Instead of expressing the phase angle in radians, such as $\pi/6$, angular degrees, 30°, are customarily used. By adjusting θ, the sinusoid can be moved left (increasing θ) or right (decreasing θ. Such movement is illustrated in figure 2.61.

Using the earlier technique of developing differential equations through circuit analysis, steady ac can be applied to pure resistance, inductance, and capacitance to observe what happens.

Alternating Current Through Resistance

Figure 2.62*A* shows a resistor of resistance R. From equation 2.79, if the current through this element is

$$i = I_m \cos(\omega t),$$

by Ohm's law, the voltage developed across the resistor is

$$v = Ri$$

or

$$v = R(I_m \cos(\omega t)) = V_m \cos(\omega t), \qquad (2.83)$$

where $V_m = RI_m$ = maximum or crest value of voltage waveform, V.

Figure 2.62*B* shows both voltage and current as functions of time. At every instant, v is proportional to i, and v and i are said to be in phase. When two sinusoidal waves are compared for phase in this manner, both must be sine waves or cosine waves; both must be expressed with positive amplitude and have the same frequency.

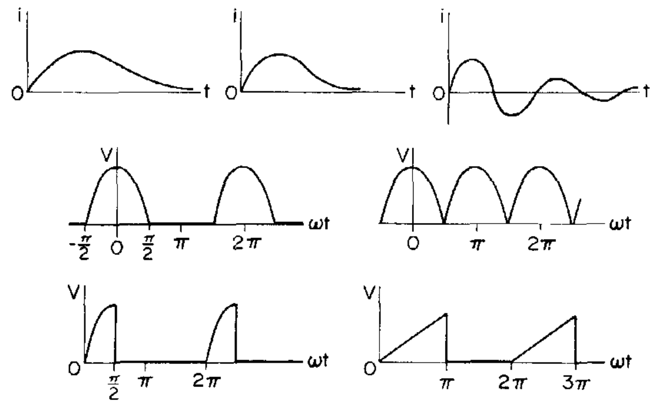


Figure 2.59.—Some time-varying electrical waves.

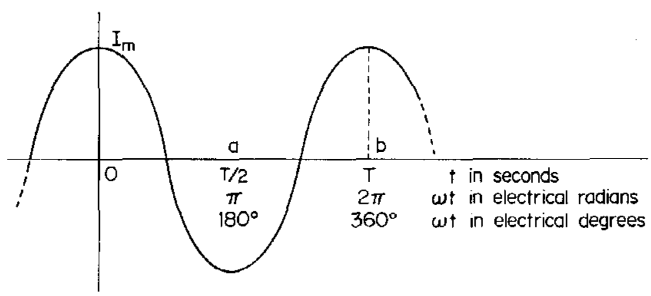


Figure 2.60.—Sinusoidal ac waveform.

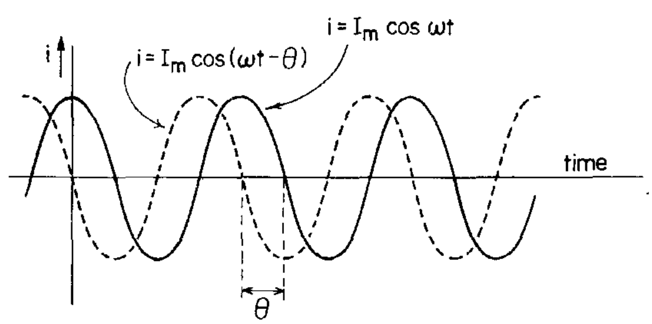


Figure 2.61.—Steady ac showing phase shift.

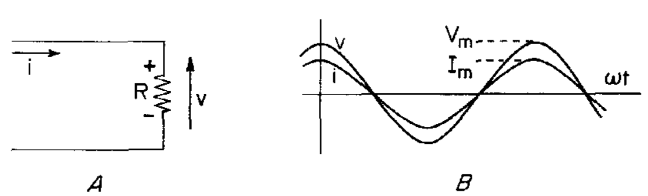


Figure 2.62.—Steady ac through resistance.

Alternating Current Through Inductance

Suppose that current through the pure inductance of figure 2.63*A* is again as in equation 2.79. The voltage across the element is

$$v = L\frac{di}{dt}$$

or
$$v = L\frac{d}{dt}(I_m\cos(\omega t))$$

or
$$v = LI_m\frac{d\cos(\omega t)}{dt}.$$

Differentiating, $v = -\omega LI_m\sin(\omega t)$

or $$v = \omega LI_m\cos(\omega t + 90°) = V_m\cos(\omega t + 90°), \quad (2.84)$$

where $V_m = \omega LI_m$ = maximum or crest voltage.

The term ωL is used so frequently that it is provided with a special name, *inductive reactance*, and is designated "X," where

$$X = \omega L = 2\pi fL \quad (2.85)$$

and
$$V_m = I_mX. \quad (2.86)$$

Figure 2.63*B* compares equations 2.79 and 2.84, with i and v as functions of time. Here, it can be seen that the current crest is reached at a later time than the crest voltage. The current waveform is said to lag the voltage waveform by 90°. The phase angle is called *lagging*.

Alternating Current Through Capacitance

Consider the capacitance shown in figure 2.64*A*, and let the voltage across it be

$$v = V_m\cos(\omega t). \quad (2.87)$$

The current through the capacitor is then

$$i = C\frac{dv}{dt}$$

or
$$i = C\frac{d}{dt}(V_m\cos(\omega t)).$$

Differentiating, $i = -\omega CV_m\sin(\omega t)$

or $$i = \omega CV_m\cos(\omega t + 90°) = I_m\cos(\omega t + 90°), \quad (2.88)$$

where $I_m = \omega CV_m$ = maximum or crest current through the capacitor.

As with the inductive resistance, ωC is also provided a special name, *capacitive susceptance*, and symbol, "B." Thus,

$$B = \omega C = 2\pi fC \quad (2.89)$$

and
$$I_m = BV_m. \quad (2.90)$$

The relationship between the current and voltage waveforms (fig. 2.64*B*) is the reverse of the inductance situation; the current waveform is now leading the voltage waveform

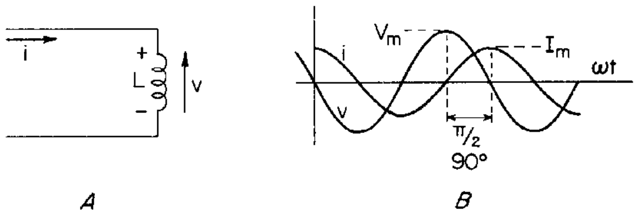


Figure 2.63.—Steady ac through inductance.

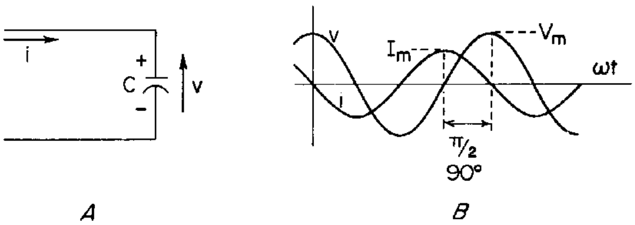


Figure 2.64.—Steady ac through capacitance.

by 90°. The phase angle is also called leading. The importance of current and voltage waveforms being compared for lagging and leading phase angles will be brought out later in this and the next two chapters.

Time-Varying Equations

The preceding discussion considered voltage and current to be steady sinusoids. But what if they are allowed to have any form? To illustrate the consequences, the fundamental laws and parameters can be applied to the simple series RL, RC, and RLC circuits shown in figures 2.65, 2.66, and 2.67, respectively.

For the series RL circuit, using Kirchhoff's voltage law

$$v = v_R + v_L.$$

Substituting in the relationships for voltages across resistance and inductance,

$$v = iR + L\frac{di}{dt}. \quad (2.91)$$

Now for the series RC circuits,

$$v = v_R + v_C.$$

Applying the elementary laws,

$$v = iR + \frac{1}{C}\int_0^t i\,dt + V_0. \quad (2.92)$$

The differential equations 2.91 and 2.92 are valid for any voltage and current, no matter what form. As before, V_0 is the initial charge on the capacitance.

Considering figure 2.67, which shows the series RLC combination,

$$v = v_R + v_L + v_C;$$

thus,
$$v = iR + L\frac{di}{dt} + \frac{1}{C}\int_0^t i\,dt + V_0. \quad (2.93)$$

To arrive at an equation that is easier to handle mathematically, equation 2.93 can be differentiated once:

$$\frac{dv}{dt} = R\frac{di}{dt} + L\frac{d^2i}{dt^2} + \frac{i}{C} + 0. \qquad (2.94)$$

This equation again describes or models the circuit for all electrical situations, as no restrictions have been placed on voltage and current.

The preceding has shown that when voltages and currents represent any form, the application of circuit relationships results in a differential equation. Through classical differential-equation methods, such equations can provide the required solution, but these techniques will not be shown because it can confuse the understanding of the vital aspects of electrical fundamental methods.

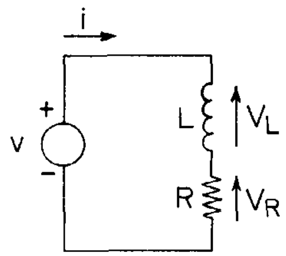


Figure 2.65.—Simple series RL circuit.

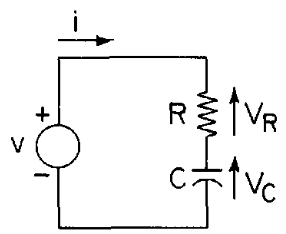


Figure 2.66.—Simple series RC circuit.

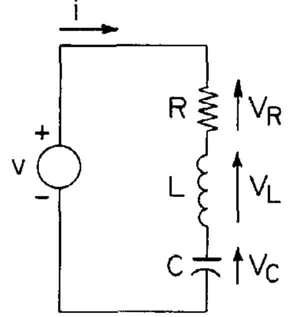


Figure 2.67.—Simple series RLC circuit.

Transients and Circuit Response

Solution of these equations for all situations yields the complete response of the circuit. For linear circuits, the solution will have two parts: forced response and natural response. The forced or steady-state response can be attributed directly to the applied source or forcing function. This is the action of voltage and current within the circuit if no changes or disturbances are made. The natural or transient response is a characteristic of the circuit only, not a result of the sources. Such action occurs when a circuit is disturbed by a change in the applied sources or in one of the circuit elements. After the change, the circuit currents and voltages undergo transition from their original state to the point where their action is again steady state. The time period involved is normally very short, and the occurrence within the transition is called a transient.

For simplicity, the forcing functions mentioned earlier in this chapter were dc, and in network analysis the study was devoted only to resistive circuits and dc sources because here only the forced response is present. When both inductance and capacitance are circuit elements, both forced and transient responses can be encountered. However, knowledge of circuit transients is not required when considering steady-state voltages and currents, as was seen in the case of steady ac. By far the majority of mine power problems only require knowledge of steady-state circuit currents and voltages, and it will be shown that even though inductance and capacitance might be present, as long as only the steady-state response is considered the solution of differential equations is not needed. However, transient circuit responses are an extremely important input in the design of mine power systems, and they will be explained in detail in chapter 11.

It has been shown in this section that any resistor, inductor, or capacitor carrying a sinusoidal current has a sinusoidal voltage developed across it. Furthermore, the sum or difference of two sinusoidal waveforms with the same frequency is another sinusoid. From these concepts, it can be shown that for a steady-state circuit, if voltage or current at any part of a linear circuit is sinusoidal (alternating at a particular frequency), voltages and currents in every part of the circuit are sinusoidal with the same frequency.

STEADY ALTERNATING CURRENT

The form of steady ac has already been shown and used in the analysis of simple ac circuits, but here the concepts of steady-state ac circuit analysis are introduced. This necessitates a review of a familiar but easily forgotten subject, complex algebra.

Real numbers such as 2, 4, and π are easy to understand in terms of physical things. Any mathematical operation on these numbers always results in another real number, except when the square root of a negative real number is taken. The term $\sqrt{-1}$ cannot be satisfied by any real number. Therefore, the square root of any negative number is called an imaginary number. Mathematicians distinguish imaginary numbers by writing "i" in front of them, but to avoid confusion with the symbol for current, electrical engineers use the symbol "j" where

$$j = \sqrt{-1}.$$

Addition or subtraction of imaginary numbers yields another imaginary number. Yet, when an imaginary number is added to a real number, a complex number is created. These have the rectangular form, x + jy (for instance, 3 + j4), where x is the real part and y the imaginary part or if

$$Z = x + jy,$$

then $$\mathrm{Re}[Z] = x \qquad \mathrm{Im}[Z] = y. \tag{2.95}$$

Complex numbers can be represented graphically by a pair of perpendicular axes as shown in figure 2.68. The horizontal axis is for real quantities, the vertical one for imaginary. Considering x + jy, if y = 0, the complex number is a pure real number and falls somewhere on the real axis. Similarly, if x = 0, the complex number (now being purely imaginary) exists on the vertical axis. Hence, complex numbers encompass all real and all imaginary numbers.

In the case of the rectangular forms

$$Z = x + jy,$$
$$W = u + jy,$$

the following common definitions and mathematical operations of complex algebra are applied.

1. Two complex numbers are equal if and only if the real components are equal and the imaginary components are equal:

$$Z = W, \text{ IFF } x = u, y = v.$$

2. To sum two complex numbers, the real and imaginary parts are summed separately:

$$Z \pm W = (x \pm u) + j(y \pm v).$$

3. The product of a real and an imaginary number is imaginary:

$$x(jy) = j(xy).$$

4. The product of two imaginary numbers is a negative real number:

$$(jy)(jv) = -yv.$$

5. The multiplication of two complex numbers follows the rules of algebra (note, an easier way to perform the multiplication will be shown):

$$(x+jy)(u+jv) = xu + jxv + juy - yv$$
$$= (xu - yv) + j(xv + uy).$$

6. By definition, the conjugate of a complex number is formed by changing the sign of the imaginary part. An asterisk denotes the conjugate:

$$Z = x + jy \qquad \text{becomes} \qquad Z^* = x - jy.$$

7. For division, the numerator and denominator are multiplied by the conjugate of the denominator (again, an easier method exists):

$$\frac{x + jy}{u - jv} = \frac{x + jy}{u - jv}\frac{(u + jv)}{(u + jv)} = \left(\frac{xu - yv}{u^2 + v^2}\right) + j\left(\frac{uy + xv}{u^2 + v^2}\right).$$

Besides the rectangular, there are three other general forms of complex numbers: trigonometric, polar, and exponential. Figure 2.69 illustrates the conversion of rectangular to trigonometric or polar forms where

$$Z = x + jy. \tag{2.96}$$

The absolute value of Z is represented by "r," and

$$x = r\cos\theta,$$
$$y = r\sin\theta,$$

where $$\theta = \tan^{-1}\left(\frac{y}{x}\right),$$
$$r = (x^2 + y^2)^{1/2}. \tag{2.97}$$

Hence, the trigonometrical form of the complex number is

$$Z = r(\cos\theta + j\sin\theta), \tag{2.98a}$$

with the conjugate

$$Z^* = r(\cos\theta - j\sin\theta). \tag{2.98b}$$

The polar form is widely used in circuit analysis and is simply written as

$$Z = r\underline{|\theta} \tag{2.99a}$$

and the conjugate,

$$Z^* = r\underline{|-\theta} \tag{2.99b}$$

Euler's theorem states that

$$\cos\theta + j\sin\theta = e^{j\theta}. \tag{2.100}$$

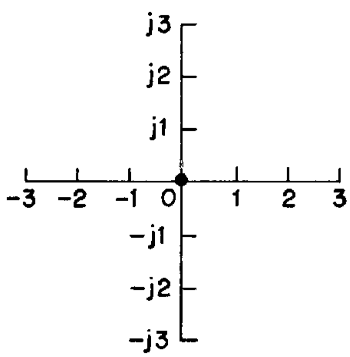


Figure 2.68.—Graphical representation of complex number.

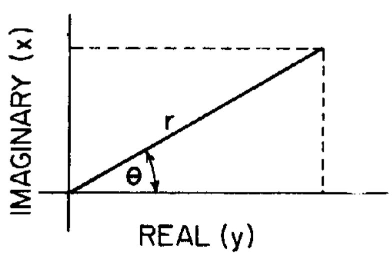


Figure 2.69.—Trigonometric or polar representation of complex number.

This expression allows a complex number to be written as an exponent, the exponential form,

$$Z = r(\cos\theta + j\sin\theta) = re^{j\theta} \qquad (2.100a)$$

and

$$Z^* = re^{-j\theta}. \qquad (2.100b)$$

All four complex forms are therefore identical or

$$Z = x + jy = r(\cos\theta + j\sin\theta) = r\underline{|\theta} = re^{j\theta}.$$

The form should be selected that gives the easiest mathematical manipulation of complex numbers. For addition or subtraction, the rectangular expression is best, but multiplication and division are much more convenient when the number is in exponential or polar form, the latter being the most used. For instance, in polar,

$$Z_1Z_2 = r_1\underline{|\theta}\, r_2\underline{|\phi} = r_1r_2\underline{|\theta + \phi},$$

$$\frac{Z_1}{Z_2} = \frac{r_1\underline{|\phi}}{r_2\underline{|\phi}} = \frac{r_1}{r_2}\underline{|\theta - \phi},$$

and in exponential,

$$Z_1Z_2 = (r_1e^{j\theta})(r_2e^{j\phi}) = r_1r_2e^{j(\theta-\phi)},$$

$$\frac{Z_1}{Z_2} = \frac{r_1e^{j\theta}}{r_2e^{j\phi}} = \frac{r_1}{r_2}e^{j(\theta-\phi)}.$$

It will be shown shortly that circuits containing resistance, inductance, and capacitance can be represented by complex numbers, and that the solution of these circuits under steady ac will use complex algebra. This can be done with almost as much ease as the dc circuit analysis presented earlier.

EXAMPLE 2.19

Find the answer to the following expression in polar and rectangular form:

$$\frac{(2 + j6)(18\underline{|21°})}{(1.63j)(2.6 + j1)}.$$

SOLUTION. Both the numerator and denominator of the above expression are multiplications of complex quantities. For ease of solution, the rectangular term should be converted to polar. This results in

$$\frac{(6.32\underline{|71.6°})(18\underline{|21°})}{(1.63\underline{|90°})(2.79\underline{|21°})}$$

or

$$\frac{(6.32)(18)}{(1.63)(2.79)}\underline{|71.6° + 21° - 90° - 21°}$$

or

$$25\underline{|-18.4°}.$$

Effective Alternating Current

The power available in the outlets of U.S. homes is a very familiar quantity: it is sinusoidal, having a frequency of 60 Hz and a voltage of 115 V. But what does 115 V actually stand for?

The voltage waveform, being a sinusoid, is not constant with time. Therefore, the voltage is certainly not instantaneous. If a measuring device could be connected to an outlet in order to visually observe the waveform, it would be found that "voltage" is not the maximum value, V_m, because this waveform crest is $115\sqrt{2}$ or 162.6 V. "Voltage" does not describe an average value either, because the average of a sine wave is identically zero. As another resort, the average throughout one positive or one negative half-cycle of the waveform could be calculated, but the result gives a measurement of 0.637 V_m or 103.5 V. To discover the meaning of the term *voltage,* the reason for measuring the voltage must be considered. In any system, current and voltage are defined in terms of what they will do. Consequently, the voltage is the effective value of the sinusoidal waveform. It is a measure of the effectiveness of the voltage source in delivering power to a resistive load. The effective value is called root-mean-square (rms).

In order to understand rms measurements, it is necessary to return to the concept of instantaneous power, where

$$p = vi.$$

If the power was being developed across a resistance, R, it was shown that

$$p = i^2R$$

and

$$p = \frac{v^2}{R}.$$

These equations have little practical value for ac as they represent the value of power for a particular instant and in ac this is ever changing. A more effective measure for the value of power is based on the fact that power is the rate of doing work. A reasonable measure is then the average rate or average power. For average power, P, consumed by the resistance, R,

$$P = \text{ave}(p) = \text{ave}(i^2R) = (\text{ave } i^2)R$$

and

$$P = \text{ave}\,\frac{v^2}{R} = \frac{(\text{ave } v^2)}{R}.$$

Average power is then an effective way to measure or quantify ac voltage and current. It has already been seen that the units of voltage and current in dc are easy to comprehend; the magnitudes are constant with time, and their ability to deliver power is constant. Therefore, it is appropriate to equate ac and dc rates of work, P_{ac} and P_{dc}, respectively, in order to determine an effective measurement for alternating voltages and currents:

$$P_{dc} = I^2R = P_{ac} = (\text{ave } i^2)R$$

or

$$I^2 = (\text{ave } i^2)$$

or

$$I = \sqrt{(\text{ave } i^2)} = \text{rms current.} \qquad (2.101a)$$

Employing the same procedure,

$$V = \sqrt{(\text{ave } v^2)} = \text{rms voltage.} \qquad (2.101b)$$

Current and voltage in ac are therefore expressed as the square root of the mean-square values or rms. They are sometimes written I_{rms} and V_{rms}. It can be shown from the voltage and current waveforms (that is, substituting

$I_m\cos(\omega t + \theta)$ into equation 2.101*a* and similarly for voltage) that

$$I_{rms} = \frac{I_m}{\sqrt{2}} \text{ or } I_m = \sqrt{2}\, I_{rms} \qquad (2.102a)$$

and

$$V_{rms} = \frac{V_m}{\sqrt{2}} \text{ or } V_m = \sqrt{2}\, V_{rms}. \qquad (2.102b)$$

Root-mean-square currents and voltages are used so often that they are directly implied when referring to an ac magnitude. They are almost always used in calculations. For simplicity, the subscripts of V_{rms} and I_{rms} are eliminated in practice, and just V and I are written to indicate rms voltages and currents. All common ac voltmeters and ammeters are also calibrated to read rms values.

The preceding analysis of average power concepts applies only to resistance. Average power in the steady state supplied to either a theoretically pure inductance or pure capacitance is identically zero. This can be proved by integrating instantaneous power to these elements to obtain an average. The results show that the energy received during one-half cycle is stored and then transferred back to the source through the balance of the cycle. The stored energy in the capacitance is greatest at the maximum of the voltage wave, while in the inductance it is maximum at the current-wave crest.

Phasors

A steady-state sinusoidal current or voltage at a given frequency is characterized by only two parameters: amplitude and phase angle. This can be seen in figure 2.70*A*, which shows two voltage waveforms separated by a phase angle. An ac quantity may also be represented graphically by a phasor, illustrated in figure 2.70*B*. The phasor is a continually rotating line that shows magnitude and direction (time). In this figure, the phasor is assumed to have a length representative of V_m, rotation about point 0, and an angle increasing with time according to $\theta_t = \omega t + \theta$. The figure shows the line as if a snapshot had been taken, freezing action. The alternating quantity, $V_m\cos(\omega t + \theta)$, is the projection of the phasor on the horizontal axis. In other words, as the phasor in figure 2.70*B* rotates, a plot of its projection on the horizontal axis with time reproduces the waveform in figure 2.70*A*. The phasor length shown here represents crest voltage but does not necessarily need to be equal to it. It is common practice to draw phasors in terms of effective (rms) values. Although voltage has been employed as an example, phasors can also represent sinusoidal current, among other things.

Voltage and current phasors are both illustrated as rotating lines in figure 2.71*A*, where

$$v = V_m\cos(\omega t + \phi),$$

$$i = I_m\cos \omega t.$$

To show both current and voltage, two phasors can be drawn, with one of them advanced by the phase angle, ϕ. Both lines rotate indefinitely about the axes, and one line will always lead the other in the same relative position; therefore, the axes are superfluous and need not be drawn. Since it is necessary to orient the phasors at a specific point in time, a convenient instant is selected as a reference. For example, in figure 2.71*B* the phasor is shown where the current phasor angle is zero. Here, the current phasor is termed a reference phasor, and all other phasors are drawn relative to it. Either voltage or current can be selected as the reference.

A phasor may be expressed in several ways. To illustrate the most used expressions, consider figure 2.72*A*, which shows one phasor displaced from the horizontal by an angle, $\omega t + \theta$. Recalling complex algebra, the horizontal axis can be assigned as a real axis and the vertical as the imaginary axis. The phasor, $\overline{V}$, is then the sum of the real and imaginary components,

$$\overline{V}_{re} \text{ and } \overline{V}_{im},$$

or

$$\overline{V} = \overline{V}_{re} + V_{im}. \qquad (2.103)$$

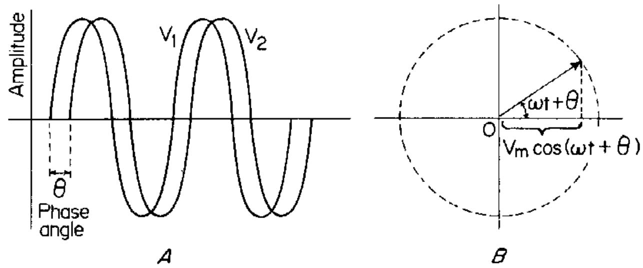


Figure 2.70.—Sinusoid versus time (*A*) and as phasor (*B*).

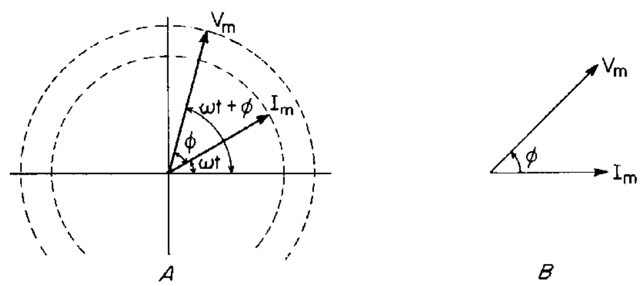


Figure 2.71.—Phasor representation of current (*A*) and voltage (*B*).

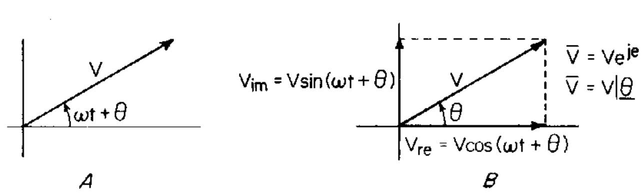


Figure 2.72.—Other expressions for phasors.

Figure 2.72*B* clearly illustrates the rectilinear form of equation 2.103. The real and imaginary components of the phasor are

$$\overline{V}_{re} = V\cos(\omega t + \theta), \qquad (2.104a)$$

$$\overline{V}_{im} = jV\sin(\omega t + \theta), \qquad (2.104b)$$

Thus $$\overline{V} = V\cos(\omega t + \theta) + jV\sin(\omega t + \theta) \qquad (2.104c)$$

or $$\overline{V} = V(\cos(\omega t + \theta) + j\sin(\omega t + \theta)). \qquad (2.104d)$$

$\overline{V}$ is used to signify that the voltage is a phasor, and once more the imaginary operator, j, signifies that $\overline{V}_{im}$ exists on the imaginary axis. Accordingly, the phasor may be considered as the vector sum of two phasors at right angles to each other. Applying Euler's theorem (equation 2.100) to equation 2.104*d*,

$$\overline{V} = Ve^{j(\omega t + \theta)} \qquad (2.105a)$$

or $$\overline{V} = Ve^{j\omega t}e^{j\theta}. \qquad (2.105b)$$

The factor, $e^{j\omega t}$, is superfluous, as it contains no unique information about the phasor, and it can be suppressed:

$$\overline{V} = Ve^{j\theta}. \qquad (2.106)$$

This is called the exponential form of the phasor. Thus equation 2.105*c* can be expressed in polar form,

$$V = \overline{V}\,\underline{|\theta}. \qquad (2.107)$$

These phasor forms are very useful in solving ac circuit problems. The terms *phasor* and *vector* are often interchanged.

Phasors and Complex Quantities

When introducing the action of time-varying sinusoids, certain voltage-current phase-angle relationships were found to exist for pure resistive, inductive, and capacitive circuit elements. In general, if a steady-state sinusoidal current has the time-domain form

$$I = I_m\cos(\omega t + \theta), \qquad (2.108a)$$

and voltage,

$$v = V_m\cos(\omega t + \phi), \qquad (2.108b)$$

current is said to be lagging voltage by the phase angle, $\phi - \theta$ (or conversely, leading voltage by the phase angle, $\theta - \phi$). Using the exponential and polar phasors, this current and voltage can also be stated

$$\overline{I} = Ie^{j(\omega t + \theta)},\ \overline{I} = Ie^{j\theta},\ \text{or}\ \overline{I} = I\underline{|\theta} \qquad (2.109a)$$

and $$\overline{V} = Ve^{j(\omega t + \phi)},\ \overline{V} = Ve^{j\phi},\ \text{or}\ \overline{V} = I\underline{|\phi} \qquad (2.109b)$$

where I and V = rms values of current and voltage, respectively.

Before steady-state circuit analysis can be performed, pure circuit elements must again be considered, this time to analyze the voltage-current relationships using complex quantities. Equations 2.108*a* and 2.108*b* are assumed to represent the general current through and voltage across each element.

EXAMPLE 2.20

A circuit has the following voltage and current waveforms applied across and through its terminals:

$$v = 282.8\cos(377t - 20°),$$
$$i = 42.4\cos(377t + 25°).$$

Write the phasor expression for voltage and current. What is the phase angle between current and voltage?

SOLUTION. The two given expressions are in the time domain, where for the voltage,

$$V_m = 282.8\ \text{V},$$
$$\phi = -20°,$$

and for the current,

$$I_m = 42.4\ \text{A},$$
$$\theta = 25°.$$

The phasors for voltage and current are then, respectively,

$$\overline{V} = \frac{V_m}{\sqrt{2}}\,\underline{|\phi} = 200\,\underline{|-20°}\ \text{V},$$
$$\overline{I} = \frac{I_m}{\sqrt{2}}\,\underline{|\theta} = 30\,\underline{|25°}\ \text{A}.$$

The current waveform is leading the voltage waveform, and the phase angle between current and voltage is

$$\phi - \theta = -20° - 25° = -45°.$$

If sinusoid current is applied to a resistance, R, the voltage across it is

$$v = Ri.$$

Applying the general time-domain expressions,

$$V_m\cos(\omega t + \phi) = RI_m\cos(\omega t + \theta)$$

or in exponential form,

$$Ve^{j(\omega t + \phi)} = RIe^{j(\omega t + \theta)}.$$

Suppressing $e^{j\omega t}$,

$$Ve^{j\phi} = RIe^{j\theta},$$

or in polar form,

$$V\,\underline{|\phi} = RI\,\underline{|\theta}.$$

In phasor form, $V \angle\phi$ and $I \angle\theta$ are the phasor polar representations,

$$\bar{V} = R\bar{I}. \tag{2.110}$$

This is the same relationship that exists for time-varying waveforms and dc. It is apparent that angles θ and ϕ are equal and that voltage and current are in phase (fig. 2.73*A*).

Suppose the same general forms of current and voltages were applied to a pure inductance where, as before,

$$v = L\frac{di}{dt},$$

then, using the general exponentials,

$$Ve^{j(\omega t+\phi)} = L\frac{d}{dt}(Ie^{j(\omega t+\theta)}).$$

Differentiating ($e^{j\theta}$ is a constant with time),

$$Ve^{j(\omega t+\phi)} = j\omega LIe^{j(\omega t+\theta)},$$

and suppressing $e^{j\omega t}$,

$$Ve^{j\phi} = j\omega LIe^{j\theta}.$$

Thus, in phasor form,

$$\bar{V} = j\omega L\bar{I}. \tag{2.111}$$

The imaginary operator, j, denotes a +90° displacement of voltage from current; such as illustrated in figure 2.73*B*. In general, if the current phasor has an angle, θ, the voltage phasor angle, ϕ, is θ + 90° for a pure inductance.

For a pure capacitance,

$$i = C\frac{dv}{dt}.$$

Employing the same process to find equation 2.111,

$$\bar{I} = j\omega C\,\bar{V}. \tag{2.112a}$$

or

$$\bar{V} = \left(\frac{1}{j\omega C}\right)\bar{I} = \left(\frac{-j}{\omega C}\bar{I}\right). \tag{2.112b}$$

In this case, −j indicates a −90° displacement of the voltage phasor from current, as shown in figure 2.73*C*.

Now that the phasor relationships of the fundamental elements have been covered, the stage is set for impedance transforms.

Impedance Transforms

The current-voltage relationships for the three fundamental elements have been found using phasors, as

$$\bar{V} = R\bar{I}, \qquad \bar{V} = j\omega L\bar{I}, \qquad \bar{V} = \frac{\bar{I}}{j\omega C}.$$

These can be rewritten as voltage-phasor to current-phasor ratios:

$$\frac{\bar{V}}{\bar{I}} = R, \qquad \frac{\bar{V}}{\bar{I}} = j\omega L, \qquad \frac{\bar{V}}{\bar{I}} = \frac{1}{j\omega C}.$$

A very important quantity, *impedance*, signified by Z, is defined as the ratio of the phasor voltage to the phasor current for a circuit or

$$Z = \frac{\bar{V}}{\bar{I}}. \tag{2.113}$$

This expression is often called Ohm's law for ac circuits. Impedance is a complex quantity with dimensions of ohms, but it is not a phasor. Therefore, the impedance of the pure passive circuit elements, resistance, inductance, and capacitance, are respectively

$$Z_R = R, \qquad Z_L = j\omega L, \qquad Z_C = \frac{1}{j\omega C}. \tag{2.114}$$

These can be applied directly to circuit analysis when a circuit is in steady state. In other words, element impedances are employed to convert or transform a time-domain circuit model into a form in which the circuit can be analyzed using only complex algebra. Hence the expressions of equation 2.114 are called impedance transforms, and the transformed mathematical model is then in the impedance (or jω) domain. As a result, no differential equations are used to solve a steady ac circuit.

All previous fundamental theorems, laws, and circuit-analysis techniques also apply to steady ac circuit analysis using impedances. Thus, an ac circuit representation in the impedance domain is analogous to a dc circuit model. On the other hand, the concept of impedance has no meaning in the time domain with time-varying voltages and currents.

To demonstrate these concepts, consider the simple RL circuit in figure 2.74*A*, now with a complex voltage source, that is, a steady-state sinusoid defined as a phasor. Here, the current through the resistance and inductance is the phasor, $\bar{I}$; therefore,

$$\bar{V}_R = \bar{I}R,$$

$$\bar{V}_L = \bar{I}j\omega L.$$

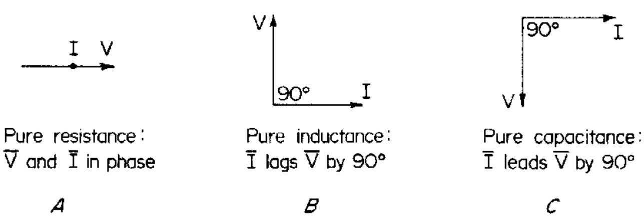


Figure 2.73.—Voltage-current phasor relationships for circuit elements.

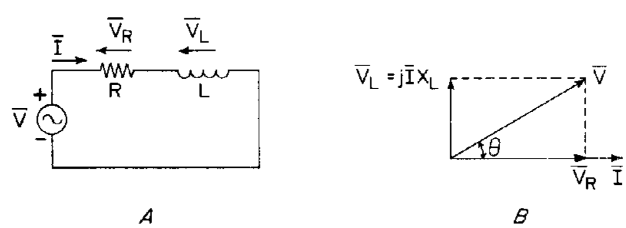


Figure 2.74.—Steady sinusoid analysis of simple RL series circuit.

By Kirchhoff's voltage law,

$$\bar{V} = \bar{V}_R + \bar{V}_L$$

or
$$\bar{V} = \bar{I}R + \bar{I}j\omega L = \bar{I}(R + j\omega L).$$

The impedance (equivalent) of the entire circuit is then

$$Z = \frac{\bar{V}}{\bar{I}} = R + j\omega L. \qquad (2.115)$$

Because impedance is a complex quantity, it also has a polar form:

$$Z = |Z| \underline{|\theta}, \qquad (2.116)$$

where $|Z| = (R^2 + (\omega L)^2)^{1/2}$, magnitude of impedance, Ω,

and $\theta = \tan^{-1}(\frac{\omega L}{R})$.

A phasor diagram for the circuit current and voltages is given in figure 2.74*B*. Note that as current is common to both elements, it could be used as the reference phasor. Here, voltage across the resistor, V_R, is in phase with current, while that across the inductor, V_L, leads current by 90°. The total circuit voltage, $\bar{V}$, can be resolved noting that

$$\bar{V} = \bar{V}_R + \bar{V}_L = V \underline{|\theta},$$

where $V = (V_R^2 + V_L^2)^{1/2}$, magnitude of source voltage, V,

and $\theta = \tan^{-1}(\frac{V_L}{V_R})$.

This last angle is identical to that found for the impedance. It should be noted that the current and voltage relationships for the inductor are as those found previously when time-domain voltages and currents were considered.

Now consider figure 2.75*A*, which shows a simple RC series circuit in which

$$\bar{V}_R = \bar{I}R,$$

$$\bar{V}_C = \bar{I}\,(\frac{1}{j\omega C}) = -j\frac{\bar{I}}{\omega C},$$

and
$$\bar{V} = \bar{I}R - \frac{j\bar{I}}{\omega C} = \bar{I}\,(R - \frac{j}{\omega C}).$$

The impedance becomes

$$Z = \frac{\bar{V}}{\bar{I}} = R - \frac{j}{\omega C} = R + \frac{1}{j\omega C}. \qquad (2.117)$$

Figure 2.75*B*, the circuit phasor diagram, shows the current-voltage phase-angle relationships with the voltage across the capacitor now lagging that across the resistor.

Continuing the process for an RLC series circuit (fig. 2.76*A*), the voltage across each element is

$$\bar{V}_R = \bar{I}R.$$

$$\bar{V}_L = \bar{I}j\omega L,$$

$$\bar{V}_C = \bar{I}(\frac{1}{j\omega C}).$$

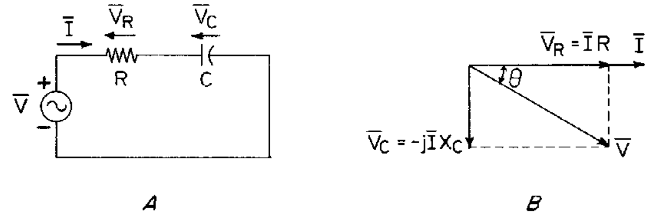


Figure 2.75.—Steady sinusoid analysis of simple RC series circuit.

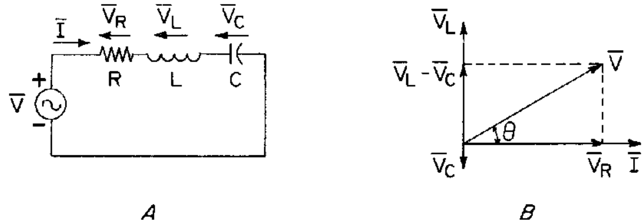


Figure 2.76.—Steady sinusoid analysis of simple RLC series circuit.

and across the entire circuit,

$$\bar{V} = \bar{V}_R + \bar{V}_L + \bar{V}_C$$

or
$$\bar{V} = \bar{I}R + \bar{I}j\omega L + \bar{I}(\frac{1}{j\omega C}), \qquad (2.118)$$

with the circuit impedance,

$$Z = R + j\omega L + \frac{1}{j\omega C}$$

or
$$Z = R + j(\omega L - \frac{1}{\omega C}). \qquad (2.119)$$

The foregoing gives the essence of impedance transforms. Each impedance shown in equations 2.115, 2.117, and 2.119 is the equivalent impedance of that circuit and has the general form

$$Z = R + jX = |Z| \underline{|\theta},$$

where R = resistance component, Ω,

X = reactance component, Ω,

$|Z| = (R^2 + X^2)^{1/2}$, magnitude of impedance, Ω,

and $\theta = \tan^{-1}(X/R)$.

Here, depending on the pure circuit elements, the reactive component is

$X = \omega L$ = inductive reactance, Ω,

$X = \frac{1}{\omega C}$ = capacitive reactance, Ω,

$X = \omega L - \frac{1}{\omega C}$ = reactance for series LC elements, Ω.

From this equation, it can be seen that resistance is constant while reactance is variable with frequency.

The time-domain expression found for a general series RLC circuit can be used to clarify the transformation process:

$$v = iR + L\frac{di}{dt} + \frac{1}{C}\int_0^t i\,dt + V_0. \qquad (2.93)$$

It has been demonstrated in the impedance domain for steady ac that

$$\overline{V} = \overline{I}R + \overline{I}j\omega L + \overline{I}\frac{1}{j\omega C}. \qquad (2.118)$$

Accordingly, time-domain differential equations can be changed to the impedance domain when the circuit is under steady ac by

1. Replacing v with $\overline{V}$ (in rms),
2. Replacing i with $\overline{I}$ (in rms),
3. Replacing $\frac{d}{dt}$ with $j\omega$,
4. Replacing $\int \ldots dt$ with $\frac{1}{j\omega}$, and
5. Letting $V_0 = 0$.

However, it is a much more efficient approach to ac circuit analysis to assign the impedances directly using equation 2.114, and soon more will be stated regarding this.

Admittance

Admittance, which is given the symbol Y, is defined as the reciprocal of impedance, Z, and

$$Y = \frac{\overline{I}}{\overline{V}}. \qquad (2.120)$$

The units are now siemens, replacing the previous designation, mhos. Admittance is therefore a complex quantity, the real part being conductance, G, and the imaginary component susceptance, B, or

$$Y = G + jB. \qquad (2.121a)$$

It should be noted that conductance is not the reciprocal of resistance unless reactance is zero, likewise for susceptance, reactance, and resistance. In general form, through equating Y and Z,

$$G = \frac{R}{R^2 + X^2} \text{ and } B = \frac{-X}{R^2 + X^2}. \qquad (2.121b)$$

Admittance affords basically the same convenience in steady ac circuit analysis that conductance provides for parallel dc circuits.

Steady-State Analysis

As previously stated, all circuit-analysis techniques that were covered for dc circuits still apply to steady ac circuits in the impedance domain. These include network reduction, Kirchhoff's laws, loop and node analysis, network theorem, plus delta-wye transforms. Impedances simply replace resistances in the concept, and steady ac sources replace dc. Even with dc, the impedance domain can be used; in other words, dc sources can be thought of as steady-state sinusoids with $\omega = 0$. Therefore, with dc, reactance has no effect.

A summary of circuit relationships follows, this time including impedance.

1. Impedances in series. A single equivalent impedance, Z, is

$$Z = Z_1 + Z_2 + Z_3 + \ldots + Z_n. \qquad (2.122)$$

2. Impedances in parallel. A single equivalent here is

$$\frac{1}{Z} = \frac{1}{Z_1} + \frac{1}{Z_2} + \frac{1}{Z_3} + \ldots + \frac{1}{Z_n}. \qquad (2.123)$$

3. Admittances in parallel,

$$Y = Y_1 + Y_2 + Y_3 + \ldots Y_n. \qquad (2.124)$$

4. Voltage distribution of series impedances,

$$\overline{V}_1 = \frac{Z_1}{Z}\overline{V},\ \overline{V}_2 = \frac{Z_2}{Z}\overline{V},\ \ldots, \qquad (2.125)$$

where $\overline{V}$ is the input voltage, $\overline{V}_1$ is across Z_1, and so on.

5. Current distribution through parallel admittances,

$$\overline{I}_1 = \frac{Y_1}{Y}\overline{I},\ \overline{I}_2 = \frac{Y_2}{Y}\overline{I},\ \ldots, \qquad (2.126)$$

where $\overline{I}$ is the total circuit current, $\overline{I}_1$ is through Y_1, and so on. Or parallel impedances,

$$\overline{I}_1 = \frac{Z}{Z_1}\overline{I},\ \overline{I}_2 = \frac{Z}{Z_2}\overline{I},\ \ldots, \qquad (2.127)$$

The overlines are removed on the above impedances and admittances simply for convenience, but it should be remembered that all are complex numbers. In essence, ac circuits in the steady state can be solved almost as easily as dc circuits employing only resistance. The major addition is that the solution now uses complex algebra.

EXAMPLE 2.21

Consider the circuit shown in figure 2.77, where

$$v = 5{,}880 \cos (377t + 53.1°),$$
$$i = 141.4 \cos 377\, t.$$

The circuit is under steady-state conditions. What are the values of R and L?

SOLUTION. The phasor representations for voltage and current are

$$\bar{V} = \frac{5{,}880}{\sqrt{2}} \angle 53.1° = 4{,}158 \angle 53.1° \text{ V},$$

$$\bar{I} = \frac{141.4}{\sqrt{2}} \angle 0° = 100 \angle 0° \text{ A}.$$

The total impedance of the circuit is then

$$Z = \frac{\bar{V}}{\bar{I}} = \frac{4{,}158 \angle 53.1°}{100 \angle 0°} = 41.58 \angle 53.1° \ \Omega$$

or in rectangular form,

$$Z = 25 + j33.25 \ \Omega.$$

The real part of this impedance must be the circuit resistance and the imaginary part equal to total reactance. Thus,

$$R = 25 \ \Omega,$$
$$X = 33.25 \ \Omega,$$

but

$$X = \omega L - 0.3.$$

Therefore, as $\omega = 377$ rad/s,

$$L = \frac{33.25 + 0.3}{377} = 0.09 \text{ H}.$$

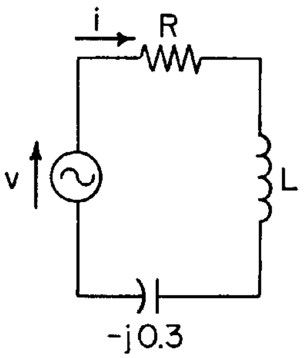


Figure 2.77.—Circuit for example 2.21.

EXAMPLE 2.22

Find the voltage, $\bar{V}$, across the 2-Ω resistance in figure 2.78.

SOLUTION. Circuit reduction appears to be the easiest way to solve the problem. Noting that $\omega = 377$ rad/s, the reactances of the impedance and capacitance are

$$X_L = \omega L = (377)(0.12 \times 10^{-3}) = 0.045 \ \Omega,$$

$$X_C = \frac{1}{\omega C} = \frac{1}{(377)(3{,}535 \times 10^{-6})} = 0.75 \ \Omega.$$

The impedance of the branch containing the impedance is

$$Z_1 = R_1 + jX_C = 1 + j0.045 \ \Omega$$

for the branch with the capacitance

$$Z_2 = R_2 + jX_C = 1 - j0.75 \ \Omega.$$

Combining these two parallel impedances in polar form,

$$\frac{Z_1 Z_2}{Z_1 + Z_2} = \frac{(1.0 \angle 2.58°)(1.25 \angle -36.87°)}{2 - j0.705} = 0.59 \angle -14.9° \ \Omega,$$

and the equivalent impedance seen by the 1,000-V source is

$$\begin{aligned} Z_{eq} &= Z + 0.59 \angle -14.9° \\ &= Z + 0.57 - j0.15 \\ &= 2.57 - j0.5 = 2.57 \angle -3.4° \ \Omega. \end{aligned}$$

Assigning the source voltage as the reference phasor, the total circuit current is

$$\bar{I}_t = \frac{\bar{V}}{Z_{eq}} = \frac{1{,}000 \angle 0°}{2.57 \angle -3.4°} = 388 \angle 3.4° \text{ A}.$$

The voltage across the 2-Ω resistance is then

$$\bar{V} = 2\bar{I} = (2)(388 \angle 3.4°) = 777 \angle 3.4° \text{ V}.$$

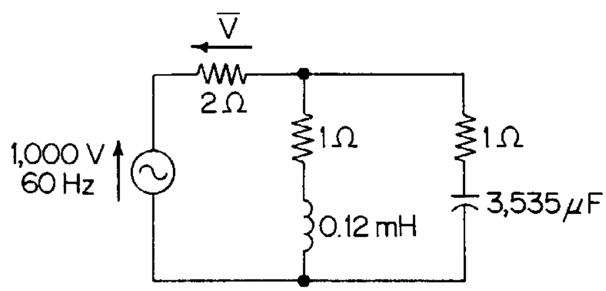


Figure 2.78.—Circuit for example 2.22.

EXAMPLE 2.23

Calculate the current, $\bar{I}$, through the branch indicated in figure 2.79 using only loop equations.

SOLUTION. Two loop currents have been assigned in the figure. Using Kirchhoff's voltage law,

$$(5 - j5)\bar{I}_1 - 5\bar{I}_2 = 1{,}000\underline{|0^\circ},$$
$$-5\bar{I}_1 + (5 + j5)\bar{I}_2 = 800\underline{|0^\circ}.$$

The solution to these simultaneous equations gives

$$\bar{I}_1 = 360 + j200 \text{ A},$$
$$\bar{I}_2 = 360 - j160 \text{ A},$$

The current through the 5-Ω resistance is then

$$\bar{I} = \bar{I}_1 - \bar{I}_2$$

or $$\bar{I} = 360 + j200 - 360 + j160;$$

thus, $$\bar{I} = j360 \text{ A}$$
or $$\bar{I} = 360\underline{|90^\circ} \text{ A}.$$

EXAMPLE 2.24

What are the Thevenin's and Norton's equivalents for the circuit shown in figure 2.80?

SOLUTION. Applying either Thevenin's or Norton's theorem, the equivalent impedance of the circuit between a and b with the internal source off is Z_s. When the steady-state voltage source is off, it acts as a short circuit, and the 4-Ω and 12-Ω resistances are effectively in parallel, and

$$\frac{(4)(12)}{4 + 12} = 3 \ \Omega.$$

This combined resistance is in series with the j10-Ω reactance, and the series combination is in parallel with the −j6-Ω capacitance, and

$$Z_s = Z_{ab} = \frac{(3 + j10)(-j6)}{3 + j10 - j6};$$

thus, $$Z_s = 12.5\underline{|-69.8^\circ} = 4.3 - j11.7 \ \Omega.$$

If the terminals a and b are shorted out according to Norton's theorem, a short circuit exists across the capacitance, and the j10-Ω impedance and 12-Ω resistance are placed in parallel. The equivalent impedance of the circuit under this shorted condition as seen by the ideal voltage source is then

$$Z_{eq} = 4 + \frac{(12)(j10)}{12 + j10} = 10.7\underline{|33.5^\circ} \ \Omega.$$

The circuit delivered by the ideal source is

$$\bar{I}_1 = \frac{\bar{V}}{Z_{eq}} = \frac{25\underline{|0^\circ}}{10.7\underline{|33.5^\circ}} = 2.33\underline{|-33.5} \text{ A},$$

and the current through the j10-Ω reactance and the shorted terminals is

$$\bar{I}_2 = \bar{I}_{ab} = \bar{I}_s = \bar{I}_1\left(\frac{12}{12 + j10}\right)$$

or $$\bar{I}_s = 2.33\underline{|33.5}\left(\frac{12}{15.62\underline{|39.8^\circ}}\right) = 1.8\underline{|-73.3^\circ} \text{ A}.$$

Z_s and $\bar{I}_s$ define the components of the Norton equivalent for the circuit in figure 2.80. By source transformation,

$$\bar{V}_s = \bar{I}_s Z_s = (1.8\underline{|-73.3^\circ})(12.5\underline{|-69.8^\circ})$$
$$= 22.5\underline{|-143.1^\circ} \text{ V},$$

which is identical to

$$\bar{V}_s = -22.5\underline{|36.9^\circ} \text{ V}.$$

Z_s and $\bar{V}_s$ define the components of the Thevenin equivalent.

It can be noted in figure 2.80 that the ideal voltage source is in series with the 4-Ω resistance in one branch. Therefore, source transformation alone could be employed to solve the problem.

The use of subscripts in this example did not follow the format previously used in the chapter. In other words, Z_s, $\bar{I}_s$, and $\bar{V}_s$ described the equivalent circuits rather than Z_0, $\bar{I}_0$, and $\bar{V}_0$. The reason is that the subscript zero has a special meaning in three-phase ac circuits, which will be discussed in chapter 4.

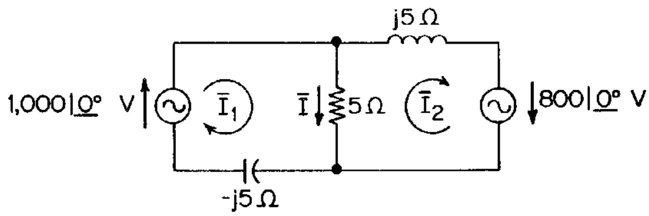


Figure 2.79.—Two-loop circuit for example 2.23.

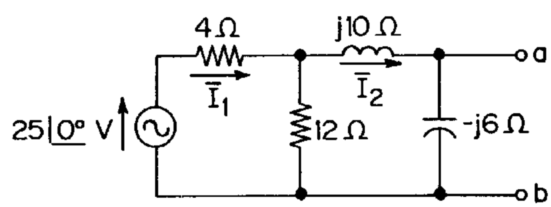


Figure 2.80.—Active circuit for example 2.24.

Chapter 2 has introduced the concepts of electrical circuit analysis. The fundamental laws were covered first, followed by numerous circuit analysis techniques, which were applied to dc circuits. Steady ac was then presented, and the chapter concluded with examples of circuit analysis on ac circuits under steady-state conditions. These concepts are fundamental to electrical engineering, regardless of application. Thus, comprehension of the contents of this chapter is vital to understanding the following chapters. The next chapter will continue the study of electrical fundamentals, with emphasis on power consumption in ac circuits.

CHAPTER 3.—ELECTRICAL FUNDAMENTALS II

The measures of instantaneous power, p, and average power, P, were introduced in chapter 2. Instantaneous power does not have application in steady ac circuit analysis, so the concept of average power has been developed to gauge the rate at which electricity does work. This chapter continues to build the foundations for mine power fundamentals that will be expanded into full comprehension in chapter 4. There, the discussion will focus on three-phase power; here, the purpose is to introduce single-phase power and transformers.

AVERAGE POWER AND POWER FACTOR

To find the average power consumed by a circuit, the resistance of each element can be examined and all the individual power consumptions computed. Reactance, either capacitive or inductive, does not affect average power. When all the average powers have been determined, their sum yields the total average power delivered to the circuit. Obviously, if the circuit elements are numerous, the process can be time consuming, but this approach is sometimes necessary.

If the average power needs to be determined for the total circuit, it would be more desirable to perform only one calculation by computing average power in terms of the terminal current and voltage in the circuit. Yet, when complex or imaginary components exist in the circuit, can they be ignored, as this implies? In other words, the voltage and current waveforms might not be in phase, and when a phase angle is involved, the product of effective voltage and current no longer equals average power. However, *instantaneous voltage and current can be used to* calculate average power and to demonstrate what occurs if a circuit has reactance.

Assume that the following current and voltage are monitored at the terminals of a circuit:

$$i = I_m \cos \omega t,$$

$$v = V_m \cos(\omega t + \theta).$$

Current is taken as reference, and the phase angle by which voltage leads current is θ. The instantaneous power consumed is then

$$p = vi = V_m I_m \cos(\omega t + \theta) \cos \omega t.$$

From the trigonometric identity for the product of two cosines,

$$p = \frac{V_m I_m}{2} (\cos\theta + \cos(2\omega t + \theta))$$

or

$$p = \frac{V_m I_m}{2} \cos \theta + \frac{V_m I_m}{2} \cos(2\omega t + \theta). \qquad (3.1)$$

The first term of equation 3.1 is constant, while the second is a sinusoid. Thus, taking the average to find average power results in

$$P = \text{av}(p) = \frac{V_m I_m}{2} \cos \theta. \qquad (3.2)$$

Realizing that $V_m = \sqrt{2}V$ and $I_m = \sqrt{2}I$, average power becomes

$$P = VI \cos \theta, \qquad (3.3)$$

in which V and I are root-mean-square (rms) voltage and current at the circuit terminals and θ is their phase angle. If the voltage and current had been dc values, the average power would just be the product of voltage and current. However, when voltage and current are sinusoidal, equation 3.3 specifies that the average power entering any circuit is the product of the effective voltage, effective current, and the cosine of the phase angle.

The function $\cos \theta$ is called the *power factor* (pf). For a purely resistive load, the phase angle is zero and the power factor is unity. Unity power factor may also exist when inductance and capacitance are present, if the effects of reactive elements cancel. If the circuit is totally reactive (either inductive or capacitive), the phase angle is a positive or negative 90°, the power factor is zero, and average power must be zero.

COMPLEX AND APPARENT POWER

When there is reactance in a circuit, a component of circuit current is used to transfer stored energy. The energy is periodically stored in and discharged from the reactance. This stored energy adds to circuit current but not to average power because average power to reactive elements is zero. In such cases, the power factor is not unity. Thus, as no work is performed by the added current, the power factor can be considered to be a measure of circuit efficiency or its ability to perform work, and average power, defined by equation 3.3, is often called *active power* or *real power*.

Power calculations can be simplified if power is defined by the complex quantity shown in figure 3.1, which is expressed mathematically as

$$\bar{S} = P + jQ, \qquad (3.4)$$

where $\bar{S}$ = complex power,
P = real power, as before,
and Q = reactive power or imaginary power.

Imaginary power accounts for the energy supplied to the reactive elements. If

$$P = VI \cos\theta,$$

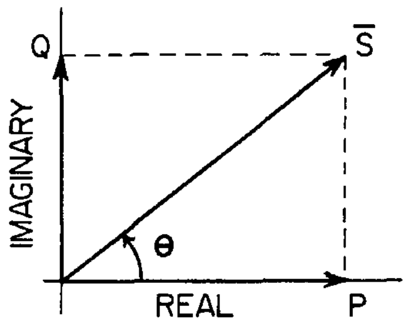


Figure 3.1.—Power represented as real and imaginary components.

then the magnitude of complex power, S, called *apparent power*, is

$$S = VI, \tag{3.5}$$

and *imaginary power* is

$$Q = VI \sin\theta. \tag{3.6}$$

Voltage and current are again rms, and θ is the phase angle. Therefore,

$$\bar{S} = P + jQ = VI\cos\theta + jVI\sin\theta$$

or $$\bar{S} = VI(\cos\theta + j\sin\theta) = VIe^{j\theta} = VI\underline{|\theta}. \tag{3.7}$$

Complex power is then simply the product of terminal rms voltage and current magnitude acting at a phase angle. Applying dc concepts, the product, VI, is the power apparently absorbed by the circuit, hence the term *apparent power*. Apparent power, real power, and imaginary power are dimensionally the same, but to avoid confusion with real power (units of watts), apparent power has units of voltamperes, and reactive power uses voltamperes reactive.

When sinusoidal voltage and current have general form, as in

$$\bar{V} = V\underline{|\theta},$$

$$\bar{I} = I\underline{|\phi},$$

instead of using equations 3.4 and 3.7, the following expression is more convenient for computing complex power:

$$\bar{S} = \bar{V}\,\bar{I}^*, \tag{3.8}$$

where $\bar{V}$ = complex voltage, V,
and $\bar{I}^*$ = conjugate of complex current, A.

Accordingly,

$$\bar{S} = \bar{V}\,\bar{I}^* = V\underline{|\theta}\,I\underline{|-\phi} = VI\underline{|\theta-\phi}$$

or $$\bar{S} = Ve^{j\theta}Ie^{j-\phi} = VIe^{j(\theta-\phi)},$$

where $\theta - \phi$ = phase angle between voltage and current.

EXAMPLE 3.1

When operating under normal conditions, an induction motor has been found to draw 100 A when 440 V is across its terminals. Current is lagging voltage by 36.87°. Find the average, reactive, apparent, and complex powers for this load.

SOLUTION. From equation 3.3, the average power is

$$P = (440)(100)\cos 36.87° = 35{,}200 \text{ W}.$$

Using equation 3.6, the reactive power is

$$Q = (440)(100)\sin 36.87° = 26{,}400 \text{ var}.$$

Equation 3.5 defines the apparent power as

$$S = (440)(100) = 44{,}000 \text{ VA},$$

and equation 3.4 yields the complex power as

$$\bar{S} = 35{,}200 + j26{,}400 \text{ VA}.$$

ALTERNATIVE SOLUTION. If voltage is assigned as the reference phasor, then

$$\bar{V} = 440\underline{|0°} \text{ V},$$

$$\bar{I} = 100\underline{|-36.87°} \text{ A}.$$

From equation 3.8, the complex power is

$$\bar{S} = (440\underline{|0°})(100\underline{|-36.87°})^*$$

or $$\bar{S} = (440\underline{|0°})(100\underline{|36.87°}) = 44{,}000\underline{|36.87°} \text{ VA},$$

where the magnitude is the apparent power, or

$$S = 44{,}000 \text{ VA}.$$

Converting the polar expression for complex power to a rectangular form,

$$\bar{S} = 35{,}200 + j26{,}400 \text{ VA},$$

which yields

$$P = 35{,}200 \text{ W},$$

$$Q = 26{,}400 \text{ var}.$$

It should be noted that the above solutions are only two of the many possible.

EXAMPLE 3.2

A load consumes 1,250 kW at 0.6 lagging power factor when 4,160 V at 60 Hz is across it. The load is connected in series with a (0.71 + j0.71) Ω impedance to a constant source. Determine the voltage and power factor at the source.

SOLUTION. From the stated conditions, the average power is

$$P_1 = 1{,}250 \text{ kW}.$$

From equation 3.3, the current through the load is

$$I_1 = \frac{P_1}{V_1 \cos\theta_1},$$

where P_1, V_1, and $\cos\theta_1$ relate the conditions for the load, or

$$\bar{I}_1 = \frac{1{,}250{,}000}{(4{,}160)\,(0.6)} = 501 \text{ A}.$$

For convenience, the voltage across the load can be assigned as the reference phasor, then

$$\bar{V}_1 = 4{,}160 \text{ V},$$

$$\bar{I} = 501\underline{|-\theta_1} = 501\underline{|-53.1°} \text{ A}.$$

The load current also flows through the series impedance. Using polar expressions, the voltage drop across this impedance is

$$\bar{V}_2 = \bar{I}_1 Z_2$$

or $\bar{V}_2 = (501\underline{|-53.1°})(1\underline{|45°}) = 501\underline{|-8.1°}$ V.

The voltage at the source is then

$$\begin{aligned}\bar{V}_s &= \bar{V}_1 + \bar{V}_2 \\ &= 4{,}160 + (496 - j71) \\ &= 4{,}656 - j71 = 4{,}657\underline{|-0.9°} \text{ V}.\end{aligned}$$

The power factor at the source can be found by first calculating the phase angle between current and voltage at the source with the current phasor taken as reference. Here,

$$\theta_s = \theta_V - \theta_I$$

or $\theta_s = -0.9° - (-53.1°) = 52.2°$.

Therefore, the power factor at the source is
$\cos\theta_s = \cos 52.2° = 0.61$ lagging.

Any circuit in steady state can be reduced to the general impedance

$$Z = R + jX$$

or $$Z = |Z|\,\underline{|\theta} = |Z|e^{j\theta} = \frac{V}{I}\,e^{j\theta}, \tag{3.9}$$

where $$\theta = \tan^{-1}\left(\frac{X}{R}\right).$$

By relating the complex power consumed by a circuit with this impedance, another useful expression can be found:

$$\bar{S} = VI\,e^{j\theta} = I^2\left(\frac{V}{I}e^{j\theta}\right) = I^2 Z;$$

hence, $$S = I^2(R + jX) = I^2R + jI^2X.$$

Because $\bar{S} = P + jQ$,

$$P = I^2R,\quad Q = I^2X \tag{3.10a}$$

or $$P = V_R{}^2/R,\quad Q = V_X{}^2/X. \tag{3.10b}$$

In equation 3.10*a*, the current I is real rms, not complex. The rms voltages in equation 3.10*b* are those existing across the individual elements, not across the total circuit.

It may already be obvious that the circuit impedance angle is identical to the power-factor angle. The following also apply.

1. If a circuit contains resistance and capacitance (a capacitive load, Z = R − jX), the current leads voltage, the phase angle is negative, and Q is negative.
2. If the circuit is an inductive load (Z = R + jX), the current lags voltage, and the phase angle and Q are positive.

In either case, the power factor ranges from zero to unity (purely reactive to purely resistive). A capacitive load is said to have a leading power factor, an inductive load a lagging power factor, as illustrated in figure 3.2.

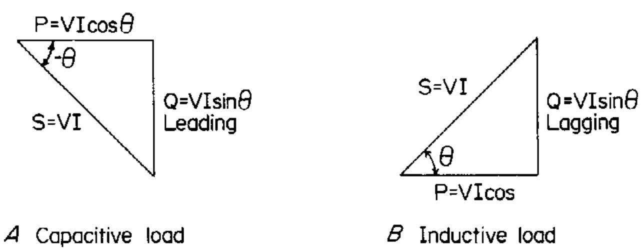


Figure 3.2.—Illustration of leading (*A*) and lagging (*B*) power factors.

The complex power delivered to several loads is the sum of the complex power consumed by each individual load, no matter how they are interconnected. This relationship can be shown using the simple circuit in figure 3.3. The total complex power to the system is

$$\bar{S} = \bar{V}\,\bar{I}^* = P + jQ,$$

but $$\bar{I} = \bar{I}_1 + \bar{I}_2;$$

thus, $$\bar{S} = \bar{V}(\bar{I}_1^* + \bar{I}_2^*)$$
$$= \bar{V}\,\bar{I}_1^* + \bar{V}\,\bar{I}_2^* = \bar{S}_1 + \bar{S}_2$$

or $$\bar{S} = P_1 + P_2 + jQ_1 + jQ_2.$$

This has extensive practical significance. For example, if a circuit has a lagging power factor, a capacitance (with a leading power factor) can be selected and then placed in parallel, so as to negate or reduce the total circuit imaginary power (with the capacitance). The net result is to reduce total circuit current, while the load consumes the same real power and thus performs the same work. This is the essence of power-factor improvement.

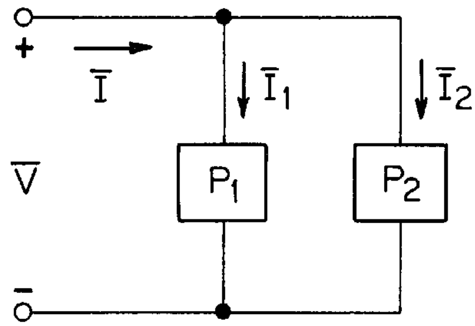


Figure 3.3.—Circuit demonstrating sum of complex powers.

EXAMPLE 3.3

Consider that the two loads shown in figure 3.3 are induction motors operating as follows:
P_1 = 50 kW at 0.6 lagging power factor,
P_2 = 25 kW at 0.8 lagging power factor.
Find the overall apparent power and power factor when these consumptions are combined.

SOLUTION. The average, apparent, and reactive power for each load are

$$P_1 = 50 \text{ kW},$$

$$S_1 = \frac{P_1}{\cos\theta_1} = \frac{50{,}000}{0.6} = 83.33 \text{ kVA},$$

$$Q_1 = S_1 \sin\theta_1 = 83{,}333(0.8) = 66.67 \text{ kvar},$$

$$P_2 = 25 \text{ kw},$$

$$S_2 = \frac{P_2}{\cos\theta_2} = \frac{25{,}000}{0.8} = 31.25 \text{ kVA},$$

$$Q_2 = S_2 \sin\theta_2 = 31{,}250(0.6) = 18.75 \text{ kvar}.$$

Complex power is then

$$\bar{S} = P_1 + P_2 + jQ_1 + jQ_2$$

or $$\bar{S} = 50 + 25 + j66.67 + j18.75$$
$$= 75 + j85.42 \text{ kVA}.$$

Apparent power is the magnitude of complex power, or

$$S = (P^2 + Q^2)^{1/2}$$
$$= (75^2 + 85.42^2)^{1/2} = 113.7 \text{ kVA}.$$

The power-factor angle is

$$\theta = \text{Tan}^{-1}\left(\frac{Q}{P}\right)$$

$$= \text{Tan}^{-1}\left(\frac{85.42}{75}\right) = 48.72°,$$

and the power factor of the combination is
$$\text{pf} = \cos\theta = \cos 48.72° = 0.66.$$

EXAMPLE 3.4

The maximum capacity of a piece of power equipment is rated by apparent power at 500 kVA. The unit is being loaded by 300 kW at 0.6 lagging power factor. The power factor must be improved to 0.8 lagging by adding capacitance in parallel with the equipment. Find the required capacitance in kilovoltamperes reactive. With the capacitance in place, find the reserve capacity that is available from the power equipment.

SOLUTION. For the load on the equipment without the capacitance,

$$P_1 = 300 \text{ kW},$$

$$S_1 = \frac{P_1}{\cos\theta_1} = \frac{300}{0.6} = 500 \text{ kVA},$$

$$Q_1 = S_1\sin\theta_1 = 500(0.8) = 400 \text{ kvar}.$$

It can be said from S_1 that the equipment is fully loaded. When pure capacitance is added, average power remains constant, and only reactive power and apparent power change. For the desired power factor, $\cos\theta_2$,

$$S_2 = \frac{P_1}{\cos\theta_2} = \frac{300}{0.8} = 375 \text{ kVA},$$

$$Q_2 = S_2\sin\theta_2 = 375(0.6) = 225 \text{ kvar}.$$

Consequently, the added capacitance causes the total reactive power to decrease. The difference between the reactive power without and with the capacitance must be the amount inserted by the capacitance. In other words,

$$Q_c = -(Q_1 - Q_2),$$

$$Q = -(400 - 225) = -175 \text{ kvar.}$$

The negative sign is used here to indicate that the capacitance adds negative reactive power to the system. Finally, the difference between the apparent power without and with the capacitance yields the reserve capacity available from the equipment, or

$$S = S_1 - S_2$$
$$= 500 - 375 = 125 \text{ kVA.}$$

It can be noted that additional average power can now be added to load on the equipment without exceeding its maximum capacity. For instance, consider that average power P will load the equipment so that the equipment is again operating at full capacity. Then, the total average power is

$$P_T = 300 \text{ kW} + P.$$

Reactive remains constant,

$$Q_T = Q_2 = 225 \text{ kvar,}$$

and apparent power changes to

$$S_T = 500 \text{ kVA.}$$

Therefore, $\quad S_T = (P_T^2 + Q_T^2)^{1/2},$

$$500 = [(300 + P)^2 + 225^2]^{1/2}.$$

Solving for the new average power,

$$P = 146.5 \text{ kW.}$$

RESONANCE

Series Resonance

Earlier, the impedance for the simple series RLC circuit shown in figure 3.4 was found to be

$$Z = R + j\omega L + \frac{1}{j\omega C} = R + j(\omega L - \frac{1}{\omega C}). \quad (2.119)$$

A special circuit phenomenon can now be demonstrated with this equation. There exists one specific frequency, ω_o, where total circuit reactance is zero and the circuit impedance is purely resistive, or

$$\omega_o L - \frac{1}{\omega_o C} = 0 \text{ or } \omega_o L = \frac{1}{\omega_o C} \quad (3.11)$$

and

$$Z_o = R. \quad (3.12)$$

At ω_o, the circuit is said to be in resonance, and

$$\omega_o^2 = \frac{1}{LC} \text{ or } \omega_o = \frac{1}{(LC)^{1/2}}.$$

Since $\omega = 2\pi f$, the resonance frequency, f_o, is given by

$$f_o = \frac{1}{2\pi(LC)^{1/2}}. \quad (3.13)$$

For a series RLC circuit in resonance, it can be shown that

1. The applied voltage, $\bar{V}$, and the resulting current, $\bar{I}$, are in phase,
2. The power factor of the circuit is unity,
3. The impedance, Z, is minimum, and
4. The current, $\bar{I}$, is maximum.

At all other frequencies that are significantly higher or lower than f_o, the series RLC circuit appears as a high impedance. With frequencies below resonance, capacitive reactance is greater than inductive reactance, so the angle of impedance is negative (total reactance is negative). Above resonance, the situation reverses and the impedance angle is positive. This can be seen clearly in figure 3.5 where circuit impedance versus frequency is plotted.

The energy stored in a resonance circuit is essentially constant, yet the energy level within the circuit may be many times higher than the energy being supplied from an external source during any period. The source itself does not supply any reactive power, only active power. The reactive power transfers energy back and forth between the resonant-circuit inductance and capacitance. The result of this energy transferral can be very high voltages, several times the terminal voltage, existing across the inductance and capacitance within the resonant circuit.

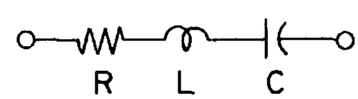


Figure 3.4.—Simple series RLC circuit for resonance.

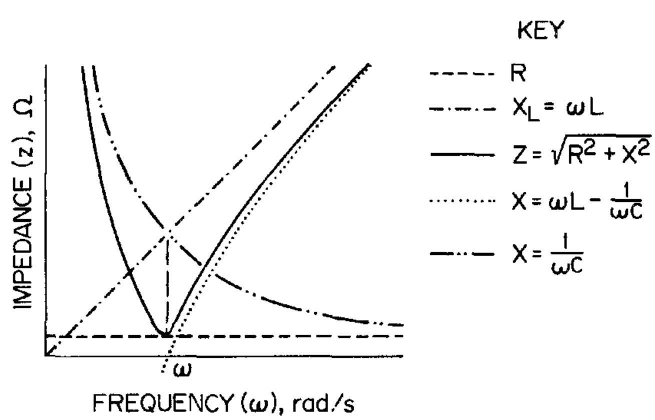


Figure 3.5.—Plot of impedance magnitude versus frequency for series RLC illustrating resonance.

This situation can be the cause of some severe overvoltages in mine power systems, and the concept will be explored further in chapter 11.

The amount of energy stored, compared with that dissipated by the resistance, is related to the shape of the curve representing impedance magnitude, as shown in figure 3.5. This curve is an example of a response curve. The quality factor of a circuit is a measure of the sharpness of the response curve and is expressed as a ratio:

$$Q_o = 2\pi \frac{\text{maximum energy stored per period}}{\text{total energy lost per period}}, \quad (3.14)$$

where the period is one complete cycle of the resonant frequency. By finding the ratio of the energy stored in either of the circuit's reactive components to the energy dissipated in the resistance, it can be shown that

$$Q_o = \frac{\text{reactance}}{\text{resistance}} = \frac{\omega_o L}{R} = \frac{1}{\omega_o CR}. \quad (3.15)$$

The quality factor normally has greater application in the communications aspects of electrical engineering than in the power aspects. For instance, the width of the response curve is also related to Q_o and has great relevance to the tuned circuits used in radio and television.

Parallel Resonance

The resonance of the simple parallel RLC circuit shown in figure 3.6*A* is very similar to that just discussed. This circuit is obviously idealized, but its performance is of general interest. The admittance can be written as

$$Y = G + j\omega C + \frac{1}{j\omega L} = G + j(\omega C - \frac{1}{\omega L}), \quad (3.16)$$

and the circuit is in resonance when susceptance B is zero. Hence, the circuit exhibits low admittance and high impedance at resonance, while the series RLC circuit had low impedance and high admittance:

$$B = \omega_o C - \frac{1}{\omega_o L} = 0 \text{ or } \omega_o C = \frac{1}{\omega_o L} \quad (3.17a)$$

or

$$Y_o = G. \quad (3.17b)$$

On the other hand, the resonant frequency is again

$$f_o = \frac{1}{2\pi(LC)^{1/2}}.$$

The statements previously given for series circuits also apply, except that current replaces voltage and voltage replaces current.

This is an example of duality. Anything stated about a series resonant circuit applies to its dual, the parallel resonant circuit, if each word in the left column below is replaced by its opposite word shown in the right column:

Series	*Parallel*
Voltage	Current.
Impedance	Admittance.
Resistance	Conductance.
Reactance	Susceptance.
Inductance	Capacitance.

Therefore, Q_o of this parallel resonant circuit is the dual of equation 3.15 or

$$Q_o = \frac{\text{susceptance}}{\text{conductance}} = \frac{\omega_o C}{G} = \frac{R}{\omega_o L}. \quad (3.18)$$

The concept also relates to many fundamentals covered in chapter 2. For example, two circuits are called duals if the loop equations for one have the same forms as the node equations for the other.

Because figure 3.6*A* is idealized (as actual inducting elements must have associated resistance), figures 3.6*B* and 3.6*C* are presented to show practical circuits that exhibit parallel resonance.

TRANSFORMERS

Early in chapter 2, the concept of mutual inductance was introduced. To review, Faraday found that a time-varying current in one circuit would induce a voltage in a nearby circuit. If the adjacent circuits are simply conductors and are labeled 1 and 2, as in figure 3.7, this statement means that

- i_1 in circuit 1 produces v_2 in circuit 2,
- v_2 in turn causes i_2 to flow (if circuit 2 is part of a complete loop), then
- i_2 induces v_1 in circuit 1.

These interrelated phenomena can be thought of as magnetic coupling between the two circuits, and it has been shown that

$$v_1 = L_{12}\frac{di_2}{dt} \text{ and } v_2 = L_{21}\frac{di_1}{dt},$$

where $L_{12} = L_{21} = M$ = mutual inductance, H.

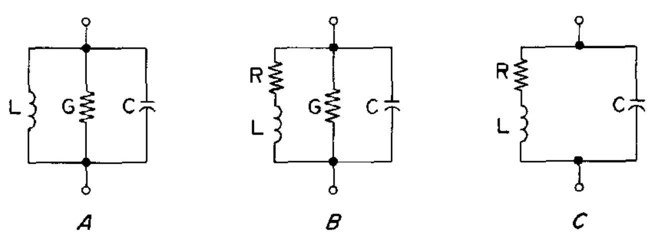


Figure 3.6.—Circuits that exhibit parallel resonance.

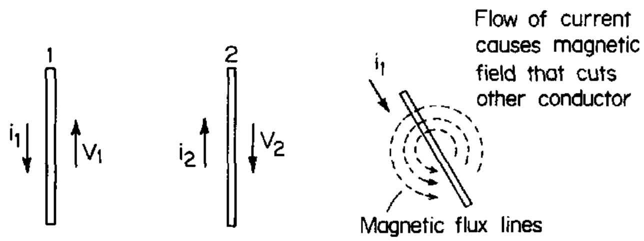


Figure 3.7.—Magnetic coupling between two conductors.

Because of the equality, M is used to represent mutual inductance. These equations are true only for straight wires, and magnetic coupling exists only if voltage and current are time varying.

The circuits considered previously were loops or meshes composed of passive and active elements, and these were conductively coupled by common branches or nodes. The following paragraphs develop the concept of magnetic coupling further and introduce the fundamentals behind one of the more important components of ac mine power systems, the transformer.

Transformers are prime examples of magnetic coupling. They are often designed to optimize this coupling, and their operation is based inherently on mutual inductance. Transformers are employed to increase the magnitude of voltage for more economical power transmission or, conversely, to decrease the level to provide voltage more suitable for electrical equipment operation. In essence, these changes can be made with either total isolation or direct conduction between circuits.

Instead of straight conductors, assume that two coils are situated side by side, and their magnetic action is passing through any environment (fig. 3.8). The current in coil 1 is then partly the result of self-inductance in coil 1 and mutual inductance from coil 2, and vice versa for coil 2. Expressed mathematically:

$$v_1 = (L_1 \frac{di_1}{dt} \pm M \frac{di_2}{dt} \pm \ldots.) \qquad (3.19a)$$

$$v_2 = (\pm M \frac{di_1}{dt} + L_2 \frac{di_2}{dt} \pm \ldots.), \qquad (3.19b)$$

where L_1, L_2 = self-inductances of coil 1 and coil 2, respectively, H,
and M = mutual inductance, H.

The additional terms implied by these equations exist only if more than two coils (or circuits, or windings) are interacting, and they are presented merely to make the expressions more general.

The plus and minus terms of the equations deserve special attention. Sign convention has been well defined for inductors, and coil 1 and coil 2 are inductors when taken individually. A current flowing into the coil produces an opposing voltage, hence the positive sign or polarity. The potential created by mutual inductance, M, however, cannot be treated in the same manner. This voltage may have either positive or negative polarity depending on the winding sense, the direction the coils are wound with respect to one another. Consider the two coils wound on a common core in figure 3.9*A*. They are wound in the same direction and therefore have the same sense. If a current is flowing into the top of the upper coil, the voltage produced by this current adds to that produced by the same current direction in the lower coil. But in figure 3.9*B*, the winding sense of the lower coil is reversed so that the same current in the top coil now creates a voltage that opposes the current produced in the lower coil. Therefore, the polarity of mutual-inductance voltages can be found by drawing physical sketches. However, this is impractical in circuit diagrams, and so magnetically coupled coils are often marked with dots that represent the direction of polarity. A dot is placed at the terminals of the coils that are instantaneously at the same polarity as a result of mutual inductance. Thus, in figure 3.10*A*, i_1 enters the dotted terminal of L_1, v_2 is sensed positively at the dotted terminal of L_2, and

$$v_1 = L_1 \frac{di_1}{dt} + M \frac{di_2}{dt}, \qquad (3.20a)$$

$$v_2 = M \frac{di_1}{dt} + L_2 \frac{di_2}{dt}. \qquad (3.20b)$$

In analyzing circuits, it may be more desirable to reference v_2 as positive at the undotted terminal of L_2, as in figure 3.10*B*. In this case,

$$v_1 = L_1 \frac{di_1}{dt} - M \frac{di_2}{dt}, \qquad (3.20c)$$

$$v_2 = -M \frac{di_1}{dt} + L_2 \frac{di_2}{dt}. \qquad (3.20d)$$

What is important is that, in either instance, the mutual voltage is produced independently from that of self-induction.

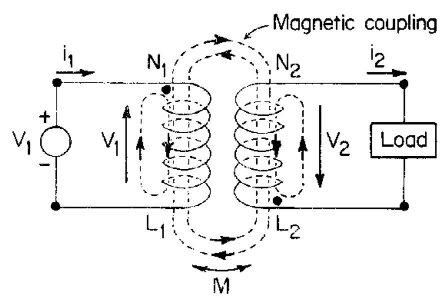


Figure 3.8.—Magnetic coupling between two coils.

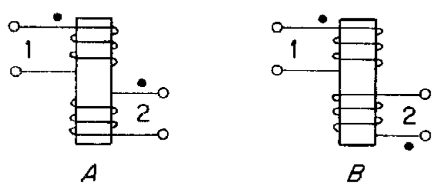


Figure 3.9.—Demonstration of coil winding sense.

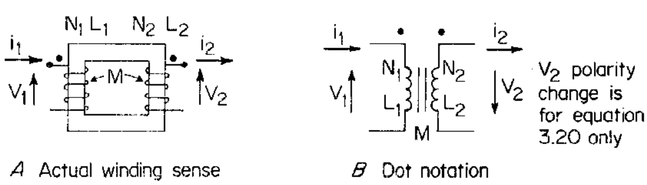


Figure 3.10.—Dot convention for mutual inductance sign.

The equations just presented are valid for any voltage or current waveform. If the currents are sinusoidal and have a radian frequency, ω, transforms can be employed so that for equations 3.20*c* and 3.20*d*,

$$\overline{V}_1 = j\omega L_1\overline{I}_1 - j\omega M\overline{I}_2, \tag{3.21a}$$

$$\overline{V}_2 = -j\omega M\overline{I}_1 + j\omega L_2\overline{I}_2. \tag{3.21b}$$

These relationships can be used to analyze circuits containing magnetically coupled elements. It should be stated that equations 3.20 and 3.21 relate only to the magnetically coupled elements; equations for complete circuits containing these devices will follow.

IDEAL TRANSFORMER

The level of mutual inductance, M, depends upon the spacing and orientation of the coils and the permeability of the medium between them. In other words, M is a function of the magnetic flux linking between the coils. More will be said about this phenomenon later in the section. In figure 3.10*A*, by comparing the power entering L_1 of the circuit with that stored or available in L_2, it can be proved from flux-linking concepts that

$$M \le (L_1L_2)^{1/2}. \tag{3.22}$$

Consequently, M has an upper limit defined by the geometrical mean of the two inductances involved. The ratio of M to its theoretical maximum is called the *coefficient of coupling*. This is by definition

$$k = \frac{M}{(L_1L_2)^{1/2}}, \tag{3.23}$$

where k can range from zero to unity. Coils having a low coefficient of coupling are said to be loosely coupled. Here the coils could be far apart or have an orientation such that little magnetic flux interacts between them. Loosely coupled circuits may have a k that ranges between 0.01 and 0.10. For tightly coupled circuits, such as air-core coils, k can be around 0.5.

A power transformer is a device having two or more tightly coupled coils or windings on a common iron core. The coils are wound and oriented to provide maximum common magnetic flux and can have a coefficient of coupling very close to 1.00. The usual range is 0.90 to 0.98. Resistance and other power losses are small. The winding receiving power is called a *primary*; that delivering power is called a *secondary*. In the circuit in figure 3.10, L_1 is the primary and L_2 is the secondary. An ideal transformer is an idealized form of transformer where $k = 1$ and losses within the device are zero. Hence, an ideal transformer can deliver all the power it receives. Many useful relationships for real transformers can be obtained by assuming the ideal transformer case.

The self-inductance of a coil has been shown to be proportional to the square of the number of turns forming the coil (N), provided that all the flux, created by the current in the coil, links all the turns (see chapter 2, "Inductance"). If a sinusoidal current, I, flows in a coil of N turns, then the voltage produced across an N-turn coil must be N times that caused in a 1-turn coil. Further, for a sinusoidal voltage, V, which is constant across an N-turn coil, the current allowed through must be 1/N times that caused in a 1-turn coil. Both these statements can be proved by magnetic field concepts, again assuming that all magnetic flux produced in a coil links all turns. It follows that for an ideal transformer with two windings:

$$\frac{L_1}{L_2} = \frac{N_1^2}{N_2^2}, \tag{3.24}$$

$$\frac{V_2}{V_1} = \frac{N_2}{N_1}, \tag{3.25}$$

$$\frac{I_1}{I_2} = \frac{N_2}{N_1}, \tag{3.26}$$

where N_1 = number of turns in primary winding,
N_2 = number of turns in secondary winding,
L_1, I_1, V_1 = primary winding inductance, rms current, and rms voltage, respectively,
and L_2, I_2, V_2 = secondary winding inductance, rms current, and rms voltage, respectively.

For this two-winding arrangement, the voltage and current can be complex sinusoids. The turns ratio of the transformer, a, is defined as the ratio of the number of turns in the secondary winding to the turns in the primary winding:

$$a = \frac{N_2}{N_1}. \tag{3.27a}$$

Hence, for an ideal transformer,

$$a = \left(\frac{L_1}{L_2}\right)^{1/2} = \frac{V_2}{V_1} = \frac{I_1}{I_2}. \tag{3.27b}$$

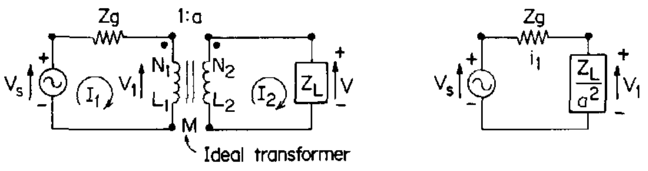
In other words, the sinusoidal voltages across the primary and secondary windings are in direct proportion to the number of turns of the windings, and the currents are related inversely to the turns. In addition, the last equation shows that the apparent power at the primary and secondary windings is indeed equal:

$$V_1I_1 = V_2I_2.$$

The magnitude of this power in voltamperes is specified for the maximum allowable or rated capacity of power transformers.

Another useful transformer relationship can be determined through a demonstration of steady ac circuit analysis with magnetically coupled circuits. Consider figure 3.11*A*, where a sinusoidal voltage source, V_s, with an internal impedance, Z_g (the combination is the Thévenin equivalent for a source), is connected to the primary of an ideal transformer. The secondary delivers power to a load impedance, Z_L. The vertical lines between the transformer windings indicate that the core is made of iron laminations. The turns ratio above the transformer symbol, 1:a, relates a convention of N_2 to N_1.

A very useful relationship is the ideal-transformer input impedance with the load connected, that is, the load



A B

Figure 3.11.—Demonstration of impedance transfer in transformers.

that the source sees through the transformer. Loop equations can be used to solve the problem. Two loops, $\bar{I}_1$ and $\bar{I}_2$ (both express complex currents), are available in the circuit; the loops are magnetically coupled through the transformer. Employing Kirchhoff's voltage law for loop 1,

$$V_s = \bar{I}_1 Z_g + \bar{I}_1 j\omega L_1 - \bar{I}_2 j\omega M,$$

and for loop 2,

$$O = -\bar{I}_1 j\omega M + \bar{I}_2 Z_L + \bar{I}_2 j\omega L.$$

M is again the mutual inductance. Notice that current enters the dot of the primary and leaves the dot on the secondary, making the sign of M negative. Rewriting these into standard loop-equation form gives

$$\bar{V}_s = \bar{I}_1(Z_g + j\omega L_1) - \bar{I}_2 j\omega M,$$

$$O = -\bar{I}_1 j\omega M + \bar{I}_2(Z_L + j\omega L_2).$$

Solving for I_1,

$$\bar{V}_s = \bar{I}_1(Z_g + j\omega L_1) + \bar{I}_1\left(\frac{\omega^2 M^2}{Z_L + j\omega L_2}\right).$$

Therefore, the impedance seen by the source, Z_{in}, is the ratio of the source voltage to terminal current, or

$$Z_{in} = \frac{\bar{V}_s}{\bar{I}_1} = Z_g + j\omega L_1 + \frac{\omega^2 M^2}{Z_L + j\omega L_2},$$

but $$M^2 = L_1 L_2,$$

then $$Z_{in} = Z_g + j\omega L_1 + \frac{\omega^2 L_1 L_2}{Z_L + j\omega L_2}.$$

There must be total coupling between primary and secondary windings for an ideal transformer; thus, the self-inductances, L_1 and L_2, have no effect in the circuit, and their value can be considered infinite. Notwithstanding, the ratio is still finite, as specified by the turns ratio:

$$L_2 = a^2 L_1.$$

For this reason, primary and secondary inductances are conventionally not specified on ideal transformers. When this is related to the input impedance expression,

$$Z_{in} = Z_g + j\omega L_1 + \frac{\omega^2 a^2 {L_1}^2}{Z_L + j\omega a^2 L_1},$$

or rearranging,

$$Z_{in} = Z_g + \frac{j\omega L_1 Z_L}{Z_L + j\omega a^2 L_1}.$$

Now allowing L_1 to tend toward infinity, the input impedance for the voltage source becomes

$$Z_{in} = Z_g + \frac{Z_L}{a^2}. \tag{3.28}$$

Equation 3.28 is significant as it shows that the source sees the load impedance, Z_L, through the transformer as Z_L/a^2. This means that an ideal transformer has the capability to change an impedance magnitude. Therefore, to assist in circuit analysis, the circuit in figure 3.11*A* can be redrawn to its equivalent, shown in figure 3.11*B*. Here, the impedance connected to the secondary is transformed to the primary. Obviously, the reverse process, primary to secondary, also holds, but the impedance is multiplied by a^2. The impedance angle remains constant in either situation.

EXAMPLE 3.5

A 60-Hz single-phase transformer has a rated capacity of 250 kVA and a turns ratio of 15:1. Assuming that the transformer is ideal, find the primary voltage if the secondary voltage is 480 V. What are the magnitudes of primary and secondary currents with these voltages applied and the transformer operating at full capacity?

SOLUTION. For the turns ratio of 15:1,

$$a = \frac{1}{15}.$$

As the turns ratio specifies the secondary voltage to the primary,

$$a = \frac{V_2}{V_1},$$

and the primary voltage is

$$V_1 = \frac{V_2}{a} = 15(480) = 7{,}200 \text{ V}.$$

The primary current for 250 kVA at 7,200 V is

$$I_1 = \frac{250{,}000}{7{,}200} = 35 \text{ A},$$

and the secondary current for 250 kVA at 480 V is

$$I_2 = \frac{250{,}000}{480} = 521 \text{ A}.$$

EXAMPLE 3.6

Consider that the circuit shown in figure 3.11*A* has the following parameters:

$$Z_g = 6 + j3\ \Omega,$$

$$Z_L = 1 + j0.5\ \Omega,$$

$$V_s = 7{,}200\text{ V, 60 Hz},$$

$$\text{Turns ratio} = 12{:}1.$$

Find the value of the load impedance (Z_L) referred to the transformer primary, the complex power at the source, the transformer secondary voltage and current, and the required transformer capacity.

SOLUTION. For the specified turns ratio,

$$a = \frac{1}{12}.$$

Transferring the load impedance to the primary,

$$\frac{Z_L}{a^2} = (12)^2(1 + j0.5) = 144 + j72\ \Omega,$$

which is the impedance referred to the transformer primary. The total impedance seen by the source is then

$$Z_{eq} = Z_g + \frac{Z_L}{a^2}$$
$$= 6 + j3 + 144 + j72 = 150 + j75\ \Omega.$$

Assigning the source voltage as the reference phasor, the transformer primary current is

$$\bar{I}_1 = \frac{\bar{V}_s}{Z_{eq}} = \frac{7{,}200\underline{|0^\circ}}{167.7\underline{|26.6^\circ}} = 42.9\underline{|-26.6^\circ}\text{ A}.$$

The transformer secondary current is

$$\bar{I}_2 = \frac{\bar{I}_1}{a} = 12(42.9\underline{|-26.6^\circ}) = 515\underline{|-26.6^\circ}\text{ A},$$

and the secondary voltage is

$$\bar{V}_2 = \bar{I}_2 Z_L = (515\underline{|-26.6^\circ})(1 + j0.5)$$

or $$\bar{V}_2 = (515\underline{|-26.6^\circ})(1.12\underline{|26.6^\circ})$$
$$= 576\underline{|0^\circ}\text{ V}.$$

The complex power delivered to the load is then

$$\bar{S} = \bar{V}_2 \bar{I}_2^{\,*}$$

or $$\bar{S} = (576\underline{|0^\circ})(515\underline{|26.6^\circ}) = 296\underline{|26.6^\circ}\text{ kVA}.$$

This may also be found from

$$\bar{S} = I^2 Z_L = (515)^2(1 + j0.5)$$
$$= 265 + j133\text{ kVA}$$

or $$\bar{S} = 296\underline{|26.6^\circ}\text{ kVA}.$$

Finally, the apparent power demanded by this load is the required transformer capacity, 296 kVA.

ACTUAL TRANSFORMERS

In actual transformers, a source must furnish the power dissipated by the secondary load plus the power needed to operate the transformer. The additional power is created from losses within the transformer circuit. The transformer capacity, the amount of power it can handle, is dependent upon the character of these losses, which are dissipated as heat in the core and the windings. Because excessively high temperatures are destructive to insulation, the capacity is limited by this rise in temperature, usually specified as an allowable temperature rise above ambient conditions.

The major losses in an iron-core transformer are winding resistance (conductor loss), leakage reactance, eddy-current loss, and hysteresis loss. This section will expand upon the ideal-transformer concept to produce a transformer equivalent circuit that accounts for these losses and is a good approximation for real-world transformer performance under any condition.

Conductor Loss

As the conductors used for the transformer windings have resistance, current flowing in the primary and secondary produces an I^2R power loss that creates heat. The loss is minimized by conductors with larger cross sections, but if the resistance is too large to be neglected, primary resistance, R_1, and secondary resistance, R_2, can be placed in series with the ideal-transformer windings as shown in figure 3.12.

Leakage Reactance

For the ideal transformer, all the flux produced by the primary must link with the secondary winding. In the real world, however, a small percentage of the total flux produced fails to link all the secondary turns; this is called leakage flux. Leakage flux can be reduced by placing the primary and secondary windings very close together, perhaps interleaving them. Further reduction comes from

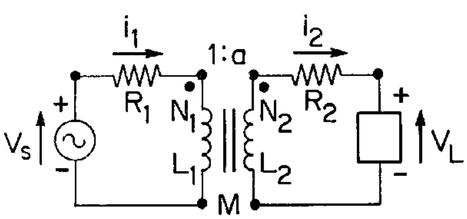


Figure 3.12.—Ideal transformer with winding resistance included.

winding the coils tightly on the core and providing a short magnetic path between them, thus creating a low-reluctance path between the coils. Nevertheless, even with the best transformer designs, leakage is significant and cannot be neglected.

Inductance is the ratio of flux linkage to the current producing the flux, or

$$L = \frac{Nd\phi}{di}, \tag{3.29}$$

where $d\phi$ = magnetic flux, Wb,
$Nd\phi$ = flux linkage of circuit, Wb,
and di = current producing flux, A.

For transformers with iron or ferromagnetic cores, current and flux do not have a linear relationship, and differentials must be used. Consider the time-varying primary current, i_1, in figure 3.13*A*, where the changing current produces the magnetic flux, ϕ_1, and

$$L_1 = \frac{N_1 d\phi_1}{di_1}. \tag{3.30a}$$

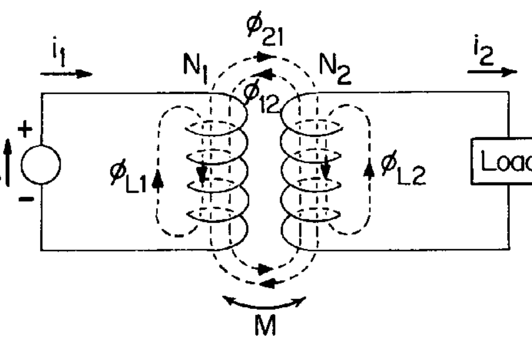
The part of ϕ_1 that links the secondary is ϕ_{12}; that which only links the primary (or is lost in terms of magnetic coupling) is ϕ_{L1}, where

$$\phi_1 = \phi_{12} + \phi_{L1} \tag{3.30b}$$

and

$$L_1 = \frac{N_1 d\phi_{12}}{di_1} = \frac{N_1 d\phi_{L1}}{di_1}. \tag{3.30c}$$

A



B

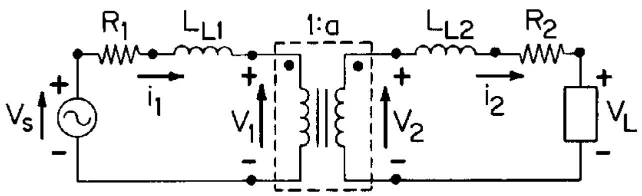


Figure 3.13.—Accounting for transformer leakage flux.

Similarly, although not shown in the figure,

$$L_2 = \frac{N_2 d\phi_2}{di_2}, \tag{3.31a}$$

$$\phi_2 = \phi_{21} + \phi_{L2}, \tag{3.31b}$$

$$L_2 = \frac{N_2 d\phi_{21}}{di_2} + \frac{N_2 d\phi_{L2}}{di_2}. \tag{3.31c}$$

Interestingly, the coefficient of coupling is also related to flux by

$$k = \frac{\phi_{12}}{\phi_1} = \frac{\phi_{21}}{\phi_2}. \tag{3.32}$$

The first term in equations 3.30*c* and 3.31*c* is the transformer mutual inductance, and the second terms are the primary and secondary leakage inductances, L_{L1} and L_{L2}, respectively, or

$$M = \frac{N_1 d\phi_{12}}{di_1} = \frac{N_2 d\phi_{21}}{di_2}, \tag{3.33a}$$

$$L_{L1} = \frac{N_1 d\phi_{L1}}{di_1}, \tag{3.33b}$$

$$L_{L2} = \frac{N_2 d\phi_{L2}}{di_2}. \tag{3.33c}$$

These equations hold for effective sinusoidal current. In steady ac analysis, the leakage inductances become leakage reactances; hence, flux leakage can be represented as an inductance or reactance. Figure 3.13*B* shows the additional elements that bring the transformer model closer to a practical transformer.

Core Losses and Exciting Current

Even with the addition of winding resistance and leakage reactance, equation 3.27*b* for an ideal transformer still applies and can be rewritten as

$$I_1 = aI_2.$$

Examination of this expression suggests that whenever I_2 is zero, I_1 must be zero. Yet, if an actual transformer primary is connected to an ac source and the secondary is left unconnected (fig. 3.14), the primary current will exist, albeit very small. Even though the secondary is open and I_2 is zero, V_2 appears across the secondary winding as a sinusoid. This implies that a changing flux in the transformer core must be produced by the current in the primary, as no other sources of changing flux are available. The portion of primary current that produces the changing flux, called magnetizing current, i_m, can be accounted for by adding an inductor, L_e, in parallel with the ideal-transformer primary winding.

The changing flux also induces small currents, eddy currents, in the transformer core material. These have an almost infinite number of closed paths and encircle practically all the flux. Since the transformer core has electrical resistance, the result is heat in the core and attendant power loss. These eddy currents flow at right angles to the magnetic field, as illustrated in the core cross section of figure 3.15*A*. The resistance along the eddy-current path is approximately proportional to the path length. Obviously, if the path length is decreased, the power dissipated in the eddy-current loop will drop. Figure 3.15*B* shows the core split lengthwise with a nonconducting layer between the two halves. The result is a desirable decrease in power loss to about two-thirds of the original power. In practice, transformers are laminated from several thin sheets of steel. Each sheet is sometimes covered with varnish to act as an insulant, but in most cases the oxide layer on each steel sheet is sufficient to produce the necessary high-resistance layers. This can substantially reduce eddy currents but cannot completely eliminate them.

As shown in figure 3.15*C*, energy is also dissipated in the transformer core each time a hysteresis loop is traversed. The energy is proportional to the area enclosed in the hysteresis loop and is called the hysteresis loss of the transformer core. In simple terms, the effect is related to the fact that the core retains some magnetism, and a coercive force is required to overcome this residual magnetism each time the ac current reverses. The loss is due to retentivity or molecular friction.

Both eddy-current and hysteresis losses are proportional to frequency and become a major consideration in high-frequency transformer applications. However, these core losses can be satisfactorily approximated at one frequency and one voltage. Good examples are 60-Hz power transformers where neither frequency nor voltage (actually, magnetic saturation of the core) changes drastically in normal operation. To account for these losses, a resistance, R_e, is again placed in parallel with the ideal-transformer primary. The sum of the currents through R_e and L_e is called exciting current, I_e, and the total current drawn by the source when the transformer is supplying power to a load is $I_1 + I_e$.

It should be noted that with sinusoidal input voltage to the primary, the exciting current is not a sinusoid but exhibits many harmonic frequencies because of the greatly varying permeability of the transformer core. However, for most purposes it may be assumed as a sinusoid with the same rms value.

The equivalent circuit shown in figure 3.16 now contains all the components necessary for it to be a useful model of a practical transformer. In summary, the important parameters for an equivalent circuit are

- R_1, primary conductor resistance,
- R_2, secondary conductor resistance,
- L_{L1}, primary leakage inductance,
- L_{L2}, secondary leakage inductance,
- R_e, a resistance accounting for eddy-current and hysteresis losses,
- L_e, an inductance accounting for magnetizing current, and
- An ideal transformer with turns ratio, $a = N_2/N_1$.

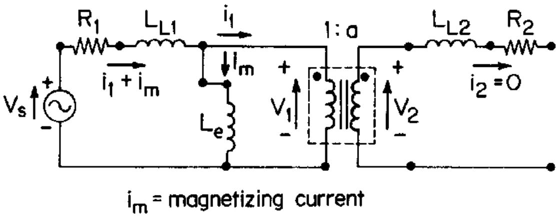


Figure 3.14.—Transformer magnetizing current.

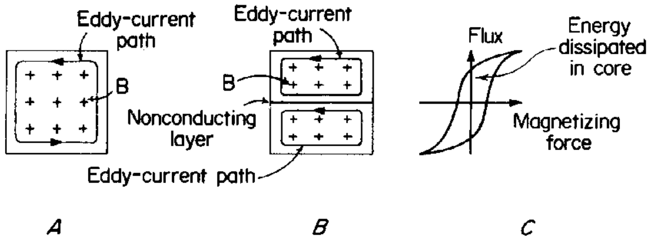


Figure 3.15.—Eddy current and magnetic hysteresis creating power loss in core.

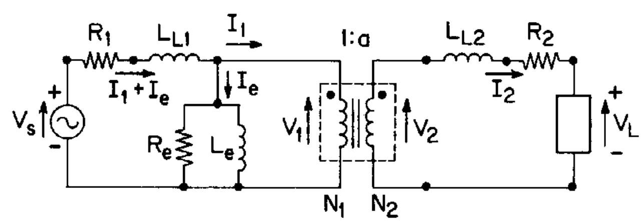


Figure 3.16.—Equivalent circuit of practical transformer.

Notice that mutual inductance, M, does not appear in the model since it is represented by the turns ratio of the ideal transformer.

Power-Transformer Construction

The two most widely used transformer types are the core and the shell. In shell construction, both primary and secondary windings are placed on an inner leg of the core. The windings are constructed in layers with an insulating barrier between them, forming a very low-leakage flux. In core construction, the primary and secondary windings are located on separate legs, thus providing maximum isolation between the coils. Both constructions are sketched in figure 3.17.

A copper or aluminum conductor is employed to construct each winding, which can have the form of an insulated wire with circular or rectangular cross section, or an uninsulated wide metal sheet. The insulated wire is continuously wound in layers, with each layer separated by a sheet of insulating material. With sheet-metal windings, the conductor is wound simultaneously with a continuous sheet of insulating material so that each adjacent conductor turn is separated by the insulation. The sheet metal is the same width as the transformer winding, and the insulation sheet is slightly wider.

Each winding is given a rated capacity, a rated current, and a rated voltage. These ratings depend upon the number of turns in the winding, the magnetic interaction with other windings, the current-carrying ability of the conductor, as well as the ability to dissipate heat through the insulation to the environment surrounding the winding. It should be obvious that the rated capacity, current, and voltage are mathematically related.

Transformer Models

Since voltage regulation, efficiency, and heating are of prime importance in mine power systems, detailed power-transformer analysis requires consideration of the complete equivalent circuit as shown in figure 3.16. However, because the exciting current, I_e, is normally very small compared with load current, I_1, a further approximation can be made by placing R_e and L_e at the transformer input terminals (fig. 3.18). This modification now allows the secondary winding resistance and leakage inductance to be transferred to the primary circuit (fig. 3.19) and combined with the primary elements. For many purposes, the exciting current is so small that R_e and L_e can be removed from the model. Figure 3.20 provides this last simplification, where the winding resistance and leakage reactance are said to be referred to the primary, and

$$R = R_1 + \frac{R_2}{a^2} \text{ and } L_L = L_{L1} + \frac{L_{L2}}{a^2}. \quad (3.34)$$

The primary is sometimes called the *high side* if its winding has a greater voltage rating (or more turns) than the secondary. The secondary is then called the *low side*. The terminology is reversed if the secondary has the higher voltage. In steady ac analysis, the inductance becomes a reactance, X_L, and

$$R = R_1 + \frac{R_2}{a^2} \text{ and } X_L = \omega(L_{L1} + \frac{L_{L2}}{a^2}), \quad (3.35a)$$

with the primary impedance simply

$$Z = R + jX_L. \quad (3.35b)$$

If desired, the primary impedance can be moved to the secondary of the ideal transformer (thus, referred to the secondary) by multiplying both terms by a^2.

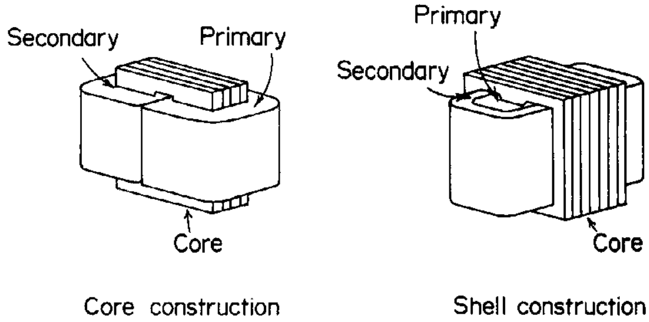


Figure 3.17.—Common power-transformer construction techniques.

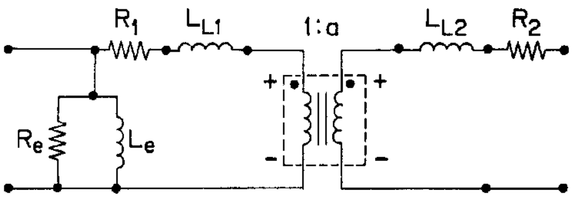


Figure 3.18.—Movement of exciting components to input.

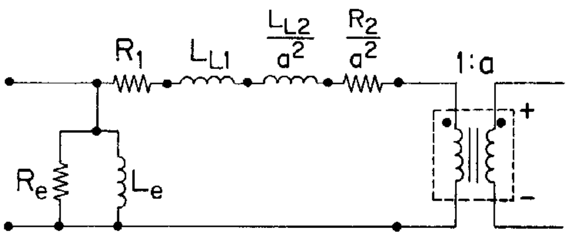


Figure 3.19.—Transferring secondary components to primary.

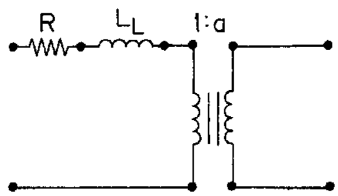


Figure 3.20.—Final simplification of practical circuit model.

EXAMPLE 3.7

A two-winding transformer has a rated capacity, primary-winding voltage, secondary-winding voltage, and frequency of 100 kVA, 2,400 V, 240 V, and 60 Hz, respectively. The primary-winding impedance is 0.6 + j0.8 Ω, while the impedance of the secondary winding is 0.005 + j0.007 Ω. The transformer is being used at the end of a feeder to step down voltage to a load. The feeder impedance is 0.05 + j0.1 Ω, and the load is 0.3 + j0.4 Ω. Find the magnitude of the voltage across the load if the voltage at the source end of the feeder is held constant at 2,400 V.

SOLUTION. As core-loss and magnetizing-current elements are not given for the transformer, they must be assumed to be negligible, with the transformer model being as shown in figure 3.20. The turns ratio is 1/10, and the transformer impedance is (equation 3.35)

$$Z_T = R_1 + jX_1 + \frac{R_2}{a^2} + j\frac{X_2}{a^2}$$

$$= 0.6 + j0.8 + \frac{0.005}{(1/10)^2} + j\frac{0.007}{(1/10)^2}$$

$$= 1.1 + j1.5\ \Omega.$$

The load impedance transferred to the primary is

$$\frac{Z_L}{a^2} = \frac{0.3 + j0.4}{(1/10)^2} = 30 + j40\ \Omega,$$

and the total impedance at the source end of the feeder is

$$Z_{eq} = Z_f + Z_T + \frac{Z_L}{a^2}$$

$$= 0.05 + j0.1 + 1.1 + j1.5 + 30 + j40$$

$$= 31.15 + j41.6 = 51.97\underline{|53.2°}\ \Omega.$$

The magnitude of current from the source is then

$$I_1 = \frac{V_s}{|Z_{eq}|} = \frac{2{,}400}{51.97} = 46.18\ \text{A},$$

which is also the current through the transformer primary. Therefore, the magnitude of voltage across the primary is

$$V_1 = I_1\left|\frac{Z_L}{a^2}\right| = 46.18(50) = 2{,}309\ \text{V},$$

and that across the secondary and the load is

$$V_2 = V_1 a = \frac{2{,}309}{10} = 231\ \text{V}.$$

Determination of Transformer Parameters

Two tests can provide the necessary elements for the transformer model in figure 3.19, where the exciting current components are at the primary terminals, and secondary parameters are referred to the primary.

The first test, the *open-circuit* or *excitation test*, is used to find the exciting-current components. The secondary of the transformer is unconnected. Rated voltage at rated frequency is applied to the primary winding, and a wattmeter (see chapter 5) is employed to measure the power, P_e, delivered by the source. An ammeter is used to measure rms exciting current, I_e (fig. 3.21*A*). The power corresponds to the core loss; in other words, P_e is dissipated by R_e. R_e can be found by

$$R_e = \frac{V^2}{P_e}, \tag{3.36}$$

where V = rms applied voltage, V,
P_e = measured average power, W,
and R_e = core-loss resistance, Ω.

From the rms value of I_e, assuming exciting current to be a sinusoid, the input admittance of R_e and L_e in parallel can be calculated from

$$Y_e = \frac{I_e}{V}. \qquad (3.37a)$$

Realizing that

$$Y_e = \frac{1}{R_e} + \frac{1}{j\omega L_e}, \qquad (3.37b)$$

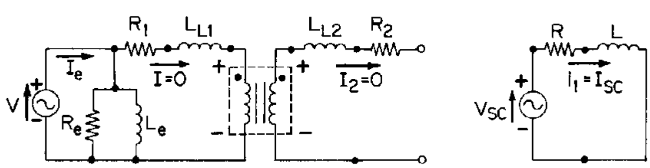
the component accounting for magnetizing current, L_e, can be obtained from

$$\frac{1}{\omega L_e} = \left(\frac{I_e^2}{V^2} - \frac{1}{R_e^2}\right)^{1/2}, \qquad (3.37c)$$

where I_e = measured rms value of exciting current, A,
$\omega = 2\pi f$ = applied frequency of source (must be rated frequency of transformer for exact results),
and L_e = magnetizing inductance, H.

Therefore, both R_e and L_e can be determined from the open-circuit excitation test.

The second test is projected at winding resistance and leakage resistance, with both primary and secondary



A Open circuit *B* Short circuit

Figure 3.21.—Transformer parameter test series.

values combined. This is termed the *short-circuit* or *impedance test*. Here, the secondary terminals are short-circuited and a source is connected to the primary. Voltage at rated frequency is applied to the transformer but at reduced amplitude, so that it produces only rated current in the primary winding and, thus, rated current in the secondary. Current, I_{sc}, and input average power, P_{sc}, are again measured.

The applied voltage for the test is typically much smaller than rated voltage. Yet the short-circuit (actually, rated) current is much greater than the exciting current, so I_e and the associated components can be neglected. As given by figure 3.21*B*, the equivalent circuit under these conditions can be simplified to a simple series RL combination. The ideal transformer is not needed because the zero load impedance (short circuit), when transferred to the primary, is still zero. Winding resistance and leakage inductance can thus be found from

$$R = \frac{P_{sc}}{I^2_{sc}} \tag{3.38}$$

and

$$\omega L_L = \left(\frac{V^2_{sc}}{I^2_{sc}} - R^2\right)^{1/2}, \tag{3.39}$$

where P_{sc} = measured average power, W,
I_{sc} = measured rms short-circuit current, A,
V_{sc} = applied rms short-circuit potential, V,
ω = $2\pi f$ = rated frequency, rad/s,
R = primary and secondary winding resistance, Ω,
and L_L = primary and secondary leakage inductance, H.

It is important to note that these values are valid only for the frequency under which the tests are made. Further, it is neither possible nor necessary to break the resulting components into primary and secondary elements.

Transformer Efficiency and Regulation

The transformer is designed to be a highly efficient device. However, the output power of a transformer is always less than its input power because of winding conductor losses and core losses. The term *efficiency* is used to measure the ability of a transformer to transfer energy from the primary circuit to the secondary circuit. The efficiency is defined as the average-power ratio:

$$\eta = \frac{P_{out}}{P_{in}} \tag{3.40a}$$

or

$$\eta = \frac{P_{in} - \text{losses}}{P_{in}} = \frac{P_{out}}{P_{out} + \text{losses}} \tag{3.40b}$$

The ratio is always less than 1 but normally in the range n = 0.95 to 0.98. Efficiency decreases when the device is operated above or below its voltampere capacity.

Voltage regulation is a characteristic of power systems that describes the voltage fluctuations resulting from varying load or current conditions. Voltage regulation is often applied to transformer secondary-voltage variations and is defined as

$$\text{V.R.} = \frac{V_{NL} - V_{FL}}{V_{FL}}(100\%), \tag{3.41}$$

where V_{FL} = transformer output voltage at full rated secondary current and rated primary voltage, V,
and V_{NL} = transformer output voltage with no secondary load but rated primary voltage applied, V.

V_{FL} and V_{NL} are also called the *full-load* and *no-load* voltages, respectively. It should be clear that voltage regulation is a function of transformer losses, impedance, and efficiency. The concept is extremely important in mine power systems as it often limits how far a mine can be safely expanded from one power source.

EXAMPLE 3.8

For the circuit shown in figure 3.22, find the complex power consumed by the transformer load, Z_L. If the figure represents the full-load condition, what is the voltage regulation at the transformer secondary? The transformer is considered ideal.

SOLUTION. The impedance seen by the 5,000-V source is

$$Z_{eq} = 1 + j1 + \frac{Z_L}{a^2}$$

$$= 1 + j1 + \frac{(0.1 + j0.1)}{(1/10)^2}$$

$$= 11 + j11 = 15.56\underline{|45^\circ}\ \Omega.$$

Using the source voltage as the reference phasor, the current delivered from the source is

$$\bar{I}_1 = \frac{5{,}000\underline{|0^\circ}}{15.56\underline{|45^\circ}} = 321.2\underline{|-45^\circ}\ \text{A},$$

and the transformer secondary current is

$$\bar{I}_2 = \frac{\bar{I}_1}{a} = 10(321.3\underline{|-45^\circ}) = 321.3\underline{|-45^\circ}\ \text{A}.$$

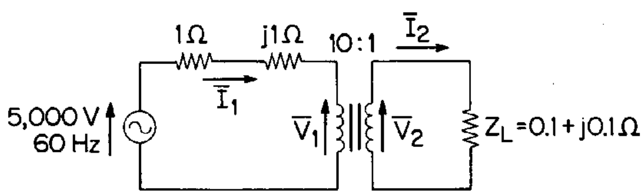


Figure 3.22.—Circuit for example 3.8.

The voltage across the transformer secondary is

$$\overline{V}_2 = \overline{I}_2 Z_L = (3{,}213\underline{|-45°})(0.141\underline{|45°})$$
$$= 454.4\underline{|0°}\ \text{V},$$

and the complex power delivered to Z_L is

$$\overline{S}_L = \overline{V}_2\,\overline{I}_2{}^* = (454.4\underline{|0°})(3{,}213\underline{|45°});$$

therefore, $\overline{S}_L = 1.46\underline{|45°}$ MVA.

If the above situation represents the full-load condition, then

$$V_{FL} = V_2 = 454.4 \text{ V}.$$

Under no-load conditions, the load impedance becomes such a high impedance that the transformer secondary current approaches zero. With no secondary current, current to the primary of an ideal transformer is also zero. Therefore, the voltage across the primary is equal to the source voltage or

$$V_1 = V_s = 5{,}000 \text{ V}.$$

The secondary voltage becomes

$$V_2 = a\,V_1 = \frac{5{,}000}{10} = 500 \text{ V}.$$

Consequently, $V_{NL} = V_2 = 500$ V,

and from equation 3.41,

$$\text{V.R.} = \frac{500 - 454}{454}(100) = 10\%.$$

AUTOTRANSFORMERS

All the transformers discussed so far have been two-winding transformers and have provided electrical isolation between the primary and secondary windings. Another type of transformer, the autotransformer, uses a single winding and does not provide electrical isolation. It is constructed from a continuous winding with a tap connected at a specific point. The autotransformer is compared with an ideal two-winding transformer in figure 3.23. The advantages and disadvantages of each type of transformer can be illustrated with reference to figure 3.24, where a normal two-winding transformer is shown on the left and is connected to operate as an autotransformer.

The two-winding transformer has the following specifications:

N_1, V_1, I_1 = primary turns, rated rms voltage, and rated rms current,

N_2, V_2, I_2 = secondary turns, rated rms voltage, and rated rms current,

and the maximum apparent power that can be delivered to a secondary load is

$$S_{out} = V_2 I_2.$$

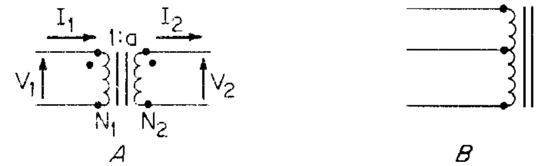


Figure 3.23.—Comparison of two-winding transformer (*A*) and autotransformer (*B*).

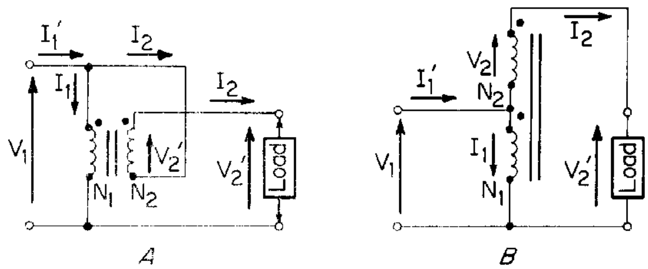


Figure 3.24.—Two-winding transformer as an autotransformer.

To help visualize the autotransformer action, figure 3.24*A* is redrawn in figure 3.24*B* with both windings placed on the same side of the core symbol. For either figure, the output voltage, V_2', is now

$$V_2' = V_1 + V_2.$$

Transformer rated output current, I_2, is still related to rated primary current, I_1, by

$$I_1 = aI_2,$$

but input current to the autotransformer is now

$$I_1' = I_1 + I_2.$$

The maximum power that can be transferred to a load at rated output current, I_2, is now

$$S_{out} = V_2' I_2 + V_1 I_2 + V_2 I_2.$$

This expression indicates that the transformer is now able to deliver an increase of $V_1 I_2$ voltamperes over the two-winding connection, yet the transformer windings are still within rated currents and voltages. The reason for the increase is that some input current is transformed by the transformer while the rest is conducted directly to the load. This is the main advantage of the autotransformer over two-winding arrangements. Because primary current is now only a portion of load current, conductor losses in autotransformers are particularly small, and voltage regulation under varying load conditions is usually good.

MULTIVOLTAGE TRANSFORMERS

The transformers considered so far have had only one secondary, but in practice many have two or more secondary windings. The transformer with two secondary windings in figure 3.25*A* is able to serve loads with different voltage requirements from one source. In such devices the magnetic interaction increases substantially over the two-winding variety because mutual inductance exists between all winding combinations. Taking this into account, the preceding theory can be expanded to model an equivalent circuit.

Another method for one transformer to serve several voltage applications is to have winding taps on the primary (fig. 3.25*B*), the secondary (fig. 3.25*C*), or both. When used on the input winding, a higher tap can be selected to account for voltage drops in the circuit that delivers power to the transformer, thus maintaining a desired output voltage. This is a common practice in mining. A special but very widely used application for secondary taps is in utility distribution transformers supplying 240- and 120-V ac service. Here, the winding is center-tapped with equal turns on either side. The voltage magnitude from either line to the tap is 120 V, and across the total winding, 240 V is available.

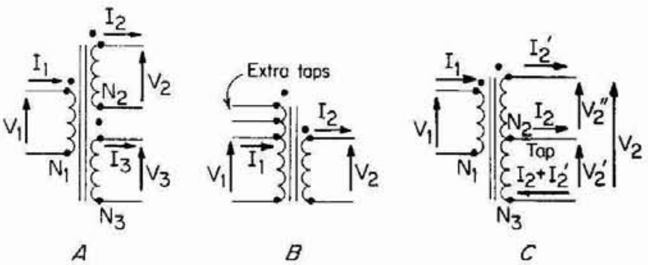


Figure 3.25.—Examples of transformers for multivoltage applications.

CURRENT AND POTENTIAL TRANSFORMERS

The prime use of transformers in mine power systems should now be apparent: to supply power at different voltage levels to system portions and equipment. Transformers are also used extensively to power control circuits, mainly to provide power for circuit breakers and associated circuitry; to power protection devices, usually relays to trip circuit breakers; and for instrumentation. Transformers employed for these applications are often given specific names: *potential transformers* (PT's) and *current transformers* (CT's). PT's are merely high-quality two-winding transformers with or without taps. The name is modified because they are used to sense voltage.

The current supplied to relays, instruments, and similar equipment is normally provided by CT's. Some CT's are like the two-winding devices that have just received so much attention. These have a primary with just a few turns of high-current-capacity conductor and a secondary with numerous turns, as illustrated in figure 3.26*A*. The turns ratio (N_2/N_1) is normally adjusted so that the secondary supplies 5 A when full-load current flows in the primary. The primary is placed in series with the circuit that is to be measured, and therefore, CT's can be considered as sensing current.

Two-winding CT's for high-voltage or high-current circuits, such as those usually found in mine power systems, are very expensive, and as a result bushing-type or donut CT's are more often used. In figure 3.26*B*, the conductor to be measured passes through a large-diameter ring-shaped laminated iron core and acts as the transformer primary. The secondary winding, which consists of several turns about the core, supplies current as before. The leakage reactance of this type of CT is obviously high and, coupled with other parameters, results in a low accuracy for current measurements. A schematic illustrating hypothetical placements of a PT and a CT in a simple circuit is provided in figure 3.27.

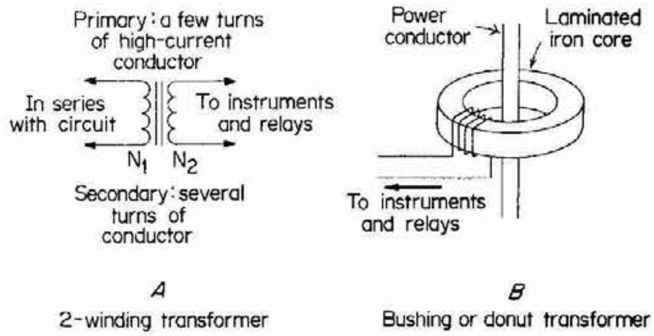


Figure 3.26.—Two types of CT's.

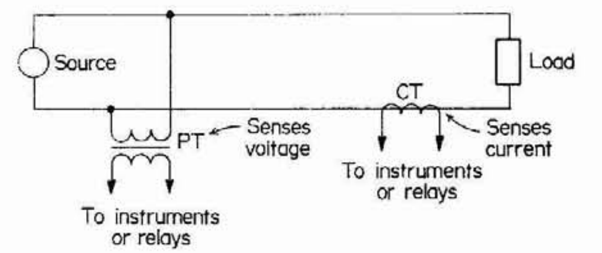


Figure 3.27.—Examples of CT and PT placement in circuit.

PT's and CT's are important components in instrumentation and protective circuitry for mine power systems. Their application for instrumentation is presented in chapter 5, while chapters 9 and 10 cover their use in protective relaying.

The purpose of the foregoing two chapters was to cover many of the basic theoretical aspects behind mine electrical systems. The content was directed towards dc and single-phase ac, and spanned fundamental electrical phenomena, the experimental laws and parameters, dc and ac circuit analysis, and finally, power transformers. Comprehension of these laws, parameters, and concepts is essential for the understanding of subsequent chapters. This will be very apparent in the next chapter, which introduces power-system concepts and three-phase circuit basics.

CHAPTER 4.—POWER-SYSTEM CONCEPTS

Power systems can be simply described as systems that transmit power from a source to the loads. For the mine, the source is often the secondary of a substation transformer and the loads are motors on mining machinery and ancillary equipment. The transmission of power is commonly performed by three-phase systems, which are by nature more complex than the dc and single-phase ac circuits introduced in the previous two chapters. The following sections are primarily concerned with three-phase power systems plus the basic tools and special mathematics needed to study them. Several references are provided at the end of the chapter. As most information is considered common electrical engineering knowledge, specific references are seldom cited but can be found in the bibliography.

BASIC POWER CIRCUIT

Many power systems or system segments can be reduced to the simple series circuit shown in figure 4.1. This familiar single-phase ac circuit consists of a source or supply voltage, an impedance, and a load or receiver voltage. Such a representation is often called the Thévenin's equivalent of the power system. Finding the series circuit may involve many simplifying assumptions or procedures, some of which are yet to be covered, but the result has numerous applications for analyzing the behavior of electrical power systems.

One specific example is analysis of voltage regulation. Here, the source voltage is kept constant, and variations of the load voltage are observed with a range of load-current conditions that cause a change in voltage drop across the impedance. Applying this example to an underground coal mine, the source could be the secondary of a power-center transformer, the impedance could be that of the trailing cable, and the load might be the motors of a continuous miner. On a larger scale, a substation output voltage, a feeder cable, and power-center primary voltages could constitute a desired Thévenin's equivalent for analysis. Both these situations are illustrated in figure 4.2. As the chapter unfolds, more applications will become apparent.

Actual analysis of the basic power circuit (fig. 4.1) can use any applicable technique already given in chapters 2 and 3. For instance, employing the impedance domain and Kirchhoff's voltage law yields

$$\overline{V}_s - \overline{I}Z - \overline{V}_L = 0 \tag{4.1a}$$

or

$$\overline{V}_s = \overline{V}_L + \overline{I}Z \tag{4.1b}$$

and

$$\overline{V}_L = \overline{V}_s - \overline{I}Z. \tag{4.1c}$$

Any variable or constant in these equations can be a complex expression. Nevertheless, the equations describe the *performance* of the power system that the circuit represents, that is, the source voltage for a specific load current and load voltage, and so forth. When three-phase systems are involved, the solution or even the finding of the equivalent circuit must also utilize the additional methods that follow.

THREE-PHASE CIRCUITS

The term *single phase* has been applied to ac systems where power is delivered from a single sinusoidal source. When power is transmitted to a load by applying two or more sinusoidal sources with fixed phase differences, the power system is called *polyphase*. The most popular system that delivers large quantities of power, including both single phase and polyphase, is the three-phase system.

The analysis of three-phase circuits can be extremely complicated. Special techniques have been developed to assist in general problem solutions, but even so, the work can be cumbersome. However, three-phase systems are purposely designed to be balanced, and if actual differences existing among phases can be neglected, the analysis of three-phase circuits can be almost as simple as analysis of single-phase circuits.

BALANCED THREE-PHASE CIRCUITS

Balanced three-phase power consists of three generated voltages, each of equal magnitude and frequency but separated by 120°. When these voltages are applied to a system of balanced impedances, balanced currents result. In other words, a balanced three-phase power system can be divided into three portions. Any voltage or current in one portion has a counterpart in another portion, which is identical but 120° out of phase.

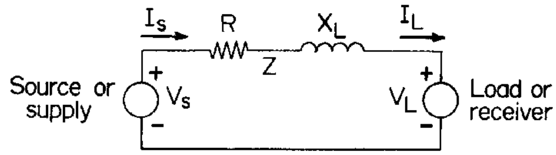


Figure 4.1.—Basic power circuit.

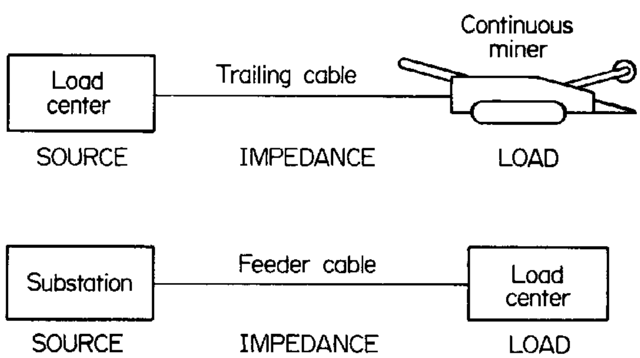


Figure 4.2.—Applications of basic power circuit.

To illustrate this voltage generation, consider the elementary three-phase generator illustrated in figure 4.3*A*. The armature consists of three single stationary conductors displaced by 120°, and a magnetic field structure rotates counterclockwise within. As the rotating magnetic flux cuts each winding, a voltage is induced. These voltages are out of phase with one another, as shown in figure 4.3*B*. A composite of these instantaneous voltages is provided in figure 4.3*C* to exemplify the phase relationships, which also can be clarified with phasors (fig. 4.3*D*). It can be noted with either representation that the voltage in winding aa' reaches a maximum first, followed by bb', and then cc'. This defines the positive sequence, abcab. . ., that is evident from the counterclockwise rotating phasors of figure 4.3*D*. If the phasors are allowed to rotate in the opposite direction (clockwise), the sequence termed negative (cbacba . . .).

An outstanding advantage of balanced three-phase systems is that they provide a more uniform flow of energy than single-phase or even two-phase systems. The 120° timing means that the individual power waves in each phase never reach zero at the same time, and more important, the total instantaneous power from all three phases remains constant. For three-phase motors, this translates to convenient starting, constant torque, and low vibration. It would seem logical that if three phases provide a substantial increase in operation efficiency, more equally spaced phases would result in even greater improvement. However, three-phase systems are generally more economical than other polyphase systems because the complications caused by additional phases offset the slight efficiency increase.

A source supplying these three-phase voltages is normally connected in either delta or wye. As shown in figure 4.4, either configuration can, in practice, be closely approximated by ideal voltage sources or in some cases by ideal voltage sources in series with small internal impedances. Three-phase sources always have three terminals, which are called *line terminals*, but may also have a fourth terminal, the neutral connection. These terminals produce three separate potentials between any two line terminals that are called *line-to-line voltages*. Also generated are three separate voltages between each line terminal and the neutral, be it a direct connection as in figure 4.4 or some imaginary neutral point. These are termed *line-to-neutral potentials*.

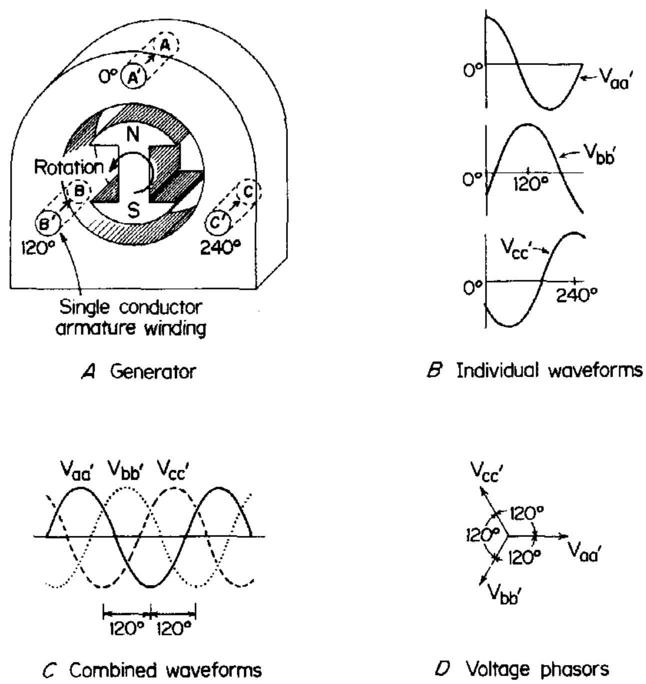


Figure 4.3.—Elementary three-phase generation.

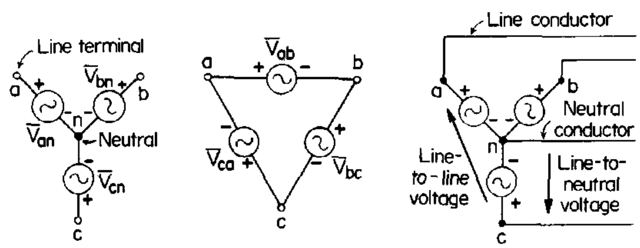


Figure 4.4.—Three-phase voltage sources.

Three-Phase System Voltages

Line-to-line voltage can be considered as a condition existing between two phases, while line-to-neutral is a condition for one phase only. Obviously, interrelationships must exist between these two voltage notations, as well as among the voltages of one notation. The wye-connected source of figure 4.5*A* can be employed to demonstrate the correspondence.

If the line-to-neutral voltages, $\bar{V}_{an}$, $\bar{V}_{bn}$, and $\bar{V}_{cn}$, are positive sequences and the phasor of $\bar{V}_{an}$ is taken as reference, then $\bar{V}_{bn}$ and $\bar{V}_{cn}$ are related to $\bar{V}_{an}$ by

$$\bar{V}_{bn} = \bar{V}_{an}\angle -120^\circ,\ \bar{V}_{cn} = \bar{V}_{an}\angle -240^\circ, \qquad (4.2a)$$

or

$$\bar{V}_{an} = \bar{V}_{bn}\angle +120^\circ = \bar{V}_{cn}\angle -120^\circ. \qquad (4.2b)$$

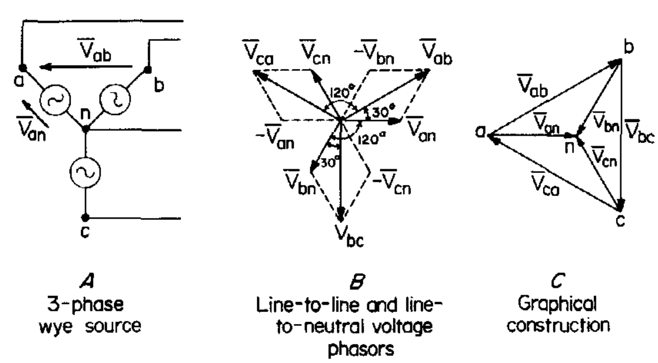


Figure 4.5.—Wye-connected source demonstrating line-to-line and line-to-neutral voltages.

Equations 4.2a and b relate that if a specific phasor representing one phase voltage is rotated 120°, it is identical to the phasor for another phase. By Kirchhoff's voltage law, the line-to-line voltage is equal to the sum of the two line-to-neutral voltages; for instance, between phases a and b,

$$\overline{V}_{ab} = \overline{V}_{an} + \overline{V}_{nb}, \tag{4.3a}$$

but $\overline{V}_{nb} = -\overline{V}_{bn}$

and $\overline{V}_{bn} = \overline{V}_{an}\underline{|-120°}$;

hence,
$$\overline{V}_{ab} = \overline{V}_{an} - \overline{V}_{an}\underline{|-120°} \tag{4.3b}$$

or
$$\overline{V}_{ab} = \sqrt{3}\,\overline{V}_{an}\underline{|+30°}. \tag{4.3c}$$

Equation 4.3c is truly significant because it states the relationship between line-to-line and line-to-neutral voltages for balanced three-phase systems. In particular, the following can be extracted:

$$|V_{ab}| = |V_{bc}| = |V_{ac}|$$
$$= \sqrt{3}\,|V_{an}| = \sqrt{3}\,|V_{bn}| = \sqrt{3}\,|V_{cn}| \tag{4.4a}$$

and
$$|V_{an}| = |V_{bn}| = |V_{cn}|$$
$$= \frac{1}{\sqrt{3}}|V_{ab}| = \frac{1}{\sqrt{3}}|V_{bc}| = \frac{1}{\sqrt{3}}|V_{ca}| \tag{4.4b}$$

It is important to note that, in addition to the foregoing identities, for a balanced three-phase system,

$$\overline{V}_{an} + \overline{V}_{bn} + \overline{V}_{cn} = 0 \tag{4.5a}$$

and
$$\overline{V}_{ab} + \overline{V}_{bc} + \overline{V}_{ca} = 0. \tag{4.5b}$$

A phasor diagram illustrating all line-to-line and line-to-neutral voltages of these systems is given in figure 4.5B. Here the correspondence by equation 4.3c is apparent.

The reasoning used for voltages can be applied to currents, and this will be handled shortly.

Load Connections

As with sources, balanced three-phase loads can be connected delta or wye. However, the interest in three-phase circuits comes from how delta or wye sources supply power to delta or wye loads. The usual combinations or systems are

- Four-wire, wye to wye;
- Three-wire, wye to wye;
- Three-wire, wye to delta;
- Delta to delta; and
- Four-wire, wye to delta.

By analyzing each combination, certain advantages and disadvantages can be seen, and some important points about balanced three-phase systems can be gained. For purposes of discussion, the lines connecting sources to loads are assumed to have no impedance, although obviously, in the real world, they must have impedance.

Figure 4.6A shows the first arrangement to be considered, the four-wire wye to wye. The source here could be either a generator or the secondaries of an ideal three-phase transformer, and the load, Z_a, Z_b, Z_c, could be a motor. These conductors are connected between the source line terminals and the load; the fourth conductor, the neutral return (or just simply, the neutral), connects the neutral of the source to the common junction of the three load impedances.

For perfect conditions, the generation is balanced, distribution impedances per phase (again assumed zero here) are equal, and the load impedance in each phase circuit is identical. Hence, the magnitude of the line currents, $\overline{I}_a$, $\overline{I}_b$, and $\overline{I}_c$, must also be equal. By Kirchhoff's

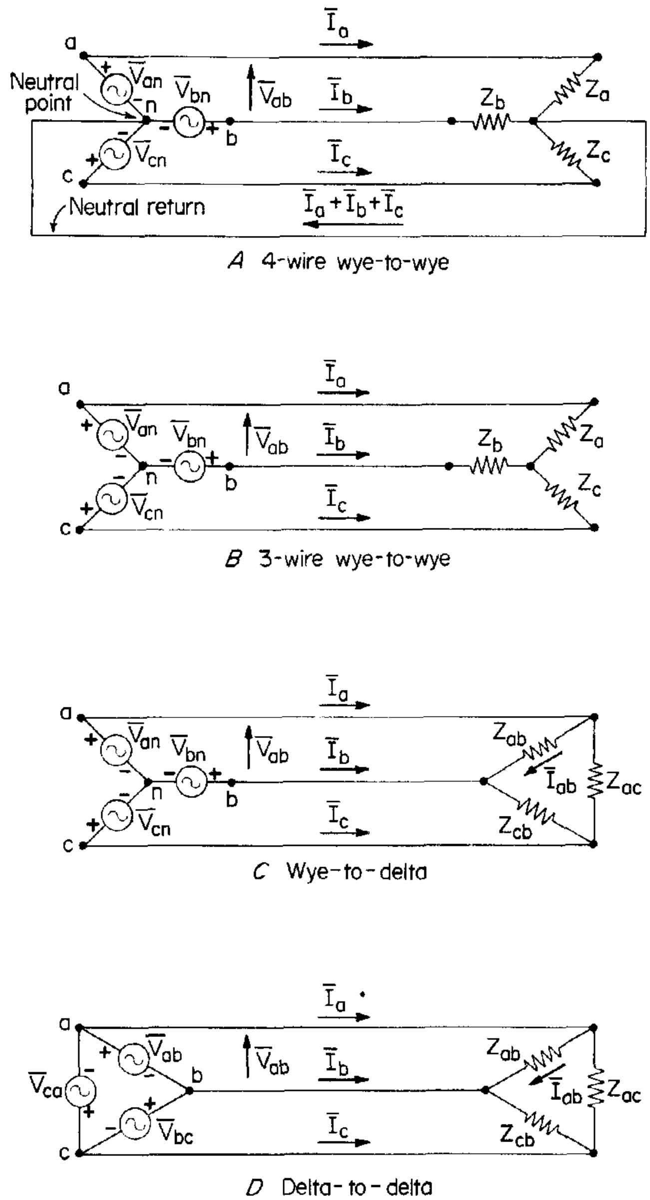


Figure 4.6.—Balanced three-phase load connections.

current law and the 120° displacement of the three line currents, the neutral-return current must be

$$\bar{I}_a + \bar{I}_b + \bar{I}_c = 0, \tag{4.6}$$

which means the neutral conductor actually carries no current under this ideal situation. Furthermore, there will be no voltage drop across the neutral, no matter what the neutral impedance is. In other words, the potential at the neutral of the source equals that of the load.

If the neutral carries no current under balanced conditions, what purpose does it really serve and can it be removed? Consider figure 4.6*B*, a three-wire wye-to-wye system, which does not employ the neutral conductor. Although this system is used in some applications, problems can arise, and the role of the neutral conductor is to minimize these problems.

In the real world, no balanced three-phase system can be perfect, and the sources, the distribution impedances, and the loads can easily become unbalanced, that is, unequal from phase to phase. The result is unbalanced currents and voltages. For example, without the neutral conductor, the neutral of the source will not equal the load neutral, and the resulting load unbalance will produce unequal voltages across the loads, no matter how balanced the source. Under this condition, a mining machine motor is likely to deteriorate and the result will be maintenance problems. In addition, safety problems can abound as a result of the unequal neutral potentials alone. Chapter 7 will investigate many of these problems in detail.

It is apparent that the neutral conductor does serve a vital role in actual three-phase power systems. Its size and current-carrying ability do not need to match those of the phase conductors in order to provide the necessary function. In a properly operating power system, normal conditions do cause some neutral current, but this is usually very small compared with the phase current. Hence, neutral conductors could be small if they were based only on the size of the neutral current, but in mining applications, this is not the only criterion. Possible system malfunctions must also be taken into account, and these will be discussed in a later section on unbalanced three-phase circuits.

A three-phase load is more likely to be delta connected than wye connected. The three-wire wye-to-delta system, shown in figure 4.6*C*, is an example of this arrangement. The prime advantage is that under unbalanced load conditions, the source will deliver power proportionately to each load. Hence the delta-connected loads need not be precisely balanced. Flexibility is increased because phase-to-phase loads may be added or removed without significantly upsetting system operation. With wye-connected loads such changes are difficult or nearly impossible to make.

A delta-connected source is shown in figure 4.6*D*. Although this arrangement can be found, it has two major disadvantages. First, a slight unbalance in the source can create large circulating currents around a delta loop (for example, source V_{ab} and load Z_{ab}). This extra current can reduce the available current capacity of the source and also increase power losses in the system. Second, it is more difficult for safety purposes to maintain metallic equipment frames at the neutral potential of the source. The logical and most economical point to employ as a ground is the neutral of the wye-connected source. This system is known as a four-wire wye-to-delta system and is illustrated in figure 4.7. It is presently the most popular three-phase power connection arrangement in mining. The neutral conductor here is more often termed a *grounding conductor.* A neutral point can also be derived from a delta source using a zig-zag or grounding transformer (see chapter 7).

Line and Phase Currents

Currents in a specific phase conductor or in one leg of a wye-connected source or load are termed *line currents.* As with line-to-neutral voltages, they can be considered as a condition of one phase only. Currents flowing between two phases are called *phase currents* (or *line-to-line currents*) and correspond to line-to-line voltages. An obvious example of phase current is that flowing through one leg of a delta-connected load. As might be assumed, for the balanced three-phase system, the magnitudes of the three phase currents through the legs of the delta are equal. Figure 4.8*A* shows a schematic of a balanced delta load with three line currents I_a, I_b, I_c and three phase currents I_{ab}, I_{bc}, I_{ca}. It can be utilized to demonstrate the relationship between line and phase currents. Considering only phase a and using Kirchhoff's current law,

$$\bar{I}_a = \bar{I}_{ab} - \bar{I}_{ca}. \tag{4.7a}$$

From the same reasoning that related line-to-neutral to line-to-line voltages,

$$\bar{I}_a = \sqrt{3}\,\bar{I}_{ab}\underline{|-30°}, \tag{4.7b}$$

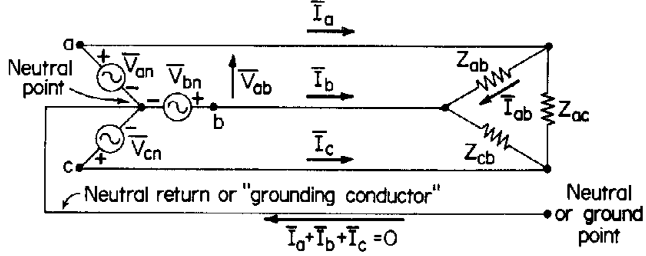


Figure 4.7.—Four-wire wye-to-delta system.

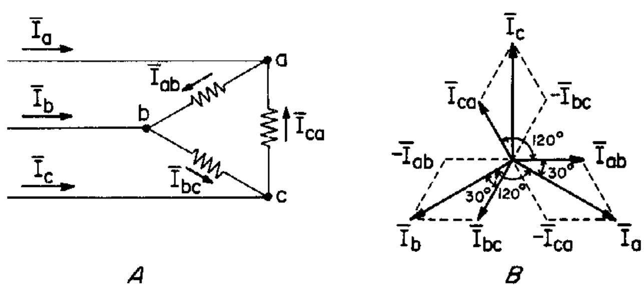


Figure 4.8.—Balanced delta load illustrating phase and line currents.

which means that in the balanced case the magnitude of line current is larger than phase current by a factor of $\sqrt{3}$. The phasors are displaced by 30°. The symmetry of phase and line currents is shown in figure 4.8*B*.

Equivalent Delta and Wye Loads

There are many instances where it is desirable to replace a balanced delta-connected load with a wye, or vice versa. The groundwork to perform this change has already been established in chapter 3. From equation 2.48,

$$Z_{an} = \frac{Z_{ab}\,Z_{ca}}{Z_{ab} + Z_{bc} + Z_{ca}}, \tag{4.8}$$

and so on for Z_{bn} and Z_{cn} in terms of the delta impedances. Equation 4.8 provides equivalence of delta and wye for all situations, including unbalanced loads. However, for balanced conditions, the expression reduces to simply

$$Z_{an} = \frac{Z_{ab}}{3} \tag{4.9a}$$

or

$$Z_{ab} = 3Z_{an}. \tag{4.9b}$$

This states that each branch of a balanced delta has three times the impedance of a balanced wye.

Now that voltages, current, and equivalent load impedances of balanced three-phase systems have been covered, these values can be compared for delta and wye loads. If the load is wye connected, the line current and load current per phase are the same, but the voltage across each load impedance is line-to-neutral, $1/\sqrt{3}$ that of line-to-line. When the load is delta connected, the voltage across one load impedance is line-to-line, while the line current is larger than the phase current through each load impedance by a factor of $\sqrt{3}$. These concepts are illustrated in figure 4.9 for equivalent delta and wye loads. It is significant to note that the three line-to-line voltages and three line currents for either connection are identical.

Three-Phase Power

Because the voltage and current are the same in each impedance of a balanced delta or wye load, the average power consumed by one impedance is one-third of the total power to the load. In a delta load as in figure 4.9*A*, current and voltage are phase and line-to-line, respectively, and the angle between them is the angle of impedance. Thus, considering phases a and b, the average power consumed by one element is

$$P_{p\Delta} = V_{ab}I_{ab}\cos\theta, \tag{4.10}$$

and total power is

$$P_T = 3\,V_{ab}I_{ab}\cos\theta, \tag{4.11a}$$

or in general,

$$P_T = 3\,V_{LL}I_P\cos\theta, \tag{4.11b}$$

where $P_{p\Delta}$ = average power consumed by each element of a delta load, W,
P_T = total power consumed by delta load, W,
V_{LL} = line-to-line (or system) voltage rms magnitude, V,
I_p = magnitude of phase rms current through load, A,
and $\cos\theta$ = power factor of load.

When the load is wye connected, line current is through each load, while the voltage is line-to-neutral. Hence, taking phase a (fig. 4.9*B*), the average power to one element is

$$P_{py} = V_{an}I_a\cos\theta, \tag{4.12}$$

and total power is

$$P_T = 3\,V_{an}I_a\cos\theta, \tag{4.13a}$$

or

$$P_T = 3\,V_{Ln}I_L\cos\theta, \tag{4.13b}$$

where P_{py} = average power consumed by each element of wye load, W,
P_T = total power consumed by wye load, W,
V_{Ln} = magnitude of line-to-neutral rms voltage, V,
I_L = line rms current magnitude, A,
and $\cos\theta$ = power factor of load.

It is important to note that the power-factor angle, θ, is referenced to the sinusoidal voltage across one load and the current through that load.

The standard measurement values for three-phase circuits are line-to-line voltage and line current, which are often the known quantities. Since for balanced systems,

$$V_{LL} = \sqrt{3}\,V_{Ln}$$

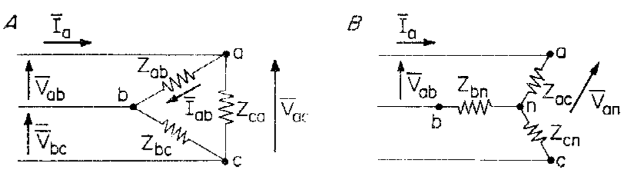
and

$$I_L = \sqrt{3}\,I_p,$$

both equations 4.11*b* and 4.13*b* are also identical to

$$P_T = \sqrt{3}\,V_{LL}I_L\ (\text{load pf}), \tag{4.14}$$

where the power-factor angle is that of a load impedance or an equivalent impedance. It is important to realize that this angle has nothing to do with the angle between V_{LL} and I_L, for example, $\overline{V}_{ab}$ and $\overline{I}_a$. Of the three three-phase average-power formulas, equation 4.14 is by far the most used.



If $Z_{ab} = Z_{ca} = Z_{bc}$, $V_{ab} = \sqrt{3}\,V_{an}$, $Z_{an} = Z_{ab}/3$ and $I_a = \sqrt{3}\,I_{ab}$

If $Z_{an} = Z_{bn} = Z_{cn}$, $V_{an} = V_{ab}/\sqrt{3}$, $Z_{ab} = 3Z_{an}$ and $I_{ab} = I_a/\sqrt{3}$

Figure 4.9.—Comparison of equivalent delta (*A*) and wye (*B*) loads.

Following the single-phase presentation of chapter 3, a balanced three-phase load has reactive power, Q_T, and apparent power, S_T, in addition to average power, P_T. The following expressions apply:

$$Q_T = \sqrt{3}\, V_{LL} I_L \sin\theta \qquad (4.15a)$$

or

$$Q_T = 3\, V_{Ln} I_L \sin\theta \qquad (4.15b)$$

or

$$Q_T = 3\, V_{LL} I_P \sin\theta \qquad (4.15c)$$

and

$$S_T = \sqrt{3}\, V_{LL} I_L = 3\, V_{LL} I_P = 3\, V_{Ln} I_L. \qquad (4.16)$$

Complex power, $\bar{S}_T$, is therefore

$$\bar{S}_T = P_T + jQ_T = S_T\underline{|\theta}, \qquad (4.17a)$$

or for phase a,

$$\bar{S}_T = 3\, \bar{V}_{an} \bar{I}_a^*, \qquad (4.17b)$$

or for phases a and b,

$$\bar{S}_T = 3\, \bar{V}_{ab} \bar{I}_{ab}^*. \qquad (4.17c)$$

It should be evident that in balanced three-phase systems, complex power, $\bar{S}_T$, does not equal $\sqrt{3}\, \bar{V}_{ab} \bar{I}_a^*$.

Basically, all power concepts presented in chapter 3 for single-phase ac also apply to balanced three-phase power. Poor power factor is worthy of critical attention because it affects the entire system operation by limiting the available power from transformers, hindering voltage regulation, and limiting the current-carrying ability of conductors and cables. Simply, the result is poor system operation and economy. Thus, power factor can be equated to an indicator of system efficiency.

EXAMPLE 4.1

An underground coal mining section contains the following three-phase equipment connected to the secondary of a transformer in the section power center:

- Continuous miner: 300 kW at 0.6 lagging pf,
- Two shuttle cars: each 60 kW at 0.8 lagging pf,
- Roof bolter: 50 kW at 0.8 lagging pf,
- Feeder-breaker: 100 kW at 0.6 lagging pf.

Find the capacity of the power-center transformer necessary to operate these machines.

SOLUTION. The complex power to several loads is the sum of the complex power consumed by each individual load. Thus, the complex power for each load must be found first. For the continuous miner,

$$P_1 = S_1 \cos\theta_1$$

or

$$S_1 = \frac{P_1}{\cos\theta_1} = \frac{300}{0.6} = 500 \text{ kVA},$$

and

$$Q_1 = S_1 \sin\theta_1$$

or

$$Q_1 = 500\,(0.8) = 400 \text{ kvar}.$$

Accordingly, for both shuttle cars,

$$P_2 = 120 \text{ kW},\ S_2 = 150 \text{ kVA},\ Q_2 = 90 \text{ kvar}.$$

For the roof bolter,

$$P_3 = 50 \text{ kW},\ S_3 = 62.5 \text{ kVA},\ Q_3 = 37.5 \text{ kvar}.$$

For the feeder-breaker,

$$P_4 = 100 \text{ kW},\ S_4 = 166.7 \text{ kVA},\ Q_4 = 133.3 \text{ kvar}.$$

The total average and reactive power are then respectively

$$P_T = P_1 + P_2 + P_3 + P_4 = 300 + 120 + 50 + 100 = 570 \text{ kW},$$

$$Q_T = Q_1 + Q_2 + Q_3 + Q_4 = 400 + 90 + 37.5 + 133.3 = 660.8 \text{ kvar}.$$

The total complex is then simply

$$\bar{S}_T = P_T + jQ_T = 570 + j661 \text{ kVA}$$

or

$$\bar{S}_T = 873\underline{|49.2^\circ} \text{ kVA}.$$

The required capacity of the transformer is equal to the apparent power of the load. Therefore,

$$\text{transformer capacity} = S_T = 873 \text{ kVA}.$$

It should be noted that, as in chapter 3, the solution to the problem cannot be assumed to be the simple summation of the apparent powers for all the loads. The only case where this is possible is where all loads are operating with the same power factor.

EXAMPLE 4.2

For the combined consumptions of example 4.1, find the necessary total capacitance in kilovoltamperes reactive to improve the overall power factor to 0.8 lagging. The capacitance will be connected across the transformer secondary.

SOLUTION. The combined complex power for the preceding problem is

$$\bar{S}_T = 570 + j661 \text{ kVA}.$$

When pure capacitance is added, average power will remain constant, but reactive and apparent power will decrease. Therefore,

$$\cos\theta_{new} = 0.8 \text{ lagging},$$

$$S_{new} = \frac{P}{\cos\theta_{new}} = \frac{570}{0.8} = 712.5 \text{ kVA},$$

$$Q_{new} = S_{new} \sin\theta_{new},$$
$$= (712.5)(0.6) = 427.5 \text{ kvar.}$$

The difference between this new or improved reactive power and that without the capacitance is the reactive power less the capacitance, or

$$Q_c = (Q_T - Q_{new}) = (661 - 427.5)$$
$$= 233.5 \text{ kvar.}$$

It can be noted that this example is much like example 3.4. The concept of power-factor improvement has been repeated here to show the similarity of most power problems, be they single phase or three phase.

THREE-PHASE TRANSFORMERS

Considerable background information about transformers was presented in chapter 3, and most of that theory is also applicable to three-phase transformers. The prime purpose is the same as with single-phase systems, to provide the different voltages required for distribution and equipment operation. The transformer can be constructed as either a single three-phase unit or a bank of three single-phase units. The only difference between the two is that the three-phase unit has all windings placed on a common core.

The connections can best be described by considering a bank of three single-phase two-winding transformers. Every coil is insulated from the rest, and there are three primary and three secondary windings, all of which can be interconnected independently. The primary and secondary windings can be delta or wye connected while complete electrical separation is retained between all the primary and secondary windings. The possible connections are wye to wye, delta to delta, delta to wye, or wye to delta. Figure 4.10 illustrates the physical connections of each combination, and figure 4.11 shows the corresponding symbols used in the three-phase circuit diagram. Any one of these combinations can be found in or about mine installations, but mine power transformers are typically delta to wye.

Delta-to-wye connections are popular in mine power systems because of the load advantages of the delta connection of the primary, which is in essence the load for the incoming power. The neutral of the wye-connected secondary provides a good grounding point for the outgoing system from the transformer and does not shift potential under unbalanced load conditions. The delta-wye winding combination does not generate third-harmonic (180 Hz for 60-Hz systems) voltages and currents that hamper delta-delta and wye-wye connections.

The second most popular transformer configuration in mines is the delta to delta. Although systems requiring a grounding neutral point create some difficulty for the delta secondaries, the delta-delta connection has one substantial advantage. If one of the single-phase transformers fails, operation can be continued by removing the defective unit and operating the two remaining transformers as open delta. This open-delta or V connection can be illustrated by the two single-phase transformers shown in figure 4.12. Although it is an unsymmetrical connection, it does provide a symmetrical three-phase power input and output. However, using the two transformers in this manner reduces capacity to 57.7% of the three-transformer kilovoltampere rating. Nevertheless, it is an effective emergency measure. The open-delta configuration is sometimes used as a temporary circuit; for example, when the completion of delta is postponed until load conditions warrant a third unit.

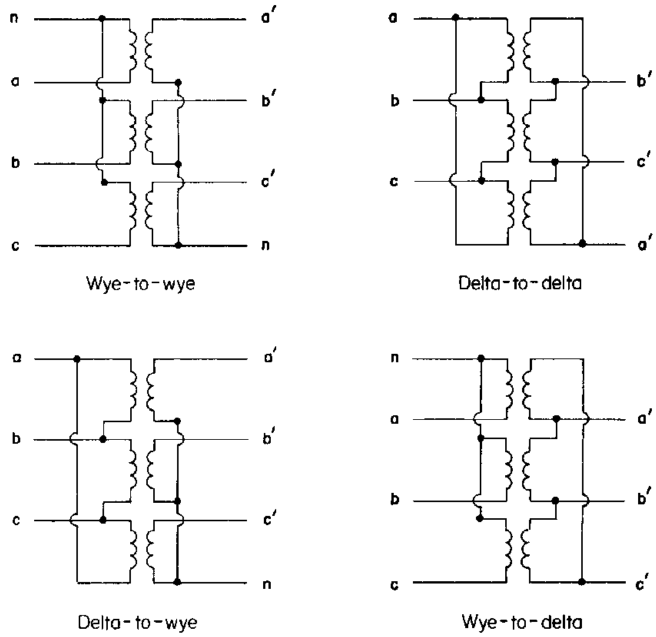


Figure 4.10.—Three single-phase transformers connected for three-phase operation.

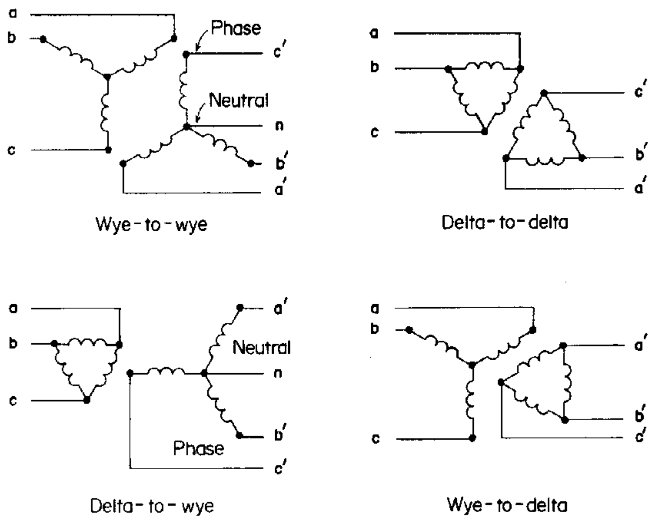


Figure 4.11.—Three-phase diagrams for the transformers of figure 4.10.

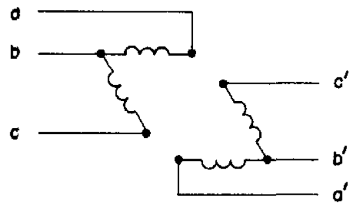


Figure 4.12.—Open-delta three-phase transformer operation.

Calculations with delta-to-delta and wye-to-wye transformers are straightforward and easy to comprehend. With delta to delta, primary line-to-line voltages and phase currents are transformed to secondary line-to-line and phase values, while for wye-to-wye transformers, line-to-neutral voltages and line currents transfer directly. Delta-to-wye and wye-to-delta combinations are different. With a delta-to-wye configuration, primary line-to-line voltages become secondary line-to-neutral, and primary phase currents transform to secondary line currents. Through this, the current and voltage for all three phases shift in phase by 30° across the transformer.

EXAMPLE 4.3

The main substation at a mine contains a delta-delta connected transformer bank composed of three identical single-phase transformers. With rated voltage applied, a 6,000-kW load at 0.8 lagging power factor is causing the transformer bank to be fully loaded. The rated primary and secondary voltages of each single-phase transformer are 36 kV and 7.2 kV, respectively.

1. Find the capacity of each single-phase transformer in the bank.
2. What are the magnitudes of the primary and secondary currents in each single-phase transformer?
3. What are the magnitudes of the primary and secondary line currents to and from the transformer bank?

SOLUTION: The problem states that the transformer bank is fully loaded by an average power, P_T. Thus, the capacity or apparent power load, S_T, of the bank is available from

$$P_T = S_T \cos\theta,$$

$$S_T = \frac{P_T}{\cos\theta} = \frac{6{,}000}{0.8} = 7{,}500 \text{ kVA}.$$

The capacity of each single-phase transformer, S_P, is one-third the total bank capacity, or

$$S_T = 3S_P$$

and $$S_P = \frac{7{,}500}{3} = 2{,}500 \text{ kVA}.$$

If the transformer is assumed to be ideal, current and voltage in the primary or secondary are related to apparent power by

$$S_P = V_P I_P.$$

Hence, the primary and secondary currents in each single-phase transformer are

$$I_{p1} = \frac{S_P}{V_{p1}} = \frac{2{,}500}{36} = 69 \text{ A}$$

and $$I_{p2} = \frac{S_P}{V_{p2}} = \frac{2{,}500}{7.2} = 347 \text{ A}.$$

It can be noted that these currents also correspond to the transformer turns ratio, a, which is 1/5 or 0.2. Because the transformer bank is delta-delta connected, the currents in each transformer are also the phase currents in the bank. Therefore, the line currents to and from the bank are, respectively,

$$I_L = \sqrt{3}\, I_P$$

and $$I_{L1} = \sqrt{3}(69) = 120 \text{ A},$$

$$I_{L2} = \sqrt{3}(347) = 601 \text{ A}.$$

BALANCED THREE-PHASE CIRCUIT ANALYSIS

By definition, any element in one phase of a balanced (or symmetrical) system is duplicated in the other two phases. In other words, currents and voltages for the other phases are equal in magnitude but displaced symmetrically in phase position. Therefore, the analysis of voltage, current, impedance, and power in one phase can provide complete knowledge about the entire three-phase system. In addition, reactions between phases, such as phase currents, line-to-line voltages, or line-to-line connected impedances, may be represented by an equivalent line or line-to-neutral value by using delta-wye transformations. The solution technique is called per-phase or single-phase analysis. The technique has wide application because almost all three-phase power systems that are operating normally are approximately balanced.

As a simple demonstration of the concept, consider figure 4.13*A*, which illustrates a wye generator connected through line resistance to a wye-connected load. In figure 4.13*B*, one phase of this circuit is extracted, and here

- V is one leg of the wye-connected source,
- R is the line resistance per phase,
- Z is a "single-leg" impedance of the wye-connected load, and
- The unconnected points, n and n', are the neutrals of the source and load, respectively.

For the balanced system, the vectorial sum of all three line currents is zero. Hence, the current between n and n' is zero, and the potential at n equals that at n'. Accordingly, the two neutrals can be joined as shown in figure 4.13*C*. This last diagram is the single-phase equivalent circuit, single-phase diagram, or per-phase reduction of figure 4.13*A*. It should be noted that figure 4.13*C* is indeed a basic power circuit, similar to figure 4.1.

Reduction of circuits containing delta-connected sources and loads is almost as easy, but one additional step is involved: the application of delta-wye transformation. Figure 4.14 demonstrates a simple example. Here, all sources and loads must be wye connected. The aim is to convert delta connections to wye using equation 4.9, and line-to-line voltages and phase currents to line-to-neutral and line, respectively. Thus, figure 4.14*B* is the per-phase equivalent of figure 4.14*A*.

The simplified representation of the balanced three-phase circuit can now be analyzed, employing all the single-phase techniques previously discussed. When the solution is found, the three-phase parameters can be determined by reversing the reduction. This need only be performed when line-to-line or phase values are required; no changes are necessary with line-to-neutral and line parameters, as can be seen in figure 4.13.

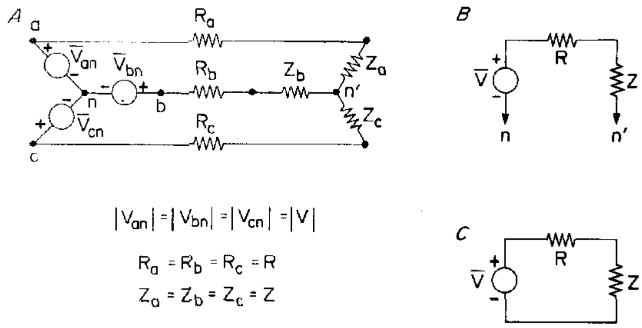


Figure 4.13.—Per-phase reduction of wye-to-wye system.

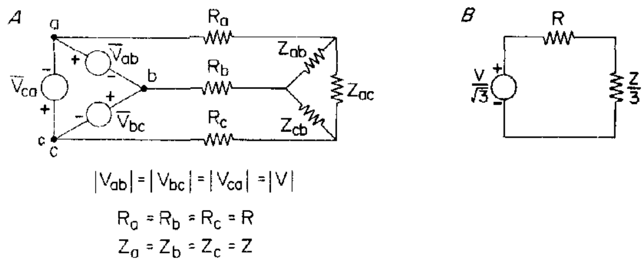


Figure 4.14.—Per-phase reduction of delta-to-delta system.

EXAMPLE 4.4

A load has a balanced delta-connected impedance of $5\underline{|45°}$ per leg. This load is connected through three balanced line impedances of $1 + j1\ \Omega$ to a three-phase source that has a line-to-line voltage of 500 V. What is the magnitude of line current delivered to the load?

SOLUTION. This problem is basically the same as that for the circuit in figure 4.14, except here a line impedance exists between the source and the delta-connected load. As a per-phase solution is called for, the delta load must be transformed to an equivalent wye:

$$Z_\Delta = 5\underline{|45°}\ \Omega,$$

$$Z_Y = \frac{Z_\Delta}{3} = \frac{5\underline{|45°}}{3} = 1.67\underline{|45°}\ \Omega,$$

or

$$Z_Y = 1.18 + j1.18\ \Omega.$$

The per-phase equivalent impedance as seen by the source is simply the sum of the line impedance and the equivalent wye impedance of the load, or

$$\begin{aligned} Z_{eq} &= Z_L + Z_Y \\ &= 1 + j1 + 1.18 + j1.18 \\ &= 2.18 + j2.18 = 3.08\ \underline{|45°}\ \Omega. \end{aligned}$$

The magnitude of this impedance divided into the line-to-neutral voltage across any one phase yields the answer.

$$I_L = \frac{V_{Ln}}{|Z_{eq}|} = \frac{500\ \sqrt{3}}{3.08}$$

$$= \frac{288}{3.08} = 94\ \text{A}.$$

EXAMPLE 4.5

A three-phase 200-hp induction motor has a full-load efficiency of 90%, power factor of 0.85 lagging, and a rated terminal voltage of 950 V line-to-line. Find an equivalent delta-connected impedance for the motor when it is operating at full load under rated voltage.

SOLUTION. Perhaps the best way to start this solution is to find the per-phase average power consumed by the motor under the stated conditions. The total average power input to the motor can be calculated from

$$P_T = \frac{(0.746)\ \text{hp}}{\eta},$$

where hp is the motor horsepower and η is its efficiency. Thus,

$$P_T = \frac{(0.746)(200)}{0.9} = 165.8\ \text{kW}.$$

As single-phase analysis is desirable, the power consumed by each element of the equivalent wye-connected load is needed:

$$P_P = \frac{P_T}{3} = \frac{165.8}{3} = 55.3\ \text{kW}.$$

Equation 4.12 can now be used to find the line current to the motor, or

$$P_P = V_{Ln} I_L \cos\theta,$$

$$I_L = \frac{P_P}{V_{Ln}\cos\theta}$$

$$= \frac{55{,}260}{(950/\sqrt{3})\,(0.85)} = 118.5\text{A}.$$

The line-to-neutral voltage divided by this line current is the magnitude of each leg of the equivalent wye-connected impedance for the motor, and the impedance angle is identical to the power-factor angle. Therefore,

$$Z_Y = \frac{V_{Ln}}{I_L}\,\underline{|\theta}$$

$$= \frac{950/\sqrt{3}}{118.5}\,\underline{|\cos^{-1}0.85}$$

$$= 4.63\underline{|31.8^\circ}\ \Omega;$$

as the equivalent delta-connected impedance is requested,

$$Z_\Delta = 3Z_Y = 13.9\underline{|31.8^\circ}\ \Omega$$

or

$$Z_\Delta = 11.8 + j7.3\ \Omega$$

EXAMPLE 4.6

A production shovel, operating at full load, uses 1,200 kW at 0.9 lagging power factor with 3,750 V line-to-line at the machine. The shovel is supplied through a trailing cable that has an impedance of 0.04 + j0.03 Ω per phase. If the voltage at the source side of the trailing cable is maintained constant, what is the voltage regulation at the machine?

SOLUTION. The per-phase equivalent circuit for this problem is again similar to figure 4.14*B*. The equivalent impedance of the shovel is not necessary, but the line currents and line-to-neutral voltage conditions for full load and no load are. Those for full load are

$$P_P = \frac{P_T}{3} = 400\text{ kW},$$

$$V_{FL} = \frac{3{,}750}{\sqrt{3}} = 2{,}165\text{ V line-to-neutral},$$

$$I_{FL} = \frac{P_P}{V_{FL}\cos\theta} = \frac{400{,}000}{(2{,}165)(0.9)} = 205.3\text{ A}.$$

The line-to-neutral voltage of the constant source, V_{NL}, can be found by computing the voltage drop across the trailing cable, $\overline{V}_{tc}$, then adding it to the voltage at the machine. The full-load voltage at the machine can be assigned a zero phase angle. Thus, based on the given power factor,

$$\overline{V}_{FL} = 2{,}165\underline{|0^\circ}\text{ V},$$

$$\overline{I}_{FL} = 205.3\underline{|-25.84^\circ}\text{ A}.$$

The voltage drop across the trailing cable is

$$\begin{aligned}\overline{V}_{tc} &= \overline{I}_{FL}\,Z_{tc}\\ &= (205.3\underline{|-25.84^\circ})\,(0.05\underline{|36.9^\circ})\\ &= 10.27\underline{|11.06^\circ}\text{ V}.\end{aligned}$$

The voltage at the constant source is

$$\begin{aligned}\overline{V}_{NL} &= \overline{V}_{tc} + \overline{V}_{FL}\\ &= 10 + j2 + 2{,}165\\ &= 2{,}175 + j2 = 2{,}175\underline{|0.1^\circ}\text{ V}.\end{aligned}$$

The subscript for this voltage is used to signify that for these conditions it is the no-load voltage at the machine. In other words, under no load, line current through and the voltage drop across the trailing cable will be zero, and the voltage at the machine will be the same as at the source voltage. Consequently, the voltage regulation is

$$VR = \frac{V_{NL} - V_{FL}}{V_{FL}}\,(100\%)$$

$$= \frac{2{,}175 - 2{,}165}{2{,}165}\,(100\%) = 0.5\%.$$

The solution may be difficult when there are delta-wye or wye-delta transformers in the system because of the 30° phase shift of voltage and current between the windings. In other words, a line-to-neutral transformer secondary voltage transfers to a line-to-line primary value and vice versa. Obviously, when the three-phase system is not balanced, the per-phase reduction method cannot be applied, and other more specialized techniques are required. These are discussed at the end of this chapter.

One-Line and Three-Line Diagrams

It is now apparent that practically all three-phase circuits consist of three conductors, three transformers, and so forth. When all these components are shown in a schematic, the drawing is called a three-phase diagram, as given in figure 4.15. Such diagrams can be extremely helpful, especially when the circuits are concentrated in a piece of machinery or power equipment, because they allow a complete view of component interconnections. They are imperative in manufacturing or troubleshooting.

However, when the circuits are large, as in a complete mine power system, three-line diagrams are not only cumbersome to draw but also exceedingly difficult to read and interpret. Furthermore, if the power system is normally balanced, a three-line diagram is unnecessary since the system is always solved as a single-phase circuit composed of one line conductor and a neutral return. In these cases, the three-line diagram is replaced by a one-line diagram in which the interconnections or conductors between components are represented by single lines plus conventional symbols. This is a further simplification of the per-phase diagram because the completed circuit through the neutral is omitted. One-line diagrams are an invaluable tool in analysis, in designing new power systems, or in modernizing existing ones. Furthermore, since circuit diagrams of coal mine power systems are required by Federal law (30 CFR 75, 77), a one-line diagram is the most practical way to comply.

Figure 4.16 shows a one-line diagram designed to convey in concise form the significant information about the system shown in figure 4.15. In such diagrams it is usually implied that all information is per-phase, unless stated otherwise. Hence it is vital to remember that each device shown is actually installed in triplicate. The conventional notations are line or line-to-neutral impedance, line current, and line-to-neutral voltage. If line-to-line values are listed, they should be stated as such. Every line, symbol, figure, and letter has a definite meaning. Consequently, when a one-line diagram is constructed, specific conventions (*1–2*)[1] must be followed to ensure that the result will be complete, accurate, and correctly interpreted by anyone. The following guidelines are recommended.

Relative geographic relationships for the power-system components should be maintained as far as practical. The typical mine map provides an excellent layout guide for mining applications. Because of the shorthand form and definite meaning of every entry, duplication must be avoided. Standard numbers and symbols are mandatory, and those commonly used in mining are shown in figures 4.17 and 4.18 and tables 4.1 and 4.2. Many of the devices listed have not yet been covered but are included here for completeness.

All known facts about the power system should be shown on the diagram, including

- Apparatus ratings (volts, amperes, power, and so on),
- Ratios and taps of current and potential transformers,
- Power-transformer winding connections,
- Relay functions, and
- Size and type of conductors.

[1] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

The amount of information shown depends on the purpose of the diagram. For instance, if the diagram is to assist in studying the power flow to loads throughout the system (a load-flow study), the location of circuit breakers and relays is unimportant. However, for the solution of other power problems, complete knowledge of these devices can be mandatory. It is important that the one-line diagram contain only known facts; implications and guesses can lead to disastrous results.

On many one-line diagrams, knowledge of future electrical plans is very helpful, and this information can be entered either diagrammatically or through explanatory notes. Finally, the diagram should include correct title data so that the installation is clearly identified and cannot be confused with another or portion thereof.

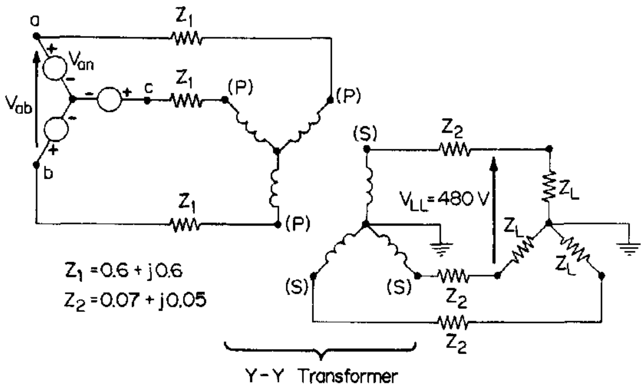


Figure 4.15.—Three-line diagram.

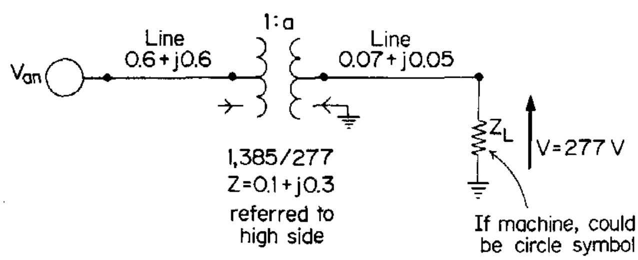


Figure 4.16.—One-line diagram of circuit shown in figure 4.15.

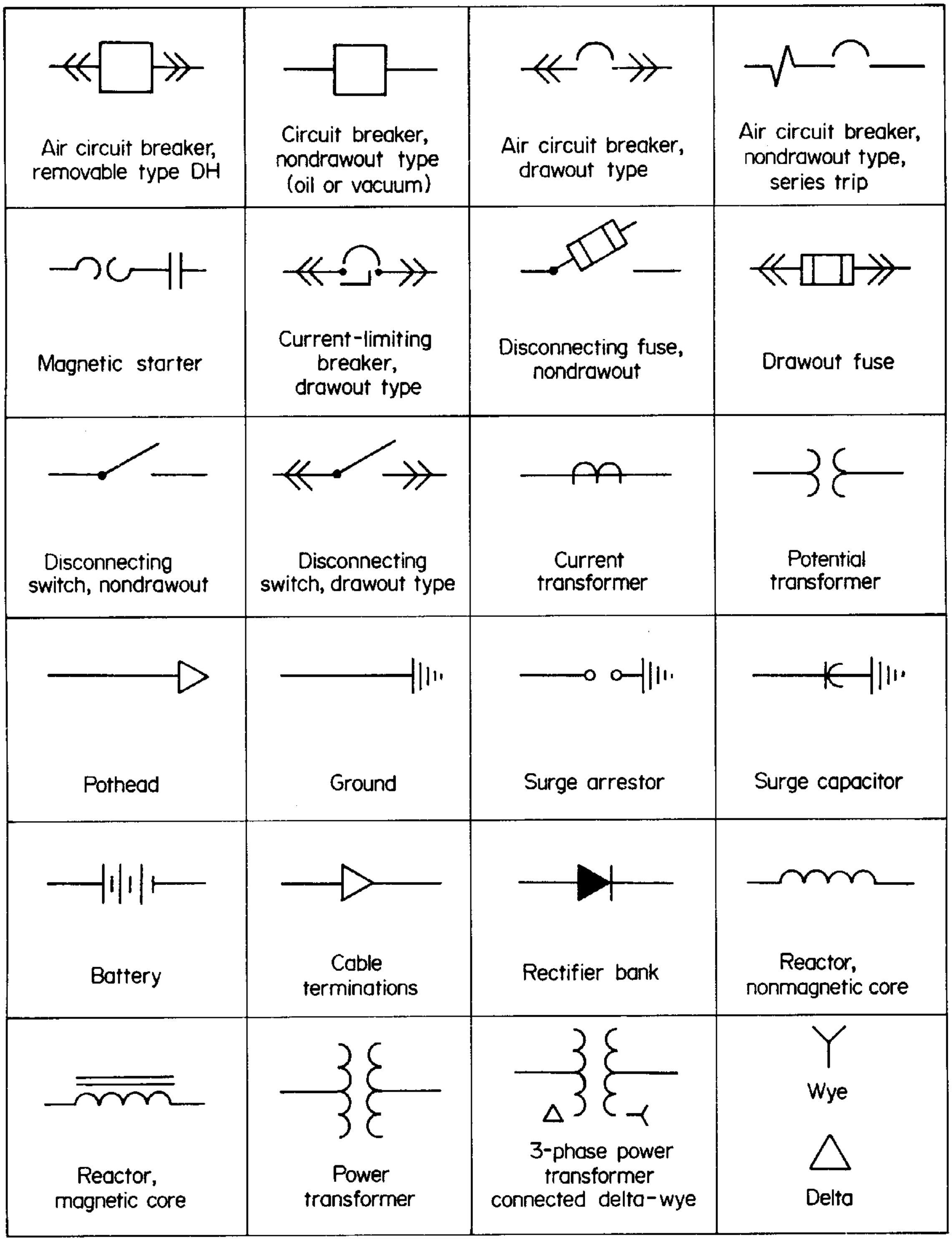


Figure 4.17.—Commonly used symbols for one-line electrical diagrams.

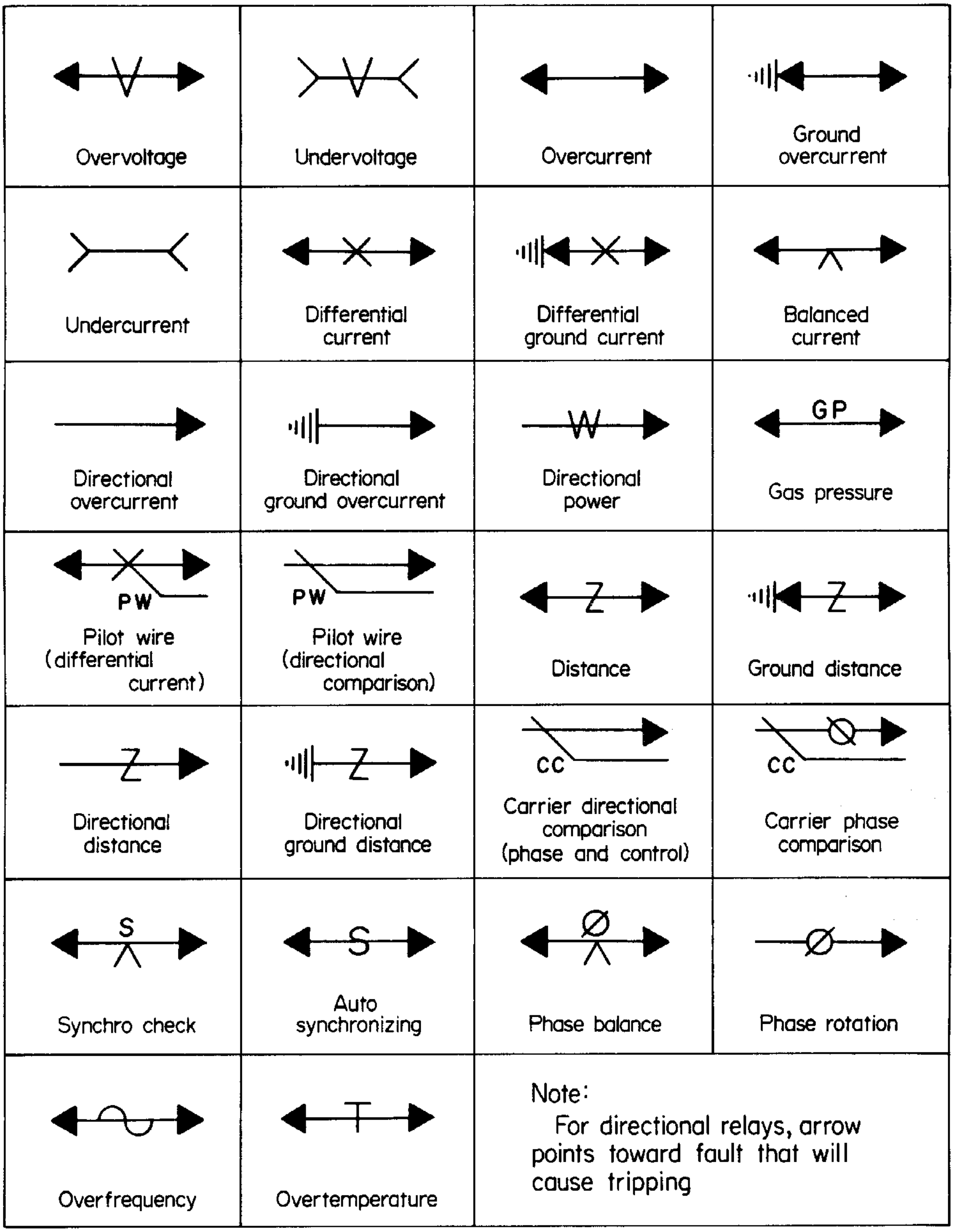


Figure 4.17.—Commonly used symbols for one-line electrical diagrams—Con.

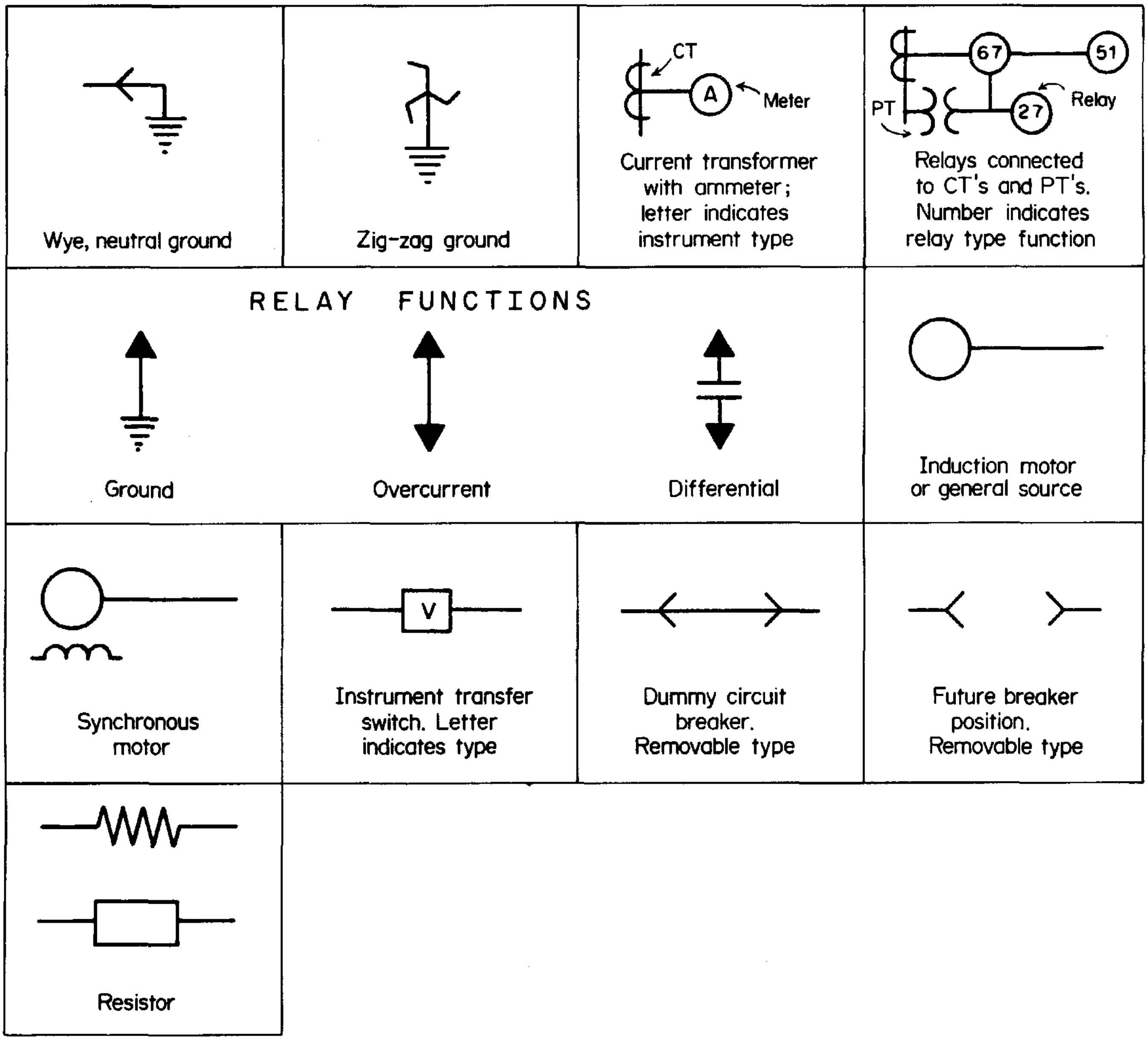


Figure 4.18.—Symbols for relay functions.

Table 4.1.—IEEE device numbers and functions (1)

Device	Function	Device	Function
1	Master element.	51	Time overcurrent relay—ac.
2	Time-delay starting or closing relay.	52	Circuit breaker—ac.
3	Checking or interlocking relay.	53	Exciter or dc generator relay.
4	Master contactor.	54	High-speed dc circuit breaker.
5	Stopping device.	55	Power-factor relay.
6	Starting circuit breaker.	56	Field application relay.
7	Anode circuit breaker.	57	Short-circuiting or grounding device.
8	Control power-disconnecting device.	58	Power rectifier misfire relay.
9	Reversing device.	59	Overvoltage relay.
10	Unit sequence switch.	60	Voltage balance relay.
11	Reserved for future application.	61	Current balance relay.
12	Overspeed device.	62	Time delay stopping or opening relay.
13	Synchronous-speed device.	63	Liquid or gas pressure level or flow relay.
14	Underspeed device.	64	Ground protective relay.
15	Speed, or frequency matching device.	65	Governor.
16	Reserved for future application.	66	Notching or jogging device.
17	Shunting or discharge switch.	67	ac directional overcurrent relay.
18	Accelerating or.	68	Blocking relay.
19	Starting to running transition contactor.	69	Permissive control device.
20	Electrically operated valve.	70	Electrically operated rheostat.
21	Distance relay.	71	Reserved for future application.
22	Equalizer circuit breaker.	72	Circuit breaker—dc.
23	Temperature control device.	73	Load resistor contactor.
24	Reserved for future application.	74	Alarm relay.
25	Synchronizing or synchronism check.	75	Position-changing mechanism.
26	Apparatus thermal device.	76	Overcurrent relay—dc.
27	Undervoltage relay.	77	Pulse transmitter.
28	Reserved for future application.	78	Phase-angle measuring or out-of-step protective relay.
29	Isolating contactor.	79	Reclosing relay—ac.
30	Annunciator relay.	80	Reserved for future application.
31	Separate excitation device.	81	Frequency relay.
32	Directional power relay.	82	Reclosing relay—dc.
33	Position switch.	83	Automatic selective control or transfer relay.
34	Motor-operated sequence switch.	84	Operating mechanism.
35	Brush-operating or slip-ring short-circuiting device.	85	Carrier or pilot-wire receiver relay.
36	Polarity device.	86	Locking-out relay.
37	Undercurrent or underpower relay.	87	Differential protective relay.
38	Bearing protective device.	88	Auxiliary motor or motor generator.
39	Reserved for future application.	89	Line switch.
40	Field relay.	90	Regulating device.
41	Field circuit breaker.	91	Voltage directional relay.
42	Running circuit breaker.	92	Voltage and power directional relay.
43	Manual transfer or selector device.	93	Field-changing contactor.
44	Unit sequence starting relay.	94	Tripping or trip-free relay.
45	Reserved for future application.	95	Reserved for future application.
46	Reverse-phase-balance current relay.	96	Do.
47	Phase-sequence voltage relay.	97	Do.
48	Incomplete sequence relay.	98	Do.
49	Machine or transformer thermal relay.	99	Do.
50	Instantaneous overcurrent or rate-of-rise relay.		

Table 4.2.—Device numbers and letters common to mining (2)

Device	Function
1	Control switch.
3	Plus interlock relay.
37	Ground-continuity check undercurrent relay.
49 D	Diode thermal relay.
49 GR	Grounding resistor thermal relay.
49 T	Transformer thermal relay.
50	Instantaneous overcurrent relay—ac.
50 G	Instantaneous overcurrent relay—dc—connected in ground wire.
50 N	Instantaneous overcurrent relay—ac—connected to neutral.
50 Z	Instantaneous current-balance relay—ac—zero sequence.
51	Time delay overcurrent relay—ac.
51 N	Time-delay overcurrent relay—ac—connected to neutral.
51 Z	Time-delay current-balance relay—ac—zero sequence.
52	Circuit breaker—ac.
59 GR	Ground protective relay—dc—unbalance relay.
72	Circuit breaker—dc.
76	Overcurrent relay—dc.
A	Ammeter.
D	Demand meter.
GD	Ground detector.
PF	Power factor.
V	Voltmeter.
VA	Volt-ammeter.
VAR	Varmeter.
W	Wattmeter.
WH	Watthour meter.

The remainder of the text will employ one-line and per-phase diagrams almost exclusively. The main thing to remember is that practically every item represents three identical or corresponding items in the actual system. Even when the normally balanced system becomes unbalanced through component failures, the same diagram is used, the only change being the notation for the specific failure.

Circuits Containing Transformers

As previously stated, solving a balanced three-phase system problem by per-phase analysis is as simple as the single-phase techniques covered in chapters 2 and 3. However, the solution is not so clear when delta-wye or wye-delta transformers are involved. Perhaps the simplest way to demonstrate the approach is first to illustrate a problem solution where the per-phase reduction is of a straightforward wye-wye transformer, then change to a delta-delta or delta-wye transformer and show the complications that might arise.

EXAMPLE 4.7

Consider figure 4.19, which shows a one-line diagram of a substation supplying power through about 1 mile of overhead line (power conductors on poles) to a three-phase wye-wye transformer bank, then through a trailing cable to a three-phase induction motor. The motor consumes an average three-phase power of 150 kW, operating at 0.8 lagging power factor. The problem is to find the rms voltage needed at the substation output to provide the rated motor terminal voltage of 480 V line-to-line.

A three-phase diagram of the circuit is shown in figure 4.20 for reference. The first step in the solution is usually to develop a per-phase circuit as shown in figure 4.21. Although the solution could be performed directly from the one-line diagram, the per-phase diagram allows direct application of single-phase techniques. The following should be noted in figure 4.21.

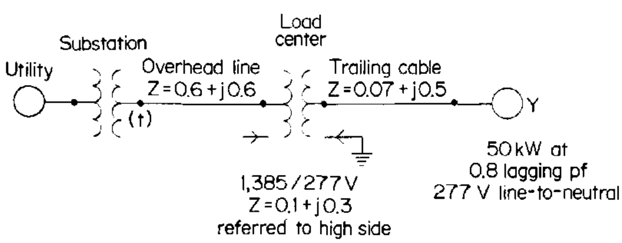


Figure 4.19.—One-line diagram for example 4.7.

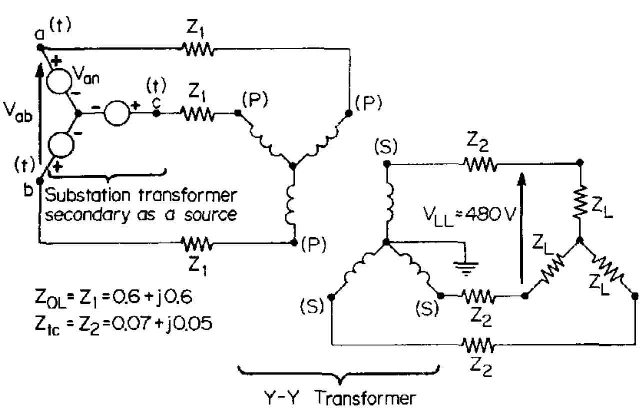


Figure 4.20.—Three-phase diagram of figure 4.19.

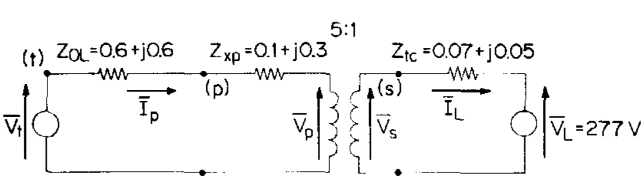


Figure 4.21.—Per-phase diagram of figure 4.19.

- The line and trailing-cable conductor impedances are now illustrated as circuit elements.
- Only one phase of the transformer bank is shown, represented as an impedance in series with the *primary of an ideal transformer*. The transformer turns ratio is computed from

$$a = \frac{V_2}{V_1} = \frac{277}{1{,}385} = \frac{1}{5},$$

where these rated voltages are line-to-neutral rms.

- The induction motor is represented by its single-phase equivalent, P_p, where

$$P_P = \frac{P_T}{3} = \frac{150}{3} = 50 \text{ kW}.$$

The solution can now follow a stepwise process.

1. Assume the motor terminal voltage, V_L, is the rated 277-V rms line-to-neutral ($480/\sqrt{3}$).
2. Compute motor line-current magnitude, I_L, using

$$P_p = V_L\, I_L \text{ (load pf)}$$

or

$$I_L = \frac{P_P}{V_L(\text{load pf})} = \frac{50{,}000}{(277)(0.8)} = 225.5 \text{ A}.$$

If the motor terminal voltage is taken as the reference phasor, this current has a phase angle determined by the load power factor. Therefore the motor current phasor is

$$\bar{I}_L = I_L \underline{|-\cos^{-1}(0.8)} = 226\underline{|-36.9^\circ} \text{ A}.$$

3. $\bar{I}_L$ is then employed to find the voltage drop across the per-phase equivalent of the trailing cable, Z_{tc}. When this is added to the motor terminal voltage, the voltage at the ideal transformer secondary, $\bar{V}_s$, is

$$\bar{V}_s = \bar{I}_L Z_{tc} + \bar{V}_L$$

or
$$\begin{aligned}\bar{V}_s &= (226\underline{|-36.9^\circ})(0.09\underline{|35.5^\circ}) + 277\underline{|0^\circ} \\ &= 20.30\underline{|-1.4^\circ} + 277\underline{|0^\circ} \\ &= 20.29\ -j\,0.50 + 277 \\ &= 297.29\ -j\,0.50\ +\ 297\underline{|-0.1^\circ} \text{ V}.\end{aligned}$$

4. For the ideal transformer with a turns ratio, a, of 1/5, the voltage across the primary is

$$\bar{V}_p = \frac{\bar{V}_s}{a}$$

or
$$\bar{V}_P = 5(297\underline{|-0.1^\circ}) = 1486\underline{|-0.1^\circ} \text{ V},$$

with the primary current being

$$\bar{I}_p = a\bar{I}_L$$

or $\quad \bar{I}_P = (\tfrac{1}{5})(226\angle{-36.9°}) = 45\angle{-36.9°}$ A.

Notice that there is no change in current or voltage phase angle across the transformer.

5. This primary current can now be used to find the voltage drop resulting from transformer and overhead-line impedances. Summing this voltage drop with the transformer primary voltage gives the desired answer to the problem, the substation output voltage, V_t:

$$\bar{V}_t = \bar{I}_P(Z_{OL} + Z_{xp}) + \bar{V}_P,$$

where Z_{OL}, Z_{xp} = overhead-line and transformer impedances, respectively.

Then,

$$\bar{V}_t = 45\angle{-36.9°}(0.6+j0.6+0.1+j0.03) + 1{,}486\angle{-0.1°}$$

or

$$\begin{aligned}\bar{V}_t &= 45\angle{-36.9°}\,(0.94\angle{41.99°}) + 1{,}486\angle{-0.1°}\\ &= 42.39\angle{5.1°} + 1{,}486\angle{-0.1°}\\ &= 1{,}528\angle{+0.04°}\text{ V.}\end{aligned}$$

Because the analysis is per phase, the result is obviously line-to-neutral voltage. If line-to-line values are required, the above answer need only be multiplied by $\sqrt{3}$. It is interesting that in this example the phase angle of the substation voltage is practically the same as that at the motor. This may not be the case in actual mine power systems.

When the transformer is delta-delta connected, problem solutions are practically the same as in example 4.7. The one-line diagram of figure 4.22 provides an illustration. When the system is represented per phase (fig. 4.23), the only additional concern is delta-wye transformation of the transformer impedance; then the solution proceeds as before. However, the process may not be as simple when delta-wye or wye-delta transformer connections are involved.

Consider the one-line diagram in figure 4.24 that shows a delta-wye transformer supplying the same motor as that shown in figure 4.19. Figure 4.25 illustrates one leg of the three-phase transformer. From this, it can be seen that secondary line currents appear as phase currents on the primary, and line-to-neutral secondary voltages become line-to-line primary voltages. In other words (fig. 4.25), for primary voltage in terms of the secondary (the ideal transformer with turns ratio, a, only):

$$\bar{V}_{AB} = \frac{\bar{V}_{an}}{a}\angle{30°}, \tag{4.18}$$

where $\bar{V}_{AB}$ = line-to-line primary voltage, V,
and $\bar{V}_{an}$ = line-to-neutral secondary voltage, V,

and for primary and secondary current,

$$\bar{I}_{AB} = a\,\bar{I}_a\angle{30°}, \tag{4.19}$$

where $\bar{I}_{AB}$ = primary phase current, A,
and $\bar{I}_a$ = secondary line current, A.

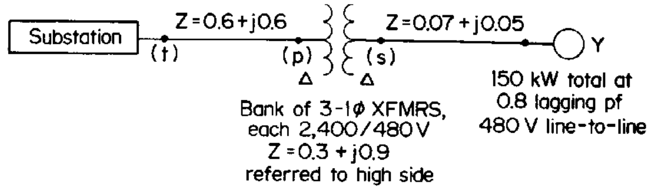


Figure 4.22.—One-line diagram with delta-delta transformer.

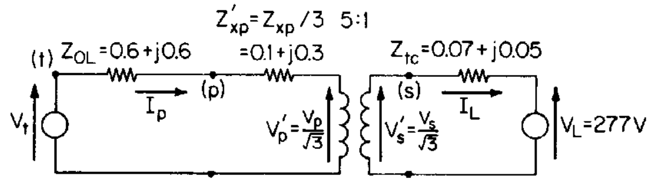


Figure 4.23.—Per-phase diagram of figure 4.22.

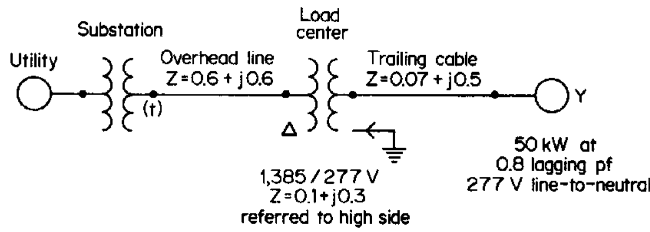


Figure 4.24.—One-line diagram with delta-wye transformer.

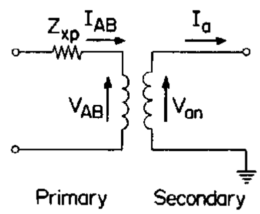


Figure 4.25.—One leg of three-phase transformer from figure 4.24.

However, when performing balanced three-phase analysis, the parameters of interest are line-to-neutral voltages and line currents. Thus, to continue the analysis in a fashion similar to that used in example 4.7 (the wye-wye transformer), $\bar{V}_{AB}$ and $\bar{I}_{AB}$ must be converted to their respective per-phase equivalents. Recalling that

$$\bar{V}_{ab} = \sqrt{3}\,\bar{V}_{an}\angle{+30°}, \tag{4.20}$$

and applying this concept to equation 4.18, the primary line-to-neutral voltage, V_{An}, is

$$\bar{V}_{An} = \frac{\bar{V}_{an}}{a\sqrt{3}}. \tag{4.21}$$

Employing equation 4.7*b* to convert equation 4.19, primary line current, $\bar{I}_A$, is

$$\bar{I}_A = a\,\bar{I}_a. \tag{4.22}$$

Equations 4.21 and 4.22 simply state that the phase shifts that occur across delta-wye or wye-delta-connected transformers do not interfere with the analysis when this is performed per phase. Analysis can be enhanced by changing the delta primary or secondary to an equivalent wye connection, thus enabling the construction of a per-phase diagram for the entire system.

Concerning the actual per-phase analysis, it has been shown that the three-phase circuit is reduced by a process no more difficult than the single-phase work covered in chapters 2 and 3. The next section will present a technique that further simplifies power-system analysis.

PER-UNIT SYSTEM

Problems related to electrical circuits should be solved in terms of volts, amperes, voltamperes and ohms. The answers to mine power problems, and indeed any electrical problem, almost always require these terms, but in the process of computations it is often more convenient to express these quantities in percent or per-unit (pu), referred to some arbitrarily chosen base. For example, if a base voltage of 100 kV is selected, voltages of 90, 120, and 125 kV have percent representations of 90%, 120%, and 125%, or per-unit values of 0.9, 1.2, and 1.25 pu, respectively. Both percent and per-unit values express a ratio of a specific quantity to the base quantity. Per-unit is given as a decimal, whereas the ratio in percent is 100 times the per-unit value. These expressions, especially per-unit, are becoming standard for equipment specifications.

There is a definite advantage in using per-unit notation over percent. Per-unit multiplication or division yields a result in per-unit. However, the product of two percent quantities must be divided by 100 to obtain a percent answer. For example, if two quantities are both 0.9 pu or 90%, then

$$(0.9\ \text{pu})(0.9\ \text{pu}) = 0.81\ \text{pu}$$

and $$(90\%)(90\%) \neq 8{,}100\%$$

but $$(90\%)(90\%) = \frac{8{,}100}{100}\%.$$

Consequently, per-unit notation will be employed almost exclusively, the only exception being where conventions dictate otherwise.

Voltage, current, voltamperes, and impedance are obviously interrelated for any specific circuit or system. As a result, the selection of any two determines the base values for the remaining two. For example, if the current and voltage bases are specific, the base impedance and base voltamperes can be found. Since three-phase circuits are usually solved as a single line with a neutral, base quantities in the per-unit system are line current, line-to-neutral voltage, per-phase impedance, and per-phase voltamperes. The mathematical interrelations of the bases are as follows:

$$V_b = I_b Z_b, \tag{4.23}$$

$$KVA_b = kV_b I_b, \tag{4.24}$$

$$I_b = \frac{kVA_b}{kV_b} = \frac{V_b}{Z_b}, \tag{4.25}$$

$$Z_b = \frac{V_b}{I_b} = \frac{(kV_b)^2 1{,}000}{kVA_b} = \frac{(kV_b)^2}{MVA_b}, \tag{4.26}$$

where V_b = base line-to-neutral voltage, V,
kV_b = base line-to-neutral voltage, kV,
kVA_b = base per-phase voltamperes, kVA,
MVA_b = base per-phase voltamperes, MVA,
I_b = base line current, A,
and Z_b = base per-phase impedance, Ω.

All these formulas are adaptations of the fundamental Ohm's law and power material; the last three are expressed in kilovolts and kilovoltamperes because of the levels normally found in power systems. It should be remembered that line-to-line voltages and total power (kilovoltamperes or megavoltamperes) are customarily specified; these must be changed to line-to-neutral voltages (dividing by $\sqrt{3}$) and per-phase power (dividing by 3) before equations 4.23 through 4.26 can be applied.

To apply the per-unit system to power problems, base values for kilovoltamperes and kilovolts are normally selected first in order to minimize calculations as much as possible. Base values for impedance and current are then found. Next, all the actual voltages, currents, impedances, and powers in the power system or system segment are expressed as a ratio to the base quantities; these are the per-unit quantities. Problems are then solved in per-unit, with the answers converted back to actual parameters. The actual values and per-unit quantities are related by

$$Z_A = Z_{pu} Z_b, \tag{4.27}$$

$$I_A = I_{pu} I_b, \tag{4.28}$$

$$V_A = V_{pu} V_b, \tag{4.29}$$

$$VA_A = VA_{pu} VA_b, \tag{4.30}$$

where Z_A, I_A, V_A, VA_A = actual values of impedance, current, voltage, and power, respectively, Ω, A, V, VA,
and Z_{pu}, I_{pu}, V_{pu}, VA_{pu} = per-unit values of impedance, current, voltage, and power, respectively, puΩ, puA, puV, puVA.

It is important to note that all impedances in a problem are referenced to the same base impedance, whether they are pure resistance or pure reactance. The same holds for all average, reactive, or apparent powers, which are referenced to the base kilovoltamperes.

Often, per-unit impedances or percent impedances of a system component have already been assigned to a base referenced to the component or power-system segment in which that component is located. These impedances can be changed to another base impedance by

$$Z_{pu} = \frac{Z_{pue}\,kVA_b}{kVA_e}\left(\frac{kV_e}{kV_b}\right)^2, \qquad (4.31)$$

where Z_{pue} = per-unit impedance of specified component, puΩ,
kV_e, kVA_e = base kV and kVA used to reference Z_{pue}, V, VA,
kV_b, KVA_b = base kV and kVA to which new per-unit impedance is to be referenced, V, VA,
and Z_{pu} = new per-unit impedance referenced to kV_b and kVA_b, pu Ω.

Transformer Impedance

Transformers are the most common devices in power systems where the component impedance is referenced to the rated power and voltage of the component. Conventionally, percent impedance is specified, but this can be converted to per-unit simply by dividing by 100.

A major advantage of using per-unit computations is seen when circuits are connected through transformers. Through the proper selection of voltage bases, the per-unit impedance of the transformer is the same no matter which winding is used. Consequently, if exciting and magnetizing currents are ignored, as they often can be in power systems, the transformers become a simple series impedance in per-unit calculations. In other words, the ideal transformer is not needed in the equivalent circuit. Example 4.8 explores this advantage.

EXAMPLE 4.8

Consider a 750-kVA power-center transformer, the approximate per-phase equivalent circuit for which is shown in figure 4.26. The per-phase ratings are 250 kVA, 5,000/1,000 V, and 5-Ω reactance referred to the high side. Following convention, the base power kVA_{b1} is 250 kVA, and the base voltage for the high side kV_{b1} is 5 kV. One kilovolt is selected as the low-side base voltage, kV_{b2}. With these, the high-side base quantities can be calculated using equations 4.23 through 4.26:

$$kVA_{b1} = 250 \text{ kVA},$$
$$kV_{b1} = 5 \text{ kV},$$
$$I_{b1} = 50 \text{ A},$$
and
$$Z_{b1} = 100\ \Omega.$$

The per-unit impedance of the transformer is thus

$$Z_{pu} = \frac{Z_{A1}}{Z_{b1}} = \frac{5}{100} = 0.05 \text{ pu}.$$

Now consider the actual transformer impedance as it would appear referred to the low side, as in figure 4.27.

$$Z_{A2} = Z_{A1}\left(\frac{N_2}{N_1}\right)^2 = Z_{A1}\left(\frac{V_2}{V_1}\right)^2$$

or

$$Z_{A2} = 5\left(\frac{1{,}000}{5{,}000}\right)^2 = 0.20\ \Omega.$$

The base quantities on the low side are

$$kVA_{b1} = 250 \text{ kVA},$$
$$kV_{b2} = 1 \text{ kV},$$
$$I_{b2} = 250 \text{ A},$$
and
$$Z_{b2} = 4\ \Omega.$$

Notice that the base power does not change and the low-side base voltage defines base current and impedance. The per-unit transformer impedance as seen from the low side is

$$Z_{pu} = \frac{Z_{A2}}{Z_{b2}} = \frac{0.2}{4} = 0.05 \text{ pu}.$$

Therefore, the per-unit impedance of the transformer is the same, regardless of the side it is viewed from, and the per-unit equivalent circuit is simply the series circuit shown in figure 4.28. Here, the input and output voltages are now expressed in per-unit since the transformer is operating at rated voltage.

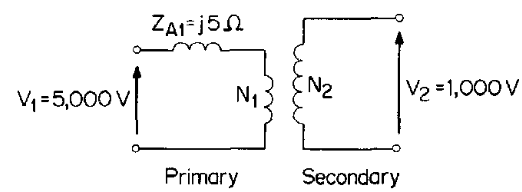


Figure 4.26.—Approximate per-phase equivalent circuit for 750-kVA load-center transformer; impedance referred to high side.

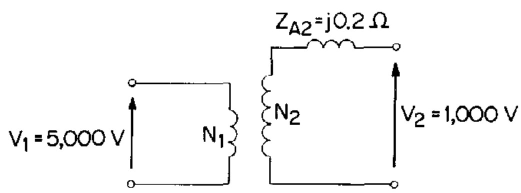


Figure 4.27.—Transformer of figure 4.26 with impedance referred to low side.

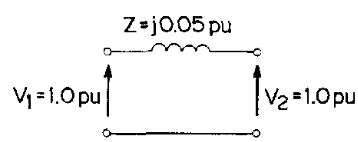


Figure 4.28.—Simplified equivalent circuit of transformer expressed in per-unit.

Three-Winding Transformers

In chapter 3 and to this point in chapter 4, equivalent circuits have been shown only for two-winding transformers, those having only one primary and one secondary winding. However, many transformers in mine power systems have three windings, with the third winding termed the tertiary or second secondary. These include power-center transformers supplying two different utilization voltages, such as 950 and 550 Vac to face equipment or 550 and 250 Vdc to machinery. The latter case not only uses a three-winding transformer but also three-phase rectification, which will be described in chapter 5.

Both the primary and secondary windings of the two-winding transformer have the same kilovoltampere capacity or rating, but the three windings of a three-winding transformer may have different kilovoltampere ratings. The impedance of each winding may be given in percent or per-unit, based on each winding rating. The three impedances can also be measured by the following short-circuit test, where rated voltage is applied to the primary for Z_{ps} and Z_{pt} and to the secondary for Z_{st}:

Z_{ps} = leakage impedance measured in primary (or first winding), with secondary (or second winding) short-circuited and tertiary (or third winding) open, Ω,
Z_{pt} = leakage impedance measured in primary with tertiary short-circuited and secondary open, Ω,
Z_{st} = leakage impedance measured in secondary with tertiary short-circuited and primary open, Ω.

The impedances of the primary, secondary, and tertiary windings are found from

$$Z_{ps} = Z_p + Z_s, \qquad (4.32a)$$

$$Z_{pt} = Z_p + Z_t, \qquad (4.32b)$$

$$Z_{st} = Z_s + Z_t, \qquad (4.32c)$$

or

$$Z_p = \tfrac{1}{2}(Z_{ps} + Z_{pt} - Z_{st}), \qquad (4.33a)$$

$$Z_s = \tfrac{1}{2}(Z_{ps} + Z_{st} - Z_{pt}), \qquad (4.33b)$$

$$Z_t = \tfrac{1}{2}(Z_{pt} + Z_{st} - Z_{ps}), \qquad (4.33c)$$

where Z_p, Z_s, Z_t = impedances of primary, secondary, and tertiary, respectively, Ω.

In equations 4.32 and 4.33, all impedances (Z_{ps}, Z_{pt}, Z_{st}) must be referred to the primary winding voltage. If Z_{st} is obtained from the described measurements, the impedance is referred to the secondary-winding voltage, hence it must be transferred.

The approximate per-phase equivalent circuit for a three-winding transformer with the winding impedances of Z_p, Z_s, and Z_t is provided in figure 4.29. Magnetizing and exciting currents are ignored. The terminals p, s, and t are the primary, secondary, and tertiary connections; the common point is unrelated to the system neutral. The three impedances must be in the per-unit system, as was the case for the equivalent circuit in figure 4.28. Hence they must have the same kilovoltampere base. Further, voltage bases for the circuits connected through the transformer must have the same ratios as the turns ratio of the transformer windings; that is, primary to secondary, primary to tertiary, which are actually the same as the ratios of the related winding voltages.

Per-Unit Method in System Analysis

As mentioned earlier, the use of per-unit equivalents in the analysis of power-system problems can greatly simplify the work involved, especially when the system contains transformers and different voltage levels. However, as per-unit calculations require the change of familiar parameters (ohms, volts, amperes, and so on) into values representing a ratio, this advantage is often difficult to comprehend. Example 4.9 will illustrate the per-unit method of analysis using the one-line diagram provided in figure 4.30, which could represent a mine power system in the early stages of development. All power levels listed are given per-phase; those shown for the mining equipment represent consumption. The voltages are all line-to-neutral.

EXAMPLE 4.9

The information required could be the voltage or current level at any point. Regardless, solution by the per-unit method first requires translation of the impedance of all components to the same base. The base selection is arbitrary, but for convenience, the largest kilovoltampere capacity of a system component is usually taken. In this case a good base would

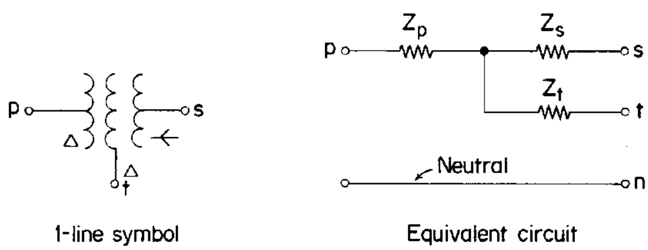


Figure 4.29.—Approximate equivalent circuit of three-winding transformer expressed in per-unit.

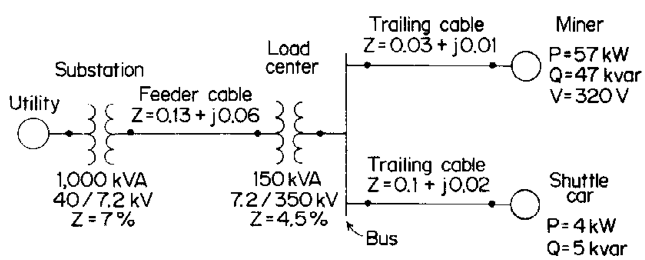


Figure 4.30.—One-line diagram of small mine power system.

be 1,000 kVA, corresponding to the per-phase capacity of the substation. But two base parameters are needed in order to define the four base quantities; as the nominal voltage for each voltage level can be or can approach a constant, the system voltages are an excellent choice for the second base parameter. For figure 4.30, these would be the line-to-neutral voltages of 40 kV at the utility, 7.2 kV at mine distribution, and 350 V at mine utilization. Note that the system voltages are usually given as line-to-line in one-line diagrams, so they must be changed to line-to-neutral values to employ the formulas given here. In any event, base voltages must correspond with the turns ratio of any interconnecting transformer. The ones selected do.

With base kilovoltamperes and base kilovolts specified, the base quantities can be calculated using equations 4.23 through 4.26.

1. For the utility:

$$kVA_b = 1{,}000 \text{ kVA},$$

$$kV_{b1} = 40 \text{ kV},$$

$$I_{b1} = \frac{kVA_b}{kV_{b1}} = \frac{1{,}000}{40} = 25 \text{ A},$$

$$Z_{b1} = \frac{(kV_{b1})^2\, 1{,}000}{kVA_b} = \frac{(40)^2\, 1{,}000}{1{,}000} = 1{,}600\ \Omega.$$

2. For mine distribution:

$$kVA_b = 1{,}000 \text{ kVA},$$

$$kV_{b2} = 7.2 \text{ kV},$$

$$I_{b2} = \frac{1{,}000}{7.2} = 139 \text{ A},$$

$$Z_{b2} = \frac{(7.2)^2\, 1{,}000}{1{,}000} = 52\ \Omega.$$

3. For mine utilization:

$$kVA_b = 1{,}000 \text{ kVA},$$

$$kV_{b3} = 0.35 \text{ kV},$$

$$I_{b3} = \frac{1{,}000}{0.35} = 2{,}857 \text{ A},$$

$$Z_{b3} = \frac{(0.35)^2\, 1{,}000}{1{,}000} = 0.12\ \Omega.$$

The per-unit representations for all components of the mine system can now be found, and the needed formulas are equations 4.27 through 4.31.

1. For the substation: percent reactance is 7% or 0.07 pu based on the transformer rated kilovoltamperes, referred to the high side, 40 kV.

$$Z_{pu} = \frac{Z_{pue}\, kVA_b}{kVA_e}\left(\frac{kV_e}{kV_b}\right)^2,$$

where $Z_{pue} = j0.07$ pu,

$$kVA_b = kVA_e = 1{,}000 \text{ kVA},$$

$$kV_e = kV_b = 40 \text{ kV};$$

thus, $Z_{pue} = j0.07$ pu.

2. For the feeder cable: actual impedance is given,

$$Z_A = R_A + jX_A = 0.13 + j0.06\ \Omega,$$

and

$$Z_{pu} = \frac{Z_A}{Z_{b2}} = \frac{0.13 + j0.06}{52} = (0.0025 + j0.0012) \text{ pu}.$$

3. For the load center: percent reactance is 4.5% or 0.045 pu based on the transformer rated kilovoltamperes, referred to the high side, 7.20 kV.

$$Z_{pu} = \frac{Z_{pue}\, kVA_b}{kVA_e}\left(\frac{kV_e}{kV_{b2}}\right)^2,$$

where $Z_{pue} = j0.045$ pu,

$$kVA_b = 1{,}000 \text{ kVA},$$

$$kVA_e = 150 \text{ kVA},$$

$$kV_{b2} = kV_e = 7.2 \text{ kV},$$

and

$$Z_{pu} = \frac{(j0.045)(1{,}000)}{150} = j0.3 \text{ pu}.$$

4. For the trailing cables: actual impedances are again given.

Miner. $Z_A = 0.03 + j0.01\ \Omega$,

$$Z_{pu} = \frac{Z_A}{Z_{b3}} = \frac{0.03 + j0.01}{0.12}$$

$$= (0.25 + j0.083) \text{ pu}.$$

Shuttle car. $Z_A = 0.1 + j0.02\ \Omega$,

$$Z_{pu} = \frac{Z_A}{Z_{b3}} = \frac{0.1 + j0.02}{0.12}$$

$$= (0.833 + j0.167) \text{ pu}.$$

5. For the machines: consumption is given in terms of average and reactive power.

Miner. $P = 57$ kW, $Q = 45$ kvar

or $\quad kVA_A = (57 + j45)$ kVA;

$$\text{thus,} \quad kVA_{pu} = \frac{kVA_A}{kVA_b} = \frac{(57 + j45)}{1{,}000}$$

$$= (0.057 + j0.045) \text{ pu.}$$

Shuttle car. $kVA_A = (4 + j5)$ kVA,

hence, $kVA_{pu} = (0.004 + j0.005)$ pu.

At this point, the entire system of figure 4.30 may be redrawn into the impedance diagram in figure 4.31.

Figure 4.31, when compared with a per-phase diagram in the impedance domain such as figure 4.21, illustrates the advantage that simplified per-unit computations lend to power-system analysis. The circuit shown is merely a series-parallel arrangement of basic electrical elements, and obviously it may be further simplified if desired, say into an equivalent per-unit impedance. This is only one example; an actual appreciation of power-system analysis by per-unit techniques can come only through experience.

The impedance diagram can be used for the solution of most power problems. Suppose currents under normal operation are desired. One method is to apply known voltages at system points and calculate the resulting currents and voltages. For instance, suppose the line-to-neutral at the miner is 320 V (about 555 V line-to-line). The per-unit equivalent of this is

$$kV_{pu} = \frac{kV_A}{kV_{b3}} = \frac{0.32}{0.35} = 0.91 \text{ pu.}$$

The line current through the miner trailing cable is then

$$I_{pu} = \left(\frac{kVA_{pu}}{kV_{pu}}\right)^*$$

or

$$I_{pu} = \frac{(0.057 - j0.045)}{0.91} = 0.063 - j0.049 \text{ pu.}$$

The conjugate of power is employed because voltage is implied as the reference phasor following the conventions stated earlier. The process is then continued through the entire system. When the desired per-unit values are obtained, they are simply converted to actual values. Considering the current in the miner trailing cable, the actual line current is

$$I_A = I_{pu} I_{b3}$$

or $I_A = (0.063 - j0.049)(2857) = 180 - j141.3$A

or $\quad I_A = 229 \angle\underline{-38.1^\circ}$ A.

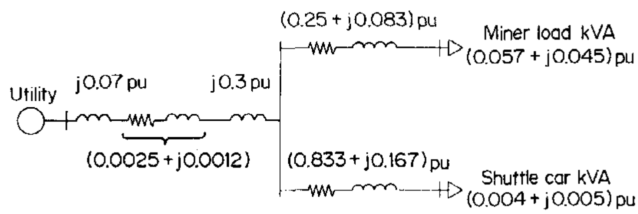


Figure 4.31.—Impedance diagram of system in figure 4.30, expressed in per-unit on a 1,000-kVA base.

UNBALANCED THREE-PHASE CIRCUITS

The solution of balanced three-phase circuit problems is usually accomplished by converting the constants, currents, and voltages to per-phase values. Because symmetry determines the magnitude and phase position of all currents and voltages, actions occurring between phases can be represented by equivalent impedances. Furthermore, currents and voltages for the other phases are equal in magnitude to those in the per-phase solution but are simply displaced symmetrically in phase position. This is extremely important because normally operating three-phase power systems can usually be approximated as balanced. However, the solution of unbalanced three-phase circuits or balanced circuits with unbalanced loads does not permit the same simplification.

Mine power systems are designed to have a high degree of reliability and therefore to operate in a balanced mode. But at times, equipment failures and unintentional or intentional disturbances from outside sources can result in an unbalanced operation. The most common sites for mine power-system unbalance are equipment trailing cables. The consequence of unbalance is abnormal currents and voltages, and if safeguards are not designed into the system to protect against these anomalies, the safety of personnel as well as equipment can be compromised. The protective circuitry within the mine power system serves as the safety valve for such hazardous malfunctions.

Power-system unbalance can occur either from open circuits or from faults. A fault occurs whenever electricity strays from its proper path. Faults can be visualized as the connecting together of two or more conductors that normally operate with a voltage between them. The connection that creates a fault can be from physical contact or an arc caused by current flow through a gaseous medium. A short circuit is one type of fault. Currents in the power system resulting from an open circuit or a fault can be exceedingly large.

An overload is not a fault. The term *overload* merely implies that currents exceed those for which the power system was designed. Such currents are usually much smaller than fault currents. Nevertheless, overloads can create equipment failures by exceeding the thermal design limits of the system. If not corrected, the overload can result in a hazard to personnel. However, such problems occur only with unattended overload operation for an extended time period, whereas faults can create an unsafe condition almost instantly.

Both circuit breakers and fuses are used to protect circuits from excessive current flow, be it a result of faulting or overloading. The circuit-interrupting operation consists of parting a pair of contacts, and since an arc is drawn between the contacts, the process must also extinguish the arc. The interruption is handled mechanically in the circuit breaker, and the excessive current is monitored electrically or thermally. Fuse operation is based on simple thermal operation. The fusible element is responsive to the heat of an overload or fault current and melts open. The fuse jacket is employed to extinguish the subsequent arc. A complete discussion of interrupting devices and the associated protective circuitry is presented in chapter 9.

Fault Types

The fault type often seen in literature is called a bolted fault, which can be described as a zero-resistance short circuit between two or more conductors. In reality most faults are not dead shorts but have some finite value of resistance.

Faults may be classed as permanent or temporary. A permanent fault is one where equipment operation is impossible and repairs are mandatory. A temporary fault is intermittent in nature. For instance, two closely spaced overhead conductors may cause trouble only on windy days, when they can be forced into contact or close proximity by the wind.

A very sinister fault type is the arcing fault. This condition is now believed to be the most common fault. When two conductors of different potential are in very close proximity, the intervening space between them can be considered as a spark gap. If the two conductors are part of an ac power system, the insulating material between the conductors may break down when the sinusoidal waveform reaches a certain value. Fault current will then flow. The potential drop across the conducting gap, which is actually an impedance, remains at a nearly constant level. It is this energy source, releasing terrific quantities of heat, that causes the devastation that is typical of an arcing fault. Soon after the sinusoid reverses polarity, the arc quenches until the spark-gap breakdown voltage or restrike potential is reattained. This repetitive arcing is almost always self-sustaining at ac voltage levels of 480 V and above.

Depending on how the fault occurs, it may be described as three-phase, line-to-line, or line-to-neutral. In mining, the cable shields and the grounding system of the equipment are at the same potential as the system neutral, and line-to-neutral faults are the most prevalent. Line-to-line faults and line-to-neutral faults are unbalanced or unsymmetrical, but the three-phase fault is balanced or symmetrical. These three basic fault descriptions are illustrated in figure 4.32. The impedance, although very small, is shown to signify its finite value.

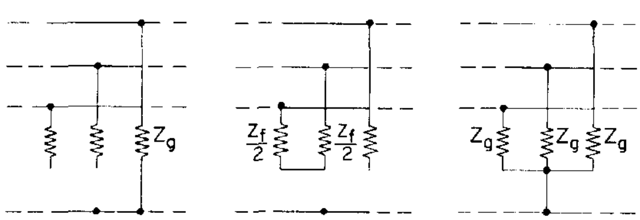


Figure 4.32.—Basic fault descriptions.

Fault Analysis

Fault analysis is a desirable and often mandatory part of any mine power-system analysis. As faulting of some nature can occur at any time, knowledge of how faults affect currents and voltages is necessary to design proper protection and to ensure personnel protection. Although faults usually occur in mine-system trailing cables, the actual fault location and time of occurrence is unpredictable. Consequently, fault analysis is frequently effected on a trial-and-error basis, searching for a worst case solution. It is necessary to assume a fault location, the configuration of power-system components prior to the fault, and sometimes the system loads. Such an effort can result in numerous calculations, to the point where digital computers can be extremely advantageous. Nevertheless, the results provide invaluable information on which to base the design of the mine power system.

Though not particularly common, fault analysis using three-phase faults has distinct advantages. Using this method, balanced faults, like balanced loads, can be investigated on an equivalent per-phase basis and therefore become as simple as faults on single-phase lines. In the majority of cases, bolted three-phase faults cause larger fault currents than line-to-line or line-to-neutral events.

Unsymmetrical faults are often of high interest in mine power systems. Instances include finding a minimum fault current or current flowing in the system neutral conductors. When an unsymmetrical fault is placed on the system, the balance is disrupted. It is possible to solve an unbalanced power system by using a three-phase diagram with symbols assigned to the quantities in each phase and carrying the phase solutions simultaneously. This complicates the problem enormously, but it can be simplified by applying the method of symmetrical components, which reduces the solution of such problems to a systematic form. The reduction is particularly applicable to balanced systems operating under unsymmetrical faults.

SYMMETRICAL COMPONENTS

The method of symmetrical components provides a means for determining the currents and voltages at any point of an unbalanced three-phase power system. In this method, the unsymmetrical phasors representing the unbalanced voltage or current are expressed as the sum of three symmetrical phasor sets. These phasor sets or symmetrical components are designated as the positive sequence, negative sequence, and zero sequence. The first two consist of three balanced phasors with equal magnitude, set 120° apart. The zero-sequence set has three phasors equal in magnitude but operating in the same time. The components are illustrated in figure 4.33, where the instantaneous values may be determined by projection upon the X-axis. The positive-, negative-, and zero-sequence components are then employed to solve the unbalanced-system problem. These sequences are so common in power-system terminology that they are often used to describe the quality of system operation.

It might be asked why the resolution of three phasors into nine phasors simplifies the solution of unbalanced power systems. The answer is straightforward. The resolution results in three symmetrical systems in which each

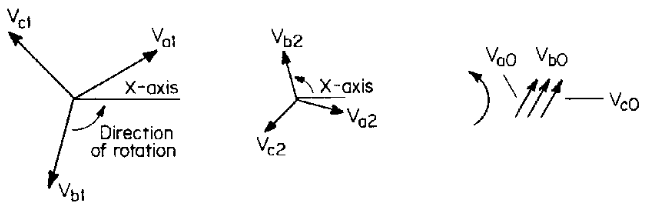


Figure 4.33.—Positive-sequence, negative-sequence, and zero-sequence vector sets.

balanced phasor set can be treated separately, just as in balanced three-phase systems. In other words, the power system can be reduced to per-phase values, then analyzed separately for each symmetrical component. This analysis hinges on the fact that currents and voltages of different sequences do not react upon each other: currents of the positive sequence produce only positive-sequence voltage drops; the same is true for the negative and zero sequences.

In addition to aiding analysis, the method of symmetrical components separates electrical parameters into parts that can represent better criteria of the controlling factors for certain phenomena. For example, the presence of negative-sequence current or voltage immediately implies that the system is unbalanced, and this can be utilized to detect malfunctioning power systems. Grounding phenomena are other good criterion examples; neutral current is very closely related to zero-sequence components.

Sequence Components

The positive sequence for voltage is composed of three symmetrical phasors, $\overline{V}_{a1}$, $\overline{V}_{b1}$, and $\overline{V}_{c1}$ for phases a, b, and c, respectively (fig. 4.33). The quantities have equal magnitude but are displaced by 120° from each other. Therefore, following equation 4.2*b*,

$$\overline{V}_{a1} = \overline{V}_{b1}\,\underline{|120^\circ} = \overline{V}_{c1}\,\underline{|240^\circ}, \qquad (4.34)$$

or rewriting in exponential form,

$$\overline{V}_{a1} = \overline{V}_{a1}, \qquad (4.35a)$$

$$\overline{V}_{b1} = e^{j240}\overline{V}_{a1}, \qquad (4.35b)$$

$$\overline{V}_{c1} = e^{j120}\overline{V}_{a1}. \qquad (4.35c)$$

The unit vector, e^{j120}, is used so frequently that it is given the symbol "a" (not to be confused with the transformer turns ratio), where

$$a = e^{j120} = 1\,\underline{|120^\circ} \qquad (4.36a)$$

and

$$a^2 = e^{j120}e^{j120} = e^{j240}. \qquad (4.36b)$$

Thus the positive-sequence vectors (equations 4.35) are customarily written as

$$\overline{V}_{a1} = \overline{V}_{a1}, \qquad (4.37a)$$

$$\overline{V}_{b1} = a^2\overline{V}_{a1}, \qquad (4.37b)$$

$$\overline{V}_{c1} = a\overline{V}_{a1}. \qquad (4.37c)$$

Equations 4.35 and 4.37 also relate to the standard practice of symmetrical-component calculations; equations are always expressed in terms of the phase a quantities.

There are several mathematical properties of the unit vector a that are useful in computations:

$$\begin{aligned}
1 &= e^{j0} = 1.0 + j0.0,\\
a &= e^{j120} = -0.5 + j0.866,\\
a^2 &= e^{j240} = -0.5 - j0.866,\\
a^3 &= e^{j360} = 1.0,\\
a^4 &= e^{j480} = e^{j120} = a,\\
a^5 &= e^{j600} = e^{j240} = a^2;
\end{aligned}$$

and for specific calculations:

$$\begin{aligned}
1 + a^2 + a &= 0,\\
a - a^2 &= \sqrt{3}e^{j90} = j\sqrt{3},\\
a^2 - a &= \sqrt{3}e^{-j90} = -j\sqrt{3},\\
1 - a &= 3e^{-j30} = 1.5 - j0.866,\\
1 - a^2 &= \sqrt{3}e^{j30} = 1.5 + j0.866.
\end{aligned}$$

These allow easy conversion to simpler forms when symmetrical components are being manipulated mathematically. For the negative and zero sequences (fig. 4.33), the symmetrical voltage sequences can be written

$$V_{a2} = V_{b2}\,\underline{|-120^\circ} = V_{c2}\,\underline{|120^\circ} \qquad (4.38)$$

and

$$V_{a0} = V_{b0} = V_{c0}. \qquad (4.39)$$

Rewriting these equations in terms of the unit vector, a, it is found that for the negative sequence,

$$V_{a2} = V_{a2}, \qquad (4.40a)$$

$$V_{b2} = aV_{a2}, \qquad (4.40b)$$

$$V_{c2} = a^2V_{a2}, \qquad (4.40c)$$

and for the zero sequence,

$$V_{a0} = V_{a0}, \qquad (4.41a)$$

$$V_{b0} = V_{a0}, \qquad (4.41b)$$

$$V_{c0} = V_{a0}. \qquad (4.41c)$$

It is important to note that in all three sequence systems, the subscripts denote specific components in each phase (a, b, or c). Furthermore, the subscripts, 1, 2, and 0 signify whether that component is part of the positive-, negative-, or zero-sequence set. Using the same reasoning, symmetrical-component equations can also be written for current.

Sequence-Quantity Combinations

The total voltage or current of any phase is equal to the vectorial sum of the corresponding components in that phase. Figure 4.34 illustrates this concept for three unbalanced voltage phasors, $\overline{V}_a$, $\overline{V}_b$, and $\overline{V}_c$. Expressed mathematically,

$$\overline{V}_a = \overline{V}_{a0} + \overline{V}_{a1} + \overline{V}_{a2}, \tag{4.42a}$$

$$\overline{V}_b = \overline{V}_{b0} + \overline{V}_{b1} + \overline{V}_{b2}, \tag{4.42b}$$

$$\overline{V}_c = \overline{V}_{c0} + \overline{V}_{c1} + \overline{V}_{c2}. \tag{4.42c}$$

Substituting in the equivalent values given by equations 4.37, 4.40 and 4.41, equations 4.42 become

$$\overline{V}_a = \overline{V}_{a0} + \overline{V}_{a1} + \overline{V}_{a2}, \tag{4.43a}$$

$$\overline{V}_b = \overline{V}_{a0} + a^2\overline{V}_{a1} + a\overline{V}_{a2}, \tag{4.43b}$$

$$\overline{V}_c = \overline{V}_{a0} + a\overline{V}_{a1} + a^2\overline{V}_{a2}. \tag{4.43c}$$

These equations state that an unbalanced system can be defined in terms of three balanced phasor sets. In other words, positive-, negative-, and zero-sequence components of phase a can be added together to obtain the unbalanced phasors. Following convention, the equations are expressed only in phase a quantities.

Similarly, three unbalanced voltages or currents may be resolved into their symmetrical components. Consider the zero sequence first. By adding equations 4.43*a*, 4.43*b*, and 4.43*c* together,

$$\overline{V}_a + \overline{V}_b + \overline{V}_c = 3\overline{V}_{a0} + (1+a^2+a)\,\overline{V}_{a1} + (1+a+a^2)\overline{V}_{a2}.$$

Since $1 + a^2 + a = 0$,

$$\overline{V}_{a0} = \tfrac{1}{3}(\overline{V}_a + \overline{V}_b + \overline{V}_c). \tag{4.44a}$$

If equation 4.43*b* is multiplied by a and equation 4.43*c* by a^2 and these results are added to equation 4.42*a*,

$$\overline{V}_a + a\overline{V}_b + a^2\overline{V}_c = (1+a+a^2)\overline{V}_{a0} + 3\overline{V}_{a1} + (1+a^2+a)\overline{V}_{a2}.$$

Therefore, $$\overline{V}_{a1} = \tfrac{1}{3}(\overline{V}_a + a\overline{V}_b + a^2\overline{V}_c), \tag{4.44b}$$

which relates the positive-sequence component of phase a to the unbalanced vectors. Finally, for the negative sequence, if equation 4.43*b* is multiplied by a^2 and equation 4.43*c* by a,

$$\overline{V}_a + a^2\overline{V}_b + a\overline{V}_c = (1+a^2+a)\overline{V}_{a0} + (1+a+a^2)\overline{V}_{a1} + 3\overline{V}_{a2}.$$

Then, $$V_{a2} = \tfrac{1}{3}(\overline{V}_a + a^2\overline{V}_b + a\overline{V}_c). \tag{4.44c}$$

Equations 4.44*a*, 4.44*b*, and 4.44*c* are therefore the reverse of equations 4.43*a*, 4.43*b*, and 4.43*c*; they allow the symmetrical components to be written in terms of the unbalanced phasors.

Symmetrical-Component Relationships

Currents in equivalent delta-connected and wye-connected loads or sources form a good basis to illustrate the existence of symmetrical components in three-phase circuits. Consider the two loads shown in figure 4.35, where $\overline{I}_{ab}$, $\overline{I}_{bc}$, and $\overline{I}_{ca}$ are the three phase currents and $\overline{I}_a$, $\overline{I}_b$ and $\overline{I}_c$ are the line currents. These may all be assumed to result from an unbalanced condition.

At the three terminals of the delta load, the following relationships are satisfied by Kirchhoff's current law:

$$\overline{I}_a = \overline{I}_{ab} - \overline{I}_{ca}, \tag{4.45a}$$

$$\overline{I}_b = \overline{I}_{bc} - \overline{I}_{ab}, \tag{4.45b}$$

$$\overline{I}_c = \overline{I}_{ca} - \overline{I}_{bc}. \tag{4.45c}$$

The zero-sequence currents of the wye-connected load are (using equation 4.44*a*):

$$\overline{I}_{a0} = \tfrac{1}{3}(\overline{I}_a + \overline{I}_b + \overline{I}_c). \tag{4.46}$$

Substituting equations 4.45 into equation 4.46 it is found that

$$\overline{I}_{a0} = \tfrac{1}{3}[(\overline{I}_{ca} + \overline{I}_{ab} + \overline{I}_{bc}) - (\overline{I}_{ab} + \overline{I}_{bc} + \overline{I}_{ca})] = 0. \tag{4.47}$$

This shows that the zero-sequence current of a three-phase circuit feeding into a delta connection is always zero. In addition, the currents to a three-phase wye-connected load with a floating neutral (fig. 4.35*B*) can have no zero-sequence component. Simply, a neutral-return circuit must be available for zero-sequence currents to flow. However, zero-sequence current may circulate in a delta connection without escaping into a neutral conductor (see figure 4.35*A*, note directions of $\overline{I}_{ab}$, $\overline{I}_{bc}$, and $\overline{I}_{ca}$).

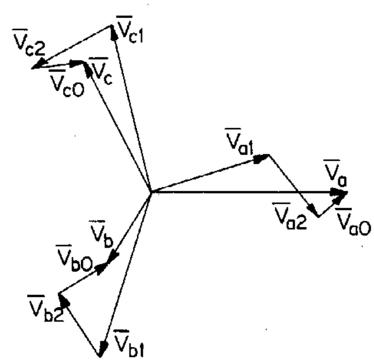


Figure 4.34.—Symmetrical component addition to obtain unbalanced three-phase set.

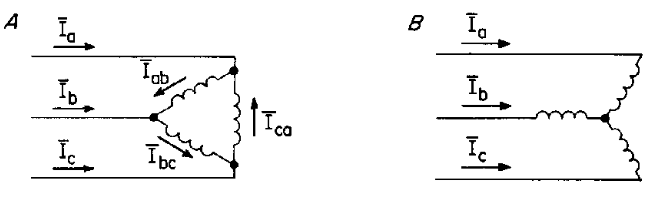


Figure 4.35.—Equivalent delta-connected (*A*) and wye-connected (*B*) loads.

For transforming positive-sequence line currents to phase currents, it can be shown from

$$\bar{I}_{a1} = \frac{1}{3}(\bar{I}_a + a\bar{I}_b + a^2\bar{I}_c), \tag{4.48}$$

which applies equation 4.44*b* to current, that

$$\bar{I}_{ab1} = \frac{j}{\sqrt{3}}\bar{I}_{a1}. \tag{4.49}$$

Using a similar process for negative-sequence currents,

$$\bar{I}_{ab2} = \frac{-j}{\sqrt{3}}\bar{I}_{a2}. \tag{4.50}$$

When the foregoing is applied to line-to-neutral and line-to-line voltages for figure 4.33, the transformation equations are

$$\overline{V}_{ab0} = 0, \tag{4.51a}$$

$$\overline{V}_{ab1} = j\sqrt{3}\,\overline{V}_{a1}, \tag{4.51b}$$

$$\overline{V}_{ab2} = -j\sqrt{3}\,\overline{V}_{a2}. \tag{4.51c}$$

where $\overline{V}_{ab0}, \overline{V}_{ab1}, \overline{V}_{ab2}$ = zero-, positive-, and negative-sequence line-to-line voltages, V,

$\overline{V}_{a0}, \overline{V}_{a1}, \overline{V}_{a2}$ = zero-, positive-, and negative-sequence line-to-neutral voltages, V.

These equations demonstrate another general relationship of zero-sequence components: a line-to-line voltage, however unbalanced, can have no zero-sequence component. Line-to-neutral voltages, on the other hand, may have a zero-sequence value.

Symmetrical-Component Impedance

Before the solution of unbalanced system problems can be discussed, the concepts of impedance under the influence of symmetrical components need to be covered. Impedance relates the current in a circuit to the impressed voltage. Symmetrical-component impedance behaves in a similar manner, except that it is sometimes affected by additional parameters. There are three likely cases for a power system: an unbalanced static network, a balanced static network, and the balanced nonstatic network. All these impedance values are created by the fact that positive- and negative-sequence currents produce only positive- and negative-sequence voltage drops, respectively. The flow of zero-sequence currents in a neutral can result in an impedance that is apparently greater than the actual impedance.

In an unbalanced static network, the sequence impedances in a particular phase are equal, but not necessarily equal to those in another phase:

$$Z_{a0} = Z_{a1} = Z_{a2}, \tag{4.52a}$$

$$Z_{b0} = Z_{b1} = Z_{b2}, \tag{4.52b}$$

$$Z_{c0} = Z_{c1} = Z_{c2}, \tag{4.52c}$$

where Z_{a0}, Z_{b0}, and Z_{c0} are symmetrical-component impedances for the zero sequence; Z_{a1}, Z_{b1}, and Z_{c1} are symmetrical-component impedances for the positive sequence; and Z_{a2}, Z_{b2}, and Z_{c2} are symmetrical-component impedances for the negative sequence.

The balanced nonstatic network is given by

$$Z_{a0} = Z_{b0} = Z_{c0}, \tag{4.53a}$$

$$Z_{a1} = Z_{b1} = Z_{c1}, \tag{4.53b}$$

$$Z_{a2} = Z_{b2} = Z_{c2}. \tag{4.53c}$$

This states that in a balanced nonstatic network the impedances in a sequence are equal, but not necessarily equal to the other sequence component impedances. Cables and powerlines are included in this case, and specifically,

$$Z_{a1} = Z_{a2} \neq Z_{a0}. \tag{4.54}$$

The last likely case is the balanced static network, where

$$Z_a = Z_b = Z_c. \tag{4.55}$$

It should be obvious that this is a situation where symmetrical components would not normally be applied.

As a general rule, positive- and negative-sequence impedances of a power system are on the same order of magnitude, but the system zero-sequence impedance may vary through a very wide range. This range is dependent upon the resistance-to-reactance ratio as seen by the zero-sequence current.

Fault Calculations

One of the most significant uses for the method of symmetrical components is the computation of voltages and currents resulting from unbalanced faults. The three-phase diagram in figure 4.36 represents a simple power system with a four-wire wye-connected source. The impedance of each phase conductor is Z, while Z_0 is the neutral-conductor impedance. A bolted line-to-neutral fault is occurring on a phase a (an x signifies the fault). The resulting current in the fault, $\bar{I}_f$, is of interest, and the following shows how symmetrical components can be used to find its value.

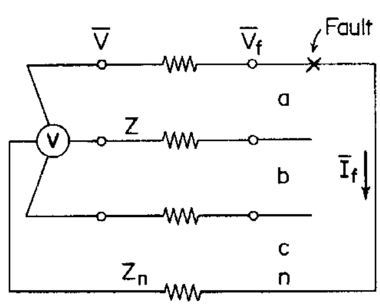


Figure 4.36.—Three-phase system with line-to-neutral fault.

It is apparent from figure 4.36 that the line currents under the fault condition are

$$\bar{I}_a = \bar{I}_f, \bar{I}_b = 0, \bar{I}_c = 0, \tag{4.56}$$

where $\bar{I}_f$ = current in fault, A,
and $\bar{I}_a, \bar{I}_b, \bar{I}_c$ = unbalanced line currents, A.

Applying equation 4.43, the symmetrical components of these currents are

$$\bar{I}_{a1} = \tfrac{1}{3}(\bar{I}_a + a\bar{I}_b + a^2\bar{I}_c) = \tfrac{1}{3}\bar{I}_f, \tag{4.57a}$$

$$\bar{I}_{a2} = \tfrac{1}{3}(\bar{I}_a + a^2\bar{I}_b + a\bar{I}_c) = \tfrac{1}{3}\bar{I}_f, \tag{4.57b}$$

$$\bar{I}_{a0} = \tfrac{1}{3}(\bar{I}_a + \bar{I}_b + \bar{I}_c) = \tfrac{1}{3}\bar{I}_f; \tag{4.57c}$$

therefore,
$$\bar{I}_{a1} = \bar{I}_{a2} = \bar{I}_{a0} = \tfrac{1}{3}\bar{I}_f. \tag{4.58}$$

To define the fault completely, it must be known whether the fault between line a and the neutral is a dead short or exhibits an impedance. Although all faults have a finite impedance, the faulting assumption states that it is bolted. Therefore, fault impedance is zero and the voltage across the fault, $\bar{V}_{fn}$, is also zero. With this, the current through the fault, $\bar{I}_f$, can be described. However, to perform the required computations, it is necessary to know the force driving the fault current and the impedance existing between this driving potential and the fault location.

The source, $\bar{V}$, is the driving potential, and it can be assumed as purely positive sequence. It is also assumed from figure 4.36 that the source has negligible internal impedance (in practical situations, however, the source impedance is of great importance). Therefore, the source line-to-neutral potentials are equal to the terminal voltages:

$$\bar{V}_{an} = \bar{V}_a, \bar{V}_{bn} = \bar{V}_b, \bar{V}_{cn} = \bar{V}_c,$$

where $\bar{V}_{an}, \bar{V}_{bn}, \bar{V}_{cn}$ = terminal line-to-neutral voltages
and $\bar{V}_a, \bar{V}_b, \bar{V}_c$ = corresponding ideal-source potentials.

The impedances involved are simply the line impedance of phase a (Z) and the neutral impedance (Z_n). With these parameters known, the process is now to convert the unbalanced system to symmetrical components, solve the problem in terms of these balanced vectors, and then reconstruct the result for the fault current.

Following convention, all work is performed in phase a quantities. Notwithstanding, phase a contains the line-to-neutral fault; thus, it is the only phase involved. Since only positive-sequence voltage is supplied by the source, the symmetrical components of the driving potential are

$$\bar{V}_{an1} = \bar{V}_{an}, \bar{V}_{an2}, = 0, \bar{V}_{an0} = 0. \tag{4.59}$$

The impedance to positive-sequence or negative-sequence current in any of the three lines is equal; thus, for phase a,

$$Z_{a1} = Z, Z_{a2} = Z.$$

Zero-sequence current follows a different path from positive or negative sequence. From the source to the fault, zero-sequence current, $\bar{I}_0$, exists in each line, but from the fault back to the source (through the neutral conductor) the current is $3\bar{I}_0$. Zero-sequence impedance, Z_{a0}, is therefore greater than Z. As was implied in the preceding section, the quantification of Z_{a0}, or just simply Z_0, is not an easy matter. However, in order to limit the amount of fault current flowing in mine power-system neutrals, large resistances are placed in the neutral circuit. With this in mind, the resistance-to-reactance ratio of the neutral impedance, Z_n, is very large, and in this instance for mine power systems under line-to-neutral faults, the impedance seen by the zero-sequence current can be approximated as

$$Z_0 = Z + 3Z_n. \tag{4.60}$$

Loop equations for each sequence current can now be expressed for figure 4.36. If a voltage is assumed to exist across the fault, for phase a,

$$\bar{V}_{an1} = Z_{a1}\bar{I}_{a1} + \bar{V}_{f1}, \tag{4.61a}$$

$$\bar{V}_{an2} = Z_{a2}\bar{I}_{a2} + \bar{V}_{f2} = 0, \tag{4.61b}$$

$$\bar{V}_{an0} = Z_0\bar{I}_{a0} + \bar{V}_{f0} = 0, \tag{4.61c}$$

where $\bar{V}_{an1}, \bar{V}_{an2}, \bar{V}_{an0}$ = sequence voltages for source, V,
$\bar{I}_{a1}, \bar{I}_{a2}, \bar{I}_{a0}$ = sequence components of fault current, A,
$\bar{V}_{f1}, \bar{V}_{f2}, \bar{V}_{f0}$ = sequence components of voltage across fault, V,
and Z_{a1}, Z_{a2}, Z_0 = sequence impedance seen by fault current, Ω.

Equation 4.59 generalizes the fault condition and is practical because of fault impedance. However, a bolted fault has been assumed to exist; thus,

$$\bar{V}_{f1} = \bar{V}_{f2} = \bar{V}_{f0} = 0. \tag{4.62}$$

All input to the problem is now available, and simultaneous solution of equations 4.57 through 4.62 shows that

$$\bar{V}_{an} = \bar{V}_{an1} = \tfrac{1}{3}Z\bar{I}_f + \tfrac{1}{3}Z\bar{I}_f + \tfrac{1}{3}Z_0\bar{I}_f \tag{4.62a}$$

or
$$I_f = \frac{V_{an}}{\tfrac{1}{3}(2Z + Z_0)}, \tag{4.62b}$$

but
$$Z_0 = Z + 3Z_n,$$

then
$$I_f = \frac{V_{an}}{Z + Z_n}. \tag{4.62c}$$

Consequently, symmetrical components have been employed to solve this unbalanced faulting problem.

This work can easily be expanded to cover other unbalanced faulting problems, and the process can be employed to solve any unbalanced three-phase or even polyphase condition. Because fault analysis is imperative in protective-device sizing, additional discussion can be found in chapter 10.

POWER TERMINOLOGY

If the sum of the electrical ratings is made for all equipment in a power complex, the result will provide a total connected load. The measure could be expressed in horsepower, but the electrical quantities of kilowatts, kilovoltamperes, or amperes are more suitable units. Note that the connected horsepower can be converted to connected kilowatt simply by multiplying by 0.746. Many loads operate intermittently, especially mining production equipment, and other equipment operates at less than full load. Accordingly, the demand upon the power source is less than the connected load. This fact is important in the design of any mine power system, as the system should be designed for what the connected load actually uses, rather than the total connected load. Obviously, these considerations have great impact on power-system investment or the capital required to build the system.

Because of the importance of assessing equipment power demands, the Institute of Electrical and Electronics Engineers (IEEE) has standard definitions for load combinations and their ratios. The important ones follow (*3*).

- *Demand* is the electrical load for an entire complex or a single piece of equipment averaged over a specified time interval. The time or demand interval is generally 15 min, 30 min, or 1.0 h, and demand is generally expressed in kilowatts, kilovoltamperes, and amperes.
- *Peak load* is the maximum load consumed or produced by one piece or a group of equipment in a stated time period. It can be the maximum instantaneous load, the maximum average load, or (loosely) the maximum connected load over the time period.
- *Maximum demand* is largest demand that has occurred during a specified time period.
- *Demand factor* is the ratio of the maximum demand to the total connected load.
- *Diversity factor* is the ratio of the sum of the individual maximum demands for each system part of subdivision to the complete system maximum demand.
- *Load factor* is the ratio of the average load to the peak load, both occurring in the same designated time period. This can be implied to be also equal to the ratio of actual power consumed to total connected load in the same time period.
- *Coincident demand* is any demand that occurs simultaneously with any other demand.

All these definitions may be applied to the units of average power, apparent power, or current. Thus they are invaluable in power-system design. A few examples are in order to illustrate their versatility.

Consider a feeder cable supplying several mining sections in an underground mine. The sum of the connected loads on the cable, multiplied by the demand factor of these loads, yields the maximum demand that the cable must carry. When applied to current, this demand would be the maximum amperage. Good demand factors for mine power systems range from 0.8 to 0.7 depending upon the number of operation sections. The lower value is used when there are fewer producing units, that is, from two to four. The demand factor can be extended to include estimates of average load. For instance, the sum of the average loads on the cable, multiplied by the demand factor, provides the average load on the cable. A prime application here is for approximating the current that a conductor is expected to carry. If, for example, 10 identical mining sections draw 53 A each; the conductors feeding all these sections would be expected to carry

$$(\text{total average load})(\text{demand factor}) = (\text{average load})$$

or

$$(10)(53\text{ A})(0.8) = 424\text{ A}.$$

The demand factor and the diversity factor can be applied to many other mine electrical areas, such as estimating transformer capacities, protective-circuitry continuous ratings, and the load that a utility company must supply.

The load factor can be used to estimate the actual loads required by equipment. Here, the total connected load multiplied by the load factor is an approximation of the actual power consumed. It should be noted that the average load factor in underground coal mining tends to be rather low, mainly because of the cyclic nature of equipment operation but also because of the employment of high-horsepower motors that are needed to perform specific functions but only operate for a small fraction of the possible running time. For instance, when cutting and loading, a continuous miner will have all motors operating, thus have a total connected load of 385 hp or $(0.746)385 = 287$ kW. The average load factor might be 0.6; therefore, the actual power consumed is $(0.6)287$ or 172 kW. The load factor can also be applied to equipment combinations.

The maximum power demand normally forms one basis that utility companies use to determine power bills; most often, 1 month is the specified time period. Demand meters are often installed at the utility company metering point.

Chapter 4 has covered a broad range of fundamentals projected towards three-phase power systems in mining. Items have included balanced three-phase circuit analysis, the per-unit system, the method of symmetrical components, and specific terminology to describe power-system operation. Comprehension of this material is vital in order to understand many chapters that follow.

REFERENCES

1. American National Standards Institute (New York). ANSI Standard Device Numbers C37.2. 1978.

2. ———. Graphical Symbols for Electrical Diagrams Y32.2. 1970.

3. Institute of Electrical and Electronics Engineers (New York). Recommended Practice for Electric Power Distribution for Industrial Plants. Stand. 141-1986.

CHAPTER 5.—BASIC SOLID-STATE DEVICES AND INSTRUMENTATION

Through the advancement of technology, the motor-generator (m–g) sets and Ignitron rectifiers for power conversion used in early mining have been all but replaced by semiconductor devices, except for m–g sets and synchronous rotary converters in specific surface mining equipment. Equipment employing semiconductors exclusively is often termed *solid state* or *static*. In mine power systems the use of semiconductors has grown from simple rectification (the conversion of power to direct current (dc)) to include such areas as motor and equipment control, protective relaying, and lighting power supplies, not to mention extensive use in communications and instrumentation.

Since the topics of solid-state devices and basic instrumentation are closely related, they are introduced together in this chapter. The discussion will be primarily informative rather than theoretical.

SEMICONDUCTORS

Semiconductors are nonmetallic elements that are characterized by relatively poor conductivity. Silicon is the most popular and germanium the second most important semiconductor in electrical or electronic applications. Semiconductors are useful in electrical circuits because they can pass current in two different conduction modes when impurities or imperfections exist in their crystal lattices. The process of carefully adding impurities to a pure or intrinsic semiconductor crystal while it is being grown is called doping. The impurities are selected for their size, so they will fit into the crystal lattice and provide either an excess or a deficiency of electrons.

For example, when a few parts per million of arsenic atoms are added to germanium, or antimony atoms are added to silicon in the crystal structure, an overabundance of free electrons is created. The result is a net negative effect, and the crystal is termed an *n-type semiconductor*. If a potential is placed across the impure crystal, conduction occurs primarily through an apparent drift of these free electrons. On the other hand, if indium or gallium is used to dope germanium or silicon, a deficiency of electrons exists, and an excess positive charge is created in the doped crystal. Thus, it is called a *p-type semiconductor*. If a potential is applied, the atoms conduct current by an apparent movement of electron sites or holes. These holes are places in the crystal lattice where an electron can be held temporarily. When there is an abundance of holes, free electrons generated within the crystal can quickly recombine with available atoms.

The free electrons in the n-material and the holes in p-material are known as majority carriers. However, because of thermal and other energies, free electrons are also found in a lesser amount in the p-type and a few holes exist in n-type semiconductors. These are called minority carriers. Nevertheless, even with the excess charge, both semiconductor types are electrically neutral.

DIODES AND RECTIFIERS

The operation of most semiconductor devices is dependent upon a p–n junction, which is the boundary formed when a piece of p-type material is joined with a piece of n-type. In the actual production, a single semiconductor crystal (or monocrystalline material) is grown so that part is doped to create a p-type region, with the balance doped for n-type. A solid-state diode or rectifier has one p–n junction; it is a device that readily passes current in one direction but does not permit appreciable current in the opposite direction. The symbol for a diode or rectifier is given in figure 5.1*A*.

Figure 5.1*B* is a simple model of a diode that can be used to explain p–n junction electrical operation. When the two semiconductor materials are joined, a charge redistribution occurs. Both the p-region and the n-region contain a high concentration of majority carriers. Electrons from the n-material diffuse across the junction to the p-material; similarly, holes migrate from the p-material into the n-material. The net result of this diffusion is a depletion region with negatively charged (acceptor) ions on the p-side and positively charged (donor) ions on the n-side of the junction. The electric field across the depletion region is established and opposes further majority-carrier diffusion, but the field creates a minority-carrier flow across the junction in the opposite direction.

Current caused by majority-carrier diffusion is called injection current, I_I, and that from minority carriers, saturation current, I_S. If no external voltage is applied to the p–n device (fig. 5.2*A*), the junction is in equilibrium because the net hole and electron flow across the junction is zero. In other words, injection current equals saturation current. However, if an external voltage is applied with a polarity such that the p-region is positive with respect to the n-region (fig. 5.2*B*), the depletion-region electric field is decreased, and a large number of majority carriers are able to cross the junction and diffuse toward the device terminals. Hence, injection current is substantially increased, and because saturation current remains constant, the result is current flow in the external circuit. In this case, the external voltage polarity is called forward bias,

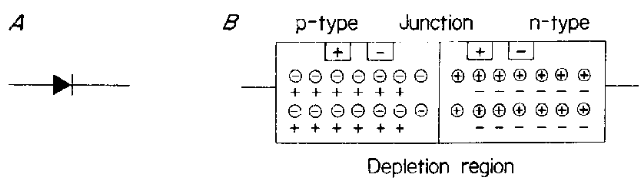


Figure 5.1.—Symbol (*A*) and operation (*B*) of a p-n junction device.

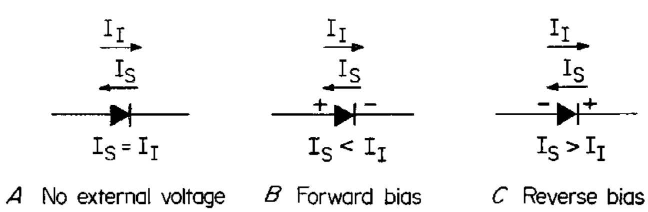


Figure 5.2.—Bias conditions and current flow for a diode.

and the current is forward current. Conversely, reverse bias, an applied voltage of reverse polarity (fig. 5.2*C*), opposes majority-carrier diffusion by enforcing the depletion-region electric field, and current is greatly reduced. As saturation current is still constant, the external reverse current is primarily a result of minority-carrier diffusion and is therefore very small.

Because there are many more majority carriers than minority carriers, the injection current, under forward bias, is orders-of-magnitude greater than the constant saturation current. As external circuit current is the algebraic sum of injection and saturation currents, forward current is significantly greater than reverse current. Furthermore, to enhance this one-directional current phenomenon, junctions are manufactured in which one side of the junction is more heavily doped than the other. Forward current is then mainly a result of majority carriers from the more heavily doped region.

The arrow portion of the diode symbol (fig. 5.1*A*) points in the same direction as forward current. As a carryover from vacuum-tube terminology, the side symbolized by the arrow is also called an *anode* (the p-region), with the opposite terminal, the *cathode* (the n-region).

Diode Equations

The number of minority carriers is dependent upon temperature and the difference in energy levels between the p- and n-regions. If the energy difference is constant, the concentration of minority carriers plus the saturation and reverse currents will vary exponentially with temperature. Therefore temperature is a limiting factor in diode operation, and the maximum rated current of a given device is determined by the heat-dissipating properties of the device mounting system.

The formula relating external and saturation current with the energy difference and temperature is

$$I = -I_S\,(e^{\frac{qV}{KT}} - 1), \qquad (5.1)$$

where I_S = saturation current, A,
q = charge of one electron, 16×10^{-20} C,
V = voltage across junction (less than external voltage, but approximately equal to it), V,
qV = energy difference between p- and n-materials,
K = Boltzmann constant, 1.38×10^{-23} J/K,
and T = junction temperature, K.

At room temperature (300 K),

$$I = -I_S(e^{39V} - 1), \qquad (5.2)$$

or at other temperatures,

$$I = -I_S(e^{(39V\,\frac{T_1}{T_2})} - 1), \qquad (5.3)$$

where T_1 = 300 K,
and T_2 = other temperature, K.

The negative sign for the saturation current denotes it as flowing in the opposite direction to forward current. The equations relate that if voltage is 0.1 V or more negative, the external current is about equal to the saturation current. Therefore, by placing a reverse bias across the device and measuring the resulting reverse current, the forward current can be predicted.

The foregoing equations result in the theoretical curve, termed a *characteristic curve*, which is given in figure 5.3. This curve diverges from that for an actual diode in one main aspect, the breakdown of the p–n junction noted at point c. Here, the external voltage meets the limit capabilities of the junction, and a greater reverse voltage will create an avalanche current that can destroy the device. As a result, p–n junctions normally require a rating for maximum reverse voltage or peak inverse voltage (PIV). Zener diodes are of special interest as they operate in this avalanche current area to regulate an applied dc voltage.

As long as the p–n junction is operated within the limits of its reverse voltage and forward current, the device can be represented by a very low resistance for forward-bias conditions and a high resistance during reverse bias. Ideally, and for the majority of applications, a diode can be assumed to have zero resistance under forward bias and infinite resistance under reverse bias.

Rectifier Circuits

A rectifier can be considered as a diode specifically designed or applied to convert power to dc. The principal application in mining is to use the unilateral properties of the rectifier for direct alternating current (ac) power conversion. With single-phase ac, there are three basic rectifier circuits to perform this function: half-wave, full wave, and bridge.

Figure 5.4*A* illustrates the circuit of a simple half-wave rectifier in which a transformer magnetically couples the source to the rectifier. This could also be direct, unisolated source connection. With a sinusoidal voltage input (fig. 5.4*B*), the rectifier acts as a switch. When forward biased (positive anode with respect to the cathode), the load, R, is electrically connected to the source, but during reverse biasing it is disconnected. In other words, low and high resistances to current exist with respect to the bias condition. These resistances create a pulsating dc waveform across the load, as shown in figure 5.4*C*. This variation of voltage is often termed *ripple*.

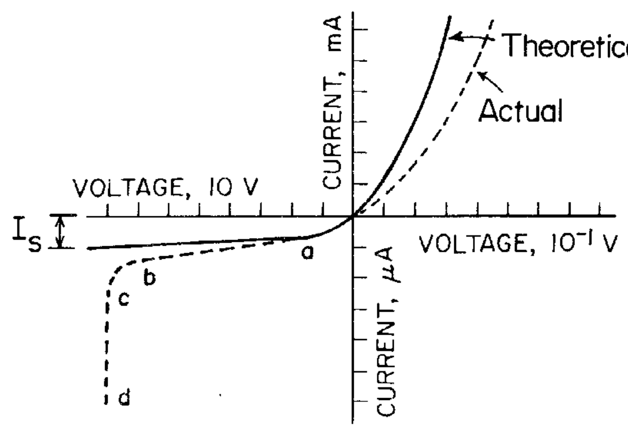


Figure 5.3.—Diode or rectifier characteristic curve.

Only the positive portions of the input sinusoid appear in the pulsating dc output, and as a result, the conversion efficiency of the half-wave rectifier leaves much to be desired. The single-way full-wave rectifier is a method of rectifying both the positive and negative portions of a sinusoidal voltage input, and it can be analyzed as two half-wave rectifiers. The circuit shown in figure 5.5*A* utilizes a center-tapped transformer secondary. When referenced to ground, the V_2 and V_2' waveforms (fig. 5.5*B*) are then 180° out of phase. Therefore, one rectifier is conducting current (forward biased) while the other is not (reverse biased). The consequence is pulsating dc power to the load during both the negative and positive portions of the ac input (fig. 5.5*C*). Conversion efficiency is greatly improved over half-wave circuits.

Full-wave rectification can also be obtained with the bridge rectifier. As shown in figure 5.6*A*, the circuit employs a transformer with a single secondary and four rectifiers. During either the positive or negative portions of the input waveforms, two of the rectifiers are effectively in series with the load resistance. For instance, when the top secondary transformer rectifiers D_2 and D_3 are forward biased but D_1 and D_4 are reverse biased, current flows from the top secondary terminal through D_2, R_L, and D_3 back to the transformer. The rectifier biasing condition reverses with the transformer secondary polarity (figure 5.6*B*, bottom), but the current through the load has the same direction. Hence, the same full-wave pulsating dc waveform in figure 5.6*C* appears across the load with only half of the secondary turns needed for the single-way full-wave rectifier.

Although the output of these three basic rectifier circuits is effectively dc and the current flow is in only one direction, the voltage fluctuation or ripple is often too great to be useful. Consequently, filtering is required to change the pulsating voltage to a relatively ripple-free potential. This filtering action is provided by inductors in series with the load, or capacitors shunting (in parallel with) the load, or both. Each of these methods will smooth the voltage output. An example of this filtering is shown in figure 5.7. It will be shown later that such filtering is not needed for dc mining equipment.

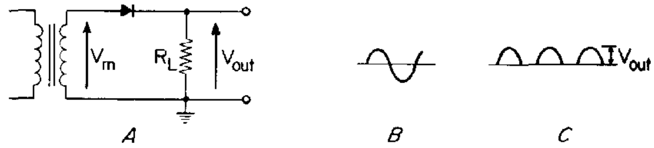


Figure 5.4.—Half-wave rectifier circuit and waveforms.

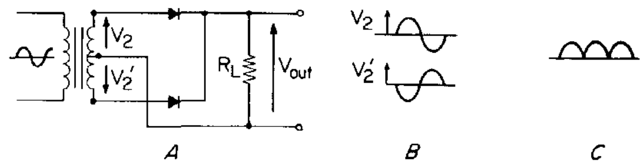


Figure 5.5.—Single-way full-wave rectifier waveforms.

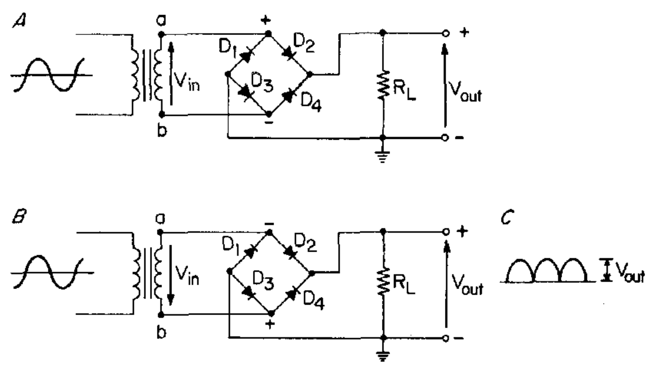


Figure 5.6.—Bridge rectifier circuit and waveforms.

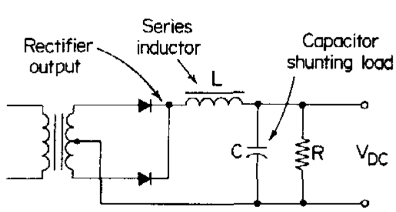


Figure 5.7.—Example of filtering a rectifier output.

Cooling

It was stated earlier that the operation of a p–n junction is highly dependent upon temperature. It follows that there exists a maximum temperature beyond which the device will be destroyed if operated. Such a point is called the maximum junction operating temperature. For silicon semiconductors, this temperature is usually around 175° to 200° C, for germanium, 85° to 110° C, but the maximum varies according to the individual device and manufacturer.

The temperature at which the junction operates is dependent upon the power dissipated in the junction, the ambient temperature, and the ability of the device to transfer heat to the surrounding environment. Devices designed and operated for small currents usually do not need cooling assistance. However, adequate external cooling is required in p–n junctions dissipating 1 or more watts. The simplest method is to mount the semiconductor case securely on a heat sink, which is commonly metal with a large surface area. Thermally conductive washers, silicon compounds, and correct bolting pressure allow good heat transfer from the device to the heat sink, and air convection transfers heat to the surrounding atmosphere. In high-power applications, forced-air cooling of the heat sink is sometimes employed to increase heat dissipation further.

Figure 5.8 illustrates a rectifier using a heat sink for this purpose. The diagram in figure 5.9 represents the typical relationships in all solid-state devices between the

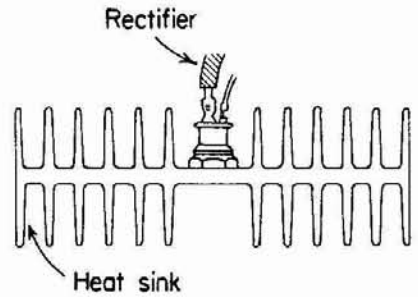


Figure 5.8.—Heat sink cooling.

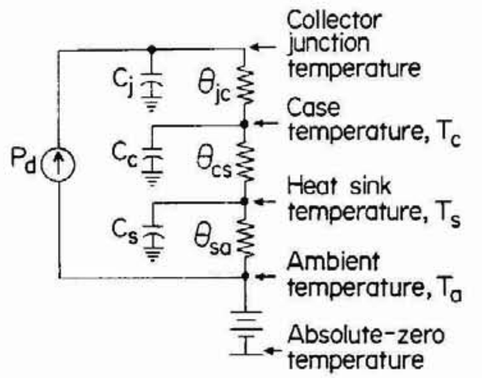


Figure 5.9.—Heat sink thermal relationships.

solid-state device, its heat sink, and the surrounding environment. The following equation relates these parameters:

$$T_j = \theta_{ja}P_d + T_a, \tag{5.4}$$

where T_j = junction temperature, °C,
T_a = ambient temperature, °C,
P_d = power dissipated by junction, W,
and θ_{ja} = ambient-to-junction "thermal resistance," °C/W.

The last item, thermal resistance, is actually composed of three parts, as shown in figure 5.9,

$$\theta_{ja} = \theta_{jc} + \theta_{cs} + \theta_{sa}, \tag{5.5}$$

where θ_{jc} = junction-to-case thermal resistance, °C/W,
θ_{cs} = case-to-heat-sink thermal resistance, °C/W,
and θ_{sa} = heat-sink-to-ambient thermal resistance, °C/W.

The junction-to-case and the heat-sink-to-ambient thermal resistances are almost always available from manufacturers. The thermal resistance between the device case and the heat sink can be neglected if the mounting is carried out correctly as described here.

Junction power can be found by the relationship

$$P_d = I_{max}V_f, \tag{5.6}$$

where I_{max} = maximum forward current, A,
and V_f = junction forward voltage drop, V.

The junction forward voltages normally range from 0.5 to 0.75 V for silicon and from 0.2 to 0.3 V for germanium, but typical values for specific devices are also available from manufacturers. When the total thermal resistance, θ_{ja}, is known, the operating junction temperature can be calculated and compared with the maximum allowed.

Overloads

The thermal relationship of figure 5.9 shows three capacitances, C_j, C_c, and C_s, which are the thermal capacitances of the p–n junction, the device case, and the heat sink, respectively. Thermal capacitance resists changes in temperature in the same way that capacitance restricts voltage change. For the p–n junction, C_j is usually very small; hence, its time constant is also small. This means that the semiconductor must not be overloaded (excessive power dissipation) for more than a few milliseconds; otherwise, the device will be destroyed. For this reason, high-speed overload protection must be applied to semiconductor devices. For rectifiers, the protection takes two forms: against excessive overloads and short circuits in load currents, and against failure in the rectifier itself (overtemperature or excessive voltage).

THREE-PHASE RECTIFICATION

Large amounts of dc power at either 250 or 500 Vdc are required for locomotives and face equipment in many mining operations. When more than a fractional kilowatt of dc power is needed from an ac source, a polyphase rectifier circuit is employed. The direct voltage is derived from three-phase ac power, most often from distribution voltages.

There are specific advantages to using polyphase rectifier circuits for dc power. As the number of ac phases driving the rectifier is increased (say, above single-phase ac), the frequency of output ripple is increased, the interval between rectifier conduction is decreased, and the ripple magnitude in the dc voltage and current waves decreases.

Transformers are almost always used between the ac source and the rectifiers. The rectifier transformer performs one or more of the following functions:

- To transform the available ac supply voltage to a value needed for the desired dc voltage;
- To provide the number of phases required to obtain the desired waveshapes of dc voltage, dc current, and ac supply current;
- To isolate the dc circuit from the ac source; and
- To limit, through transformer impedance, damaging overcurrents that might flow during malfunctions.

It is important to note that the decrease in the rectifier-conduction interval also increases the required transformer rating. The transformer utilization factor can be defined as the ratio of dc power delivered to the required transformer secondary voltampere rating. The utilization factor has been found to have a maximum value of 0.520 when three-phase ac input is used. This implies that from a transformer utilization standpoint, the most economic rectifier-conduction angle is 120°.

When power rectifiers are mentioned today, the reference is almost invariably to solid-state units using silicon rectifiers as the rectifying elements. Indeed, the silicon rectifier is virtually the only type considered for mine power installations. While there are many possible rectifier circuits, only two or three types are found in mining equipment. Circuits for silicon rectifiers are selected to make the most efficient use of the transformer, and the results usually are the single-phase full-wave bridge or the three-phase full-wave bridge. The next section will discuss three fundamental three-phase rectifier circuits, and it will be apparent why the full-wave bridge is popular.

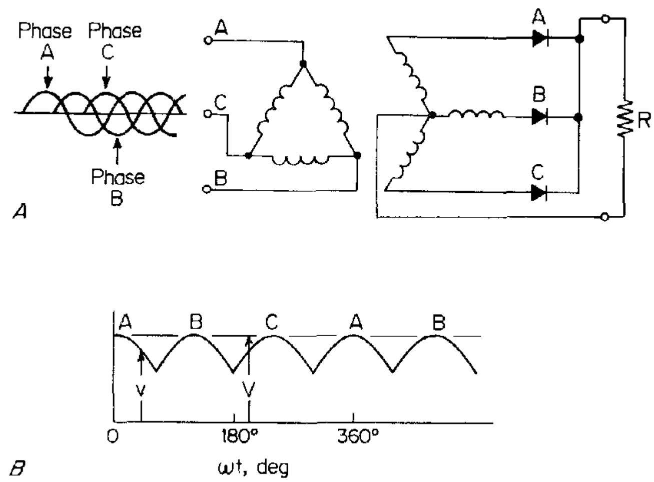


Figure 5.10.—Three-phase half-wave rectifier circuit (*A*) and output voltage waveform (*B*).

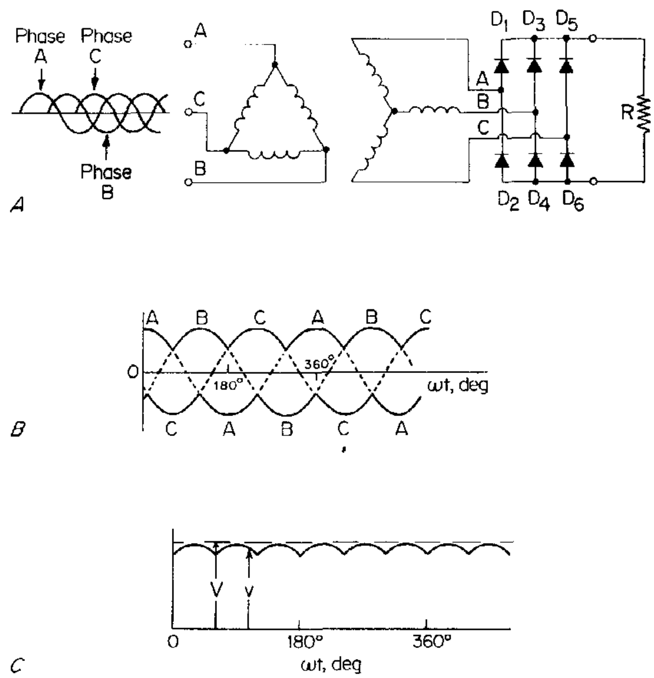


Figure 5.11.—Three-phase full-wave rectifier circuit (*A*) with input (*B*) and output (*C*) voltage waveforms.

Rectifier Circuits

Rectifier circuits can be classified as *single way* or *double way*. The phase currents of the transformer secondary (also termed the *dc winding*) are unidirectional in a single-way circuit but alternating in the double-way circuit.

The simplest three-phase rectifier circuit is the three-phase half-wave shown in figure 5.10*A*, where a delta-wye transformer is used, with each leg connected to a rectifier anode. The three rectifier cathodes are tied together to form the positive dc bus. The neutral point of the transformer winding serves as the negative connection for the load, in this case resistance, R. Being a single-way rectifier, each leg of the transformer secondary conducts current unidirectionally. If the load is pure resistance, the relationship of output voltage (that across the load) versus time is as shown in figure 5.10*B*. Each rectifier conducts for the cycle portion in which its anode has a higher positive value than the anodes of the other rectifiers. Therefore, each rectifier passes current for 120° of the input three-phase cycle. Since the current through the load is directly proportional to the output voltage, the load current has the same waveform as voltage.

Inspection of the three-phase half-wave output voltage shows that the ripple voltage is much lower than the single-phase full-wave rectifier circuit. Actually, the rms value of the ripple voltage waveform is only 18% of the average output voltage (this average voltage is the average dc load voltage, V_{dc}). If rectifier losses are ignored, since they are very small for silicon diodes, the dc output voltage and the transformer secondary voltage are related by

$$V_{dc} = 0.827V_{max} = 1.17V_{rms}, \tag{5.7}$$

where V_{dc} = average dc output voltage, V,

V_{max} = peak value of voltage applied to rectifier circuit, V,

and V_{rms} = rms value of voltage applied to rectifier circuit, V.

Both V_{max} and V_{rms} are line-to-neutral voltages. Note that the fundamental frequency is three times the ac line frequency. As a result, any filter components required to lower the ripple voltage further can be much smaller than in single-phase rectifiers.

The relationships presented here for the three-phase half-wave rectifier apply only to ideal transformers and rectifiers. In actual circuits, the voltage drop caused by dc current and the transformer secondary-winding resistance creates a dc component that pushes transformer magnetic operation toward saturation. Consequently, this simple three-phase rectifier circuit is seldom used.

Output ripple can be further reduced by a three-phase full-wave rectifier circuit, connected as shown in figure 5.11*A*. This circuit is also called the three-phase bridge or a six-phase rectifier. Being a two-way rectifier, the magnetic saturation problem in the transformer is not present. Furthermore, this configuration retains the advantage of 120° conduction for transformer economy, plus a fundamental ripple frequency of six times the ac source frequency. These characteristics make this double-way rectifier circuit of great practical value, and it is the most popular configuration for dc power in mining. The transformer dc winding may be either wye or delta connected.

In the full-wave rectifier circuit, each terminal of the transformer secondary is connected to two diodes, one at the anode and the other at the cathode. The cathodes of three rectifiers are common and form a positive dc voltage bus, while the common anode connection of the other three rectifiers represents the negative dc voltage bus. The load is connected between these two common points.

Each rectifier conducts for 120° of one input cycle, and current alternates in each transformer winding. However, current flows through a specific combination of rectifiers for only 60° of the input cycle. This combination could be D_1 and D_4 with transformer secondary terminals A and B. Therefore, the peak-to-peak voltage across the load resistance appears as six-phase ripple as shown in figure 5.11*C*.

Analysis of figure 5.11*C* shows that the rms value of the fundamental component of the ripple voltage is now only 4.2% of the average dc output voltage. In addition, the average dc output voltage for ideal rectifiers is

$$V_{dc} = 0.95V_{max} = 1.34V_{rms}, \qquad (5.8)$$

where V_{dc} = average dc output voltage, V,
V_{max} = peak line-to-line voltage applied to rectifiers, V,
and V_{rms} = rms value of line-to-line voltage applied to rectifiers, V.

The foregoing circuits are typical of most polyphase rectifier circuits, but many additional configurations are available. Because mining almost always employs full-wave rectifier circuits, coverage of more circuits is beyond the scope of this text, but the bibliography can be consulted if desired.

Parallel Rectifier Operation

The current requirements of a rectifier circuit are often too large to be handled by one rectifier for each circuit element. Two or more rectifiers must then be connected in parallel. Direct operation of two silicon rectifiers in parallel is very difficult, because unbalance between the parallel paths can be caused readily by unequal rectifier characteristics (mainly the forward voltage) and by unequal impedance in the bus bars or cables. The result is that the rectifier with the least forward voltage can be destroyed by overcurrent. To eliminate this problem, the parallel rectifiers must be forced into sharing the current equally.

The method used almost exclusively in mining equipment to force current-sharing employs paralleling reactors, sometimes called current-balancing transformers. Figure 5.12 shows how several rectifiers can be paralleled using these reactors. The combination acts as one rectifying element in a rectifier circuit such as in figure 5.11*A*. In figure 5.12, each reactor is a laminated magnetic core linked in opposing polarity by the anode currents of two rectifiers, and the cores are designed not to saturate at the highest expected current. If the two rectifier currents become unequal, the current difference excites a magnetic flux that induces an aiding voltage. This voltage is induced in the rectifier leads in a direction that will equalize the currents.

TRANSISTORS

The principal tool of the electronics industry is the amplifier, a device that can increase the power level of an input waveform or signal. An amplifier is actually an energy converter in which energy from a power supply is converted by the amplifier to signal energy. The most common device used in amplifiers is the transistor.

A bipolar transistor is formed in a manner similar to that of the junction diode, but it consists of two junctions in close proximity and parallel to each other in the same crystal. When a p-region is sandwiched between two n-regions, the device is termed an *n–p–n* transistor, the model and symbol of which are given in figure 5.13*A*. Similarly, if a thin portion of n-material is bounded by two p-regions, the transistor is termed *p–n–p*, as shown in figure 5.14*A*. As illustrated, each semiconductor region is given a name: *emitter*, *base*, and *collector*.

Transistor Operation

The operation of the transistor is dependent upon the bias voltages across the junctions. If voltages are applied to an n–p–n device as shown in figure 5.13*B*, the emitter-base junction is forward biased, and the collector-base

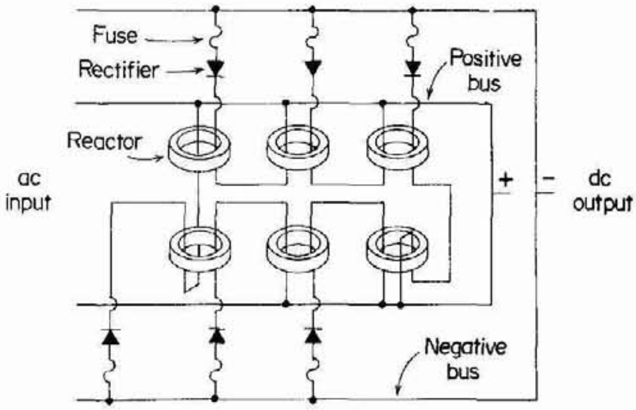


Figure 5.12.—Parallel operation of rectifiers using paralleling reactors.

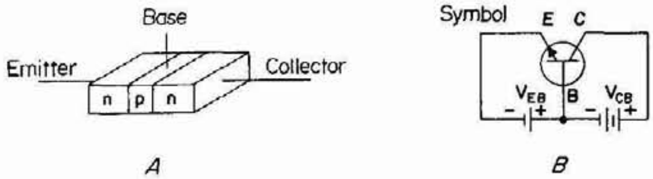


Figure 5.13.—An n-p-n junction transistor.

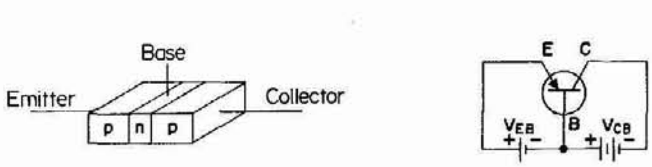


Figure 5.14.—A p-n-p junction transistor.

junction is reverse biased. These are the normal bias conditions. Electrons will flow into the base region, causing an excess of majority carriers there. Because the base region is thin and the potential existing across the two n-regions is much higher than the base-to-emitter potential, most electrons from the emitter region diffuse across the base and are accelerated into the collector region. The electrons drift across the collector and cause current flow in the collector circuit. However, a small percentage (typically, 5% or less) flows out from the base connection because of recombination with holes in the base region. This process can be considered amplification since the small base current controls the much larger collector current. A p–n–p transistor operates on the same principle, but here it is hole flow rather than electrons that causes the amplification. Consequently, the bias conditions are reversed from that for an n–p–n (the normal conditions are shown in figure 5.14*B*).

From the preceding discussion, it would appear that either end of the transistor could be called an emitter because either hole flow or electron flow creates the current amplification, but this is generally not the case. Heat dissipation is much larger in the collector-base junction because of the greater difference in potential. Therefore both p–n–p and n–p–n transistors are designed so this heat can be diffused through the collector region.

As might be expected, a saturation current resulting from thermally-generated minority carriers flows across the reverse-biased collector-base junction. In the diode, the current is designated "I_S;" in a transistor, it is termed "I_{CBO}." In the same manner as for diodes, the increase of saturation current with temperature sets the maximum operating temperature for a transistor. Heat sinks are commonly used in high-power transistor applications to diffuse collector-base junction heat and maintain temperature below critical levels. The same calculations that were presented in the preceding section on rectifiers can also be applied to transistors to determine a safe operating temperature.

The fraction of constant emitter current that reaches the collector is called alpha, α, and the collector circuit itself can be considered to be the output circuit. Since as much emitter current as possible should be collected, alpha should be as close to 1 as possible. When combined with I_{CBO}, the collector current, i_C, can be expressed in terms of emitter current, i_E, as

$$i_C = \alpha i_E + I_{CBO}. \tag{5.9}$$

Figure 5.15 shows the relationship of these currents. However, in practical applications, I_{CBO} is often so small that it can be neglected.

Since base current controls collector current, an important expression can be obtained from figure 5.15 using Kirchhoff's current law on either transistor:

$$i_B = i_E - i_C,$$

or

$$\begin{aligned} i_B &= i_E - (\alpha i_E + I_{CBO}) \\ &= (1 - \alpha) i_E - I_{CBO}. \end{aligned} \tag{5.10}$$

In terms of collector current, it can be shown that

$$i_C = \left(\frac{\alpha}{1-\alpha}\right) i_B + \frac{I_{CBO}}{1-\alpha}. \tag{5.11}$$

The term, $\alpha/(1 - \alpha)$, is called beta, β, and also the dc current amplification factor, and

$$i_C = \beta i_B + (1 + \beta) I_{CBO}. \tag{5.12}$$

This last equation shows the significant effect of temperature on transistor operation; that is, the temperature-sensitive I_{CBO} is multiplied by $(1 + \beta)$. Even though α is less than 1, β may range from 20 to 200 for amplifying transistors.

Bipolar-Transistor Amplifiers

Bipolar transistors can be operated with any one of the terminals common to the input and output, thus there are three basic circuit arrangements: common-base, common-emitter, and common-collector. The most popular is common-emitter.

Illustrated in figure 5.16, the common-base or grounded-base configuration employs the emitter and base terminals as input, with the collector and base terminals supplying output. Current gain, which is the ratio of output to input, is usually just less than 1. Because the emitter-base junction is forward biased, the circuit has low input impedance as viewed from the input terminals. Because the collector-base junction is reverse biased, the output impedance is high in comparison to the input. Hence, voltage and power amplification can be realized.

Two different circuits, signal and bias, are necessary for the operation of either of the two common-base amplifiers shown in figure 5.1. The bias voltage sources, often termed the amplifier power supply, fix the dc level for proper operation of the two junctions. If the signal input and output are not separated electrically from the bias source, as seen in figure 5.16*A*, the circuit is called a dc amplifier. Although it is beneficial in applications such as amplifying dc voltages for instrumentation, a signal with

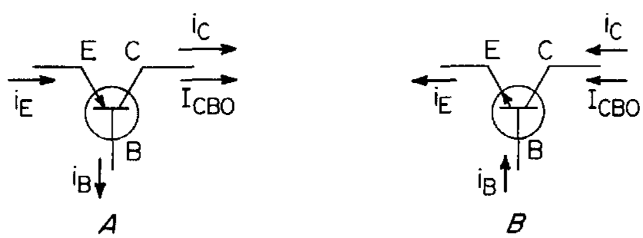


Figure 5.15.—Current relationships for p-n-p (*A*) and n-p-n (*B*) devices.

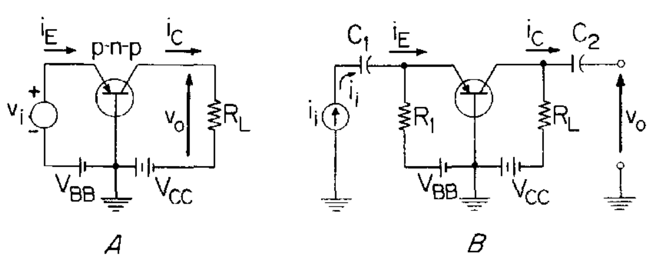


Figure 5.16.—Common-base amplifiers.

dc content or offset can interfere with correct transistor biasing. Figure 5.16*B* illustrates a popular method of removing this problem: the use of capacitors to isolate the amplifier. The capacitors exhibit high impedance to dc but low impedance to ac signals, thus they block input and output dc. As the circuit now reacts only to ac signals, it is called an ac amplifier. It can be noted that transformers can perform a similar function.

With either the dc amplifier or the ac amplifier, a small change in input voltage causes significant variation in the injection current across the emitter-base junction. As previously discussed, most majority carriers diffuse to the collector, causing collector current, i_C. If the load resistance, R_L, is small with respect to the transistor output impedance, i_C is approximately equal to i_E. The collector current creates voltage variations across the load resistance that can be much larger than the input voltage.

In the common-emitter transistor arrangement, the source signal only supplies current to the base. Because base current is much smaller than either the emitter or collector current, current amplification or gain, G_i, is high. Neglecting I_{CBO} in equation 5.12, the gain is approximately equal to

$$G_i = \frac{i_C}{i_B} = \frac{\beta i_B}{i_B} = \beta, \qquad (5.13)$$

which can be from 10 to several hundred. The input impedance is also higher than in common-base amplifiers.

Figure 5.17 shows a simple common-emitter amplifier. The control action of the base current can be demonstrated by assuming that the base-emitter forward bias is increased. This increase creates a corresponding increase in emitter-base junction current; thus, collector current is raised substantially. Because the base current is approximately proportional to but usually much less than collector current, base current is the controlling parameter of the amplifier.

The concept of characteristic curves has already been introduced in figure 5.3 in the section on diodes and rectifiers. Characteristic curves are an extremely useful tool for the graphical design and analysis of transistor circuits. Four independent transistor parameters control the number of necessary curves. When figure 5.17 is used, these parameters are as follows:

- α and β increase with V_{CE}, the collector-to-emitter voltage.
- i_B is dependent on i_C and V_{CE}.
- i_B is not a linear function of i_C.
- When V_{CE} is zero, i_C is approximately zero, regardless of i_B.

Consequently, two sets or families of curves are needed:

1. Collector or output characteristics, i_C versus v_{CE} for varying values of i_B, and
2. Common-emitter input characteristics, V_{BE} versus i_B for varying values of V_{CE}.

Figures 5.18*A* and 5.18*B* show typical output and input characteristics for an n–p–n transistor connected for common-emitter operation. The nonlinear and proportional properties of the four independent transistor parameters are evident in the graphs. These curves can be employed for design and analysis purposes. The analysis often uses a load line (the straight line in figure 5.18*A*) to observe dynamic variations of voltage and current.

The dashed line in figure 5.18*A* is very important as it delineates the safe operation boundary. Manufacturers specify maximum permissible collector voltage, current, and power dissipation, since outside this area damage to the transistor will probably result. As noted earlier, allowable power dissipation must be reduced as temperature is increased.

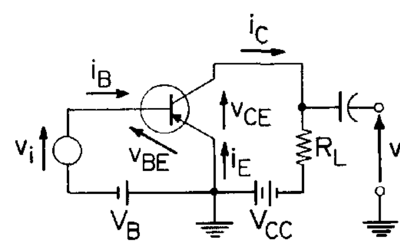


Figure 5.17.—Common-emitter amplifier.

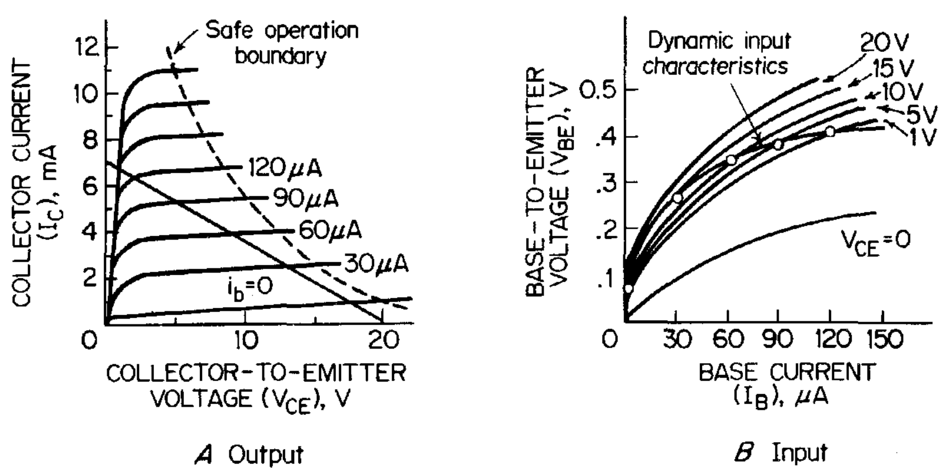


Figure 5.18.—Common-emitter characteristic curves.

Figure 5.17 illustrates an amplifier circuit with two batteries supplying dc for transistor bias, but single dc source for all bias voltages is more desirable in practical applications. Three bias techniques are frequently used for common-emitter amplifiers, and these are shown in figure 5.19. Each circuit uses resistors to supply dc bias to the base for a center bias condition about which the transistor operates. The center condition is termed the *quiescent point* of the amplifier. Of the circuits illustrated, the stabilized bias circuit (*C*) gives the best thermal stability, maintaining the quiescent point within a desired or specified range regardless of the normal operating temperature. The bypass capacitor, shown across the emitter resistor of the stabilized bias circuit, establishes a constant base bias bypassing or acting as a low impedance to time-varying voltages.

The two preceding amplifier configurations employed the collector circuit for output. In the common-collector arrangement, the output is obtained across a load resistance in the emitter circuit, as illustrated in figure 5.20. Because the source and output voltages are now in series but have opposing polarities, the circuit gives high input impedance and approximately unity voltage gain, yet current gain is about the same as in common-emitter amplifiers. A main advantage of the common-collector is that the output impedance is about equal to the load resistance, which is lower than the preceding two connections. This allows the circuit to be adjusted to fit the output needs precisely; hence, this circuit can be used for impedance matching the output of a source signal to the input of another amplifier.

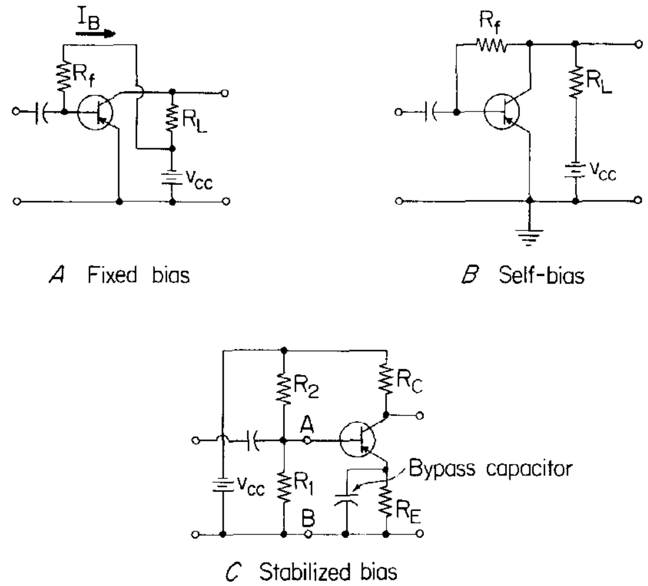


Figure 5.19.—Bias techniques for common-emitter amplifiers.

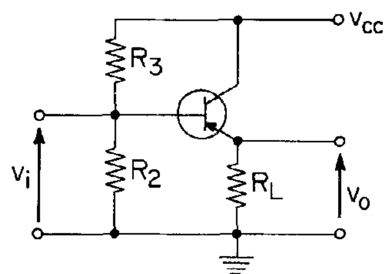


Figure 5.20.—Common-collector amplifier arrangement.

Field-Effect Transistors

The n–p–n and p–n–p junction transistors just covered contained two junctions. Field-effect transistors (FET's) have effectively only one junction but still can operate as amplifiers. These devices are voltage controlled, whereas bipolar transistors can be considered as current-controlled devices. There are two general classifications: junction FET's and metal oxide semiconductor FET's. Both have very high input impedances, much higher than bipolar transistors and approaching the input impedance of vacuum tubes.

To demonstrate the amplifying action available with FET's, consider the cross-sectional model of an n-channel junction FET, illustrated in figure 5.21*A*. The gate-to-channel junction is reverse biased by placing the voltage V_{GS} between the gate and source terminals as shown. The level of V_{GS} establishes a specific size of depletion region about the gate semiconductor and within the channel. Changing this reverse bias increases or decreases the size of the depletion region and decreases or increases the available conduction area remaining in the channel. Therefore, voltage changes between the gate and source terminals can control the allowable current through the channel from the drain to the source terminals. The action can be employed to amplify voltages or currents.

The conduction channel in the junction can be either n-type or p-type semiconductor, with the gate being p- or n-material, respectively. Figures 5.21*B* and 5.21*C* give the symbols for either junction FET type. An important advantage of FET's over junction transistors is that the source-to-drain channel is resistive without a diode effect. In essence, this allows FET's to be operated as electrically controlled resistors.

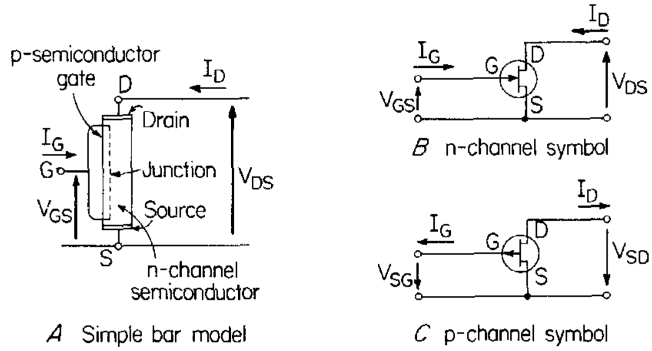


Figure 5.21.—Model and symbols for junction FET's.

As an application example, figure 5.22 shows a junction FET used in a typical amplifier circuit. The input signal is applied across the gate to the source, with output taken from drain to source. R_S is employed to set the proper dc quiescent point bias for the gate, and the capacitor in the source circuit bypasses ac, thus maintaining the bias level.

In metal oxide semiconductor FET's (or MOS-FET's), the depletion region used in the junction FET is replaced by a thick layer of silicon oxide, a good insulator, and the semiconductor employed for the gate is replaced by a metal conductor, thus forming a high-quality capacitor. A model of a MOS-FET, including the symbols, is given in figure 5.23. The operation of these transistors is similar to that of junction FET's but much more complex.

The preceding information on transistors is intended as just an introduction to a few important devices. For complete information, the bibliography must be consulted. The coverage here is justified because transistors are an extremely important, but often hidden, segment of mine power systems. The next section will cover another device that has revolutionized the control of electrical machinery.

SILICON-CONTROLLED RECTIFIERS

In past few years, the use of solid-state power equipment in mining has accelerated. One primary reason has been the introduction and acceptance of static or solid-state starting of conveyor-belt drive motors. The heart of these starters is the silicon-controlled rectifier or SCR. SCR's have many other applications; among these, the most common is in dimmers for home lighting.

SCR's, also called *thyristors*, are three-terminal semiconductor devices having a four-layer p-n-p-n material combination. Figure 5.24*A* shows a model of the SCR construction. The outer two layers act as a p-n junction and the inner layers serve as an element to control that junction. The symbol for the SCR is given in figure 5.24*B*, and figure 5.25 illustrates how the operation of the three-junction combination can be equated to two transistors connected as shown.

The equivalent circuit is represented by one n-p-n and one p-n-p transistor. When the bias on the gate, the n-p-n transistor base, is negative with respect to the cathode, the n-p-n transistor cannot conduct appreciable current. In other words, it is cut off. As no n-p-n transistor collector current can flow, the p-n-p transistor is also cut off. There is high impedance between the anode and cathode for this bias condition, and the SCR operating condition is called *OFF*. However, if the gate bias is made positive so that the n-p-n transistor conducts, current will flow into the n-p-n collector from the p-n-p transistor base. This p-n-p base current in turn causes collector current in the p-n-p transistor. The action between the two transistors has a positive feedback effect because an increase in current in one transistor creates an increase in the other. Therefore, once conduction in the SCR is established, the gate no longer has any controlling effect, and

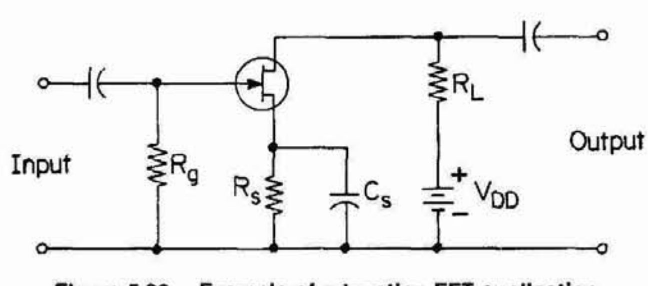


Figure 5.22.—Example of a junction-FET application.

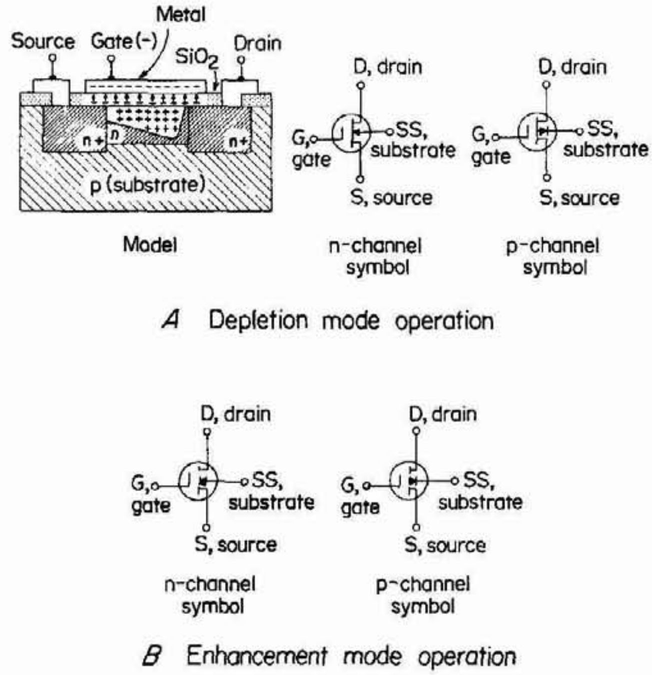


Figure 5.23.—Model and symbols for MOS-FET devices.

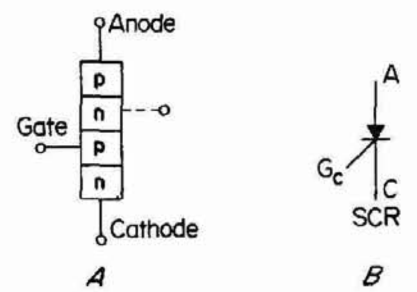


Figure 5.24.—SCR model (*A*) and symbol (*B*).

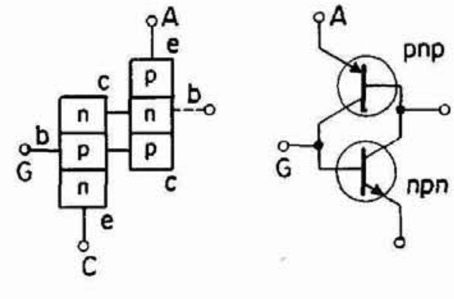


Figure 5.25.—SCR equivalent model and circuit.

the SCR is latched *ON*; that is, anode-to-cathode impedance becomes very low. The gate cannot turn the SCR conduction *OFF*. Cessation of current requires a negative gate bias and an essentially zero anode-to-cathode voltage. This allows the p–n–p transistor to cut off. The *OFF* and *ON* characteristics are apparent in the typical curve provided in figure 5.26. The breakover voltage noted here is the anode-to-cathode potential at which the SCR will turn itself *ON*.

There are many applications for the SCR or thyristor. Some of the devices and system components that the thyristor replaces include

- Thyratrons,
- Mercury-arc rectifiers,
- Saturable-core reactors,
- Relays and contactors,
- Rheostats and motor starters,
- Constant-voltage transformers,
- Autotransformers, and
- Mechanical speed changers.

Thyristor applications are a major subject in chapter 14.

INTEGRATED CIRCUITS

The semiconductor devices discussed so far are termed *discrete components* if they are manufactured as single units, for example, one diode or one transistor. They must be combined with other electrical and electronic components to perform any required function. Manufacturing processes have been refined so that several transistors, diodes, and resistors can be made in a single circuit, or in other words on one single semiconductor chip. Such devices are termed *integrated circuits* (IC's), and their study in electrical engineering is known as microelectronics. Today, many circuits requiring numerous individual transistors, such a complete amplifiers and digital computers, are packaged in a single semiconductor chip or microcircuit. When employing only one semiconductor chip, the IC is called monolithic; when the unit is created by interconnecting more than one microcircuit, the device is a hybrid IC.

The structure illustrated in figure 5.27 represents the cross section of a simple monolithic IC. The device is fabricated on a chip of p-type semiconductor, termed a *substrate*, by forming a number of junctions. The three sections shown are electrically isolated by reverse-biased p–n junctions, and the silicon surface is protected by a silicon oxide layer. A thin film of metal is deposited on top of this layer to interconnect the different regions. The top view of an actual IC is provided in figure 5.28. These devices can contain hundreds of transistors but can be small enough to pass through the eye of a needle.

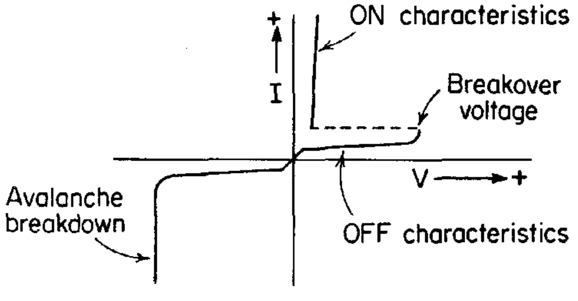


Figure 5.26.—General characteristic curve for SCR.

Except for high-power applications, IC's are preferred over discrete-component assemblies because they add reliability to equipment while reducing both size and cost. Consequently, IC's are employed where specific circuits require many transistors, diodes, and resistors. In circuit diagrams, it is accepted practice to show only the symbol for the specific application; some of these are given in figure 5.29. The use of IC's is extremely widespread in recently manufactured mining equipment, especially in control, monitoring, and communications applications.

BASIC INSTRUMENTATION

Much has been said in the preceding chapters about electrical parameters and their quantification: voltage, current, power factor, power, and so on. Instruments that measure these quantities are necessary to monitor and troubleshoot the operation of a power system and can be used to ensure optimum operation and to find malfunctions. The devices can be indicating instruments or recording instruments that are permanently installed in major

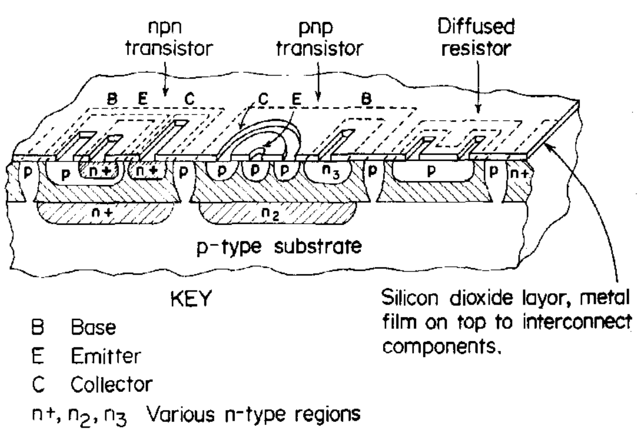


Figure 5.27.—Sketch of simple monolithic IC cross section.

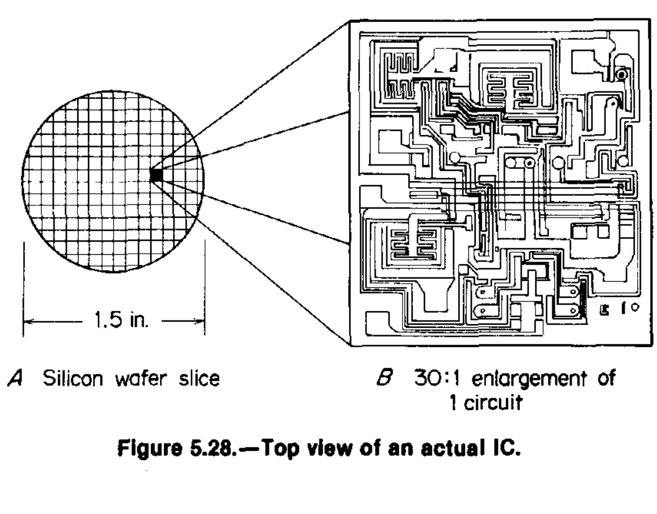


Figure 5.28.—Top view of an actual IC.

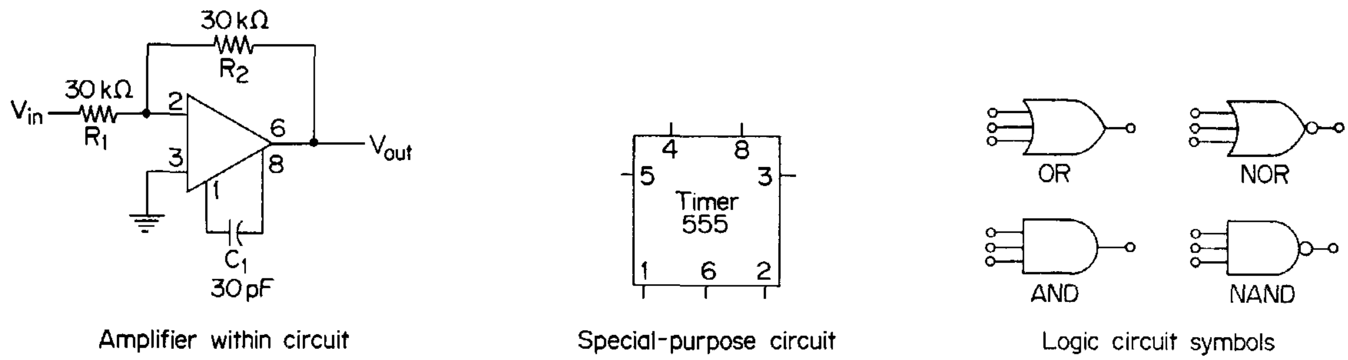


Figure 5.29.—Examples of symbols employed for IC's.

equipment or they can be self-contained and portable. It is not unusual for every piece of power equipment in or about the mine to have some form of enclosed instrumentation. The devices can range from basic meter movements to transducers connected to on-line computers that monitor the status of the entire power-system complex.

The word "meter" is often used as a suffix or part of a compound word that describes the function of the instrument. Of all the instruments designed to measure electrical quantities, the voltmeter and ammeter are the most basic. Voltmeters measure the potential difference or voltage between two points and must present a very high impedance to the circuit so as not to interfere with normal circuit operation. Ammeters measure current flow and must have a near-zero impedance. The dc voltmeters and ammeters sense average quantities, while their ac counterparts usually provide rms voltage and current values. Instrument current inputs are normally at 5 A, with potential inputs at 120 V.

The following section will explore the various instruments available to the mining industry, commencing with a description of the basic instrument or meter types and then showing how the devices are employed to monitor system quantities.

BASIC METER MOVEMENTS

A meter movement is an electromechanical device that provides the mechanical motion to an indicator in response to an applied electrical signal. Regardless of the type of meter movement, opposing magnetic fields are employed to activate the indicator or pointer. These movements can be classified as *electrostatic*, *dynamometer*, *moving iron vane*, and *permanent-magnet moving coil*.

An electrostatic movement is the only type that measures voltage directly as opposed to a voltage-produced current. This meter is basically a variable capacitor with a restoring resistor connected between a fixed and a movable plate or vane. When a difference in potential exists between the plates, the opposing charges produce a mutual attraction and the movable vane will move toward the fixed vane with the deflection proportional to the applied voltage. Upon removal or change in potential, the resistor discharges the capacitance. Thus any current through the movement is merely incidental to the operation. Electrostatic instruments can measure either ac or dc potentials; they have true rms response to ac regardless of waveform shape. Full-scale readings (maximum meter deflection) range from 100 V to 10 kV depending on the movement, with a measurement precision of 0.5% to 2%.

A dynamometer movement consist of two coils, one fixed and the other movable. The movable coil rotates in the magnetic field produced by current through the stationary coil. If the current being measured flows through both coils, (that is, they are in series), the resulting torque is proportional to the current, and the displacement is proportional to the square of current. Thus the pointer deflection indicates the rms value of current. The movement can be designed to measure dc or ac very precisely to within 0.1%. However, the dynamometer is not commonly employed as an ammeter. Its prime application is as a wattmeter, which will be described shortly.

Moving-iron-vane movements are similar to the dynamometer, except the moving coil is replaced by a soft iron vane with no permanent magnetization. Here, current through the fixed coil produces a magnetic field that induces magnetism in the soft iron vane. The magnetic fields oppose each other, producing torque that deflects the vane with a force proportional to the square of the current. The instrument can therefore measure dc or the rms value of ac, but with less precision (1% to 2%) than the dynamometer.

The last basic type of meter movement is the permanent-magnet moving-coil or d'Arsonval meter, which is a dc ammeter. The moving element is a coil of fine wire suspended so that it is free to rotate in the field of a permanent magnet. Sketches of typical movements are provided in figure 5.30. When dc flows in the coil, a torque is produced that tends to rotate the coil. The rotation is opposed by some form of spring restraint, usually a helical spring, so that coil motion and thus pointer position is proportional to the coil current. If the dc through the coil is varying so fast that the pointer cannot follow the fluctuations, the pointer will assume a position relative to the average torque, and therefore indicate the average value of current. However, if the current is a sinusoid, the average of moving-coil torque is zero, and the pointer will not be deflected. Nevertheless, d'Arsonval movements can obtain a precision of 0.1%.

For measuring current, both dynamometer and moving-iron-vane movements are often restricted to frequencies less than 200 Hz. Yet both these yield true rms readings within their frequency range. Electrostatic instruments can be extremely precise for observing voltage, but they are often very delicate and are applicable only for laboratory use. Even though d'Arsonval movements measure only dc, they are the most common type in use for both direct dc measurements and ac measurements using rectification.

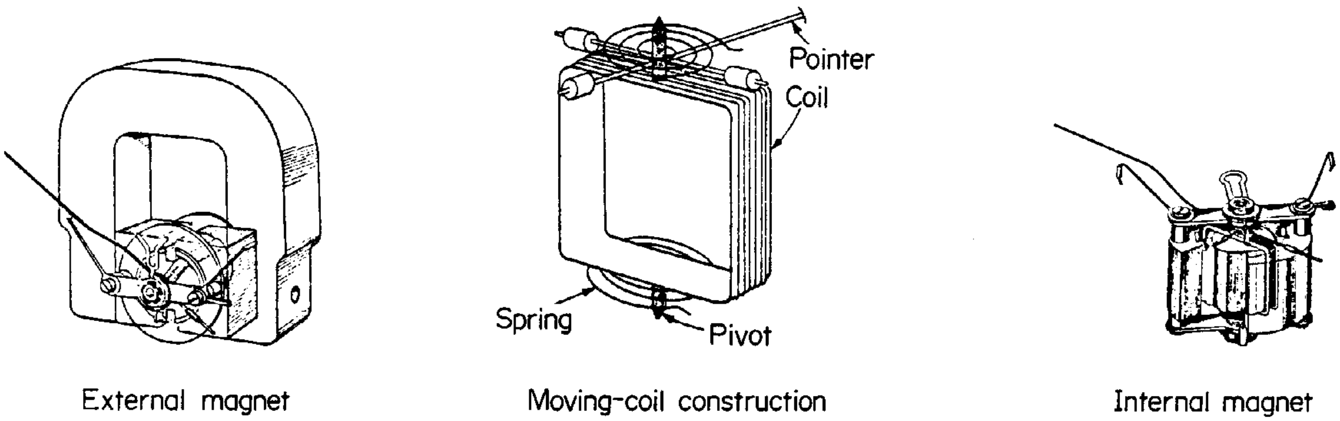


Figure 5.30.—Permanent-magnet moving-coil movements.

Meter-Movement Applications

When a d'Arsonval meter is used as an ammeter, it is inserted in series with the circuit being measured. The current range for this direct application is obviously restricted by the maximum scale reading or maximum current of the movement. D'Arsonval meters can have full-scale limits from 1.0 μA to 50 mA, although the basic movement is considered to be 1.0 mA, which allows measurement from zero to 1.0 mA. For higher current requirements, the meter is shunted with a low resistance as shown in figure 5.31. Such shunts can be tapped to provide several current ranges, or several shunts might be available, each selected by a switch to provide a specific current range. Commercially available ammeters of this type offer up to a 50-A full-scale reading.

To measure dc voltages, a d'Arsonval movement is simply placed in series with a selected high resistance, and the combination is connected between the two points where a voltage measurement is desired (fig. 5.32). Because meter deflection is still proportional to current, the meter scale can be calibrated to read the voltage required to produce a specific current. The sensitivity of such voltmeters is stated in ohms per volt. For instance, if a meter has a range of 0 to 200 μA and if the movement is to be used to measure 0 to 200 V, the total meter resistance must be

$$R = \frac{200\ \text{V}}{200\ \mu\text{A}} = 1.0\ \text{M}\Omega.$$

As moving-coil resistance, R_m, is generally on the order of 50 to 100 Ω, it can be neglected in this case. Sensitivity of the combination is therefore

$$\frac{1{,}000{,}000\ \Omega}{200\ \text{V}} = 5{,}000\ \Omega/\text{V}.$$

A higher value of sensitivity for a specific meter implies higher quality. Presently, the upper limit for the commercially available d'Arsonval voltmeter is 50 kΩ/V. The standard d'Arsonval movement of 0 to 1 mA has a coil resistance of 100 Ω; hence, it can be employed to read 0 to 100 mV directly.

External shunts are utilized for a desired maximum current when the current is higher than measurable by normal instruments with internal shunts. Figure 5.33 provides a couple of typical constructions where terminals are available for circuit as well as meter connections. These are simply standard resistance units, designed to be used with either 50-mV (0- to 50-mA) or 100-mV (0- to 100-mA) movements, in which a current through the shunts is indicated by a specific voltage drop across the shunt. For example, if a shunt is designated 100 mV, 600 A, a reading of 50 mV across the shunt signifies that 300 A is flowing in the circuit. Any time that metering or instrumentation is part of dc mine power equipment, it can almost be assumed that external shunts are involved.

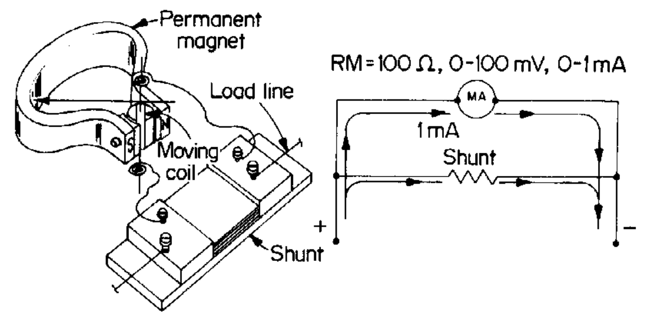


Figure 5.31.—Shunting d'Arsonval meter for high-current tests.

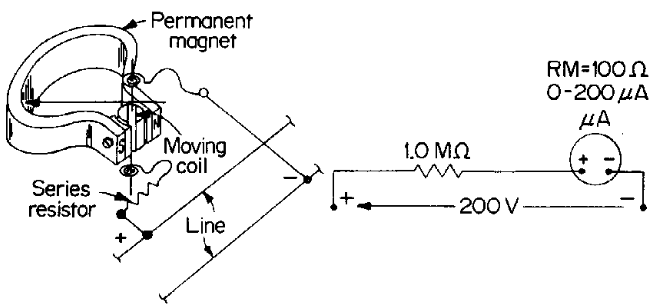


Figure 5.32.—D'Arsonval meter used to measure dc potentials.

To this point, only the measurement of circuit operation has been considered. A d'Arsonval meter can also be used to measure resistance by the addition of a dc source in the dc voltmeter circuit. Consider the circuit shown in figure 5.34, which has a dc movement in series with a dc source (usually a battery) and one or more resistors, one of which is usually variable to be used for calibration. The unknown resistance to be measured completes the loop. Meter deflection is still proportional to dc through the loop and is therefore a function of the unknown resistance. Using known resistances, the meter scale can be calibrated to read resistance directly, and different fixed resistors or multipliers can be used to extend the single scale. The combination is easily calibrated before each use by adjusting the pointer to zero using the variable resistance. The resistance desired could be a simple component or a complex circuit, but the ohmmeter should never be used on an energized circuit because of the internal source.

Combining the d'Arsonval movement with a half-wave or full-wave rectifier allows the reading of ac values in terms of dc through the coil. The full-wave or rectifier-ammeter circuit shown in figure 5.35 is the most common. Here, current through the movement is I_d, and thus, meter deflection is proportional to the average of I_d. This reading is the half-cycle average if the ac is symmetrical (that is, the dc scale of the meter will read the half-cycle average sinusoidal current). As the rms value of current is usually desired, the scale is calibrated in rms by multiplying the average current by 1.11. This is the rms value for a sinusoidal waveform only; for any other waveshape, relying on the rectifier circuit can produce large errors.

Moving-iron and dynamometer movements record rms current automatically, and many permanent meters built into power equipment to measure ac voltage and current are moving-iron types. However, the d'Arsonval meters are often preferred because of their greater sensitivity. For ac measurements of voltage or high current, the concepts of high series resistance and low parallel resistance also can be applied to the rectifier, moving-iron, and dynamometer movements, but such practices are not common except in small portable test equipment.

It can be seen in the foregoing that the d'Arsonval meter is used to measure ac or dc voltage or current as well as resistance. An instrument incorporating all these functions is called a multimeter. The selection of a specific parallel or series resistance combination provides the needed measurement function and parameter range.

Wattmeters

As mentioned earlier, the main application for dynamometer movements is in wattmeters. Figure 5.36 illustrates the wattmeter connection. Typically, the fixed coil carries circuit current while the moving coil is connected in series with a high resistance and is attached across the terminals of the circuit (the moving coil can itself be of high resistance). Circuit current flows through the fixed (or current) coil, and the current through the moving (or potential) coil is proportional to circuit voltage. Therefore, the movement torque is proportional to the product of instantaneous voltage and current, with the indication relative to the produce average or average power. The dynamometer connected as such will measure correctly the average power of a dc or ac circuit of any waveform, even when a power factor is involved.

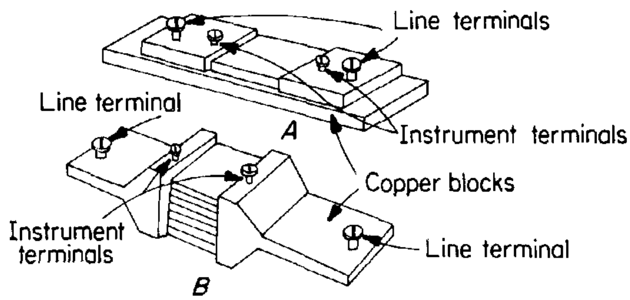


Figure 5.33.—External shunts used for high-current measurements.

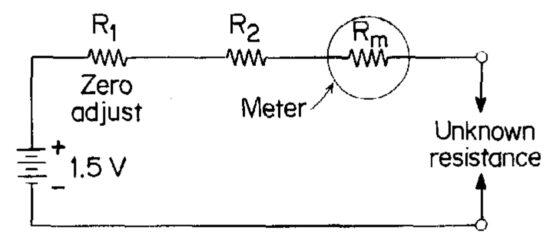


Figure 5.34.—Simple ohmmeter circuit.

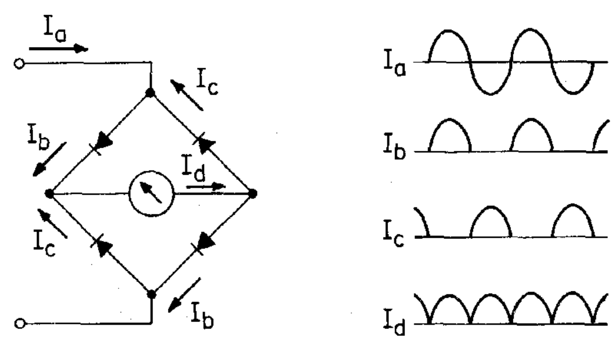


Figure 5.35.—Rectifier ammeter.

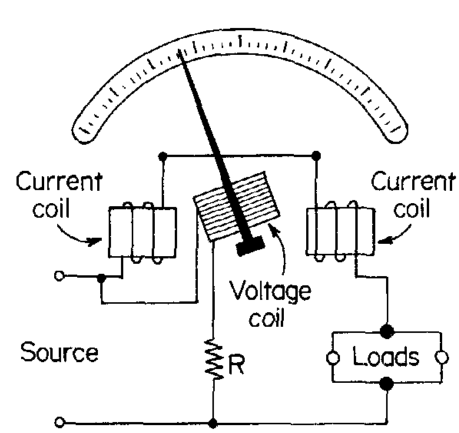


Figure 5.36.—Dynamometer connected as wattmeter.

Varmeters

In addition to being used for measuring watts, the dynamometer movement has wide application in measuring reactive power or vars. This is done in single-phase instruments by shifting the phase of the voltage coil by 90°. The voltage coil flux is then in phase with the flux produced by the reactive-current component in the current coil. Varmeters are installed in the same manner as wattmeters are.

Power-Factor Meters

A power-factor meter shows the power factor continuously and indicates whether the current is leading or lagging the voltage. The movement resembles a single-phase wattmeter but has no control spring and has two moving potential coils mounted on the same shaft 90° apart. One potential coil (B of figure 5.37) is in series with a noninductive resistor so that it produces torque proportional to the line voltage and in phase with the real component of line current. The other coil (coil A) is in series with a higher quality inductance, so its torque is proportional to the line-current reactive component. The fixed coil (coil C) is again the current coil. With unity power factor, the average torque between coils A and C is zero since the currents are 90° apart, but the currents through coils B and C are in phase, so the torque produced aligns their axes, and the pointer indicates unity power factor (1.0 pf). For leading or lagging power factors, the net torque created by currents in coils A, B, and C will swing the moving coils to the right or left, aligning the pointer in a position relative to the power factor. Meter scales are therefore calibrated so that the center position is unity power factor, and to the left and right of center are lagging and leading power factors from unity to zero.

This section has presented some direct applications for basic meter movements. Some concepts shown here apply to all electrical parameter measurements, but for ac power systems, additional components are normally employed.

POWER-SYSTEM INSTRUMENTATION

In chapter 3, the subject of current transformers (CT's) and potential transformers (PT's) was introduced. These devices actually fall under the general category of instrument transformers and serve two main functions:

- To isolate instruments, relays, and meters from line voltage, and
- To transform line currents and voltages into values suitable for measurement by standard instruments.

Thus, the normal ratings of instrument transformer secondaries are 5.0 A for CT's and 120 V for PT's. This measurement implies not only metering or actual visual readings but also sensing for such purposes as protective relaying. The following material will cover specifics of CT's and PT's as they apply to instrumentation of mine power systems. Chapter 9 will discuss the application of these transformers to protective relaying.

Instrument Transformers

Instrument transformers are connected in the power system in a manner related to the function they monitor. The primary winding of a CT is placed in series with the line conductor to be measured, or may be the line conductor itself, while a PT is placed across the line voltage to be measured (fig. 5.38). The transformers can then be used to extend the application of ac instruments in the same way that shunts and series resistors extend dc instrument usage. In this case, the ratio of a CT or PT is the ratio of primary current or voltage to secondary current or voltage under specified conditions. The secondary winding parameter is coordinated with the connected instrumentation.

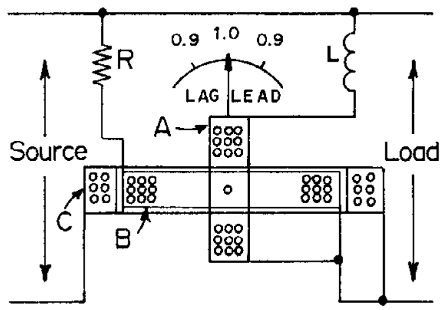


Figure 5.37.—Power-factor movement.

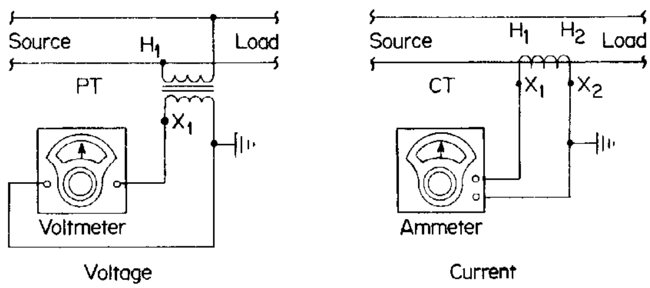


Figure 5.38.—Simple instrument-transformer connections.

To operate reliably, an instrument must receive information that accurately represents the conditions existing on the power system. When operated outside of the range for which they are intended, instrument transformers are very nonlinear devices; that is, the output from the transformer secondary can deviate from being an accurate representation of primary-winding conditions. The amount of deviation is a function of the transformer input level, secondary load, and design. To help with current application of instrument transformers so that they operate in their linear range, the American National Standards Institute (ANSI) has standardized transformer designs and secondary loads.[1] The designs are called accuracy classes, and the secondary load is called the transformer burden.

The effects of burden changes are typically more pronounced with CT's and PT's. Preferably, CT burden is expressed as a standard load impedance or its resistance and reactance components. In the past the practice was to specify the value as an apparent power (in voltamperes) at a power factor, the angle of which was referenced to a rated secondary current (for example, 0.9 pf of current lagging). Consequently, a CT burden of 0.5-Ω impedance could be

[1] Requirements for Instrument Transformers. C57.13 1968 et. seq.

expressed as 12.5 VA at 5 A, assuming the usual 5-A current. However, because of the nonlinear nature of transformers, burden impedance decreases as the secondary current increases, and a specific burden may apply only to one level of secondary current. As a result, the now nonstandard voltampere ratings are confusing. Furthermore, CT burden must be applied not only to the external load but also to all elements of that load, including the interconnecting loads. As the total burden needs to be calculated frequently, manufacturer publications usually provide the burdens of individual components. Potential transformer burden is normally stated as the total external voltampere load on the secondary at rated secondary voltage.

For the best accuracy with either PT's or CT's, the impedance of the burden should be identical to that of the instrumentation, and the accuracy limits stated by ANSI will then apply. The general rule for CT's is that if silicon steel is used for the core, the ampere turns should be at least 1,000 for good accuracy under normal conditions. When a PT has acceptable accuracy at its rated voltage, it can normally be used over a range from zero to 110% of rated voltage. Operation greater than 10% overvoltage can produce excessive errors.

Some special precautions are in order whenever current transformers are in use. A CT secondary should always be shorted or properly connected to the instrumentation (meters, relays, etc.), or dangerous potentials can occur at the secondary terminals and the core can become permanently magnetized. The flux density in the core is normally very low and can rise to saturation without a secondary current. The core can also become magnetized if dc is passed through the secondary. In either situation, the transformer ratio can be seriously changed. Furthermore, it is possible for a CT to be damaged through insulation breakdown associated with surges, overloads, and other occurrences. Therefore good practice dictates that tests be conducted prior to installation and periodically thereafter to verify transformer operation. If magnetization is suspected, the core can be demagnetized by passing rated 60-Hz current through the secondary with the primary open and gradually reducing the current to zero.

When a fault occurs on a line downstream from the CT coupling, the transformer primary current may reach several times the rated value for short periods of time. Two different techniques are available to protect against CT damage. One method is to overdesign the primary winding so that the transformer will not be damaged by the mechanical and thermal effects of moderate overload. The other design is perhaps more desirable. Here the CT is selected so that its core is close to the saturation point with normal operating primary current. When a surge current occurs, the secondary current cannot increase in proportion to the primary current and the burden is thus spared much of the shock. (See chapters 9 and 10 for further details as core saturation can seriously affect protective-relay operation.)

Single-Phase Connections

Figure 5.39 illustrates the measuring device connections needed for a single-phase circuit in order to observe voltage, current, and average power. This is a simple extension of figure 5.38. Only two instrument transformers are required: the PT drives the voltmeter and the wattmeter voltage coil, and the CT supplies current to the ammeter and the wattmeter current coil. For this arrangement, the ammeter and voltmeter would probably be moving-iron movements and the wattmeter would be a dynamometer. An alternative instrument arrangement is illustrated in figure 5.40. Here transducers are placed between the instrument transformers and the meter movements. Transducers are electronic components that present a standard burden to the transformer and provide an output compatible with the standard d'Arsonval movement. This is usually 0 to 1 mA, but 0 to 50 mV and 0 to 100 mV are also available. The transducer output is also adapted to a range of load impedances. With either arrangement, three ac power parameters can be measured and the power factor can also be calculated if desired. When any meter movement is employed, the normal reading of the meter should be one-half to three-quarters of the full-scale value in order to provide the best precision.

Note that in figures 5.39 and 5.40 the instrument transformer secondaries are grounded. The grounding is needed to prevent a high static potential, which can cause

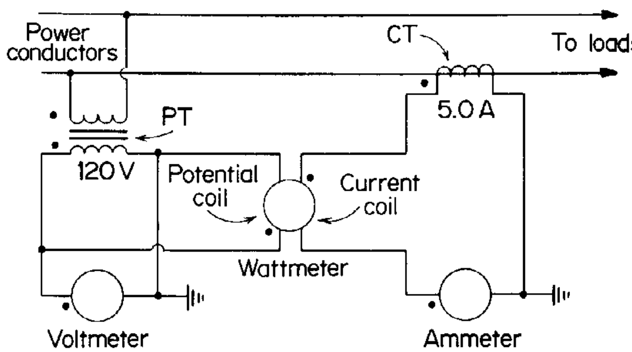


Figure 5.39.—Voltmeter, ammeter, and wattmeter arranged as single-phase system.

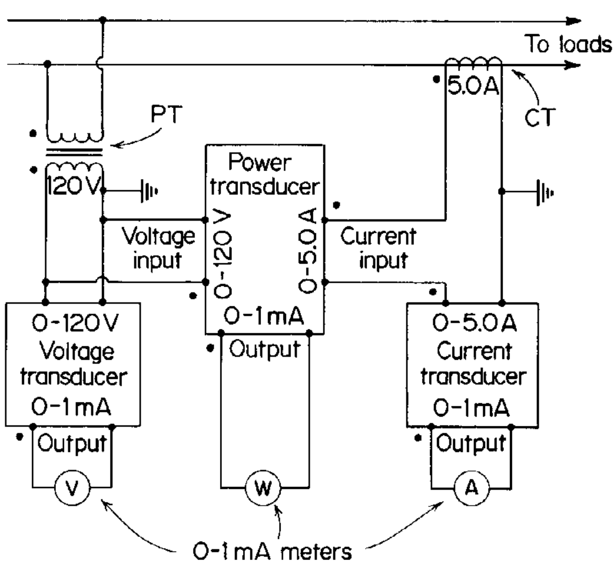


Figure 5.40.—Use of transducers with standard d'Arsonval movements.

a higher voltage than normal to appear on the secondaries. Without grounding, the transformer insulation could fail. The transformer case should also be grounded for the same safety reason.

Three-Phase Connections

When the measurement of average power in a three-phase system is required, it seems obvious to place one dynamometer wattmeter in each phase and add the results together. This is shown in figures 5.41*A* and 5.41*B* for a four-wire wye load and a three-wire wye or delta load. The sum of the meter readings is total power for either connection, for any waveform, and whether the system is balanced or not. The common connection of the three wattmeter potential coils may be placed at any potential without affecting the total power readings. If the potential is that of one phase conductor (see figure 5.42), one wattmeter becomes inoperative and thus may be omitted. The result is the two-wattmeter method of three-phase power measurements. Commercially available transducers can be used instead of the two wattmeters. The transducer inputs are two line-to-line voltages and two line currents, and the single output, which is proportional to total power as before, can be used with a standard d'Arsonval movement. A circuit arrangement for this method is shown in figure 5.43.

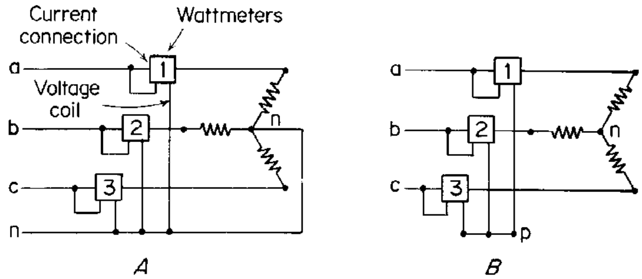


Figure 5.41.—Three-phase wattmeter connections.

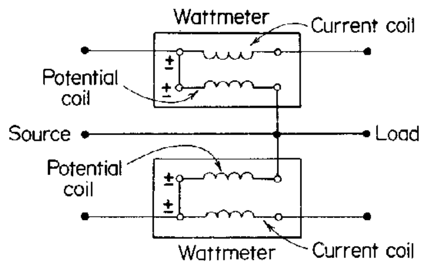


Figure 5.42.—Two-wattmeter method.

Under balanced conditions, the readings from the two-wattmeter method can be used not only for total power but also to determine the power-factor angle. It can be shown that

$$\tan\theta = \sqrt{3}\,\frac{P_2 - P_1}{P_2 + P_1}, \tag{5.14}$$

where P_1, P_2 = two power readings, corresponding to arrangement in figure 5.42,

and θ = load power-factor angle.

If P_1 represents a measurement of phase a current, equation 5.14 provides the correct sign for the power-factor angle, thereby specifying whether the load is capacitive or inductive. At times, phase sequence is hard to distinguish in practice, but the equation yields the angle magnitude and this is often sufficient information since the reactive characteristics of the load are usually known.

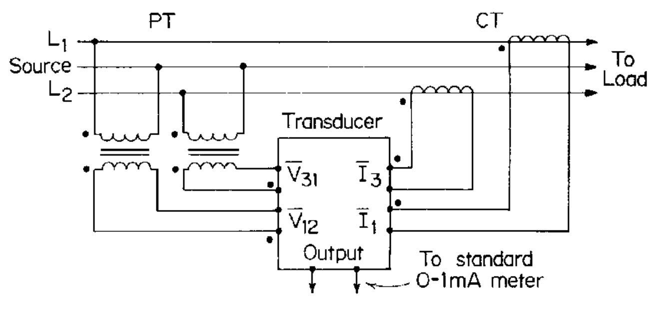


Figure 5.43.—Three-phase power measurement with transducer.

If the system is balanced or can be approximated as such, the circuit shown in figure 5.44 can be employed to measure the line-to-line voltage, line current, power factor, and total average power. The two-wattmeter approach calls for two PT's and two CT's. One PT supplies the voltmeter and one CT provides information to the ammeter, while the remaining PT and CT supply the power-factor meter so that the transformer burdens are balanced.

It is often useful to observe each line current or line-to-line voltage for major power equipment. Figure 5.45*A* provides an economical method for the line currents in which only two CT's are needed. If one CT secondary is measured, the current will correspond to the CT phase (that is, phase a or phase c), but if both CT secondaries are in parallel, the current reading is for the phase without the CT (that is, phase b). This metering is theoretically correct only for balanced voltages, but on most systems the voltage is close enough to balance that the two-CT approach gives acceptable precision. If greater accuracy is needed, three CT's should be used as shown in figure 5.45*B*. It is possible to connect the CT secondaries in delta or wye, but the burden impedances should always be wye connected. To observe all three line-to-line voltages, three potential transformers can be used as in figure 5.46*A*. The open-delta arrangement shown in figure 5.46*B* is not as accurate but gives satisfactory precision and uses only two PT's. For current or voltage with two or three instrument transformers, power-equipment metering is performed with a voltmeter or ammeter or both. The required phase is switch selected by connecting the transformer combination to the meter.

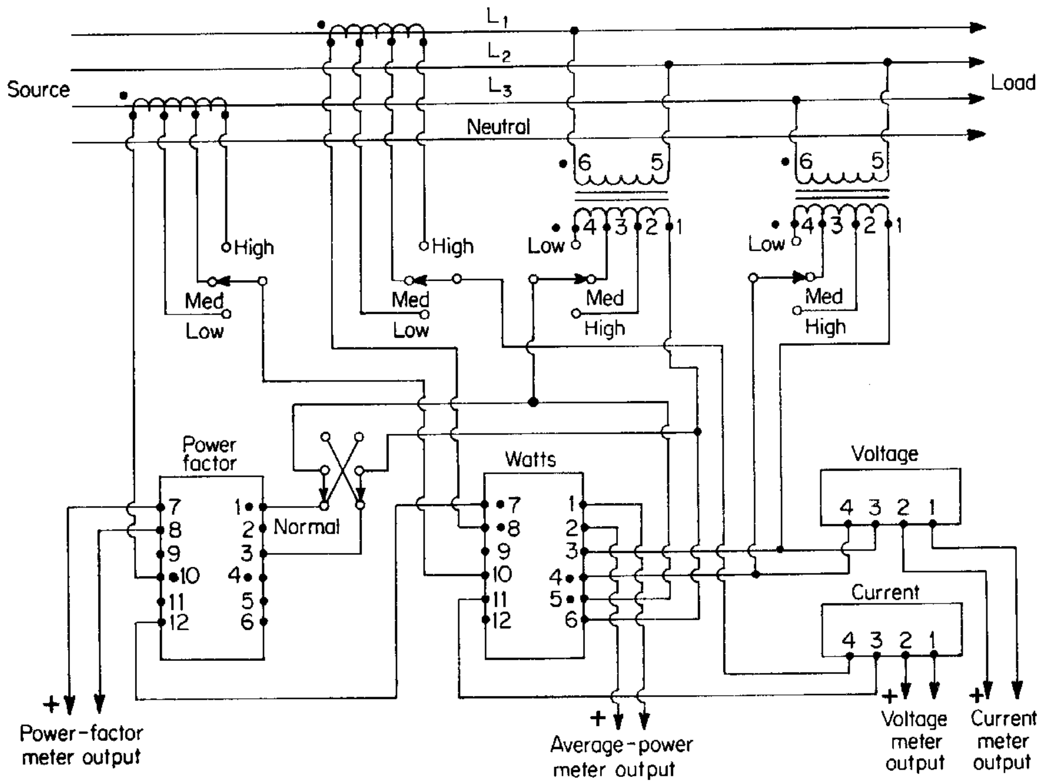


Figure 5.44.—Balanced three-phase measurement of voltage, current, and average power.

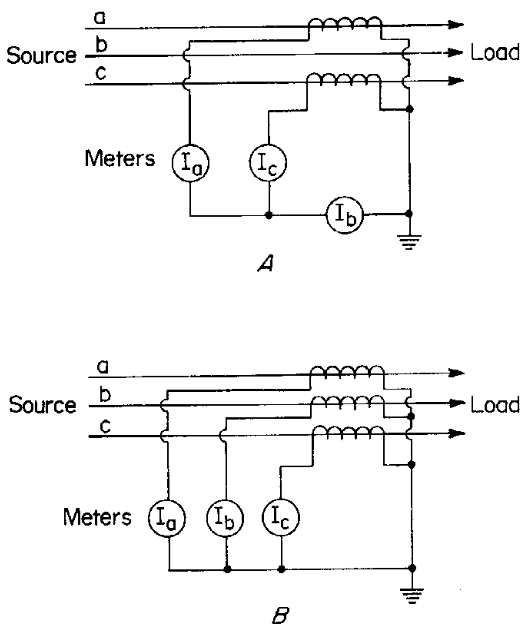


Figure 5.45.—Line current measurements with two or three CT's.

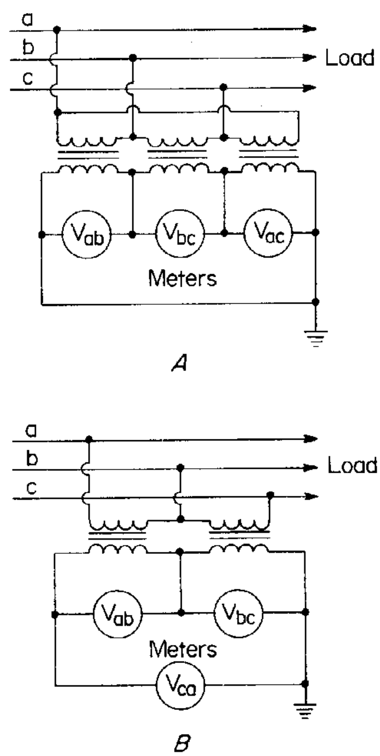


Figure 5.46.—Line-to-line voltage measurements with three or two PT's.

SPECIAL INSTRUMENTS

Several special, if not very common, instruments are available to perform measurements on specific electrical quantities. These include but are not limited to watthour meters, demand meters, bridges, megohmmeters, and phase-sequence indicators. Each of these is described in the following paragraphs.

Watthour Meters

The watthour meter is a common power instrument, used in nearly every building to measure consumed electrical energy. The typical watthour meter consists of a small induction motor with an aluminum disk that is rotated by a torque proportional to voltage times current at every instant. The principle of operation is similar to that of the dynamometer wattmeter, except the disk is allowed to turn continually with a speed proportional to average power. The number of turns is counted by a train of clocklike gears. The counter thus indicates the product of power and time, or energy, which is measured in kilowatthours. A simplified sketch of the induction mechanism is shown in figure 5.47.

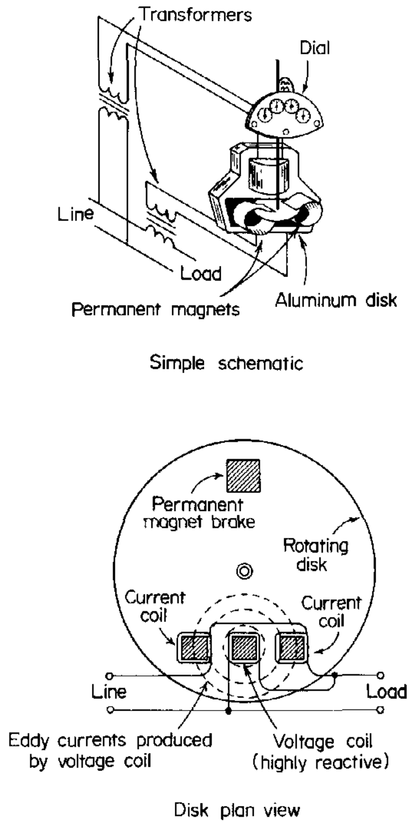


Figure 5.47.—Simplified sketch of watthour meter induction mechanism.

Demand Meters

Demand meters are usually of two types (although there are others): integrated demand or lagged demand. The readings may be indicating or recording. Integrated-demand meters consist of an integrating meter element, such as the watthour meter just described, that totals the energy used over the demand interval and drives a maximum indicating device, which can be a passive pointer, display, or chart. The meter can be reset manually, or a timing device can be used to return the drive to zero at the end of the recording period, thus leaving an indication of maximum demand. Lagged-demand meters provide a maximum demand indication that can be subjected to a characteristic time lag by either mechanical or thermal means, but usually the exponential heating curve of electrical equipment is followed. The demand interval is then defined as the time required to indicate 90% of the maximum value of a suddenly applied steady load; thus, maximum demand can be observed. Demand meters, whatever the type, can provide input to the power-system studies.

Bridges

Bridge circuits yield the most precise measurements of impedance, be it resistance, capacitance, or inductance, for two reasons: the measurements rely on null methods, and comparisons are made directly with standardized impedances that are precisely known. The term *null method* means that a zero reading or null indicates the correct value.

The Wheatstone bridge is the most widely used of these circuits. Shown in figure 5.48, the bridge is dedicated to measuring resistance, capacitance, or inductance depending on its internal components.

When the Wheatstone bridge is intended to measure resistance (figure 5.48*A*), the circuit consists of two fixed precision resistances, R_1 and R_3, which are known as the ratio arm; a variable precision resistance, R_2; and the

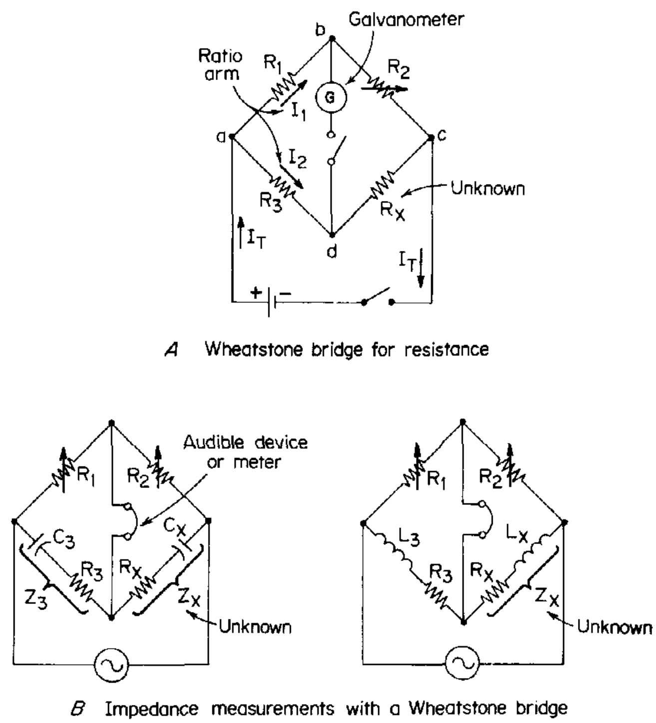


Figure 5.48.—Wheatstone bridge circuits.

unknown, R_x. A dc source supplies current to the arrangement, and a galvanometer, G, is located at the center of the bridge across points b and d. The galvanometer is simply a very sensitive ammeter with a center-scale zero-reading pointer and the ability to read very small currents in either direction. R_2 is adjusted to provide a null reading on the galvanometer, which means the potential between b and d must be zero. With this balanced condition, the unknown resistance can be calculated by

$$R_x = \frac{R_2}{R_1} R_3. \tag{5.15}$$

In commercially available bridges, R_1, R_2, and R_3 are all variable and the value of each is readily determined by calibrated dials. Thus, the bridge can measure resistances precisely over a broad range.

To measure impedance, R_3 of the resistance bridge is replaced by Z_3, and the unknown is now Z_x. An ac source is used, together with some means of measuring the potential between points b and d. This could be a sensitive ac ammeter or an audible device such as a set of headphones. R_1 and R_2 are then adjusted to provide a null, and the balanced condition means that

$$Z_x = Z_3 \frac{R_2}{R_1}. \tag{5.16}$$

Obviously, the values of Z_x and Z_3 depend upon the frequency of the ac source. The most typical value used is 1,000 Hz.

If very low resistances in the order of 10 $\mu\Omega$ to 1.0 mΩ must be measured, the Kelvin double bridge shown in figure 5.49 can be used. The circuit consists of ratio arms R_A, R_B and R_α, R_β; a connecting link or conductor, R_ℓ; a known resistance, R_s; the unknown, R_x; an adjustable dc source; and a null indicator. The indicator could again be a galvanometer. The resistances r_1, r_2, r_3, and r_4 are those of the connecting leads between the four-terminal bridge and the resistances to be compared (R_x and R_s). These lead resistances should be in the same ratio as the bridge arms to which they are connected; otherwise, the ratio unbalance will cause incorrect measurements. A small adjustable resistor can be used to balance the lead resistances. The balance equation is thus

$$\frac{R_X}{R_s} = \frac{R_A}{R_B} + \frac{R_\ell}{R_s}\left(\frac{R_\beta}{R_\alpha + R_\beta + R_\ell}\right)\left(\frac{R_A}{R_B} - \frac{R_x}{R_\beta}\right). \tag{5.17}$$

When R_x and R_s are so small that R_ℓ is comparable, the term in equation 5.17 involving R_ℓ can be significant. However, if

$$\frac{R_A}{R_B} = \frac{R_\alpha}{R_\beta},$$

then the R_ℓ term becomes zero. The source is adjustable so that current through R_x, R_ℓ, and R_s (the series resistance of which is small in comparison to the bridge) is large enough to allow a measurable milling current through the indicating device, G. An application for the Kelvin double bridge is in the measurement of cable and conductor resistances.

Megohmmeters

The preceding resistance-measuring devices can be ineffective when resistance is in the many millions of ohms. An important factor here is the resistance of insulation, such as that around conductors (fig. 5.50). One problem in these and other high-resistance measurements is to provide sufficient potential so the resulting current can be detected by an indicating device that provides resistance readings. The instrument designed to perform these tests is called a megohmmeter (fig. 5.51), where the unknown resistance is R_x, and R_1 and R_2 serve as current-limiting resistors to protect the meter from damage.

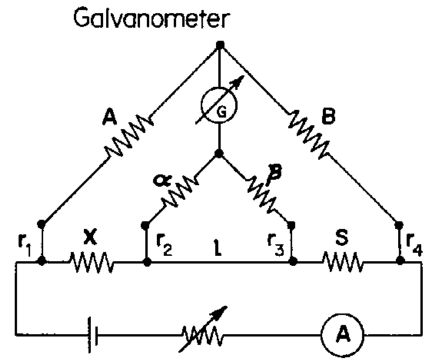


Figure 5.49.—Kelvin double bridge.

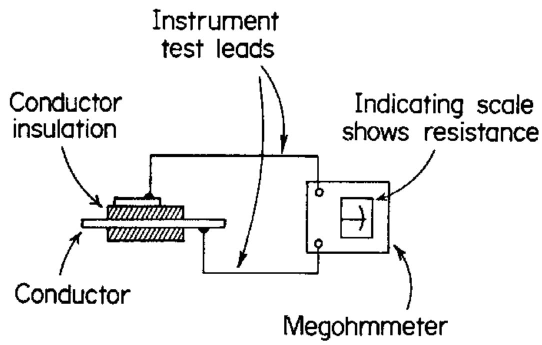


Figure 5.50.—Megohmmeter testing insulation resistance.

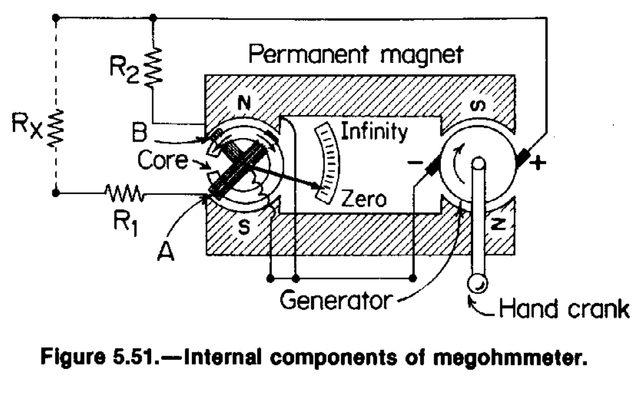


Figure 5.51.—Internal components of megohmmeter.

The most evident difference between the megohmmeter and the preceding instruments is the hand-driven generator, which supplies the needed dc potential for measurement. The generator applies from 500 to 2,500 V depending on the instrument and is tied to the resistance range desired (the higher the measured resistance, the higher the required voltage). Typically, a friction clutch is employed to restrict the generator to rated output voltage. In some megohmmeters, the potential is from batteries via an electronic power supply located within the instrument.

As shown in figure 5.51, the meter has two coils mounted over a gapped core. The movement is similar to the d'Arsonval, but there are no restraining springs, so the indicator is free to move when there is no output from the generator. If the instrument terminals are open (that is, R_x is infinite) when the generator is operated, current will flow through R_2 and coil A_1, and the torque produced will force the pointer counterclockwise to the infinite scale reading. When the terminals are shorted (R_x is zero), the torque produced by coil B is greater than that from coil A and this moves the pointer to a zero reading. For measuring an unknown resistance, the pointer location is dependent upon the opposing torque from the two coils, and the position is a function of R_x.

Another prime application for megohmmeters is the measurement of ground-bed resistances. These specialized testing procedures are covered in chapter 7.

Phase-Sequence Indicators

In order to prevent damage or incorrect operation, all conductors in a three-phase distribution system must be properly connected so they will provide the same phase sequence to all equipment. Correct interconnections can at times be difficult to accomplish in mine power systems, especially with feeder and trailing cables. At present there is no standard color coding for phase conductors. The phase-sequence indicator illustrated in figure 5.52 can be used to determine the phase relationship of energized three-phase conductors. It falls in the simplest class of testing devices: indicating instruments; other examples include a light bulb with leads to test for the presence of potential, or a battery in series with a light bulb with leads to check continuity by completing the series circuit. The phase-sequence indicator consists of two light bulbs and a capacitor connected in wye, and the lamps are labeled in the two possible phase combinations. Because of this arrangement, one lamp will burn brighter than the other depending on the connections to the power system.

The foregoing has provided information on several devices that are helpful in measuring mine electrical systems. Other instruments that are equally useful for specific applications include the split-core ac ammeter, a handheld ac ammeter that has its own CT; the synchroscope, which measures proper phase connections and the correct speed of parallel ac generators; and a frequency meter, which indicates the frequency of an electrical supply in hertz. Often there is also a need to obtain a continuous record of an electrical parameter, and the next section discusses the popular recording devices.

RECORDING INSTRUMENTS

Many of the direct-reading indicating instruments just presented are also available as recording instruments. Some of these are very similar to their indicating counterparts in that they can use the same electrical movements; they differ because the pointer is also used to provide a graphic record on a chart. These are termed chart recorders; one popular class is strip-chart recorders, so named because the electrical parameter is recorded on a strip of paper.

The similarity between the movement of the strip-chart recorder and the indicating instruments is illustrated in figure 5.53. The strip-chart recorder movement is actually a d'Arsonval type. The pen can trace on paper in several ways.

- *Inking.* The pen is a capillary tube through which ink flows from a well to the chart. This is perhaps the most used system.
- *Inkless.* The tip of the pen is a stylus that impacts the paper like a typewriter key with a regular force supplied by a cam, leaving a series of dots.
- *Thermal.* The pen tip contains a heating element that leaves a trace by heating specially treated paper.

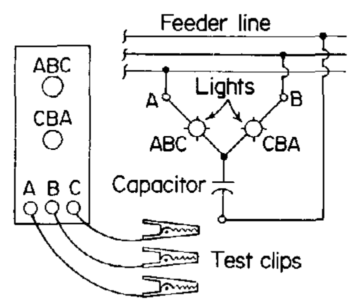


Figure 5.52.—Phase-sequence indicator.

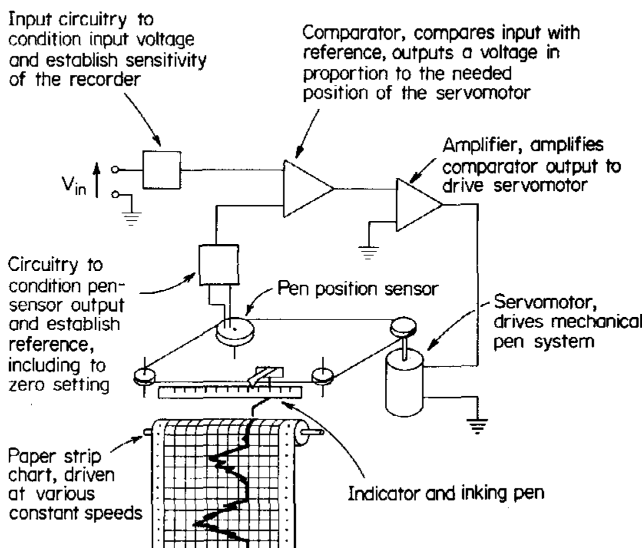


Figure 5.53.—Strip-chart recorder.

The simplest unit provides a curved recording as the pen swings in an arc, but articulated pen arms are also available that produce linear or rectilinear traces. The paper chart moves past the pen at a predetermined speed driven by an electric motor or a mechanical-spring clockwork mechanism. This recorder provides a continuous record of the average or rms value of the electrical parameter of interest, which is advantageous in obtaining records of equipment operation, for example, the electrical performance of a mining machine. A variation of these recorders uses a round chart, driven like a disk on a record player but at very slow speed. These charts can be built into major equipment to provide permanent records.

Sometimes recordings of the actual electrical waveforms are needed to study power systems. This calls for an instrument that can resolve instantaneous values of electrical parameters. Electromechanical instruments that have this resolution are called oscillographs, and the movement in most of these is a sensitive galvanometer of low mass. Two types of writing systems are normally available:

- *Direct writing.* This is similar to either the inking or thermal strip-chart recorder types. The pen has high inertia, and instrument response is about 0.5 to 100 Hz (some to dc).
- *Optical.* Instead of a pen, the movement drives a low-mass mirror that deflects a light beam that exposes a light-sensitive paper. Developing is required to obtain the record, but the system can have resolution to 10,000 Hz.

For many applications, magnetic tape recorders and oscilloscopes, both electronic instruments, find favor over oscillographs. However, oscillographs still have some practical use, especially where an extended-time hard copy is needed immediately. An example would be in measuring neutral currents existing on three-phase equipment, which can have dc as well as ac components.

ELECTRONIC INSTRUMENTS

The employment of complex and sophisticated control equipment in the mining industry is continuing to increase. Instances include solid-state motor starters, electronic protective relaying, computer logic circuits on mining machinery, and so on. These types of systems require precise voltage, current, and waveform measurements that are not possible with the preceding instruments. Certain phenomena existing on power systems, such as transients, require precise measurements with frequency response into the megahertz. Electronic measuring equipment answers this need. This section will introduce only the more popular instruments.

Electronic Meters

These instruments use many of the basic circuits that have been described for multimeters; that is, series resistances for voltage (fig. 5.54*A*), voltage-drop for resistance (fig. 5.54*B*), and shunts for current. The prime difference is that a scaled-down dc voltage, which is proportional to the actual circuit voltage, current, or resistance, is amplified by electronics. When the parameter is sinusoid, the ac is rectified before amplification. The amplified signal then drives the indicating device. In the past, vacuum tubes performed the amplification, termed a *vacuum-tube voltmeter* or VTVM; more recently, solid-state devices (IC's or FET's) have become the most popular.

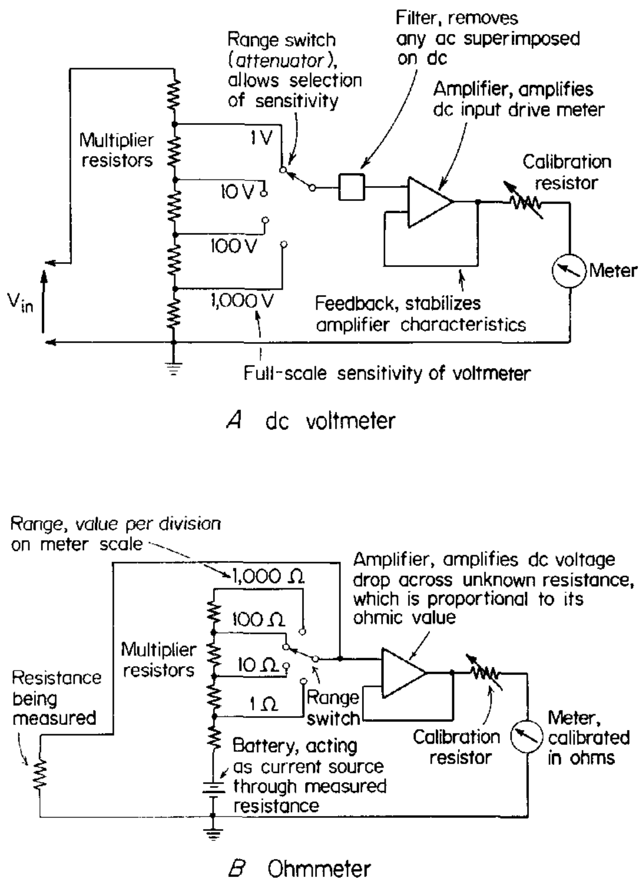


Figure 5.54.—Input circuits on electronic voltmeter.

The indicating device can be of two types: the familiar d'Arsonval movement or a digital indicator. The electromechanical displays or movements described thus far can be termed *analog*. The *digital* display is an indicating output assembly that takes the measurement results (voltage, current, average power, etc.) and through electronics gives a visual indication in a discrete number, as shown in figure 5.55. The actual display can be by Nixie tube (a gas-discharge tube), seven-segment incandescent filament, light-emitting diodes (LED), or liquid-crystal display (LCD). The electronics in the display assembly use logic or binary mathematics to convert the analog output of the measurements and drive the visual display. These digital displays are replacing their analog counterparts in many applications.

Electronic meters might appear rather complicated, but an important advantage is gained through the circuitry: the instrument can be made so that its interference with the circuit being measured is negligible. Typical input impedance of most electronic voltmeters is 11 MΩ.

Oscilloscopes

Oscilloscopes are electronic instruments that provide a real-time display of waveforms. They are available with responses from dc to hundreds of megahertz and thus can

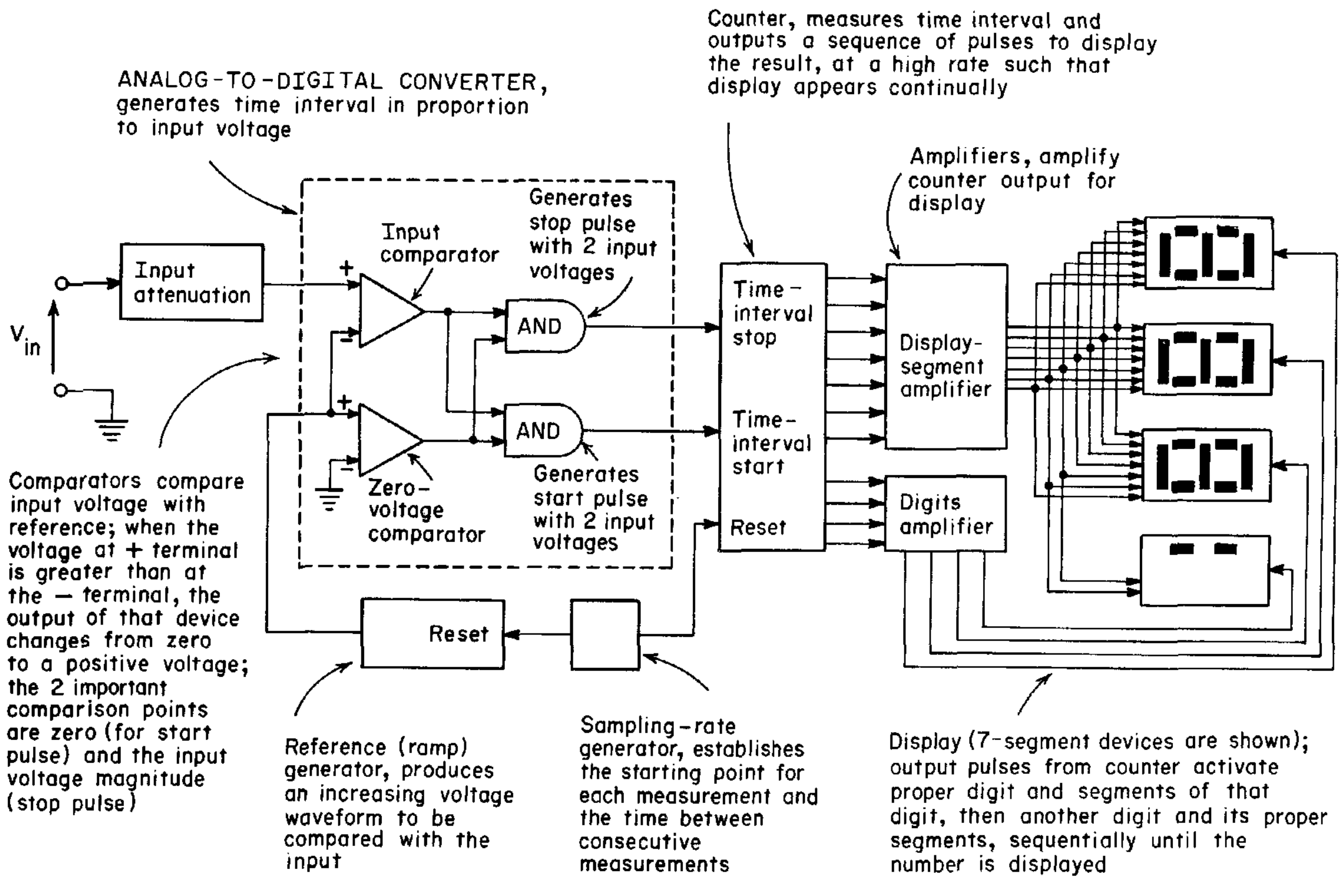


Figure 5.55.—Digital display.

be used to observe a large range of electrical phenomena including those of extremely short duration. The reason these instruments have such a broad frequency range is that they are not constrained by mechanical inertia. The heart of the oscilloscope is a cathode-ray tube or CRT (fig. 5.56). A fine beam of electrons is deflected by an electrostatic field in relationship to the voltage or current being investigated. The beam then impinges on a fluorescent screen to create a luminous display. The electrostatic field is normally established by two pairs of deflecting plates; one provides deflection vertically, the other horizontally. When a waveform is observed, the horizontal pair is driven electronically by a sweep signal to provide a time base, and the vertical pair creates a field in response to the instantaneous value of the electrical parameter. In some CRT's, additional pairs of vertical plates are available that enable more than one trace to be displayed on the screen. This allows direct comparison of two or more waveforms. A camera can be used in conjunction with the oscilloscope to provide a permanent record.

Tape Recorders

The familiar magnetic tape recorder records a signal by magnetizing a thin strip of tape. The nonmetallic tape is coated with a very thin layer of magnetic material such as iron oxide, thereby providing a relatively permanent record of a signal. The signal can be an analog recording, or digital, or in direct relationship to the measured parameter. With the digital recording, the signal is converted electronically to the binary system and the binary counterpart is recorded. The recording can be played back numerous times, and the output very closely matches the input that was observed. Analog tapes, which can have frequency ranges from dc to over 20,000 Hz, can be used as input to strip-chart recorders to provide hard copies or input to various electronic instruments that perform analysis of the electrical parameters. Digital recordings can be made compatible with digital computers for swift and elaborate analysis of the data.

Transducers

Transducers perform the transfer of information from the power system to the electronic instruments. A transducer can be described as a device that provides an electrical signal output in response to a specific measurement. Thus, potential transformers and series-dropping resistors can be considered voltage transducers, and current transformers and shunts, current transducers.

Another popular current-sensing device employs the Hall-effect principle. Hall-effect current transducers measure the effect of an electromagnetic field on a semiconductor. Basically, such devices operate on the interaction of magnetic force and the movement of charge through a semiconductor. Consider figure 5.57 in which a current, I_c, is flowing and a magnetic field is acting perpendicular to the current. The magnetic field will deflect the charge carriers in proportion to the field strength. This action produces a Hall-effect voltage, as shown, which is in

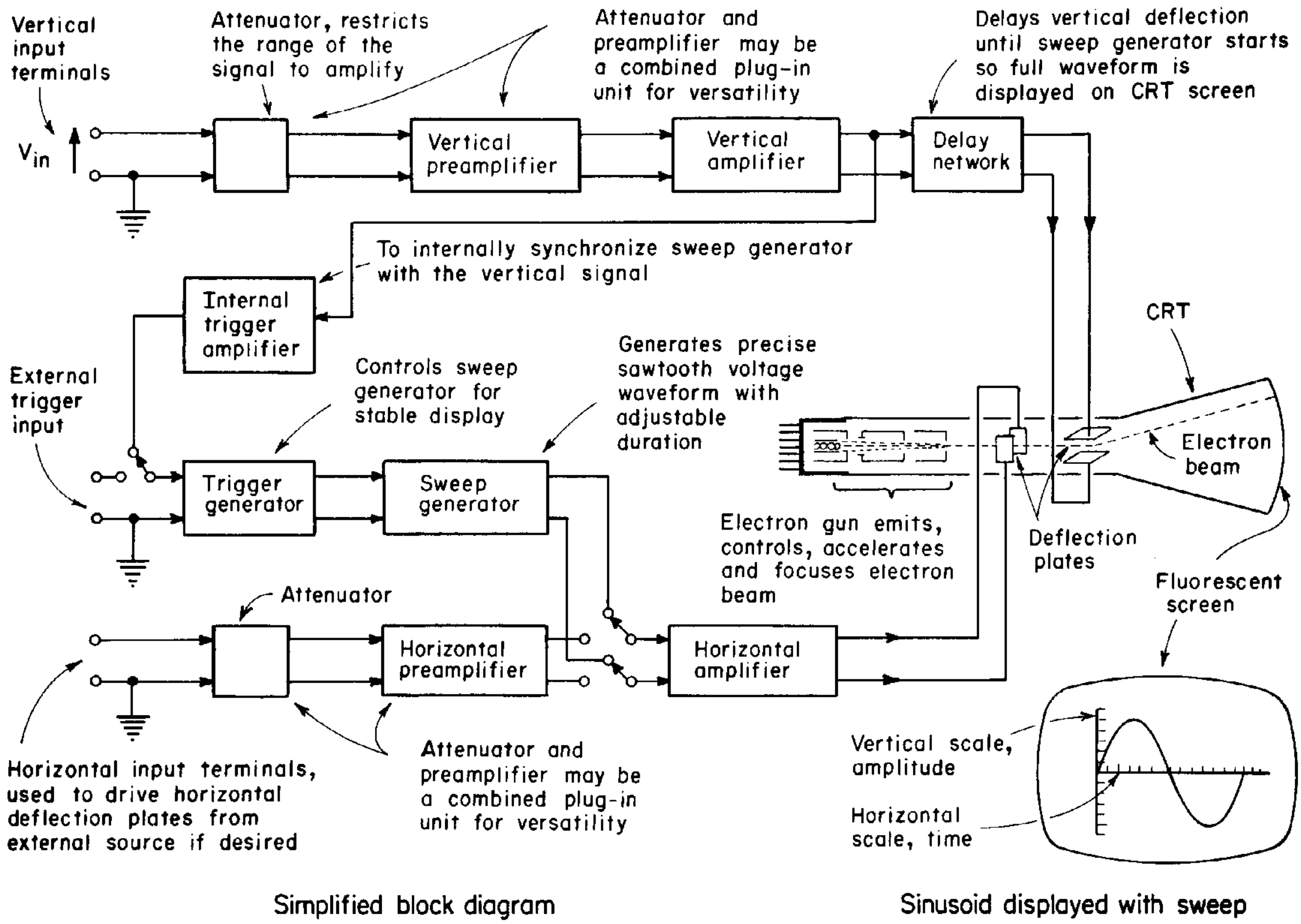


Figure 5.56.—Cathode-ray tube.

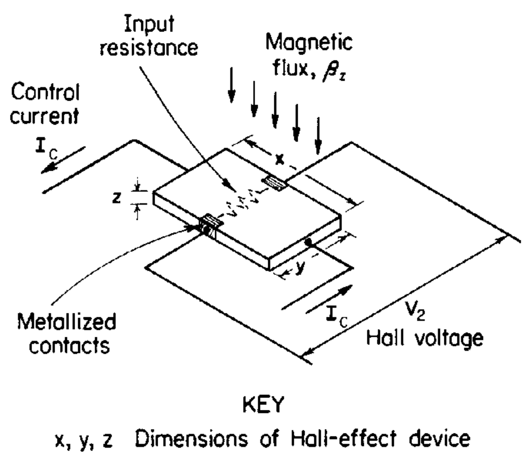


Figure 5.57.—Semiconductor illustrating Hall effect.

proportion to the magnetic field. Current flow in a conductor produces an electromagnetic field that can be measured by a Hall-effect device, thus producing a voltage output in proportion to the current. Most times, the magnetic field requires concentration. In some Hall-effect instruments, this is performed by a core of magnetic, low-retentivity material that can be clipped around a conductor. The semiconductor is mounted on the core and oriented at right angles to the induced magnetic field. The combined unit appears much like a split-core CT, and through this method dc as well as ac currents can be measured with high precision. Many instruments used for precise power-system measurements employ Hall-effect devices.

The output from a transducer is sometimes incompatible with the input of the instrument, or for safety reasons is not isolated from the power system. In these instances, the signal requires conditioning, and electronic circuitry, usually amplifiers, is called upon to perform the task.

INSTRUMENT INSTALLATIONS

It is common to find several instruments included as part of major power-equipment circuitry. As a summary to this chapter, the following describes the typical locations for measuring instruments within a power system.

1. The termination of utility transmission lines:
   Voltmeters,
   Ammeters,
   Wattmeters,
   Varmeters or power factor meters,
   Watthour meters,
   Demand meters, and
   Frequency meters.

2. Substation secondaries (outgoing distribution):
   Voltmeters,
   Ammeters,
   Wattmeters,
   Varmeters or power factor meters,
   Watthour meters (demand attachment optional), and
   Test blocks (or connection points) for portable instruments.
3. Switchhouses, load centers, and rectifiers:
   Voltmeters,
   Ammeters, and
   Test blocks for portable instruments.
4. Machinery:
   Voltmeters (optional),
   Ammeters,
   Elapsed-time meters (optional), and
   Watthour meters (also optional).

Two points must be considered when applying the above listing. At the higher transmission voltages, say 69 kV and up, it is sometimes advantageous to have the utility metering point at the substation transformer secondary. This would eliminate item 1 and might add additional instruments to item 2 for reasons of economics. The capital required for high-voltage metering can prohibit its use. The test blocks listed for portable instruments are necessary because they provide an easy avenue for maintenance and trouble shooting. In general, the test blocks consist of a series of terminals to which permanent connections are made to important circuit portions, such as major components. The block is located on the surface of the equipment, accessible only to maintenance personnel, and can eliminate the need for some permanent instruments. However, as was mentioned at the beginning, voltmeters and ammeters are considered to be the minimum permanent instrumentation within mine power equipment.

CHAPTER 6.—MOTORS AND MOTOR CONTROL

The subject of this chapter is the electromechanical conversion equipment that links electrical and mechanical systems and makes it possible to convert from one energy form to the other. The primary electromechanical devices are generators and motors. In generators, mechanical power is used to generate electrical power. Electric motors can be viewed as generators in reverse; they convert electrical power into mechanical power. The word "motor" can be applied to a device that converts energy of any form into mechanical power, but for purposes of this chapter the term is restricted to those machines that receive electrical energy.

Generators have limited but important applications in most mining operations. The principal functions are in motor-generator (m–g) sets for surface excavating machinery and mine hoists, and for providing emergency power to ventilation fans and hoisting equipment. Motors, on the other hand, are used so extensively that they are the most important mechanical source in mining machinery and the most important loads on the mine electrical system. By far the majority of mines, milling plants, preparation plants, and other related mining activities would find it virtually impossible to operate without electric motors.

Generators and motors can be studied independently of each other, but comprehension of motor operation is more easily obtained when generation is covered first. Consequently, the chapter will follow this format. In view of their relative importance, motors and their control will be the principal chapter discussion. The content will be elementary, but the objective is to provide sufficient information so that the effect of motors on the mine power system and specific motor applications in mining can be appreciated.

ALTERNATING CURRENT GENERATION

In chapter 2, it was demonstrated that a voltage is induced in a conductor when there is relative motion between the conductor and a magnetic field. This electromagnetic induction concept, called Faraday's law, is the basic principle behind the generation of voltage in electric machines. The following paragraphs return to this fundamental concept but in a slightly different fashion and serve as a transition between induced voltages in inductors or transformers and generators.

In figure 6.1, a conductor is under the influence of a magnetic field. If a force is placed on the conductor so that it moves at right angles to the magnetic-field direction, a voltage will be induced in the conductor. Obviously, when a conductor is part of a closed loop, current will flow. The direction of the current produced depends upon the directions of the magnetic flux and the conductor movement. Fleming's "right-hand rule" is often used to determine the current direction, and it also helps to illustrate the interrelationships among the three dependent parameters. The thumb, forefinger, and center finger of the right hand are stretched out so they are mutually at right angles to each other. If the hand is placed such that the forefinger points in the flux direction, with the thumb pointing in the direction of conductor motion, the center finger points in the direction of current flow.

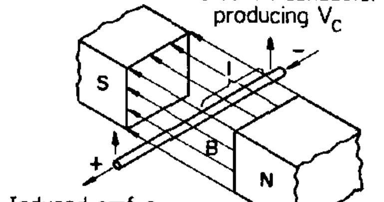


Figure 6.1.—Production of voltage from magnetic field (emf = electromotive force).

When the conductor cuts a magnetic field at a specific relative velocity, it has been found *(3)*[1] that the instantaneous magnitude of the voltage induced, e, can be calculated by

$$e = B\ell v_c, \tag{6.1}$$

where B = magnetic field flux density, T,
ℓ = conductor length, M,
and v_c = conductor velocity at right angles to magnetic flux field, m/s.

Accordingly, the magnitude of induced potential depends upon the flux density, the conductor length, and the conductor velocity relative to the magnetic field. By varying these parameters, a voltage of almost any magnitude can be theoretically produced. It is this electromechanical principle, an adaptation of Faraday's law, that is utilized in the generation of alternating and direct currents (ac and dc).

Principle of Generator Operation

Figure 6.2 illustrates the basic principle of ac generation. Consider a loop of conductor mounted on a mechanical drive shaft through an insulating block and rotated in a magnetic field *(9)*. A circular metallic ring, called a slip ring, is connected to each end of the loop, and brushes contact each slip ring surface to allow the connection of stationary conductors.

When the loop is rotated by a mechanical drive, a potential is induced in each side of the loop that is proportional to the conductor velocity at right angles to the magnetic field. At the position shown in figure 6.2, the induced voltage is at a maximum (relative right-angle velocity is maximum), but at 90° from this position there will be no induction (relative velocity is zero). The instantaneous voltages for the entire loop can be added algebraically as they are in series. Continued rotation will produce a sinusoidal voltage (fig. 6.2*B*) and thus sinusoidal current, if the loop is part of a closed circuit.

Generator Construction

It has been shown that there are basically two requirements for generation. The first is a conductor or

[1] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

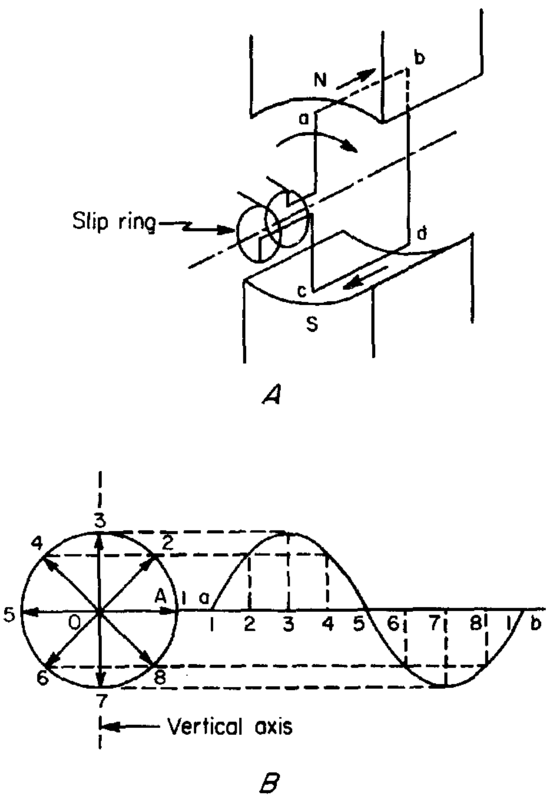


Figure 6.2.—Demonstration of ac generation.

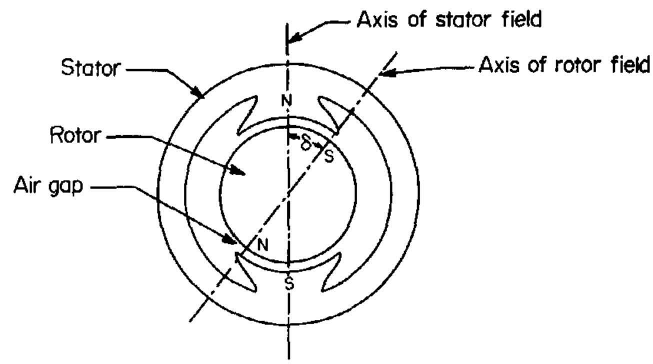


Figure 6.3.—Cross section of machine with salient poles on stator and nonsalient poles on rotor.

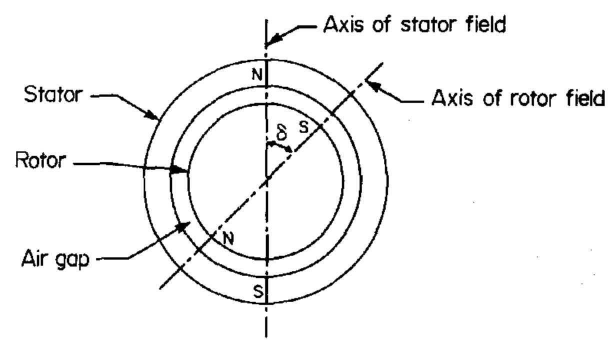


Figure 6.4.—Cross section of machine with nonsalient poles on stator and rotor.

winding in which a desired voltage is to be induced. This is termed an *armature winding*, and the structure enclosing it is called an armature. The second requirement is a magnetic-field source, and this is normally created by a field winding, although some very small machines use permanent magnets. To classify the rotating and fixed machine portions, the rotating member is referred to as the *rotor*, while the stationary portion is the *stator*.

The windings are placed on two concentric cylindrical iron cores with a small air gap between so that the flux path in the machine is as efficient as possible. The inner core usually serves as the rotor. Thin laminations, at times insulated from each other, are employed to minimize eddy-current loss, as in transformer construction. The structure of either core is one of two types: salient poles or nonsalient poles, which form the center lines of the magnetic field. Salient poles stick out from the cylinder surface (fig. 6.3) and have the windings around them but located near the core surface in the vicinity of the air gap. Nonsalient poles are part of a completely cylindrical surface, with the windings positioned in slots (fig. 6.4) (*3*). The conductors are usually insulated from the core. Surrounding the cores and windings is a structure called the frame, with some form of end enclosure. The frame serves to anchor the stationary machine elements to a foundation. The end enclosure may contain bearings, of either sleeve, ball, roller, or needle types, that support the rotor shaft and position the rotor properly with respect to the stator. Figure 6.5 is a sketch illustrating these physical components.

The function of an electromechanical machine is commonly described in terms of the number of available magnetic poles in the field. Thus, the elementary machine in figure 6.2 is a two-pole generator. Most generators, however, have more than two poles, usually even numbers of four, six, eight, and so on. Figure 6.6 shows an elementary four-pole generator (*9*). Here the armature needs only to turn 180° to produce a full sinusoidal cycle in its winding output.

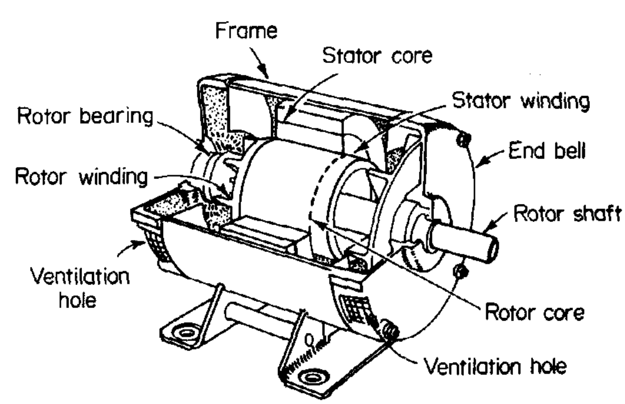


Figure 6.5.—Simplified sketch of electromechanical machine illustrating physical components.

The foregoing terminology applies to all electromechanical rotating machinery. For practically all ac generators or alternators, the armature is contained in the stator. The field winding is part of the rotor, with a dc field current supplied through slip rings, which is the reverse of the situation discussed previously.

Three-Phase Generation

A preliminary discussion of three-phase power generation has already been presented in chapter 4 but only in the context of balanced three-phase systems. This section will elaborate on its electromechanical conversion.

Consider figure 6.7, a cross-sectional view of an elementary three-phase, two-pole generator. The machine is termed *two-pole* because of the number of available magnetic poles in the field winding. Located in the stator, the armature has three single-conductor windings a, b, and c, whose axes are 120° apart. The rotor containing the field winding is turned at a constant speed by a mechanical power source connected to the rotor shaft, and the field winding is excited by dc. The magnetic-flux distribution around the air-gap circumference of the machine is designed so it forms a sine wave (*3*). Therefore, the induced voltage in each armature winding varies sinusoidally with the familiar 120° displacement among the three generated potentials.

For this two-pole generator, the sinusoidal voltage in each phase winding goes through one full cycle per rotor rotation. The waveform frequency (hertz) is identical to the rotor speed (revolutions per second). The sinusoid is thus in time with or synchronized with the mechanical speed, and such ac generators are often termed *synchronous generators*. For 60-Hz output, rotor speed is 60 r/s or 3,600 r/min.

An example of an elementary four-pole generator is shown in figure 6.8*A*. Here, the rotor poles alternate between north or south polarity when rotated. Each phase of the armature consists of two windings connected in series, as shown in figure 6.8*B*. The induced voltage per phase thus completes two cycles for each rotor revolution.

DIRECT CURRENT GENERATORS

A very elementary two-pole dc generator is shown in figure 6.9. The illustration differs from figure 6.2 only in that the dc generator has a commutator in place of the slip rings. The commutator is an annular ring that is split into parts (in this case, two), which are insulated from each

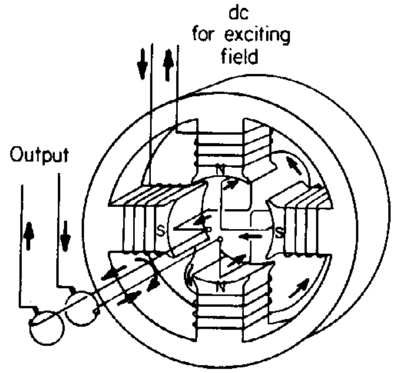


Figure 6.6.—Elementary four-pole, single-phase ac generator.

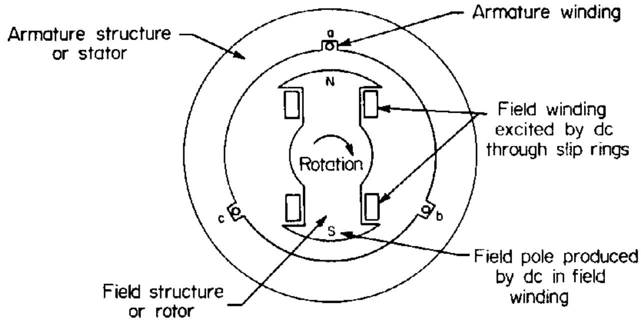


Figure 6.7.—Elementary two-pole, three-phase generator.

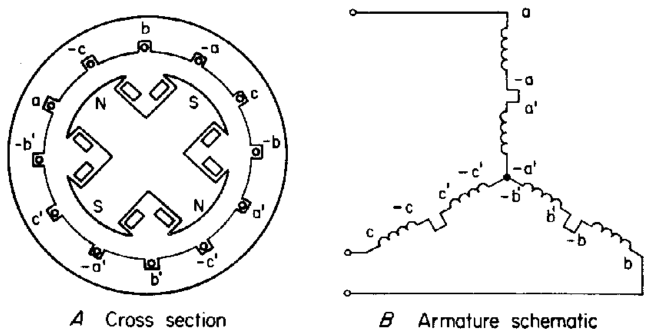


Figure 6.8.—Elementary four-pole, three-phase generator.

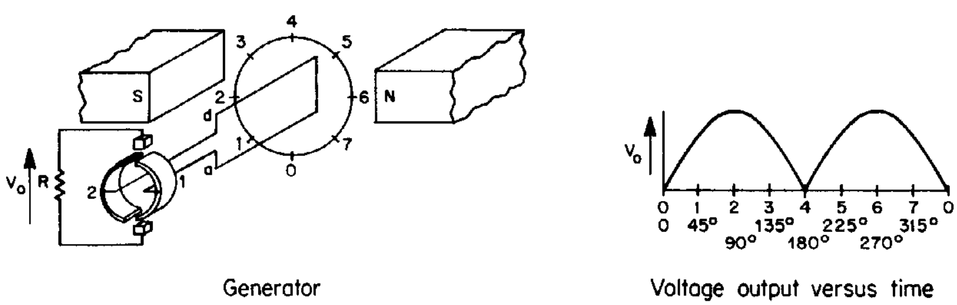


Figure 6.9.—Demonstration of dc generation.

other. Each part is termed a *commutator segment.* As before, carbon brushes contact the ring surface to allow connection of stationary conductors. During armature rotation, the voltage produced in the loop is a sinusoid, as in the ac generator. The commutator serves to rectify the waveform mechanically since at all times the positive and negative brushes are connected with the correct armature-winding polarity. In other words, the connection to the loop reverses or commutates every one-half revolution. Thus, the generator output waveform is the same as full-wave rectification (fig. 5.5). If the rotational direction reverses, so does the brush polarity.

Because of the ripple voltage, the two-pole dc generator is not realistic. In practical dc generators, the armature consists of many windings, with the commutator having a corresponding number of segments (fig. 6.10). In such a case, current from the generator will never drop to zero. When the number of armature windings is increased, the output ripple voltage decreases, and the average direct voltage will be closer to the peak voltage.

Unlike ac generators, dc generators have the armature winding on the rotor and the field winding in the stator. The field must be excited by dc provided by a source, which may be either external or internal. The internal excitation is possible because the armature is a dc source and can supply current to the field as well as the load. However, in order to start generation, the stator core of these machines must have residual magnetism. Generators connected in this way are called *self-excited.* When the source is external, the generator is termed *separately excited.* This is diagrammed in figure 6.11.

Self-excited generators have three configurations, depending on the field-winding connection: series, shunt, and compound, as shown respectively in figures 6.12, 6.13, and 6.14 The terms *series* and *shunt* relate directly to the winding connections. The compound generator has two windings, one connected in series and the other shunting the armature.

Each of the generator connections has a characteristic voltage output versus load current (*3*). Because the field, armature, and load currents are the same in series generators, the output voltage fluctuates widely with the load.

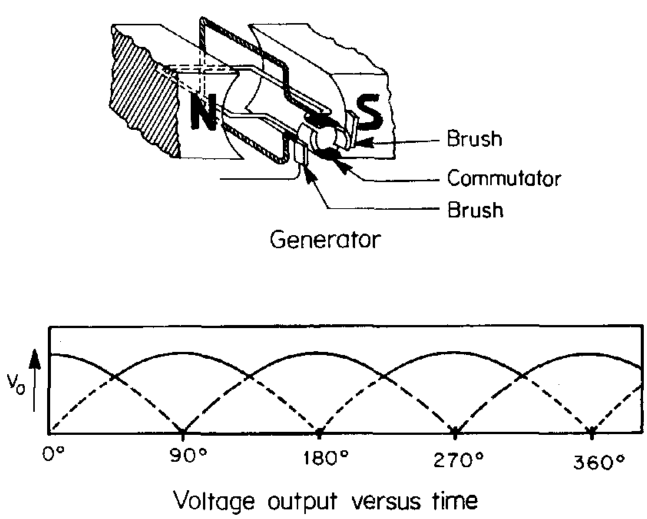


Figure 6.10.—Dc generator with two armature windings at right angles.

Hence, this connection is rarely used. Although shunt generator voltage output drops slightly as load current is increased, the regulation is satisfactory for many purposes. Compound generators are normally connected so the magnetic actions of the shunt and series windings aid each other. The resultant magnetic flux of the field can increase with load current, causing the output voltage to remain nearly constant. The level of output voltage in both the shunt and compound generators can be controlled by the variable resistance in series with the shunt field winding. The resistance in the separately excited generator provides the same function, but precise output-voltage control is obtained because the field-winding current is not a function of the load.

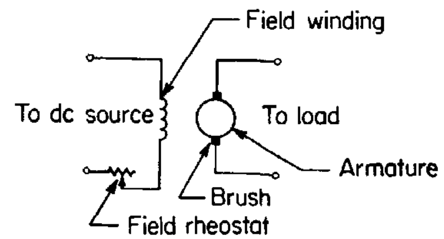


Figure 6.11.—Separately excited dc generator.

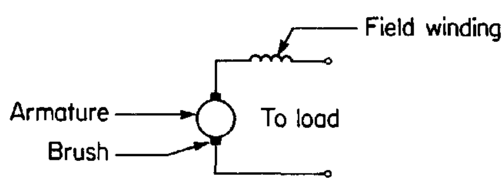


Figure 6.12.—Series dc generator.

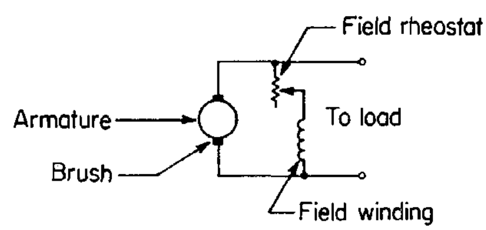


Figure 6.13.—Shunt dc generator.

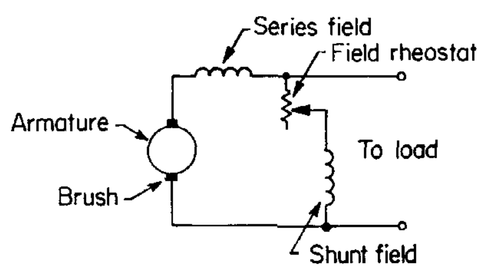


Figure 6.14.—Compound dc generator.

MOTOR BASICS

The essential motor parts are similar to those of a generator and include

- Two concentric cylindrical laminated-iron cores, separated by an air gap, to carry magnetic flux;
- Two sets of windings, wound or embedded in slots in the iron cores, either or both excited by dc or ac; and
- The inactive motor elements, including the frame, end bells, bearings, and so forth.

Combinations of these parts are found in practically all motors. Motors employ electrical energy to produce mechanical force, which is the reverse process from generator operation. The force of interest in motors is that which tends to produce rotation, or torque.

Torque

Motor torque considerations are based on the fundamental principle that a mechanical force is exerted on a current-carrying conductor in a magnetic field. A graphic example of this situation is shown in figure 6.15. Here, the magnetic field that surrounds the conductor (due to its current) interlinks with the largest magnetic field. This creates a large concentration of magnetic flux at one side of the conductor, which tends to force the conductor toward the lesser flux concentration. The result is an instantaneous force, f, at right angles to the magnetic field.

The magnitude of the force depends upon the magnetic field flux density, the conductor length, and the level of instantaneous current, and can be calculated for a straight conductor by (*3*)

$$f = B\ell i, \tag{6.2}$$

where f = force, N,
B = magnetic field flux density, T,
ℓ = conductor length, m,
and i = instantaneous current, A.

If the conductor is fixed by a radial distance, r, from the center of a rotor shaft, the associated torque, T, is (*3*)

$$T = Blri, \tag{6.3}$$

where T = torque, N·m,
and r = radial distance or moment arm, m.

For a winding, the total torque is the summation of the torques for the individual conductors or coil sides. For electromechanical machines, this mechanical quantity is termed *electromagnetic torque*, and when combined with rotation the resultant power quantities follow the rules of mechanics.

Another way of visualizing the development of motor torque is the interaction of two magnetic fields. A mechanical force is exerted on magnetic material, be it a permanent magnet or magnetism created by electric current flow. The force tends to align the material with the closest part of a magnetic field, so the north pole of one machine member is directly in line with the south pole of the other member. If the force is acting at a moment arm about a rotor shaft, torque is produced.

Even though equation 6.3 is expressed in newton meters (adhering to SI units), the quantities normally used are pound-feet, ounce-inches, and gram-centimeters. The common method of relating motor mechanics is by reference to a percent of full-load torque.

Speed-Torque Relationships

Speed-torque curves are the mechanical characteristic curves of a motor; a general example for an induction motor is provided in figure 6.16 for discussion (*15*). One application for these curves is to find the most suitable drive for a given machine. As the machine load can also be described by a speed-torque curve (see load torque in the figure), the comparison of the load and motor curves will show if the motor has the necessary characteristics to drive the load and also what the operating point will be. The operating point is the intersection of the two curves. Many other parts of the motor characteristic relate its suitability for a specific application, and some of these are listed below and are shown in the labels of figure 6.16.

1. *Locked-rotor torque.* The minimum torque developed by a motor at the instant of power application, sometimes called breakaway or starting torque.
2. *Accelerating torque.* The torque developed during the period from zero to full rated speed with rated power applied. The term is often used for the net torque between the motor and the load. It is apparent in the figure that this is a nonlinear value with speed.
3. *Breakdown torque.* The maximum torque possible from the motor with rated power input, also called maximum torque.
4. *Pullup torque.* The minimum torque developed during motor acceleration from zero to full rated speed with rated power applied. The minimum can exist in some motors at full rated speed.
5. *Full-load torque.* The torque necessary to provide rated output at rated speed with rated power applied.

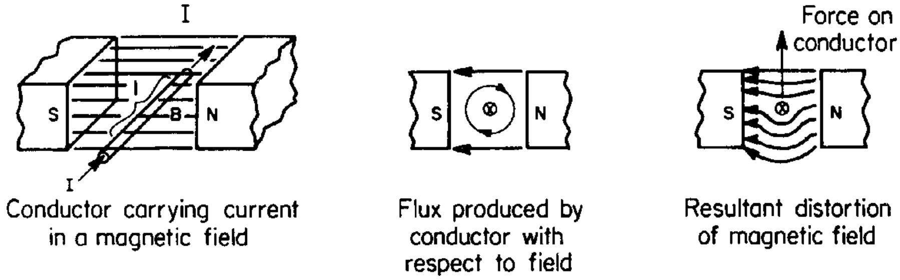


Figure 6.15.—Current-carrying conductor in a magnetic field.

6. *Pullout torque.* The maximum torque produced by a motor without stalling. This is sometimes incorrectly referred to as the maximum or breakdown torque. If a torque is applied to a motor above this value during operation, it will stall.

The other motor terms listed in figure 6.16 are tied to specific motor types or operations yet to be discussed.

Standardization

The National Electrical Manufacturers Association (NEMA) sets standards for the manufacture of electric motors (*15*), which are used throughout the mining industry. NEMA standards generally cover seven areas: speed-torque characteristics, frame size, enclosure, horsepower rating, voltage, temperature rise, and application. Although machines from different manufacturers should be directly interchangeable when they conform to a particular NEMA standard, there may still be some variation between manufacturers.

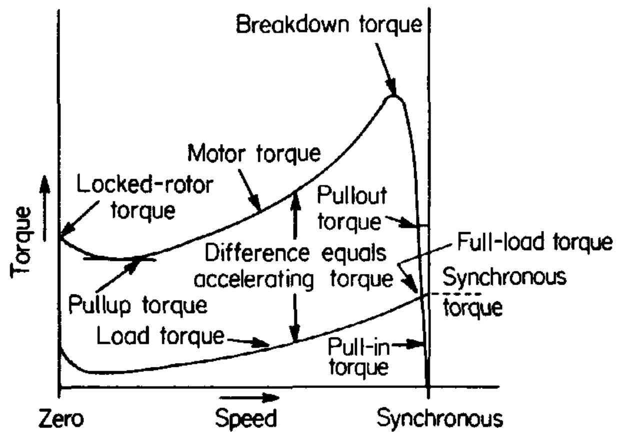


Figure 6.16.—General speed-torque motor characteristic.

Frame Size

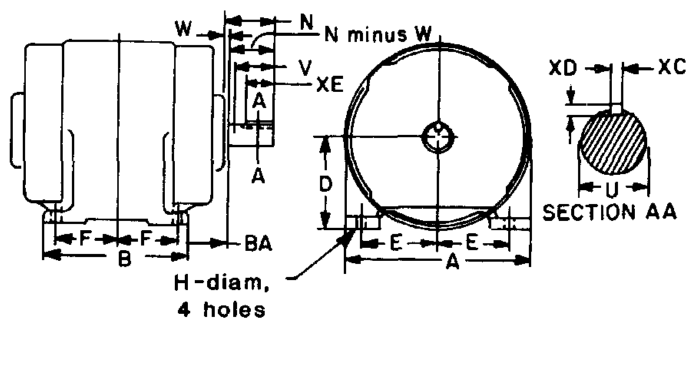
Most motors of 250 hp and under are rated according to a frame number that specifies the essential mounting dimensions (fig. 6.17) (*15*). The same frame number series covers all ac or dc motor types, and a dozen or more different motors might have the same frame.

Enclosure

Motor enclosures are usually classified as *open* or *totally enclosed.* Open motors simply have openings, usually in the end plates, to allow air cooling of the windings. Totally enclosed motors prevent passage of air into the enclosure, but these motors are not always sufficiently closed to be air tight. In this general class are the explosion-proof motors (see chapter 16), dust-ignition-proof motors, dust-tight motors, and waterproof motors. Cooling for these motors can be by air conduction on the outer frame, internal forced air through a pipe, or a liquid-cooled (water or oil) outer jacket surrounding the frame.

Horsepower

Motor horsepower is also standardized, such as 1/2, 3/4, 1, 1-1/2, 2, 3, 5, 7-1/2, 10, 15, 20, 25, 30, 40, 50, 60, 75, 100, 125, 150, 200, and 250 at speeds for 2 to 16 poles with 60-Hz operation (*15*). Above 250 hp, the standard powers are related to motor type. When a horsepower is given, it is often combined with a service factor to allow for usual fluctuations in supply voltage or slight overloading. The service factor indicates the permissible overload and is a multiplier applied to the normal horsepower rating, with values ranging from 1.0 to 1.4 depending on the size and type of motor. For instance, a 1.15 service factor would



No.	A	B	D	E	F	BA	H	NW	U	V min	XC	XD	XE
145T	7	6	3-1/2	2-3/4	2-1/2	2-1/4	11/32	2-1/4	7/8	2	3/16	3/16	1-3/8
184T	9	7-1/2	4-1/2	3-3/4	2-3/4	2-3/4	13/32	2-3/4	1-3/8	2-1/2	1/4	1/4	1-3/4
213T	10-1/2	7-1/2	5-1/4	4-1/4	2-3/4	3-1/2	13/32	3-3/8	1-3/8	3-1/8	5/16	5/16	2-3/8

Dimensions in inches

Figure 6.17.—Examples of three frame number dimensions.

indicate that a motor can carry 15% more than rated load continuously without overheating (that is, exceeding the rated temperature rise) as long as the frequency, ambient temperature, line voltage, and so on are at rated values.

An interesting situation occurs with frame sizes primarily intended for motors of less than 1 hp (*fractional horsepower*). A fractional-horsepower motor is considered to be any motor built in a fractional-horsepower frame, even if the actual horsepower rating is in excess of 1 hp. Most of these motors, however, are only available for single-phase ac.

Voltage

NEMA voltage designations are specified for most motors, be they three-phase, single-phase, or dc (*15*). A listing of the voltage ratings (also called the motor terminal voltage) common to mining is available in table 6.1. A maximum voltage variation of ±10% from rated is permitted. With ac motors, the allowable frequency fluctuation for the power supply is ±5%.

Temperature Rise

The allowable temperature rise from an ambient temperature is dependent upon the class of insulation used in the motor (*15*). The electrical insulation system is one of the most important components of a motor, as its degradation seriously affects the reliability and service life of the motor. Insulation systems are divided into four classes, A, B, F, and H, depending upon their thermal endurance. Ratings for industrial motors are typically based on a 40° C ambient temperature, but some are based on 25° C. Table 6.2 provides the allowable rise for each insulation class; also included are the common insulating materials and the maximum "hot-spot" temperature, which is the highest temperature allowable at any part of the motor. Motors built to a specific class and operated so the recommended temperatures are not exceeded may be expected to have a serviceable life of 20 yr with minimal maintenance. However, physical abuse and the electrical stresses discussed in chapter 11 can seriously shorten the motor life regardless of insulation class and operation. Class A insulated motors are rare in almost all applications, except for very small horsepowers. Class F and H insulations are most often used in motors for mining applications (*17*). The maximum surface temperature of any permissible mining motor must not exceed 150° C (see chapter 16).

If the maximum ambient temperature is greater than specified, the allowable temperature rise must be decreased by the difference in temperature above ambient. Maximum ambient is the highest temperature the motor is normally exposed to. The rise may be increased by a like amount when the maximum ambient temperature is below that specified. These specifications are applicable when operating under typical barometric pressure, as long as the altitude does not exceed 1,000 m (3,300 ft). Above 1,000 m, the allowable temperature rise must be reduced 1.0% for each 100 m (330 ft) above 1,000 m.

Additional classification standards will be discussed in the following sections, for example, applications as to load-speed and load-torque requirements. These are tied to the torque-speed characteristics, which are related to the motor type.

Table 6.1.—Motor voltage ratings common to mining

System type [1]	Nominal system voltage, V	Motor rated voltage, V
3-phase, low voltage	[2] 208	208
	230	220
	[3] 460	[3] 440
	[4] 480	[4] 440
	[3] 575	[3] 550
	[4] 600	[4] 550
3-phase, medium voltage	[4] 1,040	[4] 950
3-phase, high voltage	2,400	2,300
	4,160	4,000
	7,200	6,600
	12,470	12,470
	13,200	13,200
	13,800	13,200
Direct current	300	250
	600	550
Single phase	120	115
	240	230

[1] System voltage designations follow 30 CFR 18, 75, 77.
[2] Wye.
[3] More suitable for stationary equipment applications.
[4] Intended use is mobile mining equipment.

Table 6.2.—Motor insulation classes

Insulation class	Temp rise, °C [1]		Maximum hot-spot temp, °C	Common insulating materials [2]
	Open motors	Totally enclosed motors		
A	50	55	105	Cotton, cellulose, paper, organic, enamel-coated wire.
B	70	75	130	Mica, glass fiber, asbestos.
F	90	95	155	Do.
H	105	115	180	Mica, glass fiber, silicone elastomers, silicone resins, asbestos.

[1] Allowable rise from ambient temperature.
[2] Each class has compatible bonding agents for the materials shown.

Motor Type

The classification of motor types, which is dependent upon how the stator or rotor windings are excited, results in three general motor classes: induction, synchronous, and dc. The first two are ac machines, and for many applications these are more rugged, require less maintenance, and are less expensive than dc motors of equal horsepower and speed ratings. Ac motors can be used effectively for the majority of motor applications except when very high starting torques are required. The most widely used ac type is the squirrel-cage induction motor, so-called for its appearance. It has no slip rings, commutator, or brushes to wear out and uses the simplest kind of starting equipment. Three-phase squirrel-cage induction motors and series-wound dc motors are the most popular electromechanical machines in mining.

After the presentation in chapters 2, 3, and 4, it could be expected that three-phase motors would be more complex than their single-phase and dc counterparts and therefore more difficult to understand. However, although some parts of three-phase motor construction are more complex, their operation is simpler. As just stated, induction and synchronous machines are the two major ac motor types. Synchronous motors correspond to three-phase generators. In typical large machines, dc is applied to the field winding located in the rotor, while three-phase ac (instead of being generated) is supplied to armature windings placed in the stator. Induction motors also receive ac power at the stator windings, but ac is delivered to the rotor winding indirectly by induction, in the same manner as in a transformer.

THREE-PHASE SQUIRREL-CAGE INDUCTION MOTORS

In order to comprehend the operation of induction motors and understand important terminology, it is perhaps best to start with a simple demonstration. Although the motor does not have familiar motor components, its construction and operation do have direct application in induction-disk relays and watthour meters.

Consider figure 6.18, which depicts an aluminum disk and a horseshoe magnet (*8*). Both are mounted about the same axis and are free to rotate. When the magnet is rotated, the disk cuts the magnetic lines of force, a voltage is induced, and eddy currents will then flow. Under the magnet's south pole, the eddy currents set up north magnetic poles, and conversely for the north pole of the magnet. Because the pole attracts, the disk rotates, following the magnet. The disk can never reach the magnet speed, as there would be no relative motion between the two (that is, no induction would exist). The mandatory difference in speed for induction motors is called slip.

In conventional induction motors, the action of the disk occurs in a rotor winding, and the rotating magnetic field is supplied by the stator winding. For induction-disk relays, the aluminum disk is the same but the induction force is supplied by an ac-driven stator (see chapter 9 and also the description of watthour meters in chapter 5).

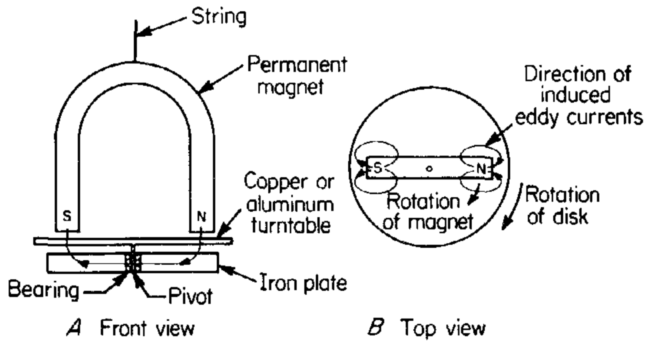


Figure 6.18.—Demonstration of induction-motor operation.

Elementary Three-Phase Motor

Figure 6.19 illustrates an elementary two-pole, three-phase squirrel-cage induction motor (*11*). The stator consists of three salient poles spaced 120° apart; the stator windings around each pole are connected in wye and energized by a three-phase system. The rotor has three main elements: a shaft (not shown), core, and winding.

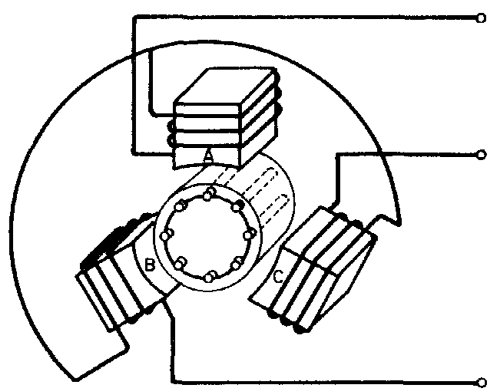


Figure 6.19.—Elementary three-phase induction motor.

As with generators, the rotor core is made of iron laminations pressed onto the shaft. The squirrel-cage winding is constructed by embedding heavy copper or aluminum bars in the core slots. The bars are connected to each other by copper or aluminum rings located on both core ends, which complete the closed circuit. In other words, there are no external connections to this rotor winding either by slip rings or a commutator. Figure 6.20 shows the winding construction.

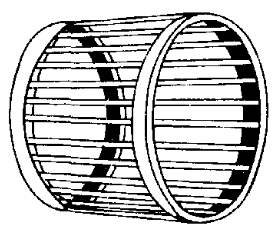


Figure 6.20.—Squirrel-cage rotor winding.

When the stator windings are powered by a three-phase system, currents through the coils reach their respective maxima at different intervals in time. Since the three currents are displaced by 120°, the magnetic field generated by each coil is also displaced from the other two by 120° (fig. 6.21*A*). The magnetic field of each winding alternates from north to south; thus, each has the action of two poles. Figure 6.21*B* shows the instantaneous direction of a stator flux as it passes through the rotor at different time intervals. At zero degrees, for instance, phase A is at maximum north, while phases B and C are weak south poles. At 60°, phase C becomes strongly polarized in the south direction and phases B and A are weak norths. The larger arrows shown in the figure represent the instantaneous direction of the resultant two-pole magnetic field. Consequently, for this example, the magnetic field is rotating counterclockwise.

In a transformer, voltages are induced in the secondary circuit by the primary. The stator of an induction motor acts in the same manner as the primary, with the rotor winding acting as the secondary winding. The rotating magnetic field of the stator cuts the rotor conductors, and motor action is developed. The relative motion, or slip, between the rotating flux and the rotor generates voltages within the rotor conductors.

According to Lenz's law, the voltage induced in each rotor bar will be in a direction opposing the relative motion of the rotating flux and the rotor. The induced-voltage direction in the rotor conductors under the influence of a two-pole rotating magnetic field is shown in figure 6.22, where positive implies that the voltage direction is toward the viewer. The voltage magnitude in each bar depends upon the stator magnetic-field density at that point. These voltages cause currents to flow through the bars, an end ring, adjacent bars, and then back through the other end ring to the origins, in complete loops. The circulating rotor currents produce magnetic fields about each rotor bar.

The interaction between the stator field and the fields around the rotor conductors results in a mechanical couple and thus motor torque. Hence, the rotor will rotate in the same direction as the stator field. A simple reversal of any two phase conductors to the stator windings of a three-phase induction motor will reverse the stator phase sequence and thus reverse the motor rotation.

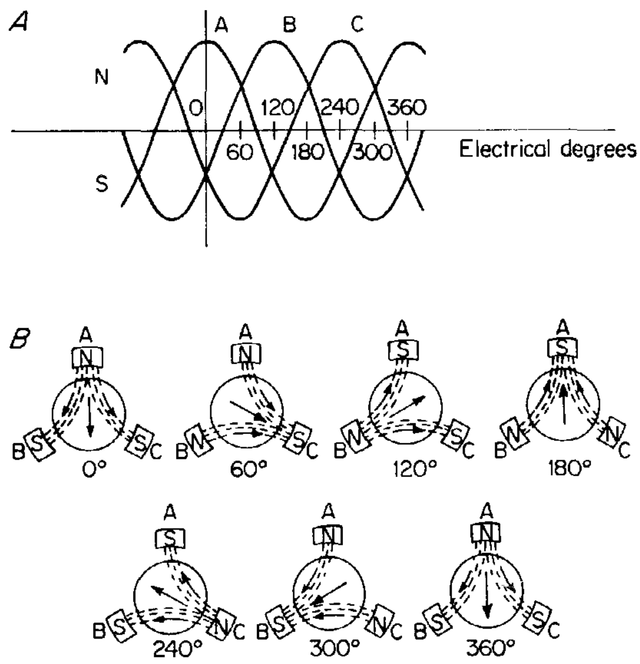


Figure 6.21.—Rotating magnetic field in elementary three-phase, two-pole induction motor.

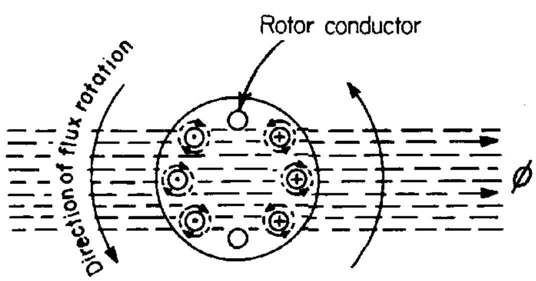


Figure 6.22.—Induced rotor potential by rotating flux.

The speed at which the stator field rotates is termed the *synchronous speed* of the motor and can be calculated from

$$n_s = \frac{120f}{p}, \tag{6.4}$$

where n_s = synchronous motor speed, r/min,
f = line frequency, Hz,
and p = number of magnetic poles presented by stator.

As slip is required to produce rotor induction, a squirrel-cage motor may approach but never obtain synchronous speed. Slip can be expressed mathematically as

$$s = \frac{n_s - n_r}{n_s}, \tag{6.5}$$

where s = motor slip, expressed as a per-unit decimal or a percent,
and n_r = actual motor speed, r/min.

Losses will occur in actual motors because of electrical and mechanical inefficiencies. Those prominent in induction motors are

- Rotor winding loss, related to I^2R;
- Stator winding loss, also an I^2R loss;
- Stator core loss, caused by eddy currents and hysteresis in the core iron; and
- Friction and windage (rotational or mechanical) losses.

These are almost pure active powers; therefore they are often expressed in watts. The losses in both windings of induction motors vary as the square of line current, core loss is nearly constant, and unless motor speed varies considerably rotational losses are nearly constant (*11*). Knowledge of machine losses allows the determination of motor heating and efficiency.

The efficiency of motor operation is a measure of the ability to convert input power to mechanical power:

$$\text{Efficiency} = \frac{\text{output}}{\text{input}} = \frac{\text{input} - \text{total losses}}{\text{input}}, \tag{6.6}$$

which may be expressed as a per-unit decimal or a percent. Slip is also related to motor efficiency, being numerically equal to the ratio of winding loss in the rotor to the total rotor power input:

$$s = \frac{\text{rotor winding loss}}{\text{rotor power input}} \tag{6.7a}$$

or

$$s = \frac{\text{rotor winding loss}}{\text{motor power input} - \text{stator losses}}. \tag{6.7b}$$

Stator losses in equation 6.7*b* include friction and windage. Equations 6.4 through 6.7 can be employed to calculate the synchronous and actual motor speeds and also the possible power and torque output, realizing that (*8*)

$$\text{power output (watts)} = 746\ (\text{horsepower output}), \tag{6.8a}$$

and $$\text{hp} = \frac{n_r T}{5250},$$

in which $$T = \frac{K I_r^2 R_r}{s}, \tag{6.8b}$$

where hp = horsepower output of motor,
n_r = actual motor speed, r/min,
T = motor torque, ft·lb,
K = a torque constant, ft·lb/V,
I_r = rotor current, A,
and R_r = rotor resistance, Ω.

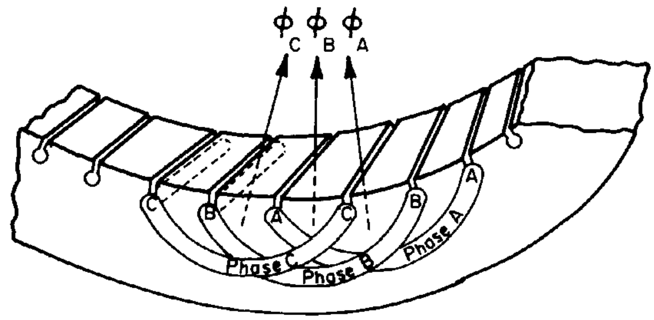


Figure 6.23.—Lapped windings of three-phase motor stator.

Motor Construction

The elementary salient-pole motor of figure 6.19 is undesirable from the standpoint of the ineffective use of material and space, as well as its overall inefficiency. The main disadvantage is coupled to the distinguishable stator poles. To overcome this problem, actual induction motors have lapped stator coils, as shown in figure 6.23, where several coils make up a stator winding that can be either delta or wye connected. The flux directions of each coil are illustrated as ϕ_C, ϕ_B, ϕ_A, and each coil contributes to the rotating flux development of the entire stator. The coils and windings are arranged to have the same effect as salient poles, but the poles are not physically distinguishable. An induction motor is assigned a specific pole number if at any given instant the stator windings set up the same number of magnetic pole fields.

The rotor core and squirrel-cage conductors are usually not insulated from each other, because the induced current is effectively contained within the conductors owing to their significantly lower resistance. The rotor core is pulled magnetically toward the stator core across the air gap. If the force is uneven when the rotor turns, the result is vibration. This is detrimental in several ways as it can lead to structural insulation failures, premature bearing failures, and misalignments with the motor load. Vibration does not occur if the magnetic effect about the rotor periphery is equal. An additional method for preventing vibration is to place rotor conductors in slots skewed to the stator slots so that a rotor slot passes gradually under a stator slot rather than abruptly. This practice also prevents "dead spots," or positions of near-zero or minimum magnetic influence. Another method of eliminating dead spots is to construct the motor so that the number of rotor slots plus the stator slots sums to a prime number.

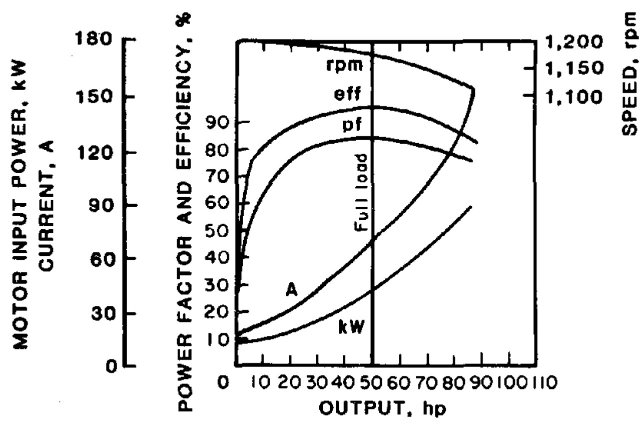


Figure 6.24.—Characteristic curves of three-phase induction motor.

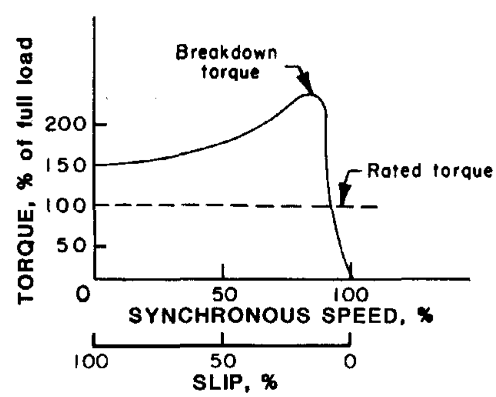


Figure 6.25.—Typical torque-speed characteristic for general-purpose induction motor.

Motor Behavior

Figure 6.24 is a graph of the speed, efficiency, power factor, power input, and current load of a typical three-phase induction motor found in mining applications. Figure 6.25 shows a representative torque-speed characteristic for a similar machine. These curves can be used to describe the electrical and mechanical operation of induction motors under loading.

From the typical torque-speed curve, the torque at locked rotor is approximately 150% of rated. The level increases steadily with rotor acceleration to the maximum or breakdown torque. With applied power input, the rotor continues to accelerate until the slip reduction reduces the rotor current to a point where torque is equal to the load torque.

Consider the motor running with no load. As the motor is loaded, slip increases, causing an increase of induction in the rotor. Hence, rotor current rises, resulting in a stronger rotor magnetic field and motor torque. Torque continues to increase with the increased shaft load until breakdown torque is reached. Any further load results in a slip value that decreases torque. If the high load is sustained, the rotor will stop.

Because the induction motor operates basically as a transformer, its electrical characteristics, as seen by the power source, will be a reflection of those occurring in the stator winding. Figure 6.26 shows phasor diagrams for rotor current and voltage during three operation points; these are referenced to the flux-density phasor of the stator, B_{stator} (*11*).

The rotor bars are embedded in the steel core so they have a high reactance (*3*). At locked-rotor conditions (rotor stationary), the stator magnetic field rotates past the motor at synchronous speed, and the induced voltage in the rotor conductors has the same frequency as the stator (or line frequency). The result is a high ratio of rotor reactance to resistance, and stator current lags stator voltage by a large amount (fig. 6.26*A*). During rotor acceleration, slip decreases, which also lowers the frequency of rotor current and voltage according to the following relationship (*8*):

$$f_r = sf, \tag{6.9}$$

where f_r = frequency of sinusoidal voltage and current induced in rotor bars, Hz,

s = slip, expressed as a decimal,

and f = frequency of voltage and current in stator, Hz.

Thus, inductive reactance drops, increasing the power factor (fig. 6.26*B*). Theoretically, if the motor could obtain synchronous speed, the rotor power factor would reach unity (fig. 6.26*C*). However, as this cannot happen in actual squirrel-cage motors, the maximum power factor is seldom greater than 0.85 (fig. 6.24) and never greater than 0.95.

Because the output torque increases with slip, motor speed decreases slightly as the load increases from no load to full load. Yet efficiency and power factor drop rapidly on low load conditions. Hence, an induction motor should not be operated at much below rated load for any length of time. It is apparent from figure 6.24 that efficiency diminishes when motor load increases above a given value. Consequently, an induction motor should not be overloaded for any extended period. Power-factor and efficiency curves normally follow roughly the same path; thus, power factor can be considered as an estimate of motor operating efficiency.

The torque developed by a three-phase induction motor varies as the square of the stator supply voltage, or

$$\text{Torque} \propto V^2_{stator}. \tag{6.10}$$

Therefore, a 10% reduction from rated stator voltage will cause a 19% reduction in available torque output.

Insulation

Insulation in motors normally has five forms: strand, turn, lead, crossover, and ground (*15*). Since the rotor conductors are uninsulated, the insulation of the stator winding conductors is the critical concern. The primary insulating system is that between the windings and the stator core or ground, and the secondary insulation is in strands, turns, leads, and crossovers.

Copper magnet wire, and to a much lesser extent aluminum magnet wire, is used to construct the stator winding or coils. Strand insulation is most frequently a resinous coating on the wire. Turn insulation is applied after strands are wound into coils (or the actual windings), and this may be a resinous coating, resinous-film taping, paper taping, or a fibrous wrapping. These types of turn insulation are utilized for applications of 6,600 V and less; for higher voltages, additional layers of mica or varnished cloth tape can be used. Crossover insulation is employed to protect wires that cross each other. The crossovers are often the weakest point in winding construction; thus, they require additional protection. Lead insulation is simply insulation about the conductors leading to the windings. Lastly, ground or ground-wall insulation is the major insulation system of the motor and isolates the windings from the core. This insulation is always subjected to the highest potential difference and requires the most attention.

Design Characteristics

Figure 6.27 illustrates the standard NEMA torque-speed characteristics for squirrel-cage induction motors. The shapes of these curves depend primarily on the ratio of rotor conductor resistance to reactance. For instance, to obtain a greater locked-rotor torque, as well as a greater slip over the unable load range, rotor conductor resistance may be increased by decreasing the conductor cross-sectional area, or inductive reactance may be decreased by placing the bars closer to the rotor surface. On the other

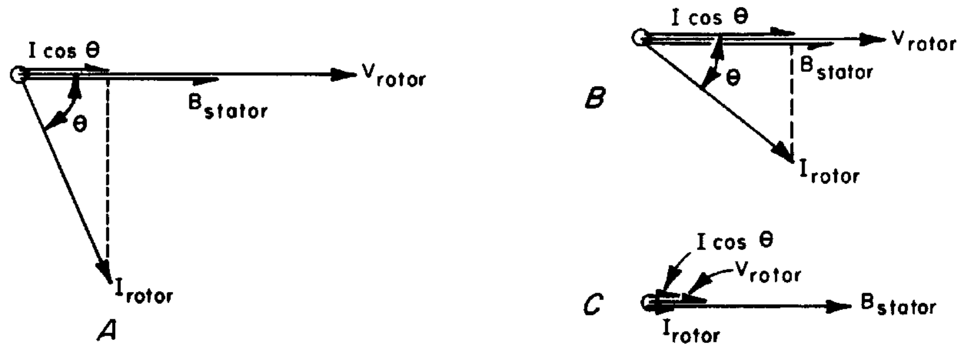


Figure 6.26.—Phasor diagrams of rotor and stator flux density for induction motor.

hand, an increase of conductor resistance will decrease overall motor torque and the stator current drawn during locked-rotor conditions.

Single-cage rotors, as previously described, are the most rugged and the most used. Double-cage rotors use two conductors, one over the other, per rotor slot (fig. 6.28*A*) and provide higher starting torques with higher load efficiency and lower running slip than the single cages (*14*). Here, the higher conductor would have high resistance and low reactance, while the lower set would have low resistance and high reactance. Double-bar rotor conductors are often susceptible to damage on loads with long accelerating times. To overcome this problem, deep-bar rotors (fig. 6.28*B*) can be used. These have a thermal advantage in that the full conductor area is available for heat dissipation, but the design still approximates the performance of the double bar. Regardless of the design, the torque-speed curves are matched to the squirrel-cage rotor construction, which is fixed for a specific motor.

In addition to rotor design changes, the actual values of breakdown and locked-rotor torque vary with the horsepower, frequency, and speed ratings of the motor. Although the operating characteristics are a function of rotor impedance, the horsepower rating is mostly dependent upon the power (or kilovoltampere) capacity of the stator and rotor windings. As rotor losses are constrained to the rotor cage, rotor thermal capacity is limited. Therefore, motor designs that create large rotor currents, such as high-torque high-slip, may have intermittent time ratings or a limited number of allowed successive starts. Unless these constraints are heeded, improper operation will burn out the rotor winding.

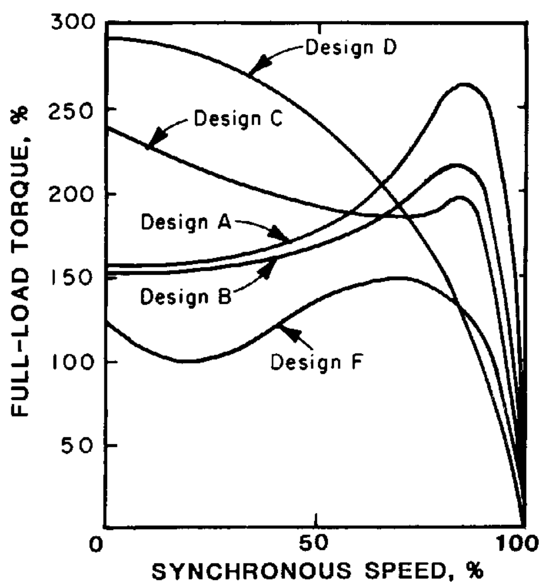


Figure 6.27.—Typical torque-speed characteristics for NEMA-design three-phase squirrel-cage motors.

The different rotor designs have led to a variety of speed-torque characteristics. To distinguish among the various types, NEMA uses a code letter system that signifies specific rotor constructions (*8*). Design B serves as the comparison basis for the motor performance of other designs and is often called the general-purpose motor. This design has relatively high efficiency even at light loads and a reasonably high power factor at full load. It has single rotor bars located rather deep in the core but with large-area slots for good heat dissipation. Starting currents range from 4.5 to 5 times the rated full-load current. The design B motor has the broadest industrial application field.

Design A has characteristics similar to those of design B, except that it has a higher breakdown torque. The rotor conductors are shallower, which decreases rotor reactance but increases the starting current, being five to seven times rated current. As a result, design B motors are often preferred over design A for large motor applications. As shown in figure 6.27, design A motors have the best speed regulation, as evidenced by the steep curve portion between synchronous speed and breakdown torque (*8*).

Design C motors have a double-cage rotor construction that results in higher locked-rotor torque and lower breakdown torque than those of design B. Starting currents are about 3.5 to 5 times rated current (*8*). These characteristics are well suited for conveyor belt drives and other applications that have sudden large load increases, but low or normal starting inertia. The motors are not suited for heavy high-inertia loads because the thermal dissipation is limited and high rotor current tends to concentrate in the upper bars (*8*). Accordingly, frequent starting of these motors can cause rotor overheating.

Very high locked-rotor torque and high slip are found with design D characteristics. Design D's principal application is for high-inertia loads. The rotor is of high-resistance design with bars located close to the surface (*8*).

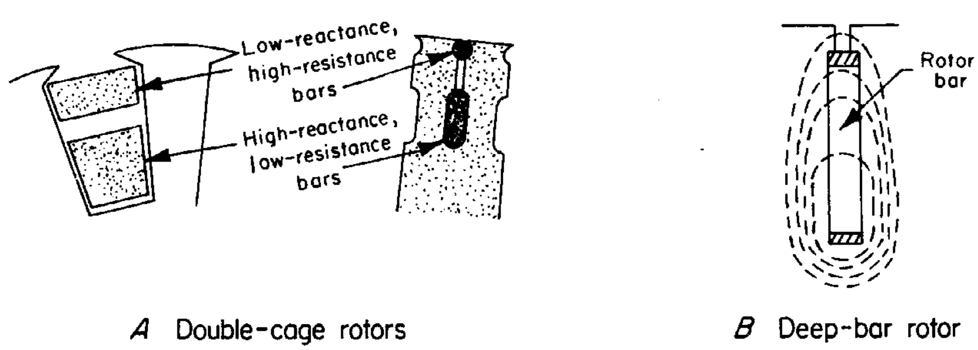


Figure 6.28.—Other rotor-conductor designs.

Starting currents range from three to eight times rated load current. The motor is suited for heavy-duty starting, but again, the poor heat dissipation of the rotor design means that starting cannot be frequent.

Design F has lower locked-rotor and breakdown torques than does design B. Design F motors also use a double-cage rotor with high resistance in both conductors, which reduces both starting and running current (*8*). The locked-rotor current is the lowest of all motor designs. Thus, design F motors are applied when starting-current limitations are severe and both starting and maximum torque requirements are low. The design, however, has poor speed regulation, low overload capacity, and usually low full-load efficiency.

Induction-Motor Starting

From the foregoing it can be seen that if an induction motor is started by directly connecting it to a power system, the momentary starting current can range from three to eight times the full load current. While this will not damage the motor, the high current can cause a significant disturbance on the power system, and, in some cases, activate overcurrent protection devices. However, most induction motors in mining applications are started by directly connecting them to the power system, especially those within mining machines such as continuous miners. The system usually has enough impedance that protective devices can be set above the in-rush current to prevent nuisance tripping. This, however, is a major problem, which is further discussed later in this chapter and in chapter 10. Full-voltage starting can usually be performed on 440- to 550-V motors up to 1,600 hp. NEMA standard magnetic starters for this range are shown in table 6.3 (*3*). The jogging service listed in the table refers to frequent stop-start or plugging (reversing under load) applications.

As shown in figure 6.29, the across-the-line starter is simply three contacts driven by a solenoid, also called a contactor. Pressing the start button energizes the solenoid, which closes the M contacts. An auxiliary contact set (M_b) simultaneously closes and bypasses the start switch. Pressing the stop button deenergizes the solenoid.

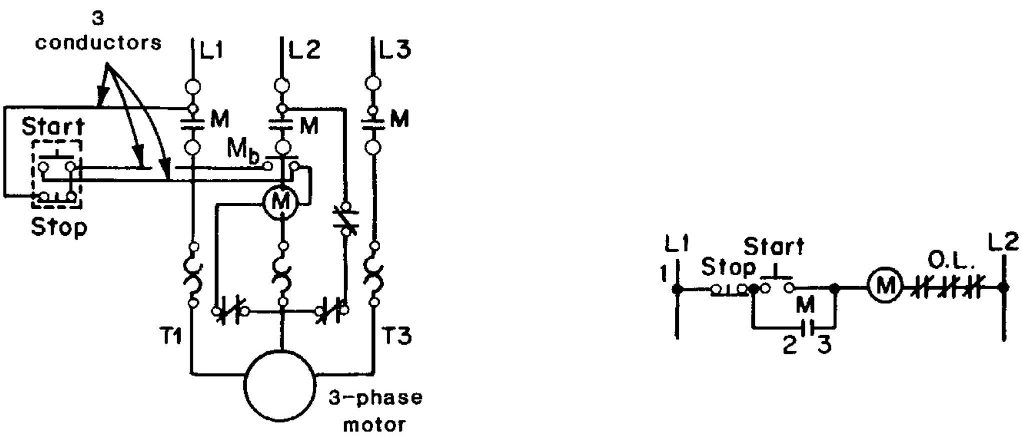
Above 1,600 hp (but sometimes lower), full-voltage starting becomes impractical even when the load connected to the motor can withstand the stress. Common methods for starting these large induction motors are shown in figure 6.30. In basic terminology, all these methods can be called reduced-voltage starting. In figure 6.30*A*, an autotransformer is used to start the motor at reduced voltage (50% to 80% of rated), thus limiting starting current and torque. When almost at full speed, contactors quickly change the motor from the autotransformer to the full-voltage supply. Primary resistor or reactor starting (fig. 6.30*B*) inserts fixed or variable impedances in series with the motor; these are shorted out after acceleration. For the wye-delta technique (fig. 6.30*C*), the motor is started as a wye connection, which places about 58% of the rated delta terminal voltage across the windings, limiting line current to 58% and torque to 35%. After acceleration, motor operation is with a delta connection. Part-winding starting requires that the motor have two identical stator windings (fig. 6.30*D*). Starting uses only one winding and limits starting current to about 65% of normal, torque to 45%. After acceleration, the second winding is switched in.

There are many systems that cannot take the shock of full-voltage starting. One instance is a conveyor belt drive

Table 6.3.—NEMA class A standard starters for three-phase induction motors

Controller size	Continuous current rating, A	Maximum horsepower for normal service		Maximum horsepower for jogging service	
		220 V	440–550 V	230 V	460–575 V
00	9	1.5	2	NAp	NAp
0	18	3	5	1.5	2
1	27	7.5	10	3	15
2	45	15	25	10	30
3	90	30	50	20	60
4	135	50	100	30	150
5	270	100	200	75	300
6	540	200	400	150	NAp
7	810	300	600	NAp	NAp
8	1,215	450	900	NAp	NAp
9	2,250	800	1,600	NAp	NAp

NAp Not applicable.



3-phase diagram Control circuit

Figure 6.29.—Across-the-line magnetic starter.

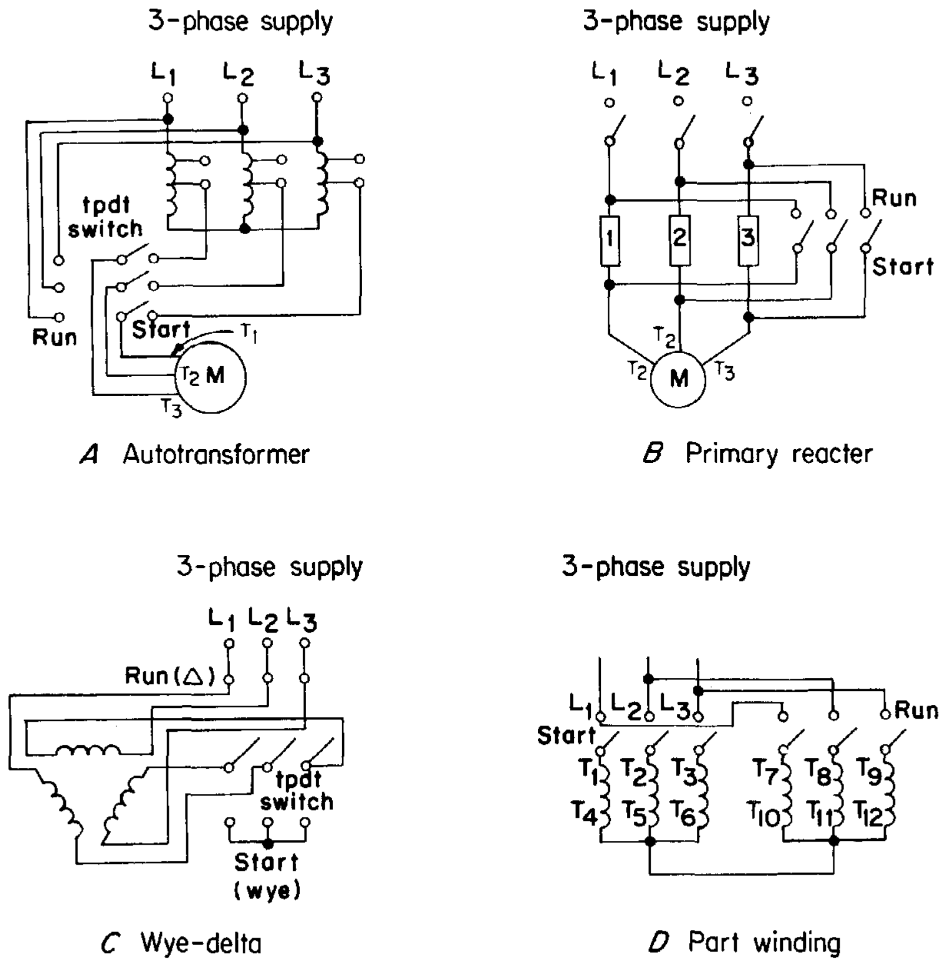


Figure 6.30.—Starting methods for induction motors.

where the horsepower limit for full-voltage starting is perhaps as low as 50 hp. The wound-rotor motors described in the next section provide an alternative.

WOUND-ROTOR INDUCTION MOTORS

As mentioned earlier, the starting and running characteristics of an induction motor may be adjusted by varying the resistance-to-reactance (R/X) ratio of the rotor conductors. Instead of rotor bars and end rings, the wound-rotor motor has insulated windings much like the stator, with the same number of poles and windings placed in the rotor slots. The windings are usually connected in wye with the ends connected to three slip rings mounted on the rotor shaft. The brush and slip-ring circuit is completed through a wye-connected set of variable resistances, as shown in figure 6.31. Thus, the external resistance can be used to vary the speed-torque characteristics by changing the rotor R/X ratio. The stator of the motor is the same as for a squirrel-cage machine.

A typical family of wound-rotor motor characteristics is illustrated in figure 6.32 (*11*). As external resistance is increased, the starting current is decreased and starting torque is increased. For a given shaft load, the reduction in rotor current will result in a speed decrease. Thus when starting a wound-rotor motor, a maximum resistance is inserted in the rotor circuit (R_9 curve). As the rotor accelerates, the resistance is reduced until the desired speed is obtained, or if full speed is required, the resistance is brought to zero (R_1 curve). Therefore the wound-rotor motor can be considered a variable-speed machine. Thermal considerations do place a lower speed limit on it, and for self-ventilated motors, continuous rated torque operation below 70% of rated full speed is not recommended (*15*). This lower limit may be reduced to 50% if the motor load is 40% of rated.

Applications for wound-rotor motors include loads that require constant-torque, variable-speed drives or for which a sequence of slow-speed steps is needed to limit motor current during acceleration, such as for high-inertia or high-torque loads. Since they are suited to high-torque loads. Since they are suited to high-torque loads, these motors have found extensive used in the mining industry to operate crushers, grinders, ball and roller mills, conveyor belt drives, and hoists.

The automatic starting method for these motors uses definite-time acceleration where a series of fixed resistances are shorted out one at a time on a predetermined schedule (*12*). This step starter is shown in a simplified schematic in figure 6.33. When the starting sequence is initiated, all resistors are in series with the rotor winding; then the relay

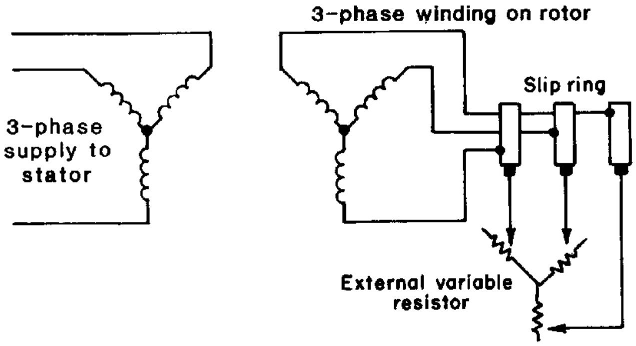


Figure 6.31.—Schematic of wound-rotor induction motor showing external resistance controller.

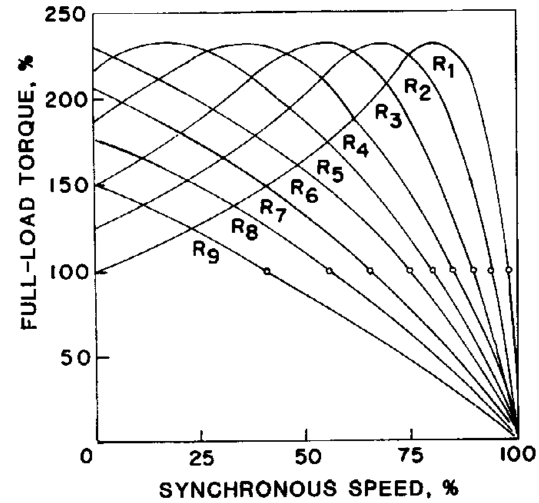


Figure 6.32.—Torque-speed characteristics for wound-rotor motor with stepped-resistance controller.

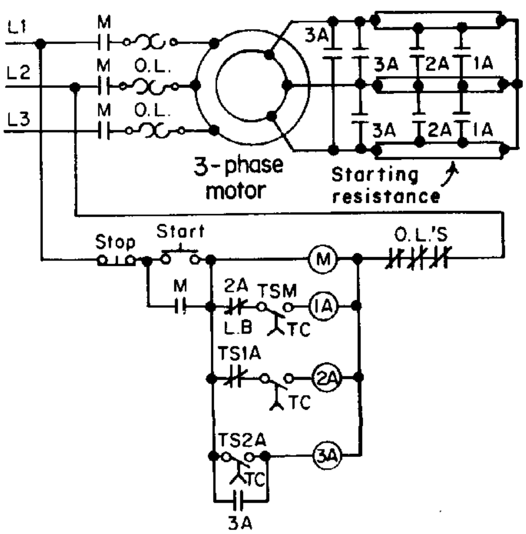


Figure 6.33.—Simplified step starter using individually timed magnetic relays.

contacts 1A, 2A, and 3A are sequentially closed, resulting in four speed-torque characteristics. The last effectively shorts out the rotor winding. Since the sequence proceeds regardless of motor speed, the method requires close coordination with motor characteristics (*15*). The actual operation of the relays is discussed in chapter 9.

Whether started automatically or manually, the wound-rotor motor continues to find application for the functions previously mentioned. However, for conveyor belt drives, these motors are now tending to be displaced by squirrel-cage induction motors equipped with solid-state starters (see chapter 14). Reasons for this change involve maintenance problems and a desire to eliminate the failures inherent with brushes, slip rings, and relay contacts.

THREE-PHASE SYNCHRONOUS MOTORS

The three-phase synchronous motor has a stator and rotor and is similar to the induction motor. The stator and stator winding have the same basic construction and purpose: to receive the power to drive a load (*15*). However, in this motor, the rotor consists of field poles connected in series, parallel, or series-parallel combinations and terminated at slip rings. The field windings are excited by an external dc source, the exciter. The number of field-winding poles equals the number of magnetic poles present in the stator. A sketch of a typical large synchronous motor is shown in figure 6.34 (*12*).

Rotor field excitation is often supplied from a small dc generator mounted on the same rotor shaft, as diagrammed in figure 6.35. Alternatively, dc supply can be obtained from a three-phase full-wave bridge rectifier, as illustrated in figure 6.36, or by a separate m–g set.

Pure synchronous motors are not self-starting and are generally accelerated in the same manner as inductor motors. Salient-pole rotors commonly have a squirrel-cage winding (fig. 6.34) to produce the necessary induction motor action. Low-speed cylindrical rotors closely resemble a wound-rotor induction motor, but with five slip rings

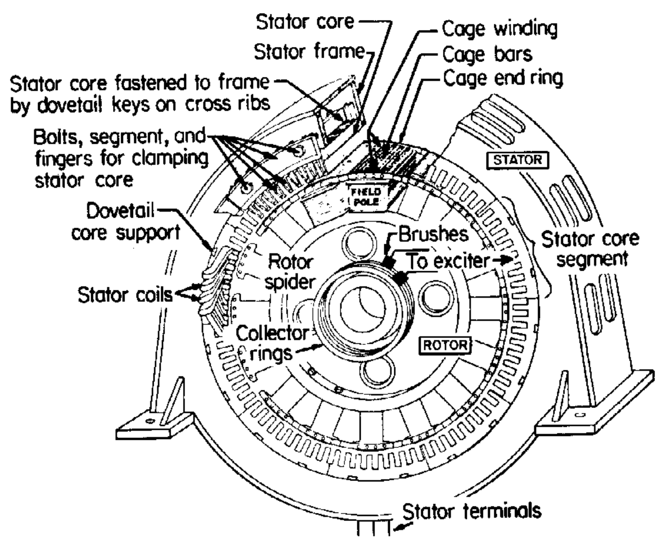


Figure 6.34.—Sketch showing construction of salient-pole synchronous motor.

(*15*). As figure 6.37 illustrates, three rings are used for a wound-rotor circuit, the other two for the dc field (*8*). These cylindrical-rotor motors can provide high starting torque to accelerate high-inertia loads. The use of squirrel-cage windings, however, is intended only to accelerate the motor rotor, not specifically to develop induction-motor torque to external loads. Some large synchronous motors are accelerated by a small induction motor mounted on the synchronous-motor shaft (*12*). The induction motor must have fewer stator poles than the synchronous motor in order to reach the required speed.

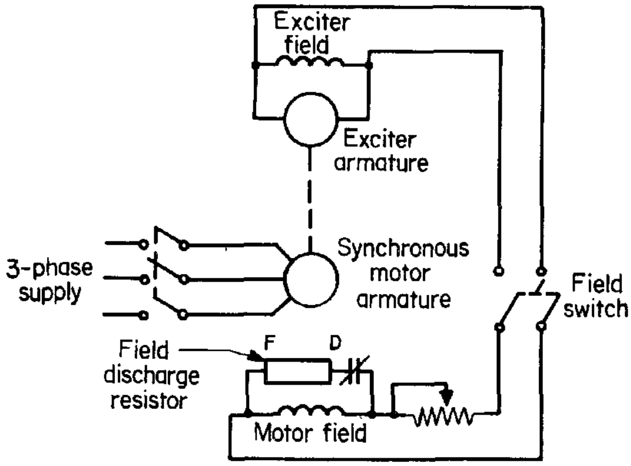


Figure 6.35.—Simplified diagram of synchronous motor using generator for field excitation.

Synchronous-Motor Starting

Figure 6.38 demonstrates the general method of starting a synchronous motor (*12*). Pressing the start button energizes the CR relay, which in turn closes the CR contacts. One set of contacts electrically locks in the start sequence (which can be terminated by pressing the stop button), and the other set energizes the M relay. The M contacts close, and three-phase power is applied to the stator winding. This allows the machine to accelerate as an induction motor. In the simplest procedure for the motor shown in figure 6.34, the motor is allowed to accelerate to the maximum induction speed, where the slip between the stator rotating field and the rotor is very small. A switch is then closed manually to apply dc to the rotor field winding (figs. 6.35–6.37). A steady rotor magnetic field is thus established that can lock in step with the rotating field of the stator. Thus, the rotor will turn at synchronous speed, which gives the motor its name. However, if dc is applied before maximum induction speed is

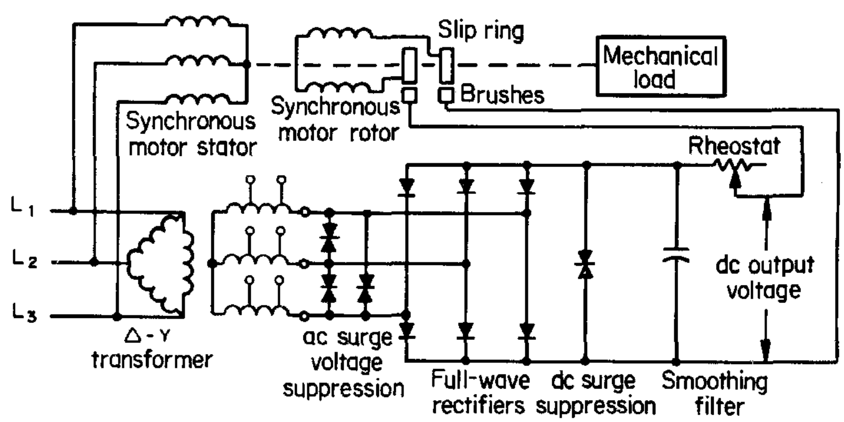


Figure 6.36.—External solid-state supply used to provide field excitation.

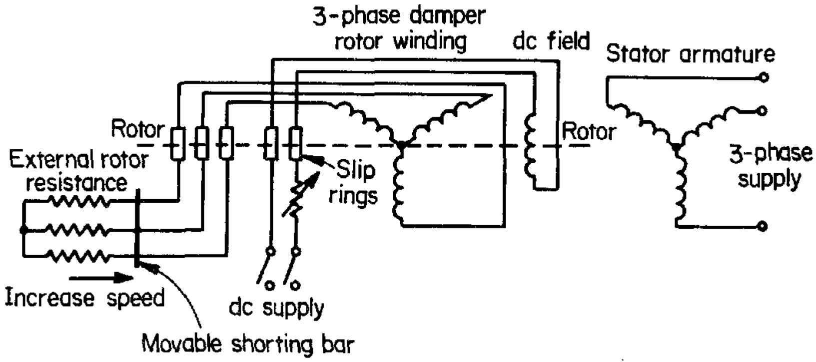


Figure 6.37.—Schematic of low-speed cylindrical-rotor synchronous motor.

achieved, the rotor may not pull into synchronization and severe vibration can occur, caused by repulsion every time a rotating pole passes a stator pole. As a result, most synchronous-motor starters do not rely on manual control but instead automatically excite the rotor field at the appropriate time.

An approach widely used for automatic starting is synchronization based on frequency (*12*). This technique uses the voltage induced in the field winding during acceleration and before the dc is applied (*1*). Again referring to figure 6.38, a resistor (R) and inductor (X) are placed across the field winding, with relay FR across the inductor. The inductance of the relay coil is selected to be much lower than that of X. Immediately after starting commences, a high-frequency potential is induced in the field winding, and the majority of current flows through the resistor and the relay coil because the inductance, X, exhibits high reactance. The FR relay opens the FR contacts faster than the interlock contact M_b of relay M closes. As the motor accelerates, the frequency of the induced current decreases. When close to synchronous speed, the frequency has decreased to the point where most current flows through the inductor, and the voltage is reduced to the point where the FR relay cannot hold its contacts open. Consequently the FR contacts close and energize relay FS. The FS contacts then close to apply dc excitation to the field winding and remove the resistor from the circuit.

Synchronous-Motor Torque

Under load, the synchronous motor behaves much like a nonslip direct magnetic coupling. The rotor does not develop induction-motor torque; it is magnetically locked to the stator rotating field and is pulled around at basically the same speed. The torque developed is dependent on the hold-in pole strength. Hence, the lock-in torque may be increased by simply increasing the dc supplied to the rotor field winding. If there is a load change, an instantaneous speed change occurs but only for a few cycles, after which the rotor again attains synchronism (*12*). The use of a squirrel-cage rotor winding also helps to dampen out speed changes, and it is therefore often called a damper winding.

A typical speed-torque characteristic for a synchronous motor containing a damper winding is shown in figure 6.39. Because the synchronous-motor portion cannot start itself, the starting torque comes from the damper winding. When the external loading does not exceed the pull-in torque value, the motor can be started and accelerated to synchronous speed. However, if no significant external load exists and then a load equal to the pull-in torque is applied, the rotor will momentarily drop to about 95% of full speed and then regain synchronism. During the loading and the momentary drop in speed, the rotor assumes a new position and continues to rotate at synchronous speed but a few degrees behind the no-load position (α in figure 6.40) (*9*). This sequence can occur for any applied

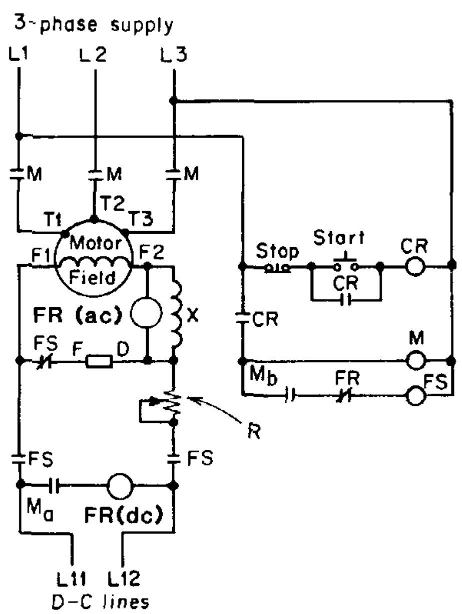


Figure 6.38.—Controller used to demonstrate general starting method for synchronous motor.

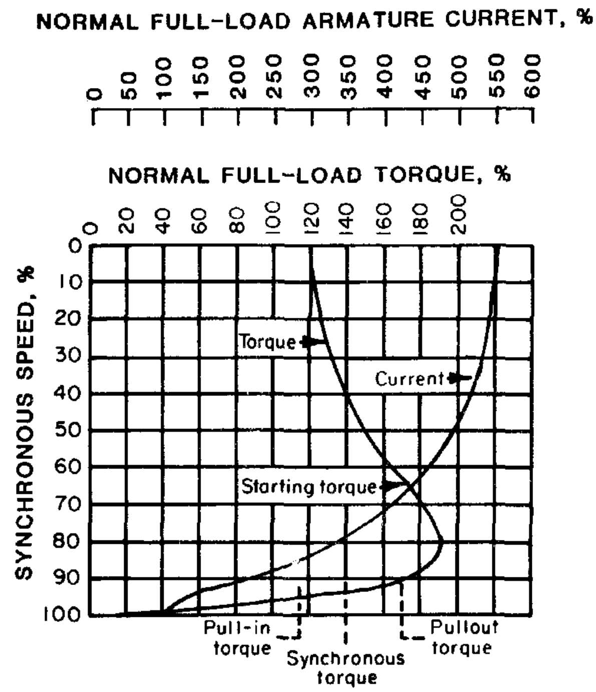


Figure 6.39.—Typical torque-speed characteristic for synchronous motor with damper winding.

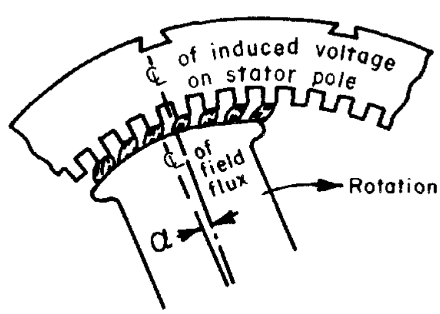


Figure 6.40.—Effect of load on rotor position.

load up to the synchronous torque level, above which the restoration of synchronous speed is questionable. If the load requirements exceed the pullout torque, the motor loses synchronism, average torque drops to zero, and the motor stops (*12*).

Generated Voltage

After excitation has been applied to the field winding, the revolving magnetic field of the rotor cuts the stator conductors and induces a voltage in opposition to the applied voltage. Figure 6.41*A* shows an equivalent per-phase circuit of a synchronous motor that demonstrates the effect of the generated voltage on the electrical performance of the machine. From Kirchhoff's voltage law (*11*),

$$\bar{V}_T = \bar{V}_c + \bar{I}R_e + j\bar{I}X_L = \bar{V}_c + \bar{V}_{XR}, \qquad (6.11)$$

where $\bar{I}R_e$ = voltage drop due to effective resistance of armature (stator) winding, V,

$\bar{I}X_L$ = voltage drop due to inductive reactance of armature winding, V,

$\bar{V}_T$ = voltage supplied to motor, V,

and $\bar{V}_c$ = generated voltage produced by rotor field winding modified by armature reaction, V.

Two important phenomena connected with synchronous machines (and some others as well) are immediately evident in the equation. Under dynamic loading conditions, if the load delivers a torque to the motor shaft, the rotor produces a generated voltage that is greater than that of the supply, and power is delivered back or regenerated into the line. Secondly, if the supply voltage is then removed, the load acts as a prime mover, and $\bar{V}_c$ will be generated as long as field excitation exists and until the load dissipates its energy. This last phenomenon is especially important when the supply voltage is lost because of a short circuit, since the synchronous motor can deliver significant current to the malfunction (see chapter 10 for further information).

Power Factor

Another important effect results from the generated voltage. In an ideal synchronous motor under no-load conditions, $\bar{V}_c$ can be equal in magnitude and frequency to $\bar{V}_t$ but 180° out of phase. Hence, with this ideal situation, the motor does not draw current ($\bar{I}$). Obviously, practical motors have such losses as windage and friction, which cause a small shift in angular position, α, between the rotor and the rotating magnetic field. Here, the phasors $\bar{V}_c$ and $\bar{V}_T$ are no longer opposite in position, since $\bar{V}_c$ is shifted clockwise by α as shown in figure 6.41*B*. The change causes the motor to draw line current ($\bar{I}$) to maintain the rotor in synchronism with the stator flux. Under heavier loading, α increases and the motor draws more current. Note that the rotor field actually does no work, and the dc energy supplied to maintain the field is dissipated as a small I^2R heat loss.

A change in the rotor field strength, say by adjusting the resistance in figures 6.35, 6.36, or 6.37, changes the magnitude of $\bar{V}_c$ but not its angular position. The difference between $\bar{V}_T$ and $\bar{V}_c$, or $\bar{V}_{XR}$ in figure 6.41, determines the angular position of motor current. When $\bar{V}_c$ is adjusted to produce a motor current that lags applied voltage (fig. 6.41*B*), the motor is said to be *underexcited*. However, increasing the dc field strength with the same shaft load can shift $\bar{V}_{XR}$ such that the reactive component of current will change from a lagging phase angle to leading (fig. 6.41*C*). In this condition, the rotor field is termed *overexcited*, and the motor appears as a capacitive load. The leading power factor is one of the most outstanding features of a synchronous motor, as it can be used for power-factor correction. The ability to operate at unity power factor should be obvious; the field winding is referred to as *normally excited* in this case.

Because the phase angle of operation depends upon both field excitation and motor load (angle α), the characteristics of synchronous motors are often represented graphically in a form called V-curves. Illustrated in figure 6.42, they allow the selection of a field-excitation current for a load to produce a desired power factor (*8*). The lines drawn to show equal power factor are termed *compounding curves* (*12*).

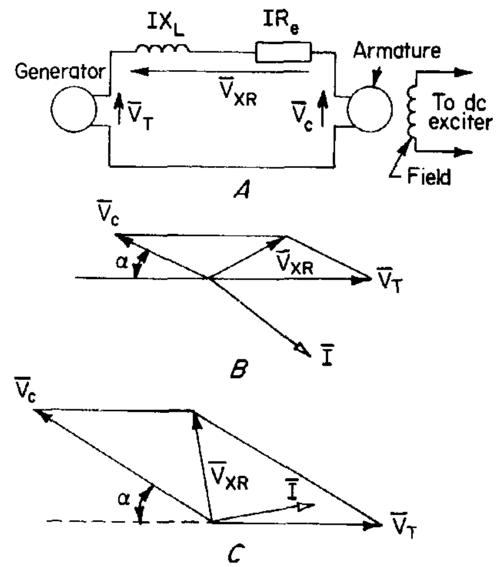


Figure 6.41.—Equivalent per-phase circuit of a synchronous motor (*A*) and phasor diagrams for (*B*) underexcited and (*C*) overexcited field winding.

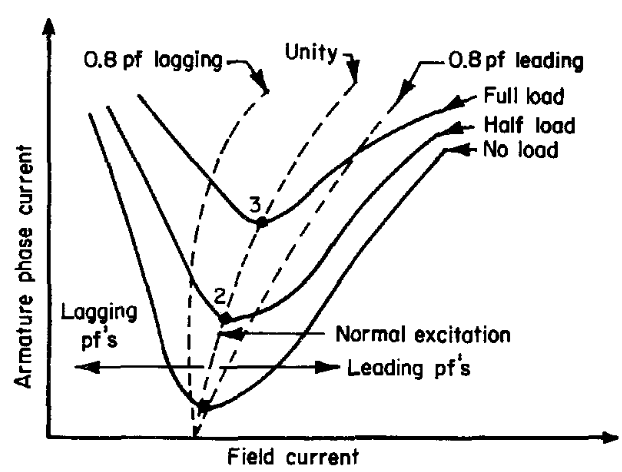


Figure 6.42.—V-curves for synchronous motor.

Applications

In the past, wide use was made of synchronous motors in the mining industry to take advantage of their constant speed and available leading power factors. Applications included ventilation fans, pumps, compressors, grinders, mills, and drive motors on m–g sets to provide power for dc equipment. However, static capacitors have now replaced motors for power-factor improvement in almost all situations because of their flexibility and ease of installation, and silicon rectifiers have supplanted m–g sets for power conversion. Nevertheless, one very important use of synchronous motors remains today: as the main drive motor in surface excavators. Here, one or more motors directly drive dc generators, which in turn power the dc motors serving the various functions on the machine. Figure 6.43 provides a plan view of a typical mining shovel where one synchronous motor drives three dc generators and the motor exciter (*10*). This subject will be continued at the end of the presentation on dc motors.

DIRECT CURRENT MOTORS

The dc motor is the most versatile of all electrical machinery. On advantage over all the preceding motors is that its speed may be easily adjusted. The dc motors accounted for the majority of motors within the mine until the 1940's, when ac distribution systems started to replace dc. Induction machines then substituted dc motors, mainly because the ac-to-dc conversion of reasonably large power quantities was cumbersome. However, dc motors continued to hold prominence for some specific loads. In recent years, because ac-to-dc conversion is now very easy, dc motors have replaced some of their induction counterparts. The reasons behind the extensive use of dc motors in mining will become apparent in the following paragraphs.

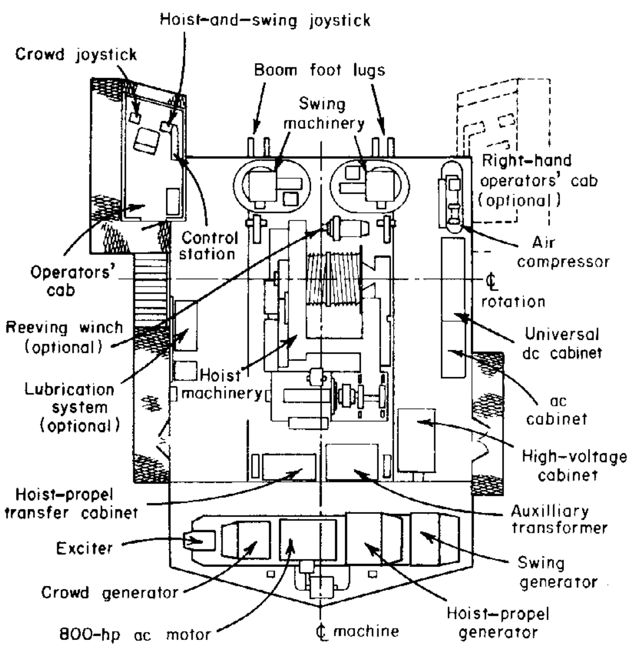


Figure 6.43.—Plan view of typical mining shovel showing m-g set.

Elementary Motor

Figure 6.44 illustrates an elementary two-pole dc motor with a one-coil armature. With the armature current flow as shown, the reaction of the armature magnetic field to that of the main field produces forces on conductors A and B, and the torque results in clockwise rotation. The commutator acts as a switch to reverse the armature current each time the conductors pass the neutral plane. To reverse armature rotation, the armature current flow is simply reversed.

The two-pole motor is rather impractical. Torque is maximum when the plane of the armature conductor is parallel to the plane of the field, zero when at right angles. Figure 6.45 shows a four-pole armature with a four-segment commutator, but still with a two-pole main field. Here, motor torque does not drop to zero because an armature conductor is always under the magnetic influence of the main field. Actual dc industrial motors have many commutator segments, armature conductors, and

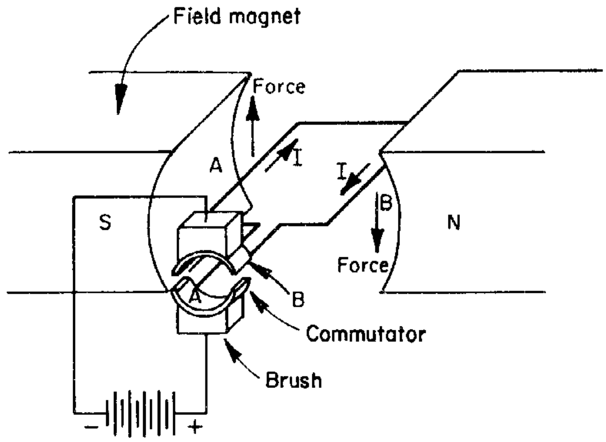


Figure 6.44.—Elementary two-pole dc motor.

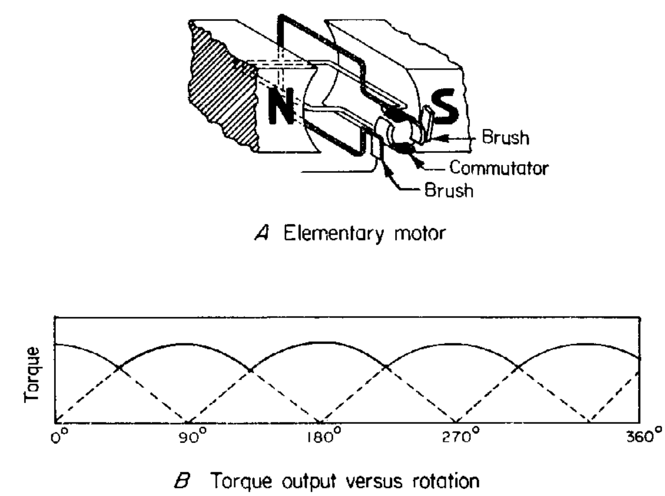


Figure 6.45.—Elementary four-pole dc motor.

main field poles. The result is nearly constant torque output.

As with the synchronous motor, the dc motor field does not do useful work. It merely provides the necessary medium for the armature windings to push against when developing rotary motion. In all but the very smallest machines, the field is supplied by dc through field windings. The energy expended in these windings forms an I^2R heat loss.

Actual Motor Construction

The essential motor parts are the armature (rotor), the commutator, and the main field frame and windings (stator). The armature is constructed of steel laminations pressed onto the shaft, with slots parallel to the shaft. The armature windings are placed in the slots and connected to the segments of the commutator, which is located at one shaft end. Carbon brushes, mounted on but insulated from the motor frame or one end bell, rise on the commutator segments. The main field windings surround laminated pole pieces that are bolted around the periphery of the motor frame.

Interpoles (or commutating poles) are mounted between the field poles (fig. 6.46), and the windings are connected in series with the armature (*12*). Their purpose is to improve commutation by opposing armature reaction, the distortion of the main magnetic field by the rotating armature field (fig. 6.47). The interpole windings produce a small magnetic field that opposes the main field in the same plane as the brushes. This reduces the magnetic field that is cut by the armature conductors undergoing

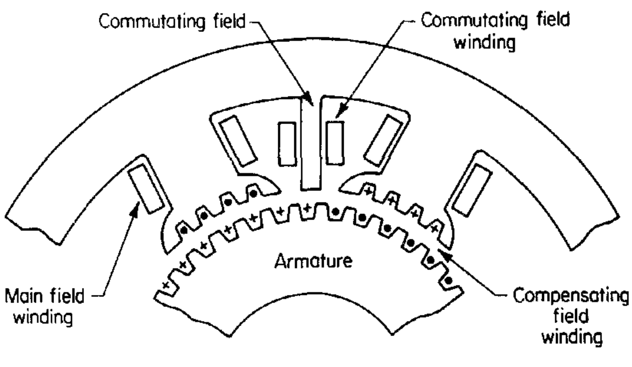


Figure 6.46.—Cross-sectional sketch of dc motor showing interpole and compensating windings.

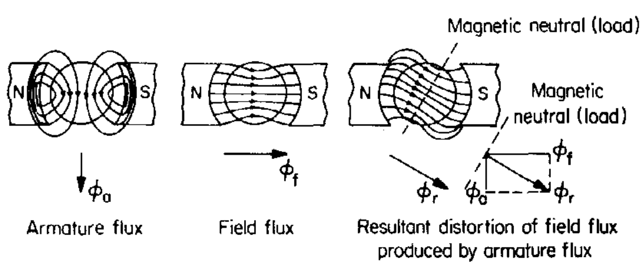


Figure 6.47.—Interaction between armature and main-field flux to produce main-field distortion.

commutation (current reversal) and thus reduces brush sparking (*3*).

The armature reaction can be further neutralized through the use of compensating or stablizing windings (*12*), which are placed in slots on the ends of the main field poles next to the armature (fig. 6.46) and are again connected in series with the armature. These windings are especially useful in motors intended for variable-speed or reversing operation. Without the compensation, armature reaction from large loads can neutralize the main field flux (*12*).

Although interpole and compensating windings serve valuable functions in dc motor operations, the machines can work without them. As they can obscure the presentation of motor operation, these windings will not be included in the following discussion.

Torque

The torque developed by any electric motor is a measure of its ability to pull against a load. In dc motors, torque is a function of armature current and the magnetic flux density of the main field or (*8*)

$$T = K\Phi I_a, \tag{6.12}$$

where T = motor torque, N·m (times 0.738 = lb·ft),
Φ = magnetic flux per main field pole linking the conductors, Wb,
I_a = total armature current, A,
and K = a proportionality constant, N·m to WbA,

and where
$$K = \frac{T_{rated}}{(I_{A\ rated})(\Phi_{rated})},$$

where T_{rated}, $I_{A\ rated}$, Φ_{rated} = rated value for torque, armature current, and flux for motor, respectively.

The above equation can be used to find the torque output from a machine, if the rated torque and the changes in armature current and the field flux are known.

Motor Connections and Performance

Exactly like dc generators, dc motors can be connected as separately excited, shunt, series, and compound. These connections are shown in figure 6.48. The performance of a separately excited motor is similar to that of the shunt, and its importance in mining applications is primarily with regard to motor control; thus, this motor will be discussed later. The speed-torque characteristics of shunt, series, and compound motors can change drastically depending on the connection. Typical curves are illustrated in figure 6.49 (*8*).

Shunt Motors

The shunt motor has the main field winding connected in parallel with the armature (fig. 6.48). Since the field winding is connected across the supply, its resistance must be rather high, but because of space constraints the armature windings have a much smaller resistance. When the motor is energized, armature current, I_a, is limited only by its winding resistance and is thus much higher

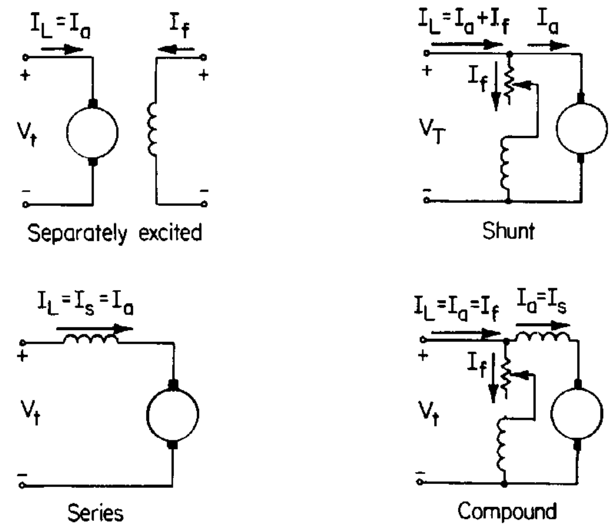


Figure 6.48.—Four connections for dc motors.

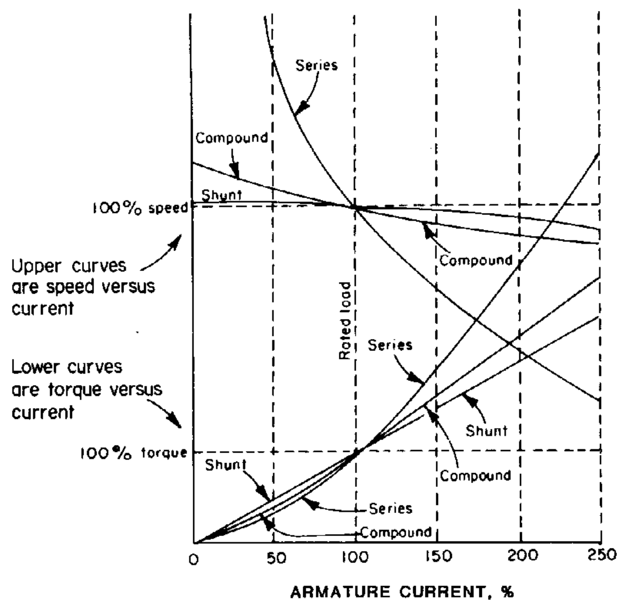


Figure 6.49.—Typical characteristics for shunt, series, and compound motors of equal horsepower and speed ratings.

than field-winding current, I_f. However, as soon as the armature starts to rotate, its conductors cut the main field magnetic flux and a counterelectromotive force (cemf) is generated in the windings. This cemf opposes the applied armature voltage and begins to limit armature current. The opposition to current flow can be seen by applying Kirchhoff's voltage law:

$$V_T = V_c + I_a R_a, \tag{6.13}$$

where V_T = supply voltage, V,
V_c = cemf induced in armature winding, V,
and I_a, R_a = armature current, A, and resistance, Ω.

The cemf is proportional to the speed of the armature, or

$$V_c \propto n\Phi, \tag{6.14}$$

where n = armature speed, r/min,
and Φ = magnetic flux per main field pole, Wb.

As the armature accelerates, the cemf rises, and the armature current drops. Yet, according to equation 6.12, motor torque decreases. A final speed is reached when the cemf is almost equal in magnitude to the supply voltage.

If the motor is unloaded, the difference between the terminal voltage and the cemf will allow only enough armature current to overcome friction, winding, and core losses. Under motor loading, the armature slows down, cemf decreases, and more current enters the armature. However as shown in figure 6.49, the speed of the shunt motor remains relatively constant from no-load conditions up to 100% rated and slightly beyond. The speed can be easily adjusted by changing a resistance in series with the field winding. From equation 6.14, weakening the field flux by decreasing field current increases motor speed. Yet, for a constant field flux, torque varies linearly with armature current (that is, $T \propto I_a$).

If across-the-line starting was attempted with the shunt motor shown in figure 6.48, the cemf would probably not build up fast enough to limit armature current to a safe value, and hence, damage to the commutator, brushes, and the armature winding could result. For this reason, a starting resistance is used in series with the armature (fig. 6.50A) for all dc motors except those of fractional horsepowers. The resistance is usually selected to limit armature current from 150% to 250% of rated current depending on the starting torque required. The shunt winding is always connected across full line voltage when starting so less armature current is needed to develop the rated torque.

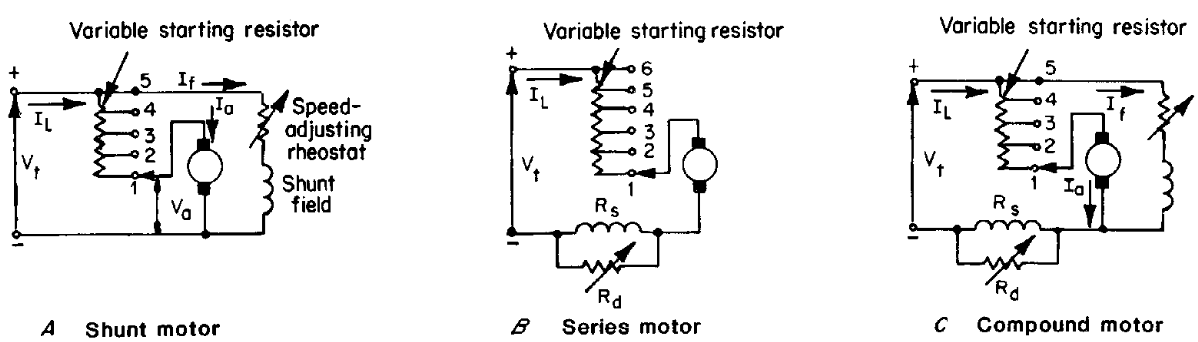


Figure 6.50.—Simplified dc motor schematics with starting resistances.

In mining, manual controllers are found on many dc machines. These are available in three general forms: faceplate, multiple-switch, and drum controllers. Schematics for these are shown in figures 6.51, 6.52, and 6.53.

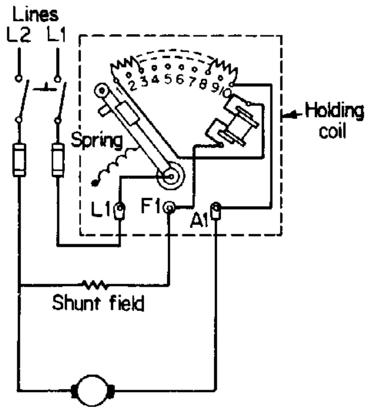


Figure 6.51.—Faceplate manual starter.

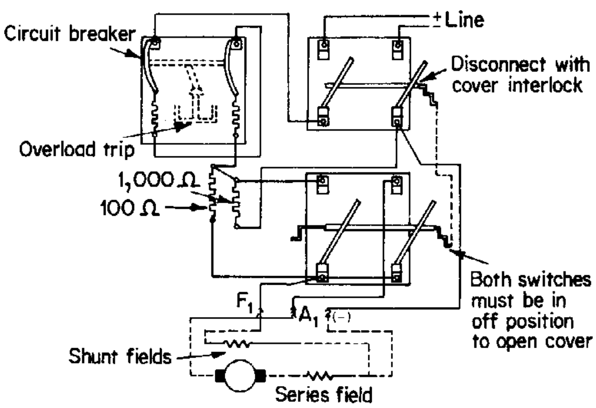


Figure 6.52.—Multiple-switch starting.

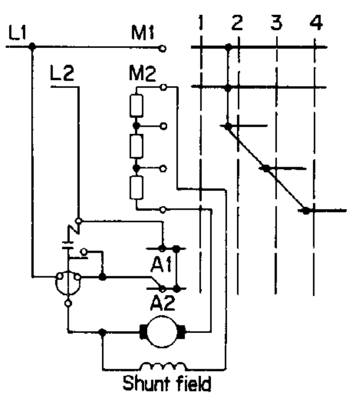


Figure 6.53.—Drum-type starter.

The faceplate starter is often used with small stationary dc motors. The level is advanced (to the right) in steps, momentarily stopping at each position to allow the motor to accelerate, until the resistance is removed. A holding coil then maintains the lever in the last position. A spring is used to return the lever to the off position during a power failure or if the lever is left in an intermediate position.

One method of multiple-switch starting, shown in figure 6.52, uses two double-pole, single-throw switches. The upper switch is closed first, energizing the shunt field through a 100-Ω resistor. This allows the main field flux to build up to some extent before the armature is connected to the line. Initial inrush current is thus reduced, which helps to prevent brush arcing and the possibility of commutator flashover. The lower switch is then closed, energizing the armature through a second resistor, and the motor accelerates. The armature resistor remains connected during running. The two line switches are mechanically interlocked so the upper switch must always be closed first. A variation of this technique is to use relay contacts or contactors to supply main field excitation, insert the starting resistance, then bypass the resistance.

Drum controllers (fig. 6.53) are frequently used on mine locomotives but are also found on some dc mining machines. A handle-controlled rotary shaft is connected to the switch segments indicated by dark lines in the figure. These segments are of various lengths so contact with the stationary contacts can be made at different intervals. When starting, the M1 and M2 contacts engage first, energizing the shunt field and inserting all resistors in series with the armature. The resistors are then removed one at a time by advancing the controller. Although not shown, an additional drum or reversing controller is usually available to reverse armature current and thus motor direction.

The use of fixed resistance starting has widespread application in mining. Here the starting resistance remains in series with the armature for running. An instance would be a small dc motor, such as a pump in a remote location. The resistance gives poor speed regulation, but the motor can be started unattended.

Dynamic Braking

If a shunt motor is running under load and the armature circuit is opened, the inertia of that load will drive the machine as a dc generator. Dynamic braking simply connects a resistance across the armature to dissipate the available energy and decelerate the load (fig. 6.54). The braking action is most effective at high armature speeds, becoming negligible at low speeds. The value of resistance, R, is selected from (*12*)

$$R = \frac{V_c - I_a R_a}{I}, \qquad (6.15)$$

where $V_c - I_a R_a$ = armature voltage at start of braking, V,

and I = dynamic braking current, depending upon desired braking level, A.

The normal value for I is 150% of rated motor current but I may be as high as 300% for quick stopping.

Series Motor

The armature and main field winding are connected in series and both carry load current in a series motor. The magnetic flux, Φ, now produced in the main field winding, is proportional to the armature current. Thus, motor torque varies as the square of armature current ($T \propto I^2_a$). Furthermore, the main field strength will change with load, causing a speed decrease with increased load.

When a series motor is started, cemf builds up as the armature speed increases. During the initial acceleration, the cemf is small, armature and field current are high, and the torque is very high. When the curves in figure 6.49 are compared with the material presented earlier, it can be seen that the starting torque of the series-wound motor is higher than that of any other motor type. Because of this, it is often said that the series machine has the best traction or starting characteristics. Thus, it is the most used motor for traction purposes in mining; examples include locomotives, shuttle cars, and diesel-electric trucks. A problem with this motor, however, is that at light loads, motor speeds may become excessively high; therefore, series motors must be directly connected to loads that cannot be removed freely. Otherwise, the motor may race to destruction.

The method for starting the series motor is similar to that for shunt machines. The arrangement of the starting resistance is shown in figure 6.50*B*. A direct application of contactor-controlled multiswitch starting of a traction motor in a mining machine is illustrated in figure 6.55 (*7*). After the power-source contacts (M1) close, the motor is accelerated with both resistances in series with the armature. The same control that activated the M1 contacts simultaneously energizes a definite-time relay. After a preset time (about 1.0 s), the relay closes its contacts, and that in turn energizes the M6 contactor, which shunts its starting resistor. The M7 contacts can be used to provide a second step before full speed is obtained or can be used to enable two-speed operation of the motor. In the latter case, control circuitry is arranged so that the M7 contacts cannot close before the M6 contacts. Figure 6.56 shows a one-step starting arrangement with the addition of forward-reverse control (contacts 1F and 2F close for forward, 1R and 2R for reverse) (*7*).

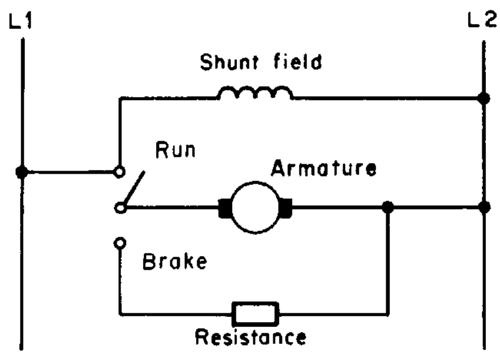


Figure 6.54.—Simplified diagram of dynamic braking applied to shunt motor.

The procedure for dynamic braking is identical to that already described, with the exception of excitation for the main field. The simplified circuit in figure 6.57 is one approach and shows the switches closed for motoring (*12*). Upon dynamic braking, the switches place the armature, series field, and braking resistance in a loop circuit. The series-field connections are reversed to maintain current flow in the same direction.

Compound Motors

Compound motors have both shunt and series field windings installed on the same poles. The series winding may be differentially or cumulatively compounded, that is, subtracting from or adding to the magnetizing force of the shunt field. This causes either reduced or increased armature speed with load. Only the cumulative compound motor characteristics are shown in figure 6.49.

Cumulative compounding gives greater torque than is possible with the simple shunt motor, because of the greater amount of main field flux available (*8*). The increased flux, however, causes the speed to drop off more rapidly than for a shunt motor, but not as much as for a series motor. Therefore, the cumulative compound motor will develop a high torque with any sudden increase in load, but at light loads it will not run away because the shunt field provides a constant field flux. These motors are often applied to loads requiring high starting torque but fairly constant operating speed under normal conditions. Thus, cumulative compounding combines the characteristics of both series and shunt motors.

The differential compound motor produces torque that is always lower than that of the shunt motor (*8*). The amount of series winding can be adjusted to offset any drop in speed as loading increases, or it may be sufficient to give a slightly higher speed than normal at full load. A motor having constant speed from full load to no load is called flat compounded, while that with slightly higher speed than normal is called over compounded.

Again, armature current is traditionally limited by resistance when starting. Figure 6.50*C* shows the process in elementary form, and figure 6.58*A* illustrates an actual application for a mining machine hydraulic pump motor (*7*). It can be seen that the two circuits are almost identical. A fixed starting resistor is used for acceleration and as in figure 6.55, the resistor is shunted by the M3 contacts after a definite time period (usually 1.0 s). A semiautomatic variation of this scheme is illustrated in figure

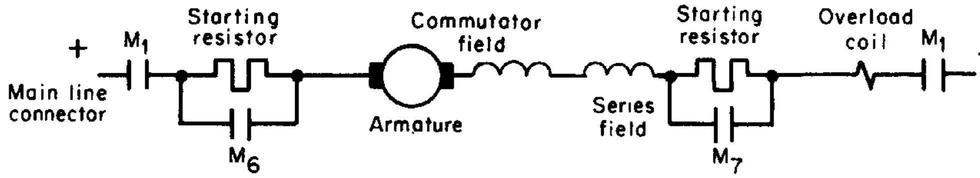


Figure 6.55.—Two-step resistance starting of series-wound motor.

6.58*B*: semi-automatic means that under certain conditions the starter requires some attention. The accelerating contacts (A) are as before, but the contactor coil is placed across the armature circuit. As cemf increases during acceleration, the voltage across the coil causes contact closure at the proper time.

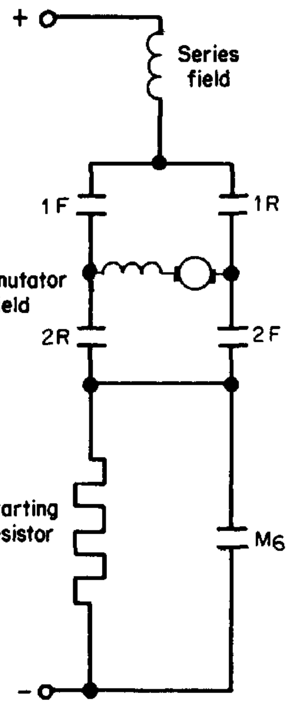


Figure 6.56.—Forward-reverse switching of series-wound motor.

Dynamic braking employs a resistance to dissipate energy generated in the armature, involving either the series field (fig. 6.57) or simply the armature itself (fig. 6.54).

Ward-Leonard System

For large-motor applications, the Ward-Leonard system provides one of the finest techniques for controlling motor speed over a wide range and in both rotational directions (*3*). Two specific examples where it is used are mine hoists and surface excavators (*2, 4*). Figure 6.59 illustrates the basic system. The dc generator is driven at constant speed, typically by a synchronous motor, but some systems employ induction-motor drives for smaller horsepower applications. The generator and motor field windings are separately excited (see exciter in figure 6.43), and the motor is excited with a constant field current.

Because the main field of the motor is constant, the speed is directly proportional to its armature cemf (V_m). The magnitude of V_m is directly dependent upon the generator output voltage (V_g) less i_aR_a, where i_a is the

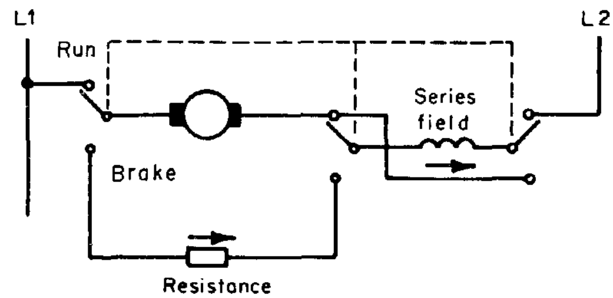


Figure 6.57.—Dynamic braking applied to series-wound motor.

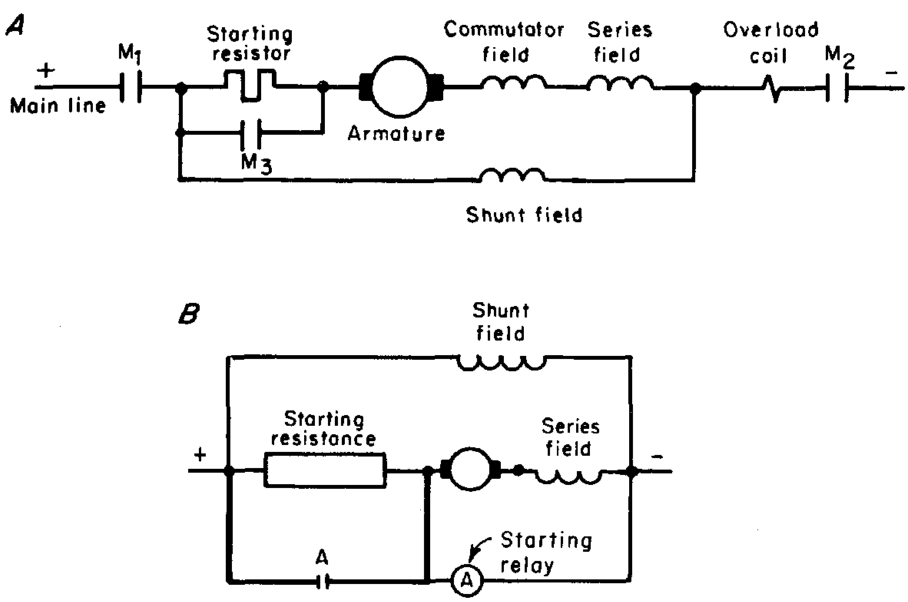


Figure 6.58.—One-step starting of compound-wound motor.

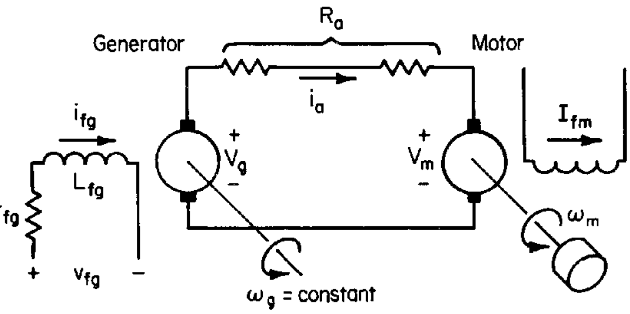


Figure 6.59.—Basic Ward-Leonard system.

armature current and R_a is the combined resistances of the generator and motor armatures. As a result, excellent control of all motor speeds and both acceleration and deceleration is obtained by adjusting the generator field strength. The generator field-winding resistance is high, and so the required level of control power is relatively low. Motor reversing is obtained by changing the current direction in the generator field. Braking is performed by reversing or reducing the generator field current.

MINE MOTORS

Many mining uses for industrial motors have been covered so far in the chapter; this section serves to clarify some additional applications, but mostly for underground mining equipment.

Applications

Mine motor functions can usually be divided into two groups: auxiliary and face (*17*). Auxiliary motors are employed for fans, pumps, conveyors, hoists, compressors, and other vital functions in mines aside from the actual process of mineral extraction. These operations commonly call for direct use of general-purpose industrial motors, and as their loads are often well defined and continuous, the motor characteristics covered so far are applicable. Face motors are associated with mining equipment, such as continuous miners, shuttle cars, loaders, roof bolters, and locomotives, where they are mounted in the machine. Their duty usually involves cyclic or random loading as well as the possibility of shock loading. The result is higher electrical and mechanical demands than those placed on equipment in other industrial applications.

The horsepower rating for a motor is based on the maximum winding temperature for continuous duty or intermittent duty. The temperature rise parameters have already been covered, but the meaning of a duty cycle has not. *Continuous duty* is quite obvious and refers to a substantially constant load (torque) over an indefinitely long period. *Intermittent duty*, however, means that loading is at alternate intervals of load and no load (motor running idle); load and off; or load, no load, then off (*9*). Each portion of the cycle is equal and the time interval is specified. In some cases, face motor intermittent duty is given a definite time interval of 15, 30, or 60 min, but it is often just listed as "mine duty" (*6, 17*). Tsivitse (*17*) states that a very successful horsepower rating for face motors has used both the continuous rating and the 60-min rating. The continuous duty is matched to the average or rms requirements of the load, and peak horsepower loading is limited to the 60-min value. The rms value for horsepower can be defined as

$$hp_{rms} = \left(\frac{\Sigma(hp_i)^2 \Delta t_i}{\Sigma \Delta t_i}\right)^{1/2}, \tag{6.16}$$

where hp_i = mean horsepower during time segment Δt_i,
and $\Sigma \Delta t_i$ = total time interval.

The ac motors in mining machines are normally four or six pole with synchronous speeds of 1,800 and 1,200 r/min, whereas dc motors often have comparable base speeds of 1,750 and 1,175 r/min (*6, 17*). These speeds are high enough to provide adequate horsepower but low enough to have reasonable reliability. Series-wound motors for traction are built to withstand rotation up to 6,000 rpm, such as might occur during maintenance.

Table 6.4 contains a listing of common applications for different motor designs to accommodate the various functions found in mining equipment (*5, 17*). Some additional information is warranted. The locked torque of traction motors is set so that the wheel or crawler-tractor treads will lose traction before the motor stalls. Ac motors that are mechanically paralleled, as for coal cutting with a continuous miner, are often sequence started with a delay to limit starting currents. Further, the high-slip characteristics mentioned in the table are for load sharing as well as to limit the rate of torque rise during shock loading. The dc motors used in load sharing often have matched speed-torque characteristics. Otherwise, the motors are compounded with a differential field that is in series with the armature of the second motor.

Table 6.4.—Common motors for mining equipment

Function	ac [1]	dc
Traction (direct coupled) ...	Design D	Series-wound.
Hydraulic pump	Design B or A	Compound or stabilized series. [2]
Conveyor	Design C	Compound.
Loading arm	([3])	Heavily compounded. [4]
Cutting	([3])	Do. [4]

[1] NEMA designs for induction machines.
[2] Speed regulation from no load to full load from 10% to 15%.
[3] Similar to NEMA design A but sometimes with higher locked-rotor and breakdown torque and higher slip per torque value.
[4] Speed regulation from no load to full load from 30% to 35%.

Actual Equipment Operation

Because mining equipment operates at the tail end of the distribution system, voltage drop becomes an important factor in the selection and utilization of motors. This is more critical for ac equipment than it is for dc, and this section will discuss some ramifications for two types of machines.

Continuous Miners

Of all electrical equipment used in U.S. underground coal mines, the continuous miner is the most concentrated simple load. This machine is the heart of present underground coal mining systems from both a production and electrical standpoint; hence, determining the load demands it makes on the system, or the load factor, is of great importance. The machine load factor can be defined as the ratio of actual power consumption to rated motor power. The rated power for the squirrel-cage induction motors used on ac continuous miners is set by the manufacturer for one motor or a combination of motors. The motors may or may not be built to NEMA standards. Regardless, torque and power are the only common ratings available to judge motor utilization. Horsepower is directly proportional to the product of motor speed and torque, and this power rating can be employed to determine three-phase motor performance. The load factor can be used not only to investigate the effective operation of a particular machine, but also to compare different equipment of a specific type, no matter what the rated power.

At low load conditions, motor efficiency and the power factor drop off rapidly. Since the motor functions on the steep portion of the power-factor curve, a small load variation will cause a relatively large motor current fluctuation. This can produce detrimental current peaks and stresses in trailing and feeder cables, particularly where conductors have marginal size. Poor power-factor operation requires correction capacitors or results in utility company penalties.

To analyze the power factor of continuous miners, a recent study (*13*) investigated the actual operation of 26 different ac continuous miners. These machines had utilization voltages of 440, 550, and 950 V, and total rated motor powers from 100 to 535 hp. All were operating in the Appalachian coalfield, with production ranging from 50 to 770 raw tons per shift. Average load factors were determined for each machine and particular attention was paid to the cutting-and-loading machine cycle because here power consumption is the most demanding. The average cutting load factors ranged from 0.26 to 1.17 and averaged 0.52 for the measured machines. It is significant that this average load factor is much less than the assumed design level of 0.85 that has been popularly used in the industry. When employing all hydraulic, mechanical, and electrical machine components, a 0.60 load factor might be considered satisfactory for the continuous miners studied. Hence, the implication was that many were being used inefficiently. However, drawing conclusions about machine efficiency and utilization based only on the load factor could be misleading because of the numerous factors involved.

Many of the low to moderately powered machines in the study had higher-than-average load factors, and some were considered to be totally adequate. During field measurements, close attention was paid to the performance of the machine operator, and in almost every case it was found that the operator was pushing his machine as much as possible during sumping (the cutting cycle), because traction approached full slip. From the standpoint of adequate employment, some continuous miners could be termed *overpowered*, particularly in the case of high-powered machines cutting friable coal.

When a high-horsepower machine (500 hp or more) is used on a low-voltage system, the demand for large current can create considerable trailing-cable voltage drops and a machine voltage condition that not only hampers operation but reduces the safety levels of the system on which it operates, perhaps by causing a poor power factor or substantially reducing motor torque capabilities. In the study, good machine voltage conditions (that is, as close to $\pm 10\%$ as practical) almost invariably resulted in good motor load factors. To obtain the favorable operation required not only good voltage but a strong utilization system, that is, using the largest practical trailing-cable conductors and shortest practical trailing-cable lengths. When distribution voltage regulation was bad, poor machine load factors also occurred. The situation was most evident on 4,160-V distribution systems that had been extended beyond their limits. More information on these subjects is presented in chapters 8, 12, and 13.

Traction Locomotives

A specific case study that was associated with the preceding work involved measurements on main-line traction locomotives. The results of the study can be applied to all series-wound motor traction. The company involved was experiencing numerous motor armature failures on their locomotives—up to 47 in 1 yr. Two avenues were explored to determine the problem: inspection of the failed motors and electrical measurements on a typical operating machine. Examination of the motors showed that the commutators were heavily pitted and charred, which was a direct indication of overloading. Subsequent electrical measurements substantiated this suspicion. The locomotive apparently experienced severe continuous stress that caused abnormally low motor voltage every time it encountered a particular curve located on a steep upgrade. When provided with this information, the mining company was able to reduce the locomotive trailing load, and the motor failures diminished.

In this case, the very low motor voltage provided the clue to identifying the problem. Unlike ac motors, dc series-wound motors can still operate under low voltage, although their control circuitry might not function properly. Here, the low voltage indicated high current because the trolley system was well maintained and had adequate capacity and properly spaced rectifiers.

There is another method that indicates if the motors within a vehicle are being overstressed while performing a specific duty cycle. Every manufacturer supplies characteristic curves for its mining equipment. The example in figure 6.60 shows motor characteristics for a small locomotive using two series-wound motors. The technique consists of finding the current needed by the job and using the characteristic curves to compare the needed value with the maximum value allowed per motor. The classical method employs rms current and makes the following assumptions:

- The vehicle operates under constant velocity while performing a specific function;
- The motors heat during acceleration and cool for deceleration; and
- The ampere rating for the motors stabilizes after 8 h of operation.

The method requires complete knowledge of the entire duty cycle, which in mining is the repetitive process that places individual demands on the machine. For instance, the track profile for a locomotive under loaded and then unloaded conditions can be divided into segments of equal demand, such as the grade for a specific haulage distance. For each portion of that duty, the torque or tractive effort demand must then be found. Stefanko (*16*) contains methods to calculate this input information.

The characteristic curves are then used to find the current demand and the actual time the machine operated at that current for each portion of the duty cycle. For instance, consider that the locomotive of figure 6.60 is operating on 600 ft of track with a +0.5% grade and has a tractive effort per motor of 1,020 lb. From the tractive effort curve, the current demand is 81 A, and using the speed curve, the machine speed is about 8.1 mi/h. By simple calculations it can be found that the locomotive would take 0.84 min drawing 81 A to move the 600 ft. The technique is continued until all times and currents are known for each duty cycle portion.

A problem occurs in the above procedure when the speed obtained from the curves is greater than that allowed. For example, assume that the next portion of the track profile has a length of 2,400 ft at a −0.5% grade, and the tractive effort for each motor has been found to be 340 lb. Referring to figure 6.60, current is 42 A, and speed is 12.5 mi/h. However, say that the maximum allowable speed is 10 mi/h. In this case, the locomotive would be commonly operated on-off, on-off, and so on, to maintain but not exceed 10 mi/h throughout the haulage portion. The time the motors are on and off can be calculated precisely by

1. Finding the time required to travel the distance at maximum allowed speed (2,400 ft at 10 mi/h yields 2.73 min),
2. Determining the time it would take at the speed found from the curve (2,400 ft at 12.5 mi/h gives 2.18 min), and
3. Subtracting the results of item 2 from 1 (0.55 min).

Item 2 provides the time the motors are on (42 A at 2.18 min), while item 3 gives the off time (zero current for 0.55 min).

With all currents and times known, including those times at zero current, the following equation provides the rms current demanded by the duty cycle:

$$I_{rms} = \left(\frac{\Sigma I_i^2 t_i}{\Sigma t_i}\right)^{1/2}, \qquad (6.17)$$

where I_i = current demand for each duty cycle portion, A,
t_i = time involved for respective current demand, min,
and I_{rms} = effective current demand for duty cycle, A.

For additional functions performed by haulage locomotives, the following factors can be assumed:

1. Switching operations have zero current demand but one-quarter of the actual time is applied in the above summation.
2. If the locomotive is used to load and unload its cars, the maximum tractive effort is employed for the loading process, the minimum for unloading. One-half the actual time involved for each is used for the effective time (t_i).
3. Delays are taken as zero current and zero time. Normal delays are assumed to not allow effective motor cooling.

Mine motors are presently standardized at a 90° C allowable temperature rise based on a 25° C ambient temperature (*17*). Older motors, as in figure 6.60, may have a 75° C temperature rise limit but are still based on

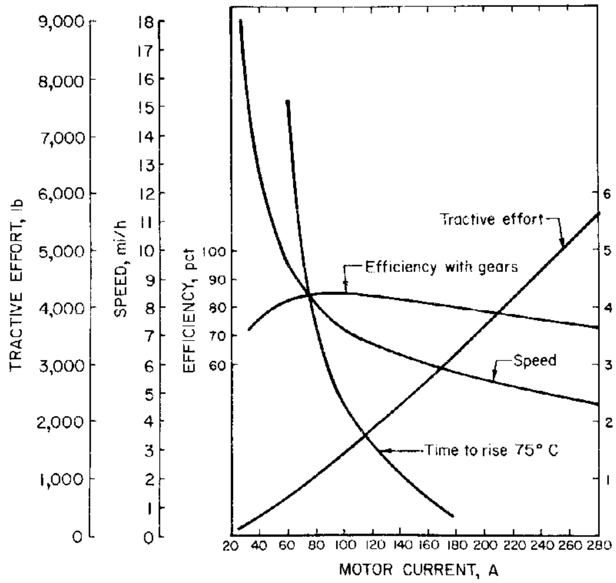


Figure 6.60.—Typical characteristic curves for each motor in traction locomotive (8-ton, 250-V motor, characteristic curves on 250 V; pinion, 13 teeth; gear, 69 teeth; wheel diameter, 29 in).

the 25° C ambient. The base temperature closely fits the typical conditions found in underground operations.

As mentioned earlier in the chapter, the allowable temperature rise is effective to elevations of 3,300 ft (1,000 m); above this, the allowed is reduced 1% for every 330 ft (100 m), or

$$\%\ \text{reduction} = \frac{\text{elev} - 3{,}300}{330}. \qquad (6.18)$$

In addition, for maximum ambient temperatures exceeding 25° C, the allowable rise must also be reduced by the difference above the base temperature. For example, a motor with 75° C temperature rise insulation, operating at 6,600 ft elevation in 30° C ambient temperature, has only a 62.5°C allowable temperature rise.

Consequently, if the locomotive is operating at 3,300 ft or less in a maximum ambient of 25° C or less, the characteristic curves can be used directly to find if the duty cycle demands exceed that allowed. In other words, the rms current found by equation 6.17 can be compared with the time-to-rise temperature curve. If the resulting time is greater than the actual time involved, the locomotive will work under that duty cycle. For example, if I_{rms} is 80 A, 7.5 h of operation is allowed (fig. 6.60).

However, if the allowable temperature rise must be reduced, the manufacturer curves can no longer be used directly and must be corrected. Fortunately, the time-rise curve is very nearly parabolic. Thus, any allowable temperature rise curve can be closely approximated by a straight line through two points plotted on log-log paper, with the two axes representing motor current and operation hours. This process can be time consuming. Using the parabolic relationship, the following formula also gives the allowable effective (rms) current for the total operating time when curve correction is necessary:

$$\ln(I) = \frac{[\ln(H_T) - W][\ln(I_1) - \ln(I_2)]}{Z - W} + \ln(I_2), \quad (6.19)$$

$$\text{where } W = \ln\left\{\frac{[(I_2^2 H_2)^{1/2} (T_p/T_c)]^2}{I_2^2}\right\},$$

$$Z = \ln\left\{\frac{[(I_1^2 H_1)^{1/2} (T_p/T_c)]^2}{I_1^2}\right\},$$

H_T = total operation time for motor, h,
T_c = rated allowable temperature rise, °C,
T_p = corrected allowable temperature rise (due to elevation or ambient temperature), °C,
H_1, I_1 = a point taken from manufacturer's time-to-rise temperature curve, h, A,
H_2, I_2 = a second point taken from curve, h, A,
and I = allowed rms current for total operation time H_T, A.

If the allowable current from equation 6.19 is less than I_{rms} from equation 6.17, the motor is overstressed for that duty cycle. Even though the foregoing was applied to dc motors in traction locomotive, the same concepts can be adapted to any mine motor application.

SINGLE-PHASE MOTORS

Although the vast majority of mine motors are three-phase and dc, single-phase motors do find widespread use for auxiliary functions aside from the mining process. As a general rule, single-phase induction motors have one running speed and require a separate means for starting rotation, usually a separate stator or starting winding. Motors are classified by their starting method. The most used techniques are split phase and capacitor start, which will be discussed briefly in this section.

Rotating Stator Field

When a single-phase ac voltage is applied to one stator winding, the current flow produces a magnetic field with a resultant direction that alternates on a line, as in line OP in figure 6.61. If a squirrel-cage rotor winding is in the stator field, a voltage will be induced in the rotor conductors, but the current produced will create a magnetic field that coincides with the stator field (fig. 6.62). As no magnetic interaction occurs, no torque is developed, and the rotor remains stationary (*9*).

If the rotor is moved by some means, the rotor conductors cut the stator magnetic field, and the induced voltages are in phase with the current through the stator winding. However, the rotor winding impedance appears as almost pure inductance, and rotor current will lag the induced voltage by almost 90° (fig. 6.63). Thus the rotor magnetic field is now 90° from the stator field and is termed a *cross-magnetizing field* (*9*). The rotor and stator fields combine to produce a resultant field that rotates at synchronous speed. The cross field strength is proportional to the rotor speed, and is about equal to the stator field strength at synchronous speed. The same operational principles that have been given for three-phase induction motors also hold for single phase; slip must always exist between the rotating field and the rotor. Because of the cross field, the slower rotor speed causes the rotating field to pulsate. Accordingly, vibration and noise are inherent with single-phase induction motors.

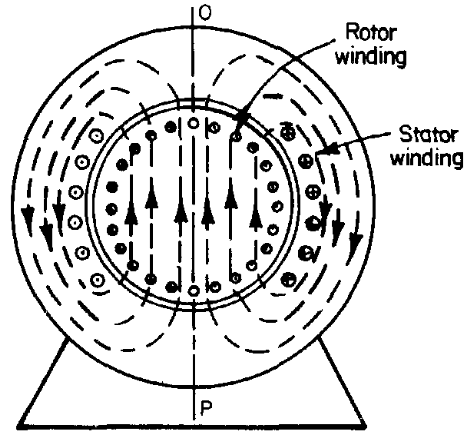


Figure 6.61.—Stator field of two-pole, single-phase induction motor.

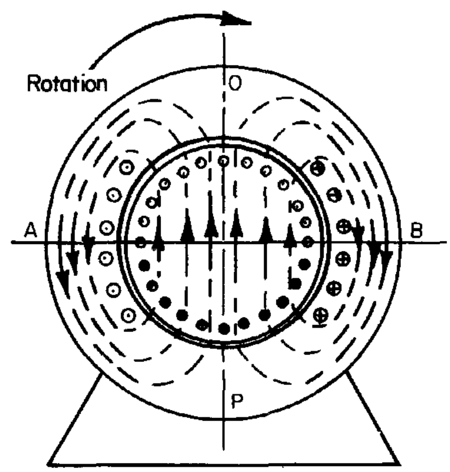


Figure 6.62.—Rotor field of stationary two-pole, single-phase induction motor.

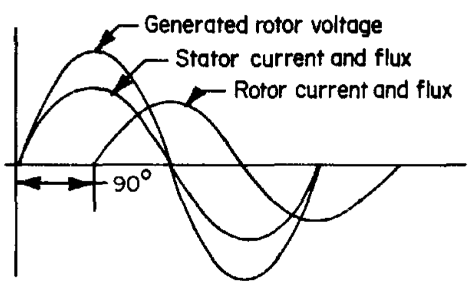


Figure 6.63.—Phase relationships between stator and turning rotor.

Split-Phase Starting

Split-phase motors have two stator windings connected in parallel, as shown for the two-pole motor in figure 6.64. The impedance of each winding is such that the currents through them are out of phase. One winding, the auxiliary or starting winding, is usually constructed of small-gauge wire and has high resistance and low reactance. The running or main winding has a heavier gauge conductor so the winding is of low resistance and high inductance. When energized, the phase angle between the currents through the two windings is only about 30°, but this is enough to produce a rotating magnetic field. The rotating field pulsates, and starting torque is small.

Once the motor is started, the rotor cross field is produced. Thus the starting winding is no longer needed, and it is usually disconnected when the rotor speed reaches 70% to 80% of synchronous (*9*). A centrifugal switch mounted on the rotor shaft is almost always used (fig. 6.65).

The starting direction determines the final rotating direction. Unlike three-phase motors, single-phase induction motors must be stopped and the starting-winding connections reversed, then reenergized to produce a rotating field in the opposite direction.

Capacitor-Start Motors

The capacitor-start motor also has two stator windings. The main winding is arranged for direct connection to the power source, and the auxiliary winding is connected in series with a capacitor. With this arrangement, the currents through the two windings can be as high as 90° out of phase. Hence, starting torque can approach 100% of rated (*9*). Typically, the starting winding is disconnected at 70% to 80% of synchronous speed. A centrifugal switch or a relay sensing current through the main stator winding may be used (fig. 6.66). Apart from the high starting torque, the operation of capacitor-start motors is basically the same as split phase. However, popular split-phase motors have an upper power limit of 1/3 hp, whereas capacitor-start machines can be obtained up to 10 hp.

This chapter has introduced the operation and characteristics of the motors in common use in the mining industry. Although elementary in nature, the contents of the chapter should not be discounted. The electrical power systems in and about mines have the purpose of adequately serving motors. If the characteristics of these loads are not precisely known, it is doubtful that a safe and effective mine power system can be achieved.

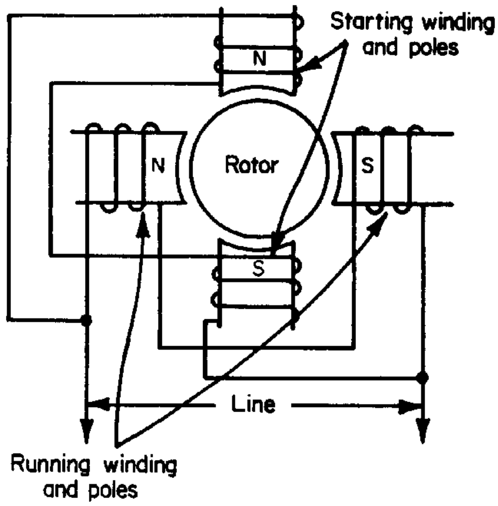


Figure 6.64.—Starting and running stator windings.

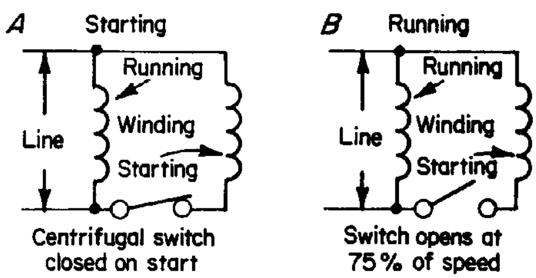


Figure 6.65.—Centrifugal switch to remove starting winding.

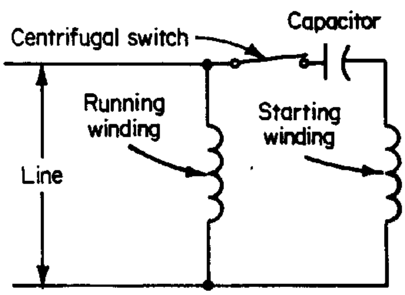


Figure 6.66.—Capacitor-start motor.

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CHAPTER 7.—GROUNDING[1]

A vital part of any mine power distribution system is the connection to earth or ground, which is referred to as the *mine grounding system.* It consists of grounded or grounding conductors, extending from ground beds to equipment. A grounded conductor is a power conductor tied to the grounding system; a grounding conductor is separate from the power conductors and is used only to ground exposed metallic parts of the power system. A ground bed, also termed a *ground mesh* or *grounding electrode,* as well as other names, is a complex of conductors placed in the earth to provide a low-resistance connection to "infinite" earth. The grounding system serves to protect personnel and machinery from the hazards associated with electrical equipment that is operating improperly. The protection afforded can be divided into the following four functions, which are the main purposes behind grounding the system.

First, the grounding system must limit potential gradients between conducting materials in a given area (*38*).[2] During a ground fault, for instance, a phase conductor comes into contact with a machine frame, and current flows through the equipment; subsequently, the potential of the equipment tends to become elevated above ground potential by an amount equal to the voltage on the conductor. If a person touches the machine, while being simultaneously connected to ground in some manner, the body's potential can become elevated, possibly to a lethal extent. The maximum potential to which a person could be exposed when touching a machine frame is equal to the voltage drop along the grounding conductors. Thus, the grounding system must provide a low-resistance path for the fault current to return to the source, and the ground conductors should have low resistance so they can carry the maximum expected fault current without excessive voltage drop. An example of the exposed potential in a surface mining situation is illustrated in figure 7.1 (*38*).

Second, the grounding system should limit the energy available at the fault location. Heavy arcing or sparking can ignite nearby combustible material. The air itself can become ionized, making it capable of carrying tremendous amounts of current. A high-energy fault can vaporize breakers, switchgear, and phase conductors, and protective enclosures may be blown apart with explosive force (*21*). Controlling the maximum allowable fault current significantly reduces the danger of fire and holds equipment damage to a minimum.

Third, the control of overvoltages is essential. An overvoltage condition may occur by accidental contact of equipment with a higher voltage system, or from transient phenomena due to lightning strokes, intermittent ground faults, autotransformer connections, or switching surges (*4*). The maximum ratings for cable insulation, transformer windings, relay contactors, and so forth may be temporarily exceeded in these cases. This does not usually result in an immediate breakdown of equipment, but component parts of the electrical system are successively overstressed and weakened by repeated exposure (see chapter 11). This leads to premature failures, reduced component life, and mysterious "nuisance trips," which can occur without apparent reason. By providing a path between the transformer neutral and ground, most of the sources of transient overvoltages can be reduced or possibly eliminated.

Last, a grounding system should isolate offending sections by selective relaying of ground faults (*44*). The sensitivity and time delays of the protective circuitry should be adjusted so a fault in a certain area will cause the local breaker to sense the malfunction and quickly remove power from only the affected section. If the relative tripping levels and speeds are not established correctly, nearby breakers may not trip when they should, and a small problem could escalate into a large calamity. Consequently, power to half a mine may go out because of poor relay coordination, and much time could be lost in the effort to trace and locate the trouble spot. Thus, the relaying system must be arranged so, even at the lowest level of the power-distribution chain, sufficient fault current can flow to enable the protective circuitry to sense it and take remedial action.

Chapters 9 and 10 cover the protective circuitry used to provide the function of section isolation, while chapter 11 describes the devices employed with the grounding

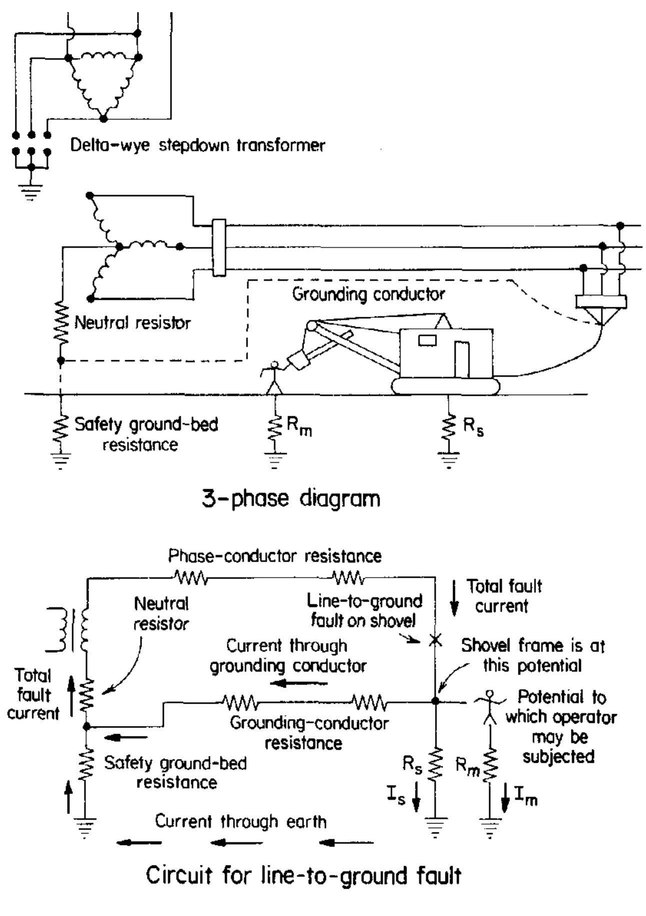


Figure 7.1.—Illustration of electrical shock hazard.

[1] The author wishes to thank Alan M. Christman, who prepared the original material for many sections of this chapter while he was a graduate student at The Pennsylvania State University.

[2] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

system for transient overvoltage control. As an introduction, this chapter looks mainly at the first three purposes and presents the common methods of system grounding, the effect of electric shock on human beings, mine ground system characteristics, and ground-bed construction. Extensive information about grounding is contained in practically all subsequent chapters.

GROUNDING SYSTEMS

Over the past few decades, several different grounding philosophies have held sway in the electrical industry, each with its own advantages and disadvantages (*20*). These methods of grounding are discussed below. Note that reactance-grounded systems are not presented in the following paragraphs, as they are not normally used in industrial power systems.

Ungrounded Neutral

The ungrounded system was probably the first to be used because of its simplicity. Here there are no intentional ground connections in the system whatsoever. However, a perfect ungrounded system cannot exist, since any current-carrying conductor may be coupled to ground through numerous paths, including the distributed capacitance of its wiring, or through motor windings (*49*). This phenomenon is shown in figure 7.2 (*20*). The first line-to-ground fault on such a system will have very little effect (*27*) because there is no way for the fault current to find a complete circuit back to the source, and its magnitude will be very small or nil. Very low fault current means no flash hazard and no equipment damage. Circuit operation continues normally with no interruption of power, an important consideration in industries where downtime is critical (*60*). The first fault is often hard to locate because its effects are negligible. Often no repair effort is made until a second fault occurs, with its concomitant hazards of arcing, heavy current flow, and equipment damage. Since the entire system is "floating," there is no control of transient overvoltages. Except for the problem of accidental contact with a higher voltage system, all the other overvoltage sources mentioned previously are enhanced because of distributed capacitance to ground (*20*).

Solidly Grounded Neutral

An alternative is the solidly grounded neutral (*20*). The first ground fault produces a substantial neutral current flow, which may be quickly sensed by protective circuitry, thereby shutting down the bad section. Overvoltages are controlled since the system, as illustrated in figure 7.3, now has its neutral solidly referenced to ground (*20*). The hazards of this system are due to the magnitude of the fault current. Detection equipment must be sensitive enough to detect low-level fault currents and fast enough to disconnect bad circuits before heavy faults can disrupt system integrity. Large fault currents, typically several thousand amperes, can explode protective enclosures, destroy equipment, and start fires, which is an excellent reason for not using this technique in explosive atmospheres.

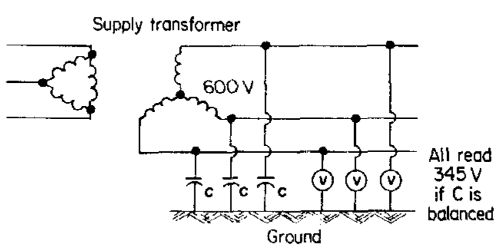


Figure 7.2.—Capacitance coupling in ungrounded system.

Low-Resistance Grounded Neutral

The low-resistance grounded-neutral system is established by inserting a resistor between the system neutral and ground. The resistance is such that ground-fault currents are limited from 50 to 600 A, but are commonly about 400 A (*20*). Transients are controlled by the ground connection, and ample fault current is available for actuating protective relays. The flash hazard is not as serious as in the solidly grounded neutral system, but a current flow of 400 A can still do considerable damage. To limit damage, the least sensitive ground relay should respond to 10% of maximum ground-fault current. A schematic diagram of this method is shown in figure 7.4 (*20*).

High-Resistance Grounded Neutral

Perhaps the best technique, and that required by law in coal-mining applications on portable or mobile equipment, is the high-resistance grounded system, often referred to as the safety ground system. The neutral grounding resistor is sized according to the system voltage level,

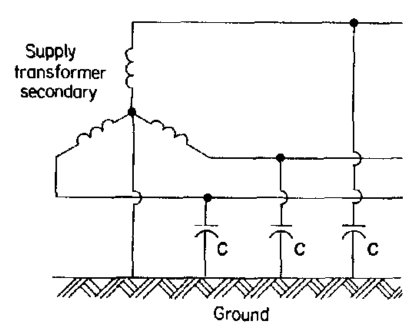


Figure 7.3.—Solidly grounded system.

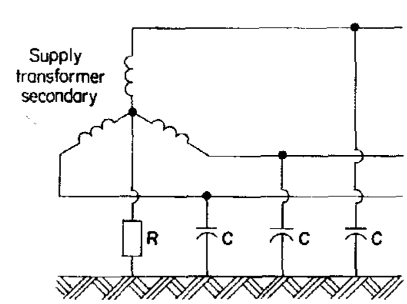


Figure 7.4.—Resistance-grounded system.

in general to limit ground-fault current at 50 A or less. Where the line-to-neutral potential is 1,000 V or less, the grounding resistor must limit fault current to 25 A or less; above 1,000 V, the voltage drop in the grounding circuit external to the resistor must be 100 V or less under fault conditions, With this system, sensitive relaying must detect faults on the order of a few amperes to provide fault isolation and facilitate quick location of the trouble spot (*60*). The level of fault current is also low enough to practically eliminate arcing and flashover dangers. The ground connection also serves to limit the amplitude of overvoltages. However, loads cannot be connected line to neutral, as the grounding conductor must not carry any load current.

ELECTRIC SHOCK

For a safe grounding system to be efficiently and economically designed, voltage and current levels that are harmful to human beings must be determined. With the trend toward larger and more powerful mining machinery, distribution voltage and current levels have risen proportionately. Constant vigilance is required when using electricity if the hazard of electrocution is to be avoided. Even if a shock is nonlethal, involuntary movement caused by the shock may lead to serious injury or death. As an example, a man standing upon a ladder may come into contact with a live wire and fall from his perch (*12*).

Physiologically speaking, the muscles of the body are controlled by electrical impulses transmitted from the brain via the nervous system. These pulses occur at a rate of about 100 per second and may be of positive or negative polarity. From this, it can be seen that the human "internal power supply" operates at about 50 Hz, which is exactly the frequency of the electric power generated in Europe, and is only 10 Hz removed from the U.S. power generation frequency of 60 Hz. This is an unfortunate coincidence, for tests have shown that the most dangerous frequencies to which a person can be exposed are power frequencies in the range of 50 to 60 Hz (*12*).

How sensitive are human beings to the flow of electricity? Tests have indicated that for an average male holding No. 7 AWG (American Wire Gauge) copper-wire electrodes in his hands, 60-Hz ac is first perceived at a level of about 1 mA (*12*). By intermittently touching or tapping an electric conductor, currents of only 1/3 mA can be felt. In the case of dc, the threshold of perception for the average male is 5.2 mA. Sensitivity levels for women in the cases mentioned above can be found by multiplying the male values by a factor of two-thirds (*13*). It is generally agreed that the magnitude and duration of the current are the important shock parameters, rather than the potential difference or voltage (*12*), as can be seen in table 7.1 (*42*).

As current magnitude is increased above the level of perception, many test subjects have reported a tingling sensation, the intensity of which increases as the current rises. Generally, muscles in the vicinity of the current path start to contract involuntarily, until finally a point is reached where the subject being tested can no longer release his grip on the conductor (*14*). The maximum current magnitude that a person can withstand while still able to release the live conductor through the use of muscles stimulated directly by the current, is called the let-go current (fig. 7.5) (*14, 16*). Tests performed on hundreds of volunteers have shown that the maximum let-go current for a healthy adult male is 9.0-mA ac and 60-mA dc. The corresponding values for women are 6.0-mA ac and 41-mA dc. These safe-limit values apply to 99.5% of the sample population (*11*). The value of a specific individual's let-go current is virtually constant, even with repeated exposures to that current level. In addition, these multiple exposures can be tolerated with no ill effects (*16*).

It has been stated that human tissue possesses a negative resistance characteristic. In other words, an increase in current magnitude or contact duration leads to a decrease in the value of skin resistance (*17*). In any case, if a person has grasped a live conductor and realizes that he/she cannot let go, fear-induced perspiration will cause a lowering of the body's resistance, and more current will flow. For ac, when the current level across the chest reaches more than 18 to 22 mA, the chest muscles tighten involuntarily and breathing ceases. Although circulation of blood by the heart is unimpaired, death by asphyxiation can occur within minutes (*43*).

If an individual's initial contact with a live wire causes a current flow ranging from about 50 to 500 mA, ventricular fibrillation may result (*48*). Under normal conditions, the heart beats with a strong, coordinated rhythm. However, a current passing through the heart when the ventricles (the heart's two large pumping chambers) are just starting to relax after a contraction, can cause the various fibers of the heart muscle to beat weakly in an uncoordinated manner (*43*). In this condition, known as ventricular fibrillation, the heart is almost totally incapacitated and blood circulation decreases practically to nothing. Within 2 min, the brain begins to die because of oxygen deficiency. Once initiated, ventricular fibrillation almost never stops spontaneously, and treatment by trained medical personnel must be secured if the victim is to survive.

Obviously, people cannot be used as test subjects in ventricular fibrillation experiments because of the high risk involved. Numerous tests have been carried out on several species of animals and the results extrapolated,

Table 7.1.—Current range and effect on a typical man weighing 150 lb

Current	*Physiological phenomena*	*Effect on man*
Less than 1 mA	None	Imperceptible.
1 mA	Perception threshold	Mild sensation.
3 mA	Pain threshold	Painful sensation.
10 mA	Paralysis threshold of arms and hands	Person cannot release hand grip; if no grip, victim may be thrown clear. Tighter grip because of paralysis may allow more current to flow; may be fatal.
30 mA	Respiratory paralysis	Stoppage of breathing, frequently fatal.
75 mA	Fibrillation threshold (depends on time)	Heart action uncoordinated, probably fatal.
4 A	Heart paralysis threshold (no fibrillation)	Heart stops on current passage, normally restarts when current interrupted.
Greater than 5 A	Tissue burning	Not fatal unless vital organs are burned.

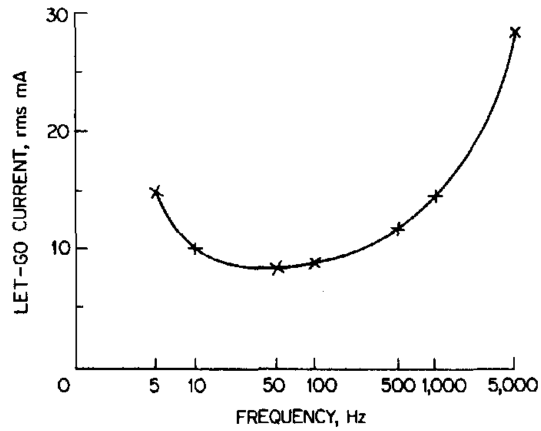


Figure 7.5.—Effect of frequency on let-go current for men.

based upon body weight, to cover human beings (*15*). It was found that fibrillating current is proportional to body weight and inversely proportional to the square root of the shock duration. Using 50 kg (110 lb) as a body weight, it has been proposed that the value of current that can be safely endured by 99.5% of normal adults without causing ventricular fibrillation is (*16*)

$$I = \frac{116}{\sqrt{t}}\Bigg|_{8.3\ \mathrm{ms}}^{5\ \mathrm{s}} \tag{7.1}$$

where I = rms ac, mA,
and t = shock duration, s.

As noted, this equation is valid for values of time between 8.3 ms and 5.0 s (*15*).

It may be seen from the above equation that for a 1-s contact time, the ventricular fibrillation threshold current is about 116 mA. Since a normal person has a pulse rate between 60 and 80 beats per minute, the critical phase of the heartbeat (when a person is vulnerable to ventricular fibrillation) occurs about once each second. Therefore during a shock lasting for 1 s or more, the heart must pass through this critical phase (*48*). As a result, it is thought that ventricular fibrillation is the leading cause of death by electric shock.

Higher currents on the order of a few amperes will freeze both the chest and heart muscles, thereby preventing the onset of ventricular fibrillation. Generally, the heart will restart upon the cessation of current flow (*48*). These current magnitudes are less dangerous statistically than the lower values where fibrillation is prevalent. Further increases in current level, to 5A and above, may produce serious burns leading to shock and possible death, while current levels that substantially elevate body temperature produce immediate death (*16*).

In an electric-shock situation, the victim's electrical resistance plays an important role in determining how much current will flow, as indicated by Ohm's law:

$$I = \frac{V}{R}. \tag{7.2}$$

For a human being, at least three components of resistance have been isolated: contact resistance, skin resistance, and internal resistance (*43*). Contact resistance, as illustrated by table 7.2, depends upon the degree of skin moistness and the area of contact with the live conductor (*42*). Values of 40,000 to 50,000 Ω/cm^2 are given for dry skin and 1,000 Ω/cm^2 for wet skin (*13*). Skin resistance depends upon the physical condition of the tissues: A person who does rough, heavy outdoor work may have a skin resistance of 10,000 Ω, while a value of 1,000 Ω is typical of a sedentary office worker (*43*). Internal resistance is the resistance of the body's interior and is generally accepted to be about 500 Ω between major extremities (*25*).

Table 7.2.—Typical resistance for various contact situations, ohms

Contact [1]	Dry skin	Wet skin
Finger touch	500,000	20,000
Hand on wire	50,000	10,000
Finger-thumb grasp	20,000	5,000
Hand holding pliers	20,000	2,000
Palm touch	10,000	1,000
Hand holding 1.5-in pipe	2,000	500
2 hands holding 1.5-in pipe	500	NA
Hand, immersed	NAp	200
Foot, immersed	NAp	100

NA Not available. NAp Not applicable.
[1] Skin surface only; resistance may be lower when skin is cut, blistered, or abraded.

Voltage magnitude has some effect upon the body's reaction to electric shock, although current is by far the most important parameter. Potentials greater than about 240 V simply puncture the skin, thereby negating the effects of skin resistance (*12*). There is also some evidence that overall body resistance varies inversely with the applied voltage, although this is subject to disagreement. The relationship is given by (*43*).

$$R \propto E^{-n}, \tag{7.3}$$

where R = resistance, Ω,
E = potential, V,
and n = 1.5 to 1.9.

Above about 2,400 V, tissue damage due to burning becomes the major cause of electric-shock injury (*42*).

Thus it can be seen that the body's response to electricity is extremely complex, and currents on the order of a few milliamperes can be fatal if long continued.

CHARACTERISTICS OF MINE GROUNDING SYSTEMS

The concept of protecting mine electrical equipment and personnel against the consequences of ground faults by suitable grounding has existed since electricity was first introduced into coal mines. As early as 1916, the Bureau of Mines recommended equipment frame grounding as a means of preventing electrical shock to miners working on or around electrical equipment (*6*). For the coal mining industry, a suitable grounding system has always been a difficult problem, more complex and difficult than in other industries.

Ground Beds

For mine usage, the electrical distribution cables and overhead transmission circuits carry into the mine one or more grounding conductors in addition to the phase conductors. Each piece of ac equipment has its frame solidly connected via these grounding conductors to a safety ground bed commonly located near the main surface substation and consisting of buried horizontal conductors or driven rods, or a combination of both. The neutral of the substation transformer secondary is also connected to the safety ground bed through the neutral grounding resistor, as shown in figure 7.6. It should be noted that many important components are missing from this diagram, and chapter 13 covers substation circuitry in detail.

The substation actually requires two ground beds, maintained some distance apart. Lightning discharges and other transformer primary surging conditions are directed to the system or station ground. The system and safety grounds must be kept separate so current flow intended for one will not enter the other. It is essential for the safe operation of the mine power system that the resistance of the beds be maintained at 5.0 Ω or less (*3, 39, 44*). A ground bed with this resistance range is often termed a *low-resistance ground bed.*

To demonstrate one reason for a low-resistance bed, consider a situation where lightning strikes the substation, and 10,000 A is discharged through the surge arresters into the system ground bed. If the ground bed is of 5.0-Ω resistance, a potential of 50,000 V is developed, and the grounding grid of the ground bed becomes elevated to 50 kV above infinite earth. Depending upon the physical extent of the grid, a person walking through the area underlain by the grid could bridge a lethal potential gradient with his or her feet (*2*). Metallic objects within the potential gradient field can also be elevated to dangerous potentials and become lethal to the touch. Typical step and touch potentials are illustrated in figures 7.7 and 7.8 (*2*).

These step and touch potential hazards are applicable to both the system and safety ground beds. However, the dangers of a high-resistance safety ground bed are not found close to the bed but at the mining equipment. The most insidious feature of the safety ground system is that the equipment connected to it is maintained not at earth potential, but at the safety ground-bed potential. Unless the bed has low resistance, any safety ground-bed current flow can render every piece of mine equipment potentially lethal. The flow can be created by faults to earth, coupling from lightning strokes to the system ground, lightning strokes to safety grounded machinery, and stray currents from dc haulage systems. Three such cases are illustrated in figures 7.9, 7.10, and 7.11 (*9*). Consequently, with high-resistance ground beds, an elevated frame potential is a problem not just on the machine where it occurs, but everywhere (*10*).

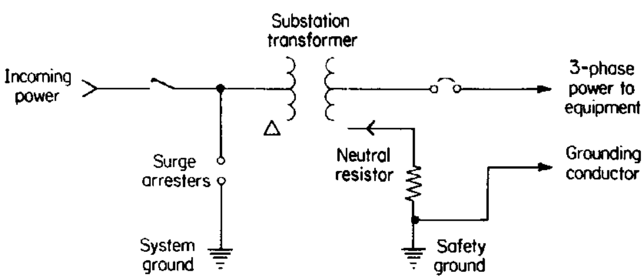


Figure 7.6.—Simplified one-line diagram of substation.

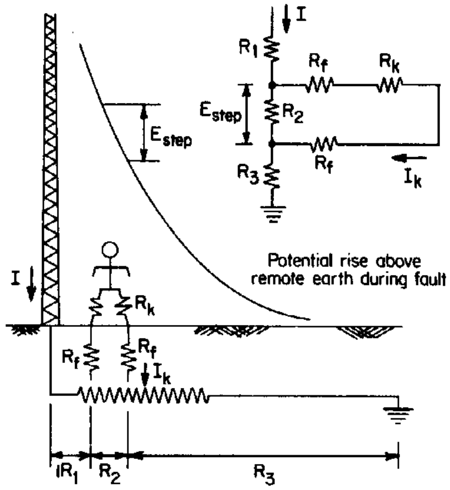


Figure 7.7.—Step potentials near grounded structure.

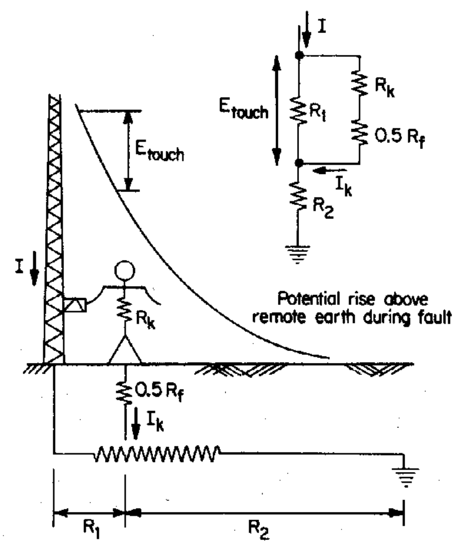


Figure 7.8.—Touch potentials near grounded structure.

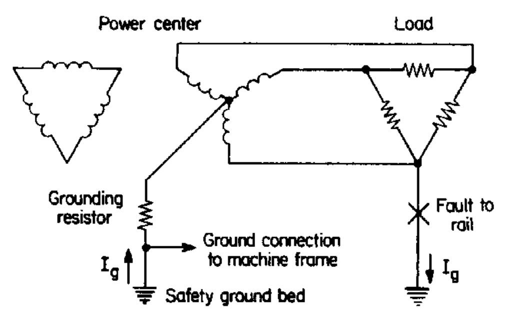


Figure 7.9.—Line-to-earth fault resulting in current flow through safety ground bed.

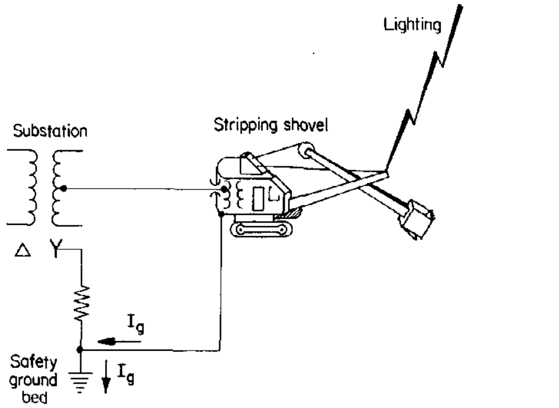


Figure 7.10.—Lightning stroke to equipment causing current flow through safety ground bed.

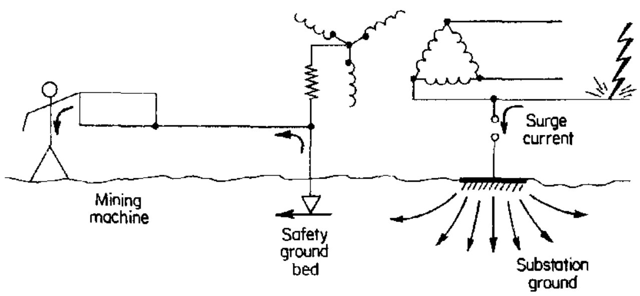


Figure 7.11.—Lightning stroke current through system ground bed causing elevation of safety ground bed.

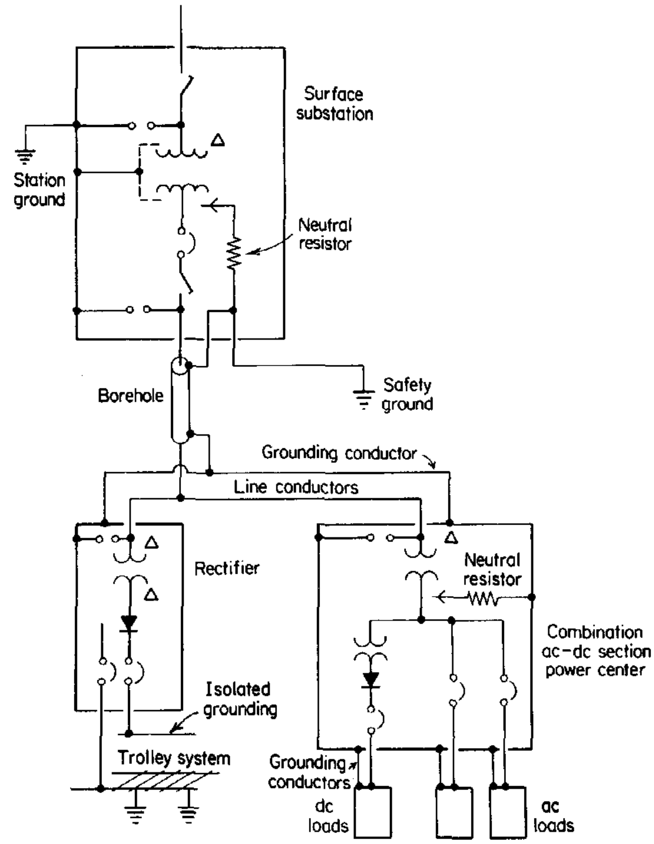


Figure 7.12.—One-line diagram of simplified mine power system.

Grounding in Underground Mining

Early practice in underground coal mining was to drive a metal rod into the mine floor and use that as a ground. In almost every case this arrangement proved to be totally unacceptable, with test measurements indicating 25-Ω or more resistance (*28*). With the exception of pumps, the contact resistance of mining machinery with the mine floor also proved to be too high for adequate grounding. Rail haulage track systems, even though often poorly bonded, showed much lower resistance to ground than most metallic rods driven specifically for that purpose. As a solution, Griffith and Gleim (*28*) in 1943 stated that ". . . consideration should be given to a grounding circuit carried to the outside of the mine." Present coal mine practice does just that.

A simple form of the bipower (mixed ac–dc) system in use in underground coal mines today is illustrated in figure 7.12. After transformation, three-phase ac power enters the mine to supply the various three-phase ac loads. Some of the ac power is converted to dc at rectifier stations to power the locomotive system and, occasionally, dc face equipment. More often, any dc face machinery is powered from rectifiers located in the mine section. Except for the trolley system, all dc as well as the ac equipment frames are connected to a common junction, which is tied to the surface safety ground bed. In order for the system to be effective, grounding conductors must be continuous and this continuity must be verified. Ground-check monitors ensure this.

Trolley locomotives generally utilize the overhead trolley wires as the positive conductor and the tracks as the negative. Neither of these is tied to the rectifier-station frame ground. However, because the track is in contact with the mine floor, the negative conductor for the trolley system is grounded. The dc system that supplies power to face equipment normally employs trailing cables that have neither the negative nor positive conductor grounded. Thus, this subsystem is often ungrounded unless the supply is obtained from the trolley system. Note that in diode-grounded systems, the negative conductor is grounded.

At each transformation step within the power system, such as in a power center, an additional neutral point must be established on the transformer secondary. The neutral is tied through a grounding resistor to the equipment frame and thus via the grounding conductors to the safety ground bed (an exception will be discussed later).

Even with all these grounding points, the ac grounding system must be isolated from separate dc power systems. If it is not, dc may appear in the ac grounding system, thus elevating it above true ground potential. If an ac ground current is present, it will be offset by the dc level. The principal concern is with trolley installations, where isolation is achieved by having no common points between the ac and dc systems. Various techniques have been tried to maintain separation or to eliminate dc offsets while grounding dc face equipment frames.

Face Equipment Grounding

When a working section utilizes an ac continuous miner energized from a section power center and dc shuttle cars powered from the trolley system, the ground potentials of the dc and ac equipment frames are not necessarily equal, because of the voltage drop in the track. Jacot *(33)* suggested that this problem could be solved by isolating the low-voltage ac neutral point from the power-center frame and also the high-voltage grounding system, and connecting it via an insulated cable to the track, as shown in figure 7.13. The low-voltage neutral point remains connected to the ac face-equipment frames. This technique should make the low-voltage ac and dc equipment frame potentials the same, thus eliminating dc offset problems. Difficulties can still arise with this method. If any track rail bonds are bad between the ac and dc low-voltage ground points, the dc frame potentials might be elevated with respect to the ac frames. Further, the power center must be constantly maintained at a safe distance from the tracks to preserve isolation between the track and high-voltage grounding systems.

Another method is shown in figure 7.14*A*. Here, a section power center supplies power to ac face equipment and also, through a rectifier, to dc machinery (usually shuttle cars). The rectifier is isolated within the section power center, but the dc output is grounded through a center-tapped current-limiting resistor. All dc equipment frames are then grounded by trailing-cable grounding conductors, which in turn are connected to the center tap of the grounding resistor. The latter point is connected to the high-voltage grounding system. This has been considered a very safe dc ground protective system because it permits the use of protective circuitry to trip the rectifier breaker in case of a dc ground fault (see chapter 9). However, the use of the center-tapped resistor has been criticized *(46)*. On such a system, any failure to maintain grounding-conductor conductivity or accidental connection of a wrong conductor when splicing cables may lead to a hazard. Nevertheless, an important advantage of the method is that the dc and ac frame potentials can be the same. A more recent method for limiting dc ground-fault current is similar to high-resistance ac grounding and is illustrated in figure 7.14*B*.

In 1963 the Bureau of Mines accepted the use of silicon diodes as a means of grounding dc face equipment frames. When a diode is used, the grounding resistors are not needed because the frame is grounded through the diode to the negative conductor, as illustrated in figure 7.15. The diode circuit also includes a ground protective device, which will interrupt the power if a current flows from a positive power conductor to an equipment frame (again, see chapter 9). According to Jones *(37)*, diode grounding should ensure good ground continuity since the same conductor acts as both a dc negative conductor and the grounding connection. However, a grounding diode only protects the dc system against ground faults within the equipment frame. Current leakage to ground or faults within trailing cables can still present hazards.

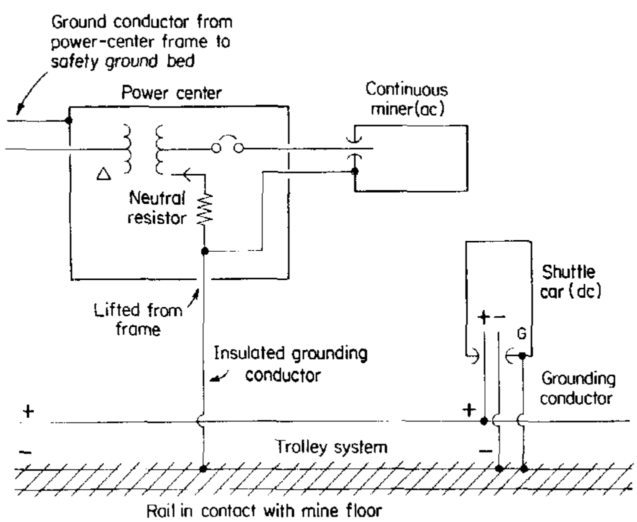
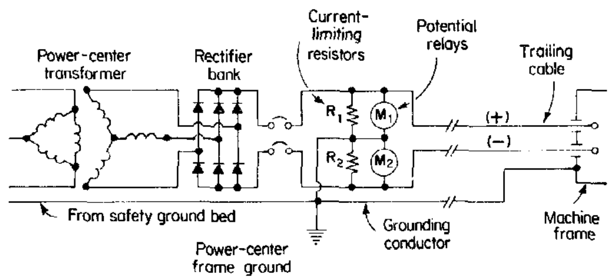
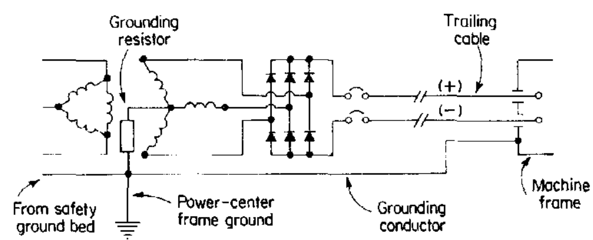


Figure 7.13.—Mixed ac-dc mine power system; dc load energized from trolley system.



A Resistors between dc line conductors and grounding conductor



B Resistor between transformer neutral and grounding conductor

Figure 7.14.—System grounding with current-limiting resistors.

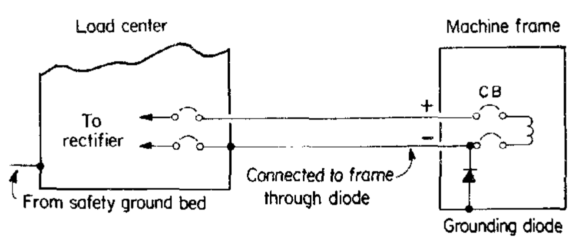


Figure 7.15.—Diode grounding of machine frame.

Track Grounding

As previously mentioned for trolley systems, the rectifier frame is grounded by the ac system, but the negative conductor is grounded to the mine floor through the track. In order to maintain isolation, there is no internal connection between the rectifier output (or the trolley distribution system) and its frame. However, if the rectifier is sitting on the mine floor, there is a possible common point from the track (dc) to the rectifier frame (ac). Ideally, the common point through the earth is a much higher resistance than the rail itself so that all rectifier current returns in the rail. When the rail resistance increases because of poor bonding or crossbonding, some current may flow through the earth to or from the rectifier frame, depending on the rail potential. Thus dc is introduced in the ac ground system.

Leakage of trolley-wire insulator to the roof or rib may have the same effect, although it is less common. This lack of effective separation can cause dc offset currents on any mining machine and electrical system whenever the sum of the mine floor resistance and equipment frame contact resistances is too low and, therefore, dc current flow is permitted through the earth. To help minimize any problems, rectifiers should be located no closer than 25 ft from the track. In severe cases, the rectifier frame can be insulated from the mine floor.

The preceding has shown that haulage conversion units are the primary source of dc offset currents. Regardless of the source, once stray dc currents occur, they can exist on all the ac grounds within the mine. This problem is further complicated since these currents may also travel through water pipes and hoses, or anything conductive.

The two most undesirable effects of dc offset currents on the ac ground system are nuisance tripping and intermachine arcing. Nuisance tripping can occur whenever the offset ac waveform is greater than the relay trip value, and it primarily affects ground-overcurrent relays and ground-check monitors. Intermachine arcing occurs when two machine frame potentials are not the same. While they are touching, a current flow is possible, but when they separate, arcing may occur. These problems are discussed further in chapter 17.

Grounding in Surface Mines

The typical grounding system for a surface coal mine is similar to that for underground mining. One or more substations with resistance-grounded secondaries are employed to transform the incoming utility voltage to the lower potential used by the mining machines. At this level, pit distribution is carried on overhead lines or cables to supply switchhouses located near the particular piece of equipment. A trailing cable completes the power circuit from the switchhouse to the machine. A switchhouse is sometimes connected via cable to a portable substation, which supplies lower voltage power to production, auxiliary, or lighting equipment.

Substation grounding includes both a system and a safety ground bed, each physically removed and electrically isolated from the other. Grounding conductors extend from the safety ground bed to all equipment frames. The neutrals of the transformer secondary of portable substations are resistance grounded to the equipment frame.

In contrast with underground coal mines where the entire secondary distribution system is underground, both the primary and secondary lines in a surface mine are out in the open where they are exposed to lightning. In fact, equipment such as draglines and shovels are subject to direct strokes (fig. 7.10). For the best possible protection from lightning, it is essential that the grounding system have as low a surge impedance as possible. The key factor here is to provide many short, direct paths to earth. The specifics of lightning protection for all mines are presented in chapter 11.

GROUND-BED CONSTRUCTION

Since the minesite is determined by the location of the rock or mineral to be extracted, the conditions required for the installation of an adequate ground bed are not always easily met. If annual rainfall is low or soil resistivity is high, an extensive array of buried metallic conductors may be necessary to assure a low-resistance connection to earth. Measurement of soil parameters can be made before the construction of a grounding grid is begun, thereby ascertaining the configuration for the metallic network that will yield the desired values of earth resistance and potential gradient. After construction, the resistance of the selected ground-bed configuration must be checked. Proper design at the time the ground bed is installed will save much time and expense in later years.

Present-day ground beds can be divided into two general categories: meshes and rodbeds. A mesh is a horizontal network of metallic conductors arranged in a grid pattern, which is embedded a short distance below the earth's surface. A rodbed is an interconnected network of vertical metallic rods driven into the earth. The metallic components for either ground-bed type are also called *electrodes*.

Ground Resistance

Any grounding system exhibits some finite resistance with respect to infinite earth, even though it is completely immersed in the soil. When a fault from a power conductor to earth occurs, current can flow through the ground-bed metallic electrodes, across the soil-metal interface, and into the ground. The greater the surface area of metal in intimate contact with the soil, the lower the resistance. Most of the actual resistance exhibited by each metallic conductor occurs within 6 to 10 ft of the electrode, as illustrated in figure 7.16 (*36*). If the surrounding soil is viewed as a succession of concentric shells, it is easily seen that the shells adjacent to the electrode have a much smaller cross-sectional area, and hence a higher resistance

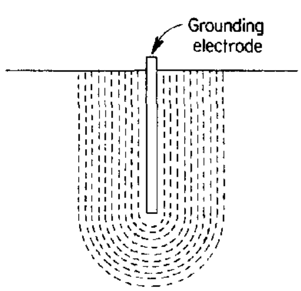


Figure 7.16.—Resistance of earth surrounding electrode.

than more distant shells. Consequently, the main factors which determine grounding-grid resistance are the physical dimensions of the system and the innate characteristics of the soil, primarily its resistivity (*51*). Figure 7.17 shows how the total resistance of a driven rod varies as it penetrates soil horizons of different resistivity (*36*).

The electric field around a current-carrying wire is analogous to the electrostatic field surrounding a charged conductor of similar shape. By calculating the capacitance of an electrode immersed in the soil, its resistance can then be determined. For a conductor buried deeply in the earth (*29*),

$$R = \frac{\rho}{4\pi C}, \qquad (7.4)$$

where R = resistance, Ω,
ρ = soil resistivity, Ω-m,
and C = capacitance, F.

If the conductor is relatively near the earth's surface, as is usually the case, the effects of the conductor image, which is located an equal distance above the surface, must be included in the formula, yielding (*29*)

$$R = \frac{\rho}{2\pi C}. \qquad (7.5)$$

For multielectrode systems, the capacitance of each conductor plus its mutual capacitance with respect to all other conductors must be calculated. By maximizing the capacitance, the resistance can be minimized, which is the desired goal.

The two predominant methods for determining the capacitance of earth-electrode systems are Howe's average potential method and Maxwell's method of subareas, each of which has a constant charge density and potential (*29*). Howe's technique assumes a uniform charge density on each electrode and then calculates the average surface potential. The capacitance can be found from (*21*)

$$C = \frac{Q}{V}, \qquad (7.6)$$

where C = capacitance, F,
Q = charge, C,
and V = potential, V.

Electrode Configuration Formulas

One of the most common ground systems is the rodbed. For a single vertical rod (*22*),

$$R = \frac{\rho}{2\pi \ell}\left(\ln \frac{4\ell}{a} - 1\right), \qquad (7.7)$$

where ρ = soil resistivity (Ω-m or Ω-ft)
ℓ = length of rod, m or ft,
and a = radius of rod, m or ft.

Figure 7.18 shows how the resistance and conductance of a typical driven rod vary as the rod length is increased (*23*). It can be seen that the resistance curve starts to flatten out, which indicates that a length in excess of 15 ft is ineffective. However the conductance curve is almost linear. Figure 7.19, which is an extended version of the previous graph, shows very clearly that even at depths of 100 ft, the conductance increases in direct proportion to the length (*23*). If the soil can be easily penetrated, deeper

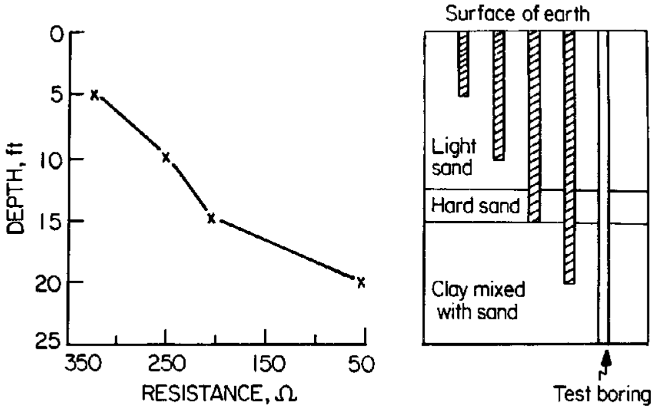


Figure 7.17.—Decrease in earth resistance as electrode penetrates deeper soil horizons.

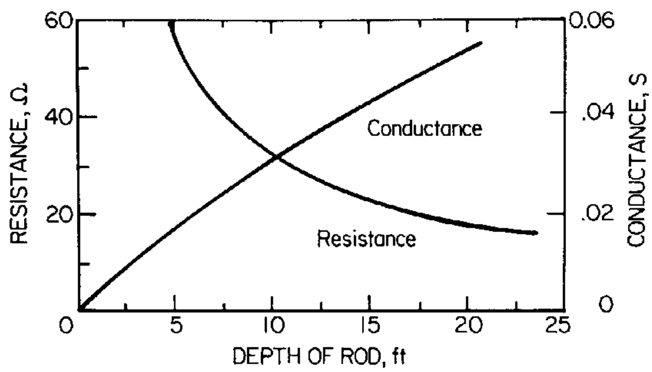


Figure 7.18.—Calculated values of resistance and conductance for ¾-in rod driven to depth of 25 ft.

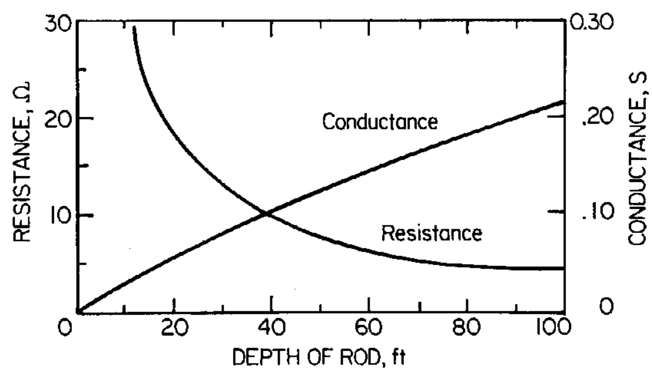


Figure 7.19.—Calculated values of resistance and conductance for ¾-in rod driven to depth of 100 ft.

rods are always better. The simple nomogram shown in figure 7.20 may be used to estimate the resistance of a driven rod without carrying out calculations (*55*). Figure 7.21 shows the effect of soil resistivity on the resistance of a driven rod, as well as the benefits gained by using longer rods (*22*). It could be pointed out that several shorter ground rods are easier to drive than one long rod of the same total length. However, when using multiple rods, the effects of mutual resistance tend to negate some effectiveness, so the resistance of the group is greater than would be expected unless very large spacings are used between electrodes (*23*). Figure 7.22 shows this effect for rods spaced at a distance equal to their length, while figure 7.23 shows the advantages that accrue when spacing is increased from 0.5 to 100 ft (*54*).

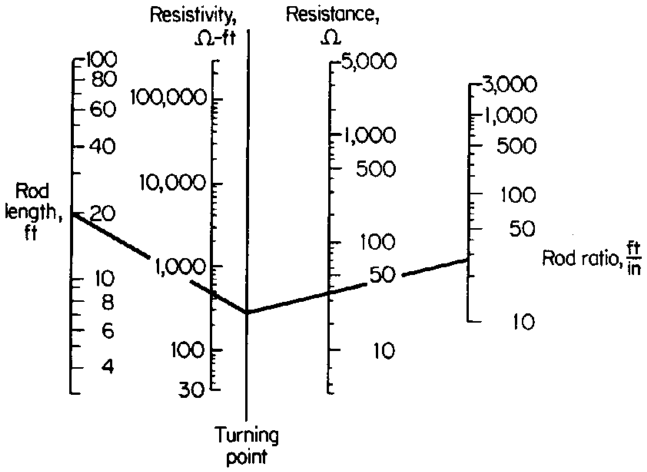


Figure 7.20.—Nomogram to provide resistance of driven rod. Rod ratio is equal to rod length (feet) divided by rod diameter (inches). Example is shown for a rod of ¾-in diameter and 20-ft length, driven in 500-Ω-ft soil resistivity, providing about 35-Ω resistance.

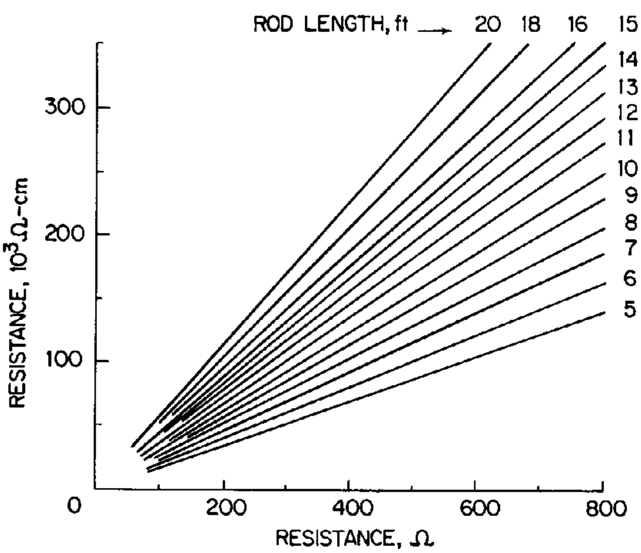


Figure 7.21.—Resistance of one ground rod, ¾-in diameter.

One of the following two formulas can be applied to systems composed of multiple rods. If the rod spacing-to-length ratio is large (spacing > > length), then (*50*)

$$R = \frac{\rho}{n2\pi\ell}(\ln\frac{2\ell}{a}), \tag{7.8}$$

where n is the number of rods and the other variables are as previously defined. If the rod spacing-to-length ratio is small (length > > spacing) (*50*),

$$R = \frac{\rho}{2\pi\ell}(\ln\frac{2\ell}{A}), \tag{7.9}$$

where $A = (a\ S_{12}\ S_{23}\ S_{34}\ .\ .\ .\ )^{1/n}$
and S_{12} = spacing between electrodes 1 and 2, and so forth.

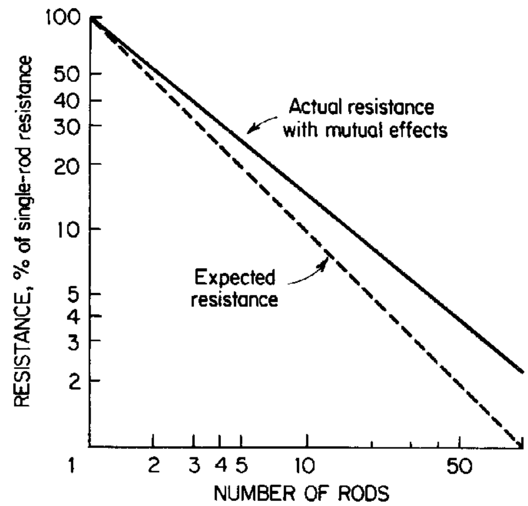


Figure 7.22.—Resistance of parallel rods when arranged in straight line or circle with spacing equal to rod length.

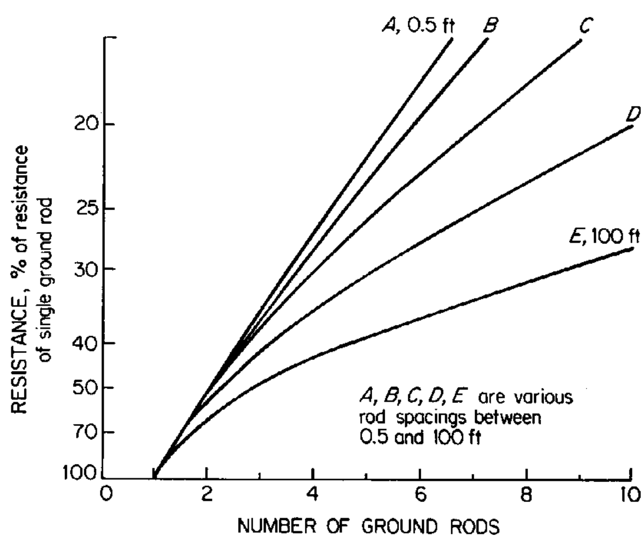


Figure 7.23.—Variation of earth resistance as number of ground rods is increased for various spacings between rods.

A formula for determining the resistance of grounding meshes is given by (*53*)

$$R = \frac{\rho}{\pi \ell}\left(\ln\left(\frac{2L}{(2az)^{0.5}}\right) + k_1\left(\frac{L}{(B)^{0.5}}\right) - k_2\right), \quad (7.10)$$

where L = total length of buried conductor,
z = burial depth,
and B = area enclosed inside mesh perimeter.

The constants k_1 and k_2 depend upon the burial depth and the length-to-width ratio of the mesh and may be determined from the graphs shown in figures 7.24 and 7.25 (*53*). However, for a typical mesh where the length and width are similar and the burial depth is a few feet or less, then $k_1 = 1.3$ and $k_2 = 6$.

In many cases, combinations of rods and a mesh are used, especially when driven rods are interconnected by bare conductors that are also buried in the soil. For these situations (*53*),

$$R = \frac{R_1R_2 - R_m^{\ 2}}{R_1 + R_2 - 2R_m}, \quad (7.11)$$

where R_1 = rodbed resistance,
and R_2 = mesh resistance.

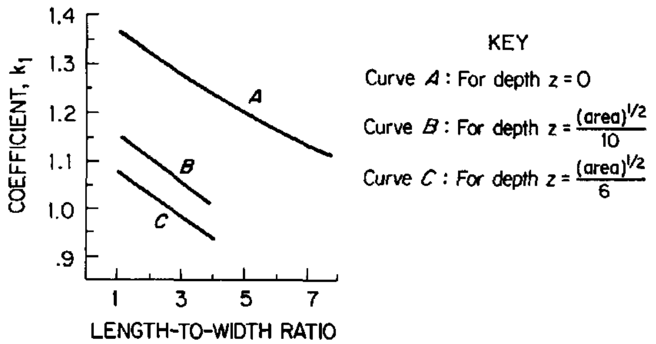


Figure 7.24.—Values of coefficient k_1 as function of length-to-width ratio of area.

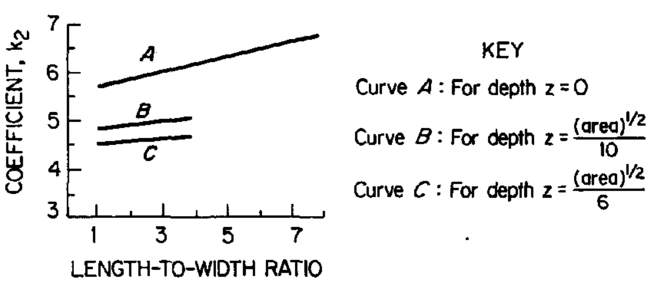


Figure 7.25.—Values of coefficient k_2 as function of length-to-width ratio of area.

The mutual resistance, R_m, is (*53*)

$$R_m = \frac{\rho}{\pi L}\left(\ln\frac{2L}{\ell} + k_1\frac{L}{(B)^{0.5}} - k_2 + 1\right), \quad (7.12)$$

where L = total length of mesh conductor,
and ℓ = length of one rod.

If the soil is of uniform resistivity, adding a mesh to a preexisting rodbed, or vice versa, cannot be justified merely from the viewpoint of reduced resistance, since the reduction in resistance will seldom amount to more than 10% to 15%. However, the addition of a mesh to a rodbed will usually smooth out the potential-gradient distribution, and the addition of a rodbed to a horizontal mesh generally attenuates seasonal fluctuations in resistance (*23, 55*).

Other electrode configurations are in use but are not as widespread as the two covered above. Table 7.3 summarizes most of the other electrode types and gives formulas for determining their resistance (*22*). As a first approximation, the Laurent formula gives a quick and fairly accurate estimate of the ground resistance of any type of system (*56*):

$$R = \frac{\rho}{4r} + \frac{\rho}{L}, \quad (7.13)$$

where L = total length of buried bare conductor,
and r = equivalent radius of the system.

The equivalent radius of a grounding system varies depending upon the exact configuration, but a safe estimate is one-half of the length of the longest diagonal line contained by the system (*55*).

Contact resistance between the surface of the electrodes and the soil is not normally a significant factor if the bed has been in existence long enough for the soil to settle and compact, but in new beds it may amount to 20% of the total resistance (*57*).

In summary, the best way to achieve a low-resistance ground is to maximize the periphery or areal extent of the grounding system. Conductor diameter has little effect upon resistance, and mechanical strength requirements should be the primary consideration. Because of wide seasonal variations in the soil resistivity of surface layers, deeply buried meshes or deeply driven rods are often preferable. This is also advisable if lower resistivity layers are known to exist at depth. Driven rods are usually preferred over buried meshes for three reasons (*39*):

- The expense of earth removal to bury the mesh is avoided.
- Rods do not require the packing of earth around the buried electrodes to ensure good earth contact.
- The use of rods can give a desired resistance more easily than using any other ground-bed form.

Note that although formulas are excellent for calculating the theoretical resistance of a grounding bed, the actual resistance should always be measured with an earth tester to ensure system integrity.

Table 7.3.—Approximate resistance formulas for various electrode configurations

Electrode configuration	Description	Formula
	One ground rod; length L, radius a	$R = \frac{\rho}{2\pi L}\left(\ln \frac{4L}{a} - 1\right)$
	Two ground rods; spacing s>L	$R = \frac{\rho}{4\pi L}\left(\ln \frac{4L}{a} - 1\right) + \frac{\rho}{4\pi s}\left(1 - \frac{L^2}{3s^2} + \frac{2L^4}{5s^4} \ldots\right)$
	Two ground rods; spacing s<L	$R = \frac{\rho}{4\pi L}\left(\ln \frac{4L}{a} + \ln \frac{4L}{s} - 2 + \frac{s}{2L} - \frac{s^2}{16L^2} + \frac{s^4}{512L^4} \ldots\right)$
	Buried horizontal wire; length 2L, depth s/2	$R = \frac{\rho}{4\pi L}\left(\ln \frac{4L}{a} + \ln \frac{4L}{s} - 2 + \frac{s}{2L} - \frac{s^2}{16L^2} + \frac{s^4}{512L^4} \ldots\right)$
	Right-angle turn of wire; length of arm L, depth s/2	$R = \frac{\rho}{4\pi L}\left(\ln \frac{2L}{a} + \ln \frac{2L}{s} - 0.2373 + 0.2146 \frac{s}{L} + 0.1035 \frac{s^2}{L^2} - 0.0424 \frac{s^4}{L^4} \ldots\right)$
	Three-point star; length of arm L, depth s/2	$R = \frac{\rho}{6\pi L}\left(\ln \frac{2L}{a} + \ln \frac{2L}{s} + 1.071 - 0.209 \frac{s}{L} + 0.238 \frac{s^2}{L^2} - 0.054 \frac{s^4}{L^4} \ldots\right)$
	Four-point star; length of arm L, depth s/2	$R = \frac{\rho}{8\pi L}\left(\ln \frac{2L}{a} + \ln \frac{2L}{s} + 2.912 - 1.071 \frac{s}{L} + 0.645 \frac{s^2}{L^2} - 0.145 \frac{s^4}{L^4} \ldots\right)$
	Six-point star; length of arm L, depth s/2	$R = \frac{\rho}{12\pi L}\left(\ln \frac{2L}{a} + \ln \frac{2L}{s} + 6.851 - 3.128 \frac{s}{L} + 1.758 \frac{s^2}{L^2} - 0.490 \frac{s^4}{L^4} \ldots\right)$
	Eight-point star; length of arm L, depth s/2	$R = \frac{\rho}{16\pi L}\left(\ln \frac{2L}{a} + \ln \frac{2L}{s} + 10.98 - 5.51 \frac{s}{L} + 3.26 \frac{s^2}{L^2} - 1.17 \frac{s^4}{L^4} \ldots\right)$
	Ring of wire; ring diameter D, wire diameter d, depth s/2	$R = \frac{\rho}{2\pi^2 D}\left(\ln \frac{8D}{d} + \ln \frac{4D}{s}\right)$
	Buried horizontal strip; length 2L, section a by b, depth s/2, b, a/8	$R = \frac{\rho}{4\pi L}\left(\ln \frac{4L}{a} + \frac{a^2 - ab}{2(a + b)^2} + \ln \frac{4L}{s} - 1\right.$
	Buried horizontal round plate; radius a, depth s/2	$R = \frac{\rho}{8a} + \frac{\rho}{4\pi s}\left(1 - \frac{7a^2}{12s^2} + \frac{33a^4}{40s^4} \ldots\right)$
	Buried vertical round plate; radius a, depth s/2	$R = \frac{\rho}{8a} + \frac{\rho}{4\pi s}\left(1 + \frac{7a^2}{24s^2} + \frac{99a^4}{320s^4} \ldots\right)$

Two-Layer Earth Structures

In many situations, the soil is not homogeneous but consists of two or more distinct layers that are approximately horizontal and possess differing resistivity values. The effect of a two-layer structure upon ground resistance depends upon the top-layer thickness, the relative conductivity of the two layers, and the dimensions of the grounding system with respect to the thickness of the first layer (*26*). Figure 7.26 shows the potentials and potential gradients for a mesh system in the first layer of a two-layer configuration where the thickness of the first layer ranges from 0.1 to 1,000 m (*18*). In this case, the first-layer resistivity ($\rho 1$) is 200 Ω-m, and $\rho 2$ is 600 Ω-m. The equivalent radius of the grounding grid is 10 m, and the reflection factor (K), as defined by the following equation, is 0.5:

$$K = \frac{\rho 2 - \rho 1}{\rho 2 + \rho 1}. \tag{7.14}$$

It may be seen that the potential gradient depends almost solely upon the first-layer resistivity if the grounding system is wholly immersed in that layer. The effect of first-layer resistivity upon ground-bed resistance increased with the thickness of that layer. Thus, if $\rho 1 < \rho 2$, ground resistance will decrease as top-layer thickness increases.

Soil-Heating Effects

The manner in which a ground bed responds to the flow of current through it depends upon the magnitude and duration of the loading. Two types of loading have been recognized and will be dealt with separately.

Long-term loading of the safety ground bed in a mine power system should consist only of currents due to unbalance, the charging of conductor capacitances, and mutual inductance between conductors. At any rate, in a properly functioning system, the current magnitude

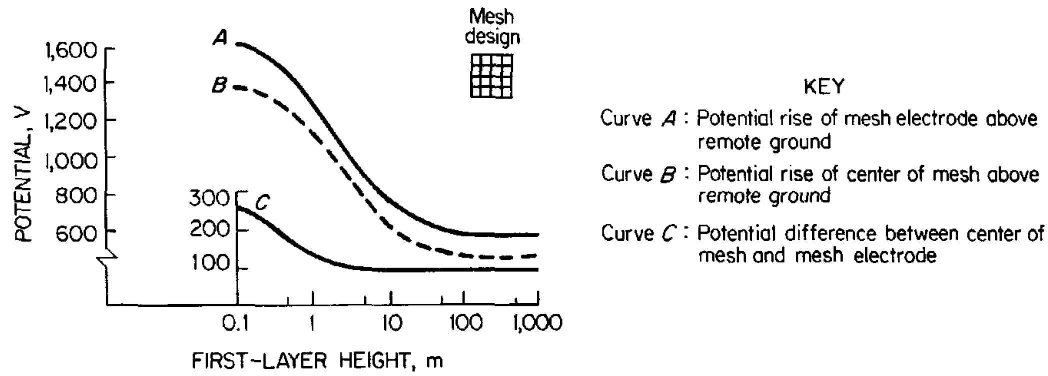


Figure 7.26.—Influence of first-layer height of potentials.

should be on the order of a few amperes. If the bed is very extensive, the dissipation of ground current in the soil may cause only a small rise in soil temperature. Because of the negative temperature coefficient of soil, the actual ground-bed resistance will decrease *(23)*. If the temperature rise is high enough to evaporate some soil moisture, then the resistance will increase somewhat. Capillary action will tend to restore any moisture, and the soil itself will also conduct away some of the heat. Eventually an equilibrium point will be reached where the system is once again stable, although the soil temperature and ground-bed resistance may be slightly altered. The maximum allowable ground-bed current is given by *(50)*

$$I = \frac{1}{R}(2\rho\lambda\theta)^{1/2}, \tag{7.15}$$

where ρ = soil resistivity, Ω-m,
λ = soil thermal conductivity, 1.2 W/(m· °C),
and θ = maximum allowable soil temperature rise, °C.

If both sides of the equation are multiplied by R, the maximum permissible applied voltage is found to be

$$V = (2\rho\lambda\theta)^{1/2}. \tag{7.16}$$

Generally, the maximum allowable temperature is 100° C, at which point total evaporation occurs. Therefore θ may be replaced by (100 − T) where T is the ambient Celsius temperature. The preceding analysis is subject to two restrictions *(57)*:

1. The thermal conductivity, λ, is somewhat temperature dependent, and
2. Soil moisture will start to evaporate at temperatures below 100° C.

Short-term overloading of the grounding system may occur during certain fault situations, but in a properly functioning system, only the grounding conductors, located inside cables and with the overhead powerlines, and the neutral resistor should be subjected to fault current. Should a situation occur in which the ground bed is called upon to handle large currents for a short time, heat conduction through the soil may be ignored because of its low rate. In this situation, the maximum allowable soil temperature rise is given by *(57)*

$$\theta = \frac{0.24 i^2 \rho T}{\delta\sigma}, \tag{7.17}$$

where ρ = soil resistivity, Ω-m,
i = current density at electrode surface, A/m^2,
T = time, h,
δ = soil density, kg/m^3,
and σ = soil specific heat, kWh/°C − kg.

So far only the effects of ac upon soil heating have been discussed. Dc causes completely different phenomena. The first of these is polarization. The flow of dc through water causes some of the molecules to dissociate into the constituent gases, hydrogen, and oxygen. The resulting gas bubbles eventually form a film on the electrode surfaces, thereby insulating them from the surrounding soil, which leads to a dramatic rise in resistance.

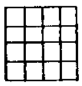
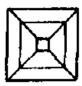
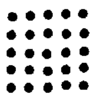
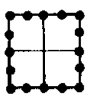
In addition, dc causes *electro-osmosis* (also referred to as *endosmosis*). Here, moisture present in the soil (which is not electrolyzed) tends to migrate toward the negative electrode of the dc source. Actually, cations present in the soil are attracted to the cathode, and the polar water molecule is normally attached to these positive ions. Again, an increase in resistance is the result.

Control of Potential Gradients

In addition to providing a low-resistance path to ground, the ground bed should also be designed so that potential gradients in the soil surrounding the bed (step and touch potentials) are held to a minimum for the protection of personnel.

As a generalization, it can be stated that meshes are superior to rodbeds as far as potential-gradient control is concerned *(18, 23)*. This is illustrated by table 7.4, which compares a variety of grounding systems, each having about the same total length of buried conductor *(18)*. The electrodes are buried to a depth of 0.6 m, and as can be seen, grid C (rodbed) shows significantly higher potential than does grid A (mesh). The potential gradients around a mesh may be decreased by making the meshes smaller. Figures 7.27 and 7.28 show the improvement which can be obtained by burying the grounding system to a greater

Table 7.4.—Comparison of grounding grids with other types of electrodes

Grid	Rod or mesh layout	Total length of buried conductor, m	Length of rods, m	Number of rods	Maximum mesh voltage, % of potential rise
A..............		160	NAp	0	18.9
B..............		174	NAp	0	12.7
C..............		175	7	25	32.4
D..............		176	5	16	23.2
E..............		176	NAp	0	17.6

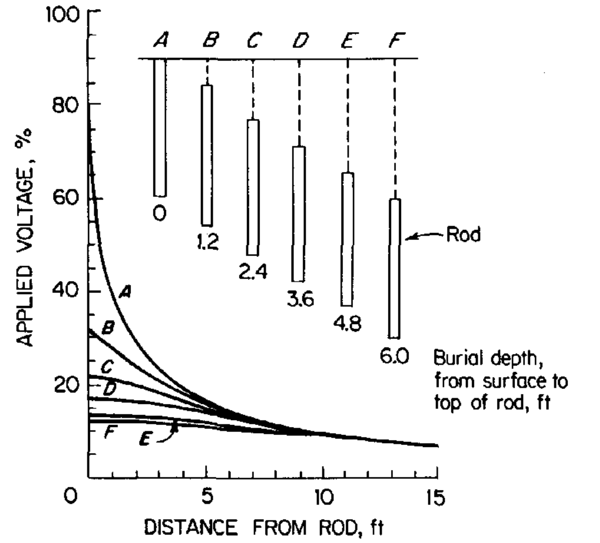


Figure 7.27.—Potential on ground surface due to rod 6 ft long and 1-in diameter buried vertically at various depths.

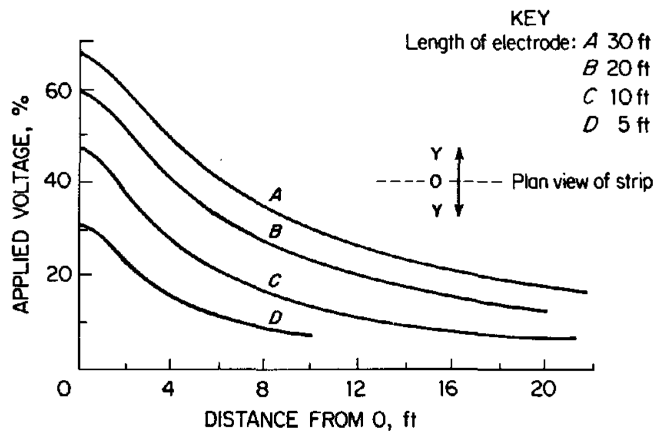


Figure 7.28.—Potential on ground surface due to strips, 1 in by 0.1 in, of various lengths buried horizontally at depth of 2 ft; values given are those along line OY perpendicular to length of strip.

depth (*24*). It is obvious that the deeper a bed can be buried, the better will be the gradient control. A rodbed, where the rods are interconnected by bare conductors with the entire system buried to a depth of a few feet, should provide both a low resistance and a safe potential gradient. Building a fence around the perimeter of the ground bed is one way of limiting human exposure to hazardous potential gradients.

GROUND-BED RESISTANCE MEASUREMENT

Measurement Method

The accepted technique for determining the resistance to infinite earth of a grounding resistance is called the *fall-of-potential method* (*36*). Figure 7.29*A* shows a drawing of this arrangement (*56*). Three terminals are required: the ground under examination, a potential electrode, and a current electrode. The current electrode is spaced far from the ground system being tested, and the potential electrode is placed at some point on a straight line between the two. The resistance-measuring equipment is operated, and a reading is taken. Here, a known current is passed through the current electrode, the voltage between the potential electrode and ground is measured and the resistance is the ratio V/I. This process is repeated as the potential electrode is moved farther and farther from the grounding electrode, toward the current electrode. A graph is then drawn in which the ground resistance is the ordinate and the distance between the ground and potential electrodes is the abscissa. Figure 7.29*B* shows two typical plots that may result (*56*). Curve *a* was taken with the current electrode at a greater distance than in curve *b*. The flat portion of curve *a* is an indication that the current electrode is now far enough away from the grounding system that the mutual effect no longer exists. This is illustrated in figure 7.30 by the *hemispheres of influence* surrounding the ground and current electrodes (*35*).

The proper spacing for the measurement probes is based upon hemispherical electrodes, so any actual ground system must first be converted to an equivalent hemisphere before the needed spacing can be determined (*56*). This may be approximated by assuming that the equivalent radius is equal to one-half the length of the longest

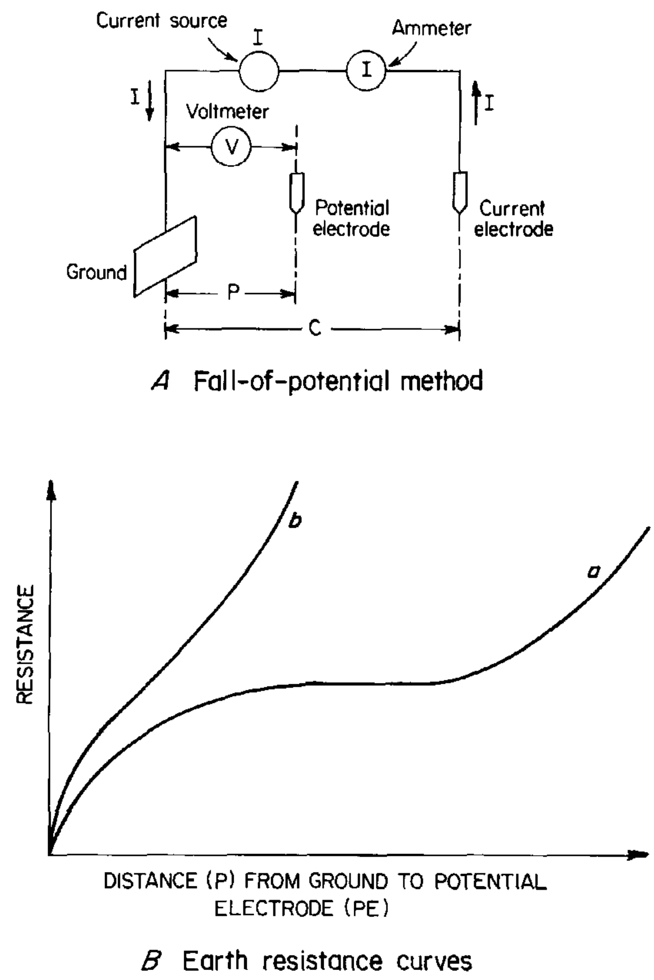


Figure 7.29.—Measuring resistance of grounding system.

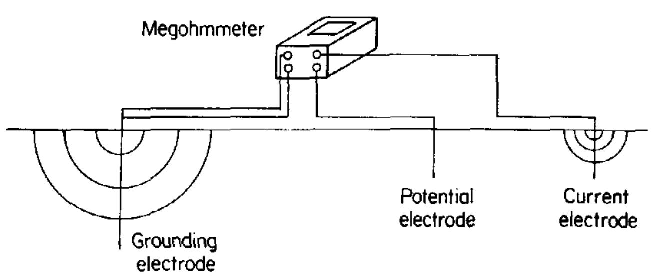


Figure 7.30.—Concentric earth shells around ground connection being tested and around current electrode.

diagonal that can be placed inside the perimeter of the system (that is, 50% of the maximum bed dimension). Figure 7.31 shows the proper spacing for both current and potential electrodes for a given equivalent radius of the grounding system (*55*). The potential-electrode spacing that yields the true value of ground resistance is equal to about 61.8% of the current-electrode spacing. For large ground systems, it may be impossible to attain the necessary spacings for potential and current electrodes resulting from this technique. In that case, the procedure

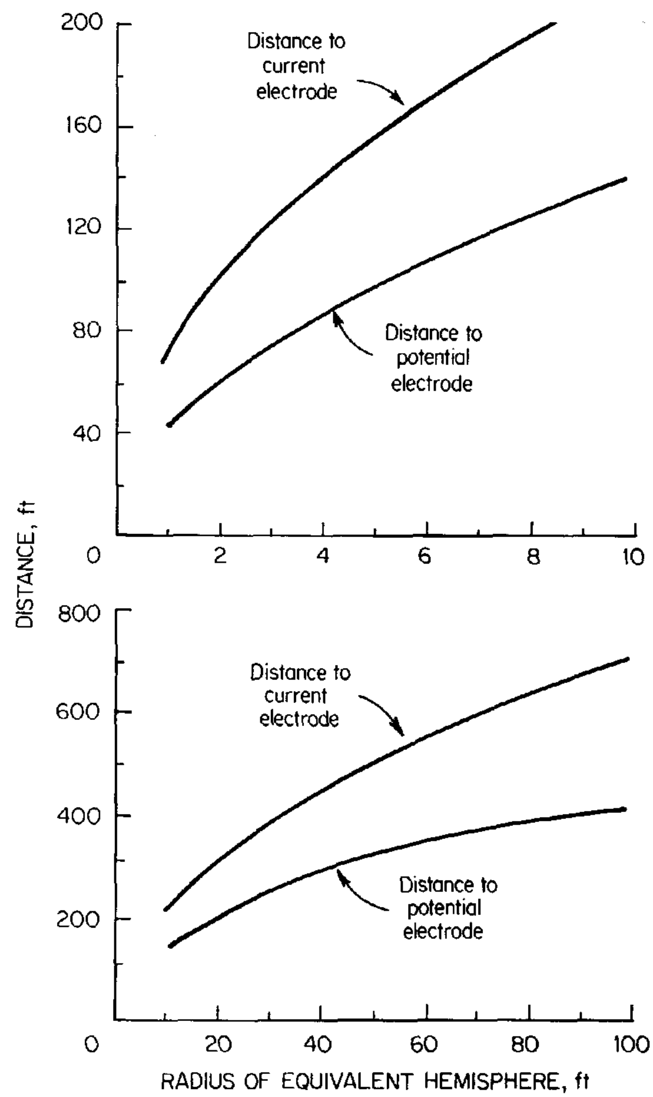


Figure 7.31.—Correct spacing of auxiliary electrodes to give true resistance within 2.0%.

outlined earlier may still be followed, that is, varying the potential electrode spacing while keeping the current electrode at some fixed spacing as far as possible from the grounding system. The true resistance may then be derived from the resulting graph using one of several available methods (*56*).

Ground Test Instruments

Certain precautions should be observed when a ground test instrument is chosen. A machine that uses dc should be avoided because of problems with polarization and electro-osmosis. Ac is satisfactory, but a frequency slightly removed from the actual power frequency is preferable so the effects of stray currents can be avoided. On the other hand, if the frequency used is too far removed from the power frequency, erroneous results may occur since ground resistance (impedance) varies with frequency (*45*). The leads from the instrument to the electrodes should be spaced as far apart as possible to minimize the effects of mutual inductance and capacitance. In a good instrument, the resistance of current and potential probes is not critical, but inferior equipment will give readings that vary widely depending upon the probe resistance. Great accuracy in measuring earth-ground resistance is not critical because the earth resistance measurement techniques themselves can never be precise or accurate.

GROUND-BED RESISTIVITY

In the discussions on resistance it was pointed out that soil resistivity, ρ, is an important parameter; specifically, ground-bed resistance is directly proportional to soil resistivity. The resisitivity of a material was defined in chapter 2 as the resistance in ohms between the opposite faces of a unit cube of that material. The value of resistivity varies widely depending upon the substance being measured; for rocks and minerals, it may range from 10^{-3} to 10^{17} Ω-cm. A general classification is shown in table 7.5 *(19)*. Efforts have been made to relate resistivity values to the geologic age of various rocks, as can be seen in table 7.6. As a rule, resistivity increases with rock age *(5)*, but there are exceptions *(54)*.

Rock structure enters into resistivity determinations, in addition to geologic age. The resistivity of a newly formed rock depends mainly upon the amount of water it contains. Young rock will generally have a large pore volume and hence a fairly significant quantity of connate water; therefore, it will exhibit a low resistivity. As time passes and the rock is subjected to forces that tend to consolidate, compress, or metamorphose it, the pore volume and water content will decrease, with a subsequent increase in resistivity *(5)*. Hard crystalline rocks are usually bad conductors, but if crushed or badly fractured, their resistivity may decrease because of greater porosity *(47)*. Resistivity values for some common soils are given in table 7.7 *(55)*.

When completely dry, most rocks and minerals are nonconducting, although some metallic ore bodies will carry current *(24)*. The main soil constituents have very high resistivities, and in fact, the oxides of silicon and aluminum are good insulators *(50)*. Figure 7.32 reviews the resistivities of some common rocks, ores, and metals *(47)*.

Factors Affecting Resistivity

Several factors can affect resistivity, and these are generally considered to include

- Moisture content,
- Dissolved salts,
- Temperature,
- Soil type,
- Grain size and distribution, and
- Location.

The level of influence for each is described in the following paragraphs.

Soil containing no moisture has a very high resistivity. The addition of water causes a sharp increase in conductivity, but the decrease in resistivity rapidly levels off once the moisture content of the soil reaches about 16 wt %, as shown in figure 7.33 *(55)*. Tests by the Bureau of Standards have indicated that resistivity increases markedly when moisture content falls below 20% *(36)*.

Table 7.5.—General resistivity classification

Conductivity characteristic of material	*Resistivity, Ω-cm*
Good	10^{-3}—10
Intermediate	10^{2}—10^{9}
Poor	10^{10}—10^{17}

Table 7.6.—Variations in resistivity with geologic age

Approximate resistivity, Ω-m	Quaternary	Cretaceous, Tertiary, Quaternary	Pennsylvanian, Mississippian, Triassic	Cambrian, Ordovician, Devonian	Pre-Cambrian and combination with Cambrian
1					
5		Loam			
10		Clay			
20		Chalk	Chalk		
30			Trap		
50			Diabase		
100			Shale		
300			Limestone	Shale	
500			Sandstone	Limestone	
700					Sandstone
1,000	Coarse			Sandstone	
1,200	sand				Quartzite
1,500	and			Dolomite	
3,000	gravel in				Slate
5,000	surface				Granite
10,000	layers				Gneiss

Table 7.7.—Typical values of resistivity of some soils

Type of soil	*Resistivity, Ω-cm*
Loams, garden soils, etc.	500– 5,000
Clays	800– 5,000
Clay, sand and gravel mixtures	4,000– 25,000
Sand and gravel	6,000– 10,000
Slate, shale, sandstone, etc.	1,000– 50,000
Crystalline rocks	20,000–1,000,000

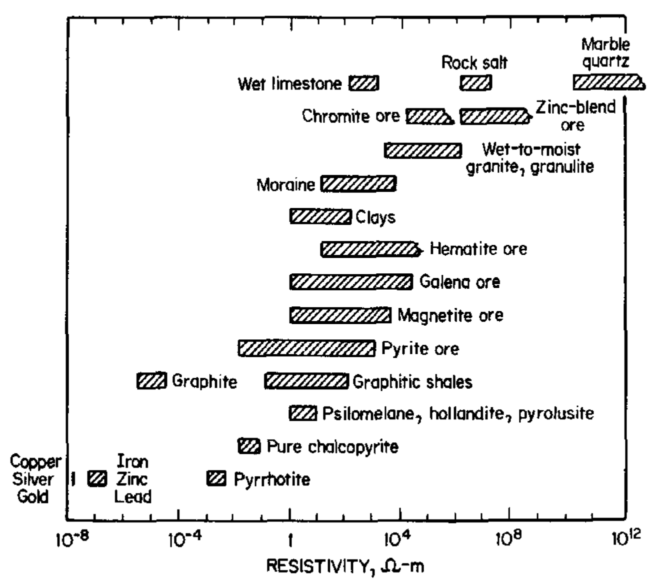


Figure 7.32.—Resistivity range of some rocks, minerals, and metals.

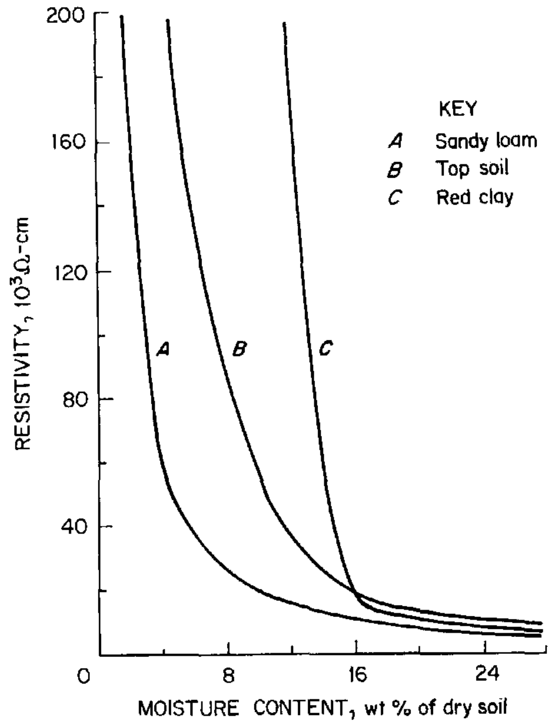


Figure 7.33.—Variation in soil resistivity with moisture content.

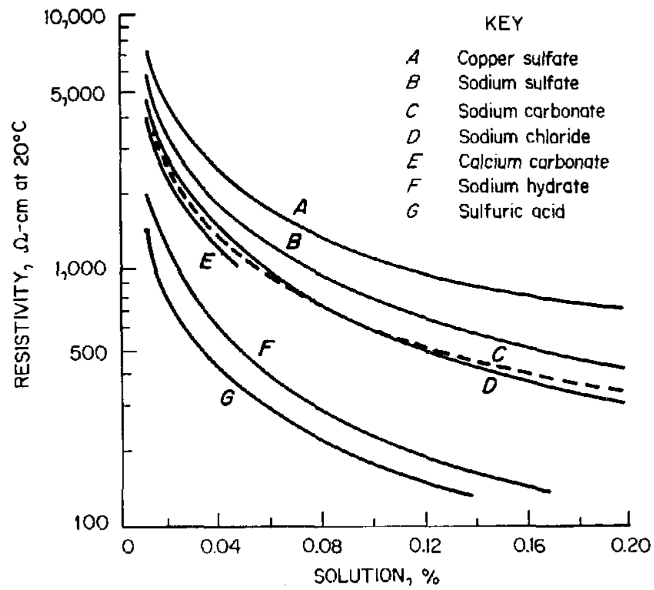


Figure 7.34.—Typical resistivity curves of solutions.

The conductivity of water is not a constant value, and it has noticeable effects on soil resistivity. Very pure water, such as may be found high in the mountains, has a poor conductivity, and as a result, mountain soil may be very wet and still possess a high resistivity (*24*). To a large extent, it is the dissolved salts present in the water that make the solution conductive. Conduction is electrolytic in nature; that is, current flows via the movement of positive and negative ions in solution. Thus, the concentration of dissolved salt, the particular type of salt, and the solution temperature all have an influence upon the degree to which a dissolved salt can lower soil resistivity. Figure 7.34 shows the effect of various salts upon resistivity (*55*).

Water has a large negative temperature coefficient of resistivity, and the transition from liquid to solid state is marked by a dramatic rise in resistivity (*31*). In addition, most electrolytes have a negative temperature coefficient of resistivity, amounting to about −2.0%/°C (*24*). Table 7.8 illustrates this effect (*34*).

Table 7.8.—Effect of temperature on resistivity of water

Temperature, °C[1]	*Resistivity, Ω-cm*	*Temperature, °C*[1]	*Resistivity, Ω-cm*
20	7,200	0 (ice)	30,000
10	9,900	−5	79,000
0 (liquid)	13,800	−15	330,000

[1] To convert to degrees Fahrenheit, multiply by 9/5 and add 32.

When a very high impulse current such as a lightning stroke enters a ground bed, the resulting voltage gradient may be so high that the soil breaks down. These current levels can be extremely damaging to the soil. Lower current levels flowing into a ground system for extended periods may heat the soil to the point where most of its moisture will evaporate. When this condition is reached, soil resistivity increases drastically.

Different soils are characterized by various resistivity levels (table 7.7). To a large extent, this is due to the previously discussed effects of structure as it pertains to conductivity. Loams and clays possess a low resistivity, while shales, sandstones, and crystalline rocks occupy the high end of the scale (*50*).

The nature of the particles making up the soil or rock is another aspect of rock structure, which influences conductivity through the rock's ability to trap and retain water. Surface tensions cause water to cling to large soil particles or grains; with small-grained substances, moisture simply fills up the multitude of pore spaces between individual particles. The range of particle sizes and their packing determines how much of the volume occupied by a particular soil will be void space and thus available for filling by water. If most of the grains are the same size, total pore volume may range from 26% to 46%, depending upon the manner in which the grains are packed (*19*).

If a particular rock structure or formation is confined to a small geographical area, then it probably has a fairly uniform resistivity, excluding areas of subsequent igneous activity. Should the formation be widespread, however, chances are that variable resistivities will be noted depending upon location. This is due to the differences in local conditions that may have prevailed over a small area during actual deposition or formation of the rock strata. This may also be caused by variations in the ground water properties from place to place within a large region (*5*).

Resistivity Measurements

The basic procedure for measuring soil resistivity involves the determination of the potential gradient on the earth's surface caused by flow of a known current through the area.

To illustrate the basic technique, assume an earth structure composed of two horizontal layers, the top one of high resistivity, ρ_1, and the lower one of low resistivity, ρ_2, as shown by figure 7.35 (*19*). The thickness of the upper layer is given by h. A power source forces current flow through the ground between the two outer electrodes. At very small electrode spacings, the apparent resistivity will approximate ρ_1 since most current flow would be confined to the upper layer. At very wide spacings (much larger than h), the apparent resistivity will be about the same as ρ_2, because the majority of the current would flow through the deeper layer.

Many methods are available for measuring earth resistivity, such as the techniques of Gish-Rooney, Lee, and Schlumberger. Most of these procedures are based on the arrangement described by Wenner (*58*), which is shown in figure 7.36 (*35*). Four uniformly spaced electrodes are used, and a current source is connected across the two outer terminals while the potential drop is measured across the inner terminals. When the electrode length b is small compared with the spacing a, then the resistivity is (*51*)

$$\rho = 2\pi aR, \tag{7.18}$$

where ρ = resistivity, Ω-m or Ω-ft,
a = spacing between electrodes, m or ft,
and R = resistance = V/I, Ω.

Some problems that may arise from the use of this method are

- Stray currents due to leakage as from motors,
- Natural currents due to electrolysis of nearby minerals,
- Polarization due to use of a dc source,
- Inductance between the lead conductors, and
- Leakage from the conductors and the instrument when in wet areas.

The first three problems are circumvented through the use of an ac source operating at the nonpower frequency of an instrument that generates the equivalent of a square wave. The use of a well-insulated instrument and conductors solves the latter two difficulties. The megohmmeter has all these features and is an excellent apparatus for use in work of this type.

To perform a resistivity survey, the megohmmeter is set up as shown in figure 7.36, the instrument is operated, and a resistance value R is read from the built-in meter. The procedure is then repeated at different electrode spacings. A graph may be made comparing the resistivity, ρ, with the electrode spacing, a, as shown in figure 7.37 (*55*). For each value of electrode spacing, there is a corresponding value of resistivity, ρ_a, seen by the instrument. This apparent resistivity is equal to the resistivity that a semi-infinite homogeneous earth would display at an equal electrode spacing and an identical value of R. In the example shown, the apparent resistivity decreases as electrode spacing increases. The overall shape of the curve indicates that the soil here is composed of two horizontal layers, with the overlying horizon having a higher resistivity then the lower one. As the electrode spacing, a, is increased, more and more of the current flow between the outer electrodes occurs in the deeper layer of the soil, and this is reflected in the continuous decrease in the apparent resistivity (*5*).

In a case like the one just described, a grounding grid composed of deeply driven vertical rods would be best, since the rods would penetrate into the underlying layer of

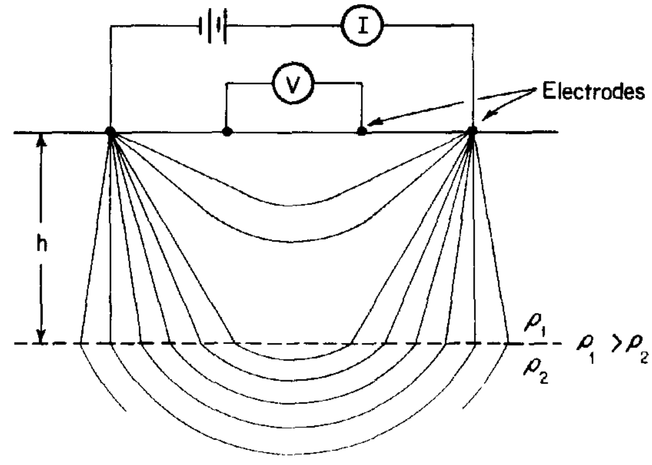


Figure 7.35.—Diagram for four-electrode resistivity survey showing lines of current flow in two-layer earth.

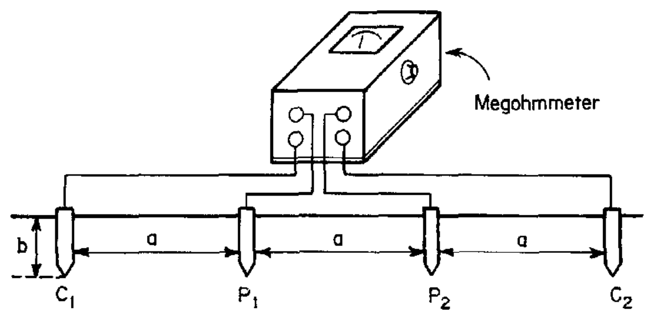


Figure 7.36.—Connections for Wenner four-terminal resistivity test using megohmmeter; distance a should be at least 20 times b.

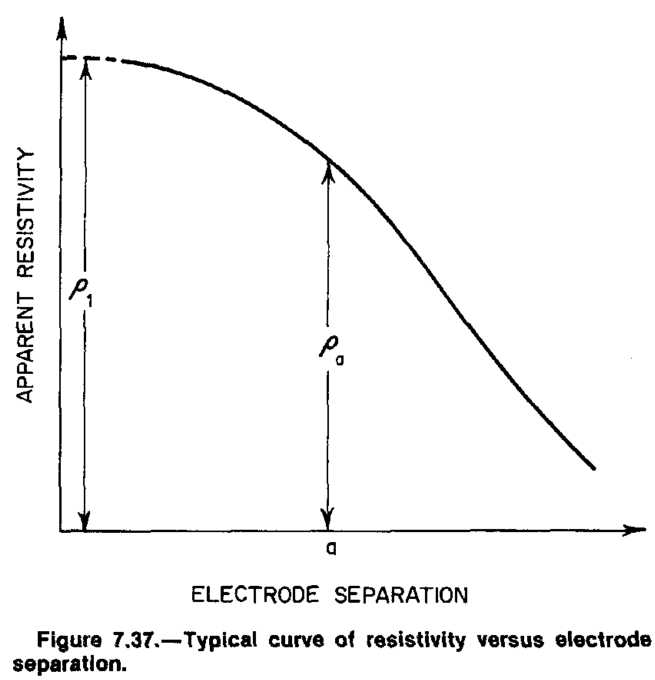


Figure 7.37.—Typical curve of resistivity versus electrode separation.

higher conductivity and thus provide a more effective ground. Additionally, soil horizons near the surface are usually subject to wide seasonal variations in resistivity due to changes in ambient temperature and moisture (*40*).

Tagg (*55*) presents several methods whereby an accurate interface-depth determination may be calculated. Values are read from a standard graph, and multiple calculations are then performed, followed by another graph construction from which the correct depth is read. Core drilling has verified that values derived in this manner agree closely with the actual conditions.

Effect of Chemical Treatment of Soils

The natural resistivity of some soils is so high that it is virtually impossible to construct a ground bed with a satisfactorily low value of resistance. By injecting into the earth a substance whose resistivity is very low, the local soil resistivity can be effectively reduced, thereby lowering the resistance of a grounding grid. Such chemical treatment acts to increase the apparent dimensions of the metallic electrodes (*7*). The result of chemical treatment is to reduce ground resistance by a considerable amount, often as much as 15% to 90%. Figure 7.38 shows an example of this effect (*36*). Generally, the percentage improvement is greater for a very high resistance ground.

Substances traditionally used as chemical additives include sodium chloride, calcium chloride, copper sulfate, and magnesium sulfate (*36*). Newer additives include gels composed of acrylamide, silicic acid, or copper ferrocyanide. In the past, electrodes were sometimes surrounded by a bed of coke, not a true chemical treatment but rather a partial soil substitute (*24*). The effectiveness of most treatments in lowering ground-bed resistance is about the same, with the ultimate selection depending upon the criteria of cost, availability, and corrosive properties.

A prime disadvantage shared by most chemical treatments is the fact that they will corrode most metals (*7*). Magnesium sulfate has little or no corrosive effect, and graphite is also innocuous. Other additives generally speed up the decay of grounding electrodes.

Another disadvantage is that chemical treatments are dissipated and carried away by neutral drainage through the soil (*36*). Acrylamide gel, which is not water soluble, is an exception (*34*). The rate at which chemical additives are washed away depends upon the soil type and porosity as well as the amount of rainfall. Useful life may range from 6 months to 5 or more years.

The cost of chemical treatment may be higher than the price of driving longer ground rods to reach deeper, lower resistivity soil layers, but in some instances it is not feasible or desirable to increase penetration depth. As shown in figure 7.39, the seasonal variations in resistance that are exhibited by grounding grids because of temperature and moisture fluctuations, are attenuated in those cases where chemical treatment has been applied (*36*).

The best method of application, illustrated in figure 7.40, is to dig a circular trench about 1 ft deep and with an inside diameter of 18 in around each ground rod (*36*). The additive is placed into the trench and then covered with earth. The area is then flooded with water to initiate the solution process. In this manner, the solution can permeate a greater volume of soil, while any corrosive action is minimized.

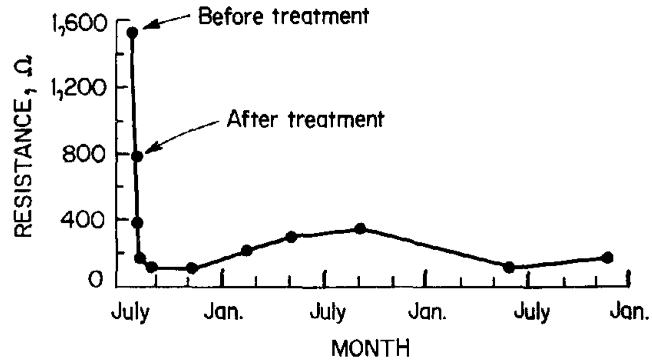


Figure 7.38.—Reduction in ground mat resistance by soil treatment.

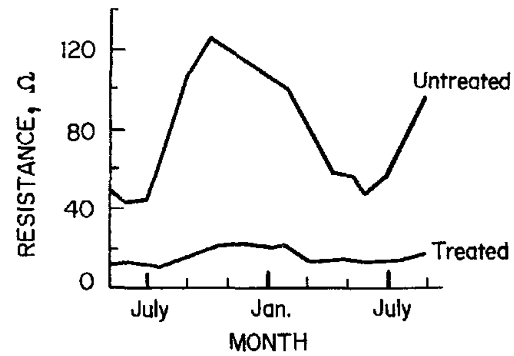


Figure 7.39.—Seasonal resistance variations attenuated by soil treatment.

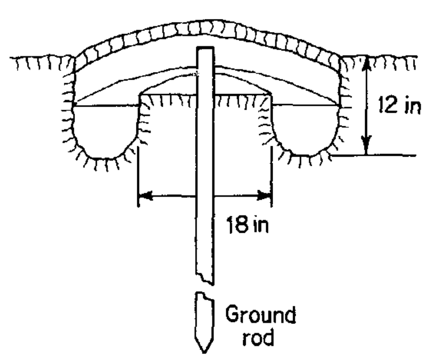


Figure 7.40.—Trench model of soil treatment.

GROUND-BED CORROSION

Corrosion is a phenomenon that must be considered in the design of a ground bed. There are three basic ways by which underground corrosion can occur (*52*):

- Dissimilar metals connected together electrically and surrounded by an electrolyte such as soil,

• Dissimilar electrolytes in close proximity to the same piece of buried metal, and

• Stray electrical current leaving a buried metal structure.

In the first mode, variations in electrochemical potential provide the key to the dilemma. The standard half-cell, upon which most corrosion work is based, consists of a copper rod bathed in a saturated copper sulfate solution. When measured with reference to the copper and copper sulfate half-cell, each metal displays a certain characteristic potential, as shown in table 7.9 (*61*). If two metals are joined and immersed in soil, the one whose potential is more negative will discharge current and be corroded, but the more positive (noble) species will collect current and be protected. When only one metal is used, corrosion can still occur because of differences in soil composition. Metal in an oxygen-rich zone will be protected, while metal in a relatively oxygen-poor soil horizon will be attacked. Foreign metallic structures in the grounding-grid vicinity, such as pipes, cable sheaths, and building frames, may also act in conjunction with the ground bed to form an anode-cathode corrosion situation.

Table 7.9.—Typical potentials of metals in soil measured from a copper and copper sulfate reference electrode

Metal	*Potential, V*
Magnesium	−2.5
Aluminum	−1.3
Zinc	−1.1
Iron	−.7
Copper	−.2

The engineer designing a ground-bed system is faced with the problem of solving two conflicting sets of demands. For safe grounding, a very low resistance is desired between the soil and the buried metallic grid. To eliminate potential-gradient hazards, all metal structures should be tied together. However, protection requires that underground metallic structures be insulated from the corrosive effects of the soil. Similarly, the soil and metallic structure should be isolated from one another (*61*). This seeming paradox may be remedied by making the correct choice of ground-bed conductor and by applying suitable preventive techniques.

Copper makes an ideal ground-bed conductor since it is corrosion resistant, has a high electrical conductivity, and is easy to clamp or weld (*61*). However, a good all-copper system is often ruined by tying it together with noncopper structures in the same locale, thereby leading to the corrosion of the less noble species (*52*). If the ground bed must be located in an area where steel or lead are present, two options are available. First, an insulating coating may be applied to the base metals. If this is not feasible, an all-steel grounding system is preferable, or one composed of steel rods connected with insulated copper wire (*8*). The idea here is to minimize the exposed surface area of the more noble metal. Normally, steel electrodes can be improved by applying a heavy zinc coating or by driving zinc electrodes in addition to the steel. Known as sacrificial anodes, the zinc conductors will be preferentially attached, thereby protecting the steel members. For extra protection, magnesium may be used instead of zinc. In highly corrosive soils, it may be necessary to utilize an external power source that supplies dc to the soil in order to nullify the natural corrosion currents. This is known as cathodic protection (*41*). For externally driven anodes, zinc or magnesium may be used; graphite and high-silica cast iron are also suitable.

It may be seen that judicious choice of grounding materials and the use of corrosion-prevention techniques such as cathodic protection can provide a ground bed that is both low in resistance and high in longevity.

GENERAL GROUND-BED GUIDELINES

The primary objective of a grounding system is "to limit the potential rise above ground that appears on the frame and enclosures of the equipment connected to the power system" (*30*). Consequently, the station ground and safety ground beds should be spaced at least 50 ft apart, even though the law presently permits only a 25-ft separation (*21*). A typical voltage-gradient representation is shown in figure 7.41 (*30*). The two ground beds must be far enough apart so current surges through the station ground bed will not cause the safety ground bed to rise to more than 100 V above infinite earth.

Once the site has been selected, the excellent guide developed by King (*39*) can be used for the design and construction of low-resistance driven-rod ground beds. The simplified procedure consists of the following four steps.

1. Using the Wenner array, earth resistivity is measured along the two lines at right angles to each other, centered across the proposed ground-bed site. Two measurements, with 6-ft and then 18-ft spacings, are taken along each line.

2. Depending on the magnitude and homogeneity of the resistivities measured, the rod length, number, and arrangement are selected from tables. These tables (*39*) are based on the same information presented earlier in this chapter but are too extensive to be reproduced here.

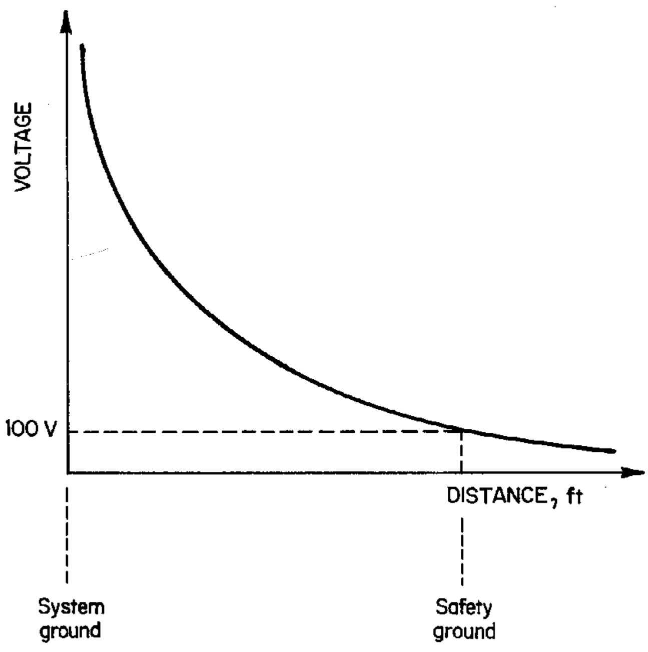


Figure 7.41.—Voltage gradients in earth during ground-fault conditions.

3. The selected rod configuration is driven, and the rods are interconnected with flexible, bare copper conductors. Recommended size is 4/0 AWG, and connection should be clamped or brazed, but never soldered (*1*).

4. The completed bed is measured by the fall-of-potential method to check that its resistance is below 5.0 Ω. If it is more, a new resistivity, ρ_n, is calculated by

$$\rho_n = \rho_o \frac{R}{5}, \tag{7.19}$$

where ρ_o = old resistivity,
and R = measured resistance.

Again, using the tables, additional rods are selected and then driven. Afterward, the resistance is again measured. Whatever the procedure used to construct the bed, the resistance should be checked not only when it is installed but periodically thereafter to ensure that it is still functioning properly.

GROUNDING EQUIPMENT

The basic resistance-grounded system consists of a resistance inserted between the power-system neutral point and ground. Specific concerns when selecting the grounding resistor are resistance, time rating, insulation, and connection. A problem also exists if there is no available connection to the power-system neutral.

Grounding Resistor

The ohmic value of the resistor is determined by the line-to-neutral system voltage and the maximum ground-current limit. As stated earlier, when portable or mobile equipment is involved, the maximum limit on low-voltage and medium-voltage systems is 25 A, and the upper current limit on high voltage is set by the grounding-conductor resistance, because flow through this conductor cannot cause any machine frame potential to be elevated more than 100 V above earth potential. However, high-voltage limits are typically chosen at 25 or 50 A (50 A is the maximum allowed in some States). For instance, if grounding-conductor resistance is 3.3 Ω, and maximum allowable ground current is 30 A, then 25 A is normally chosen. When the resistance-grounded system is feeding only stationary equipment, there is no specified maximum ground current, but industry practice sometimes specifies low-resistance grounding with a 400-A limit. For all applications, sizing the ohmic value of the grounding resistor is simply performed by dividing the line-to-neutral voltage by the selected ground-current limit; conductor impedance is neglected. The technique is justified by the method of symmetrical components for a line-to-neutral fault.

Ground current can be limited at a level less than the restricted maximum, but for high-resistance grounding the smallest value chosen has two concerns: ground-fault relaying and charging current. For maximum safety, ground protective circuitry should sense ground current at a fraction of current limit (see chapter 9). Hence, reliable relay operation with electromechanical devices can be a problem if maximum current is less than 15 to 20 A. The other limitation is that ground-fault current should always be greater than the system-charging current (*59*), the current required to charge system capacitance when the system is energized (see chapter 11). When very low ground-relay settings are used, the charging current may itself cause tripping.

The second main concern in selecting a grounding resistor is its time rating, or the ability to dissipate heat. A grounding resistor carries only a very small current under normal system operation, but when a ground fault occurs, the current may approach full value. The high current exists until the circuit breaker removes power from the faulted circuit, which may take from a fraction of a second to several seconds depending upon the protective circuitry used. With correct fault removal, the physical size of the resistor can be small, as very little heat is produced. However, protection devices have been known to malfunction, and in these instances ground current might continue to flow until the power is removed manually. Thus, the resistor must be able to dissipate the power produced from full ground current for an extended time when portable or mobile equipment is involved. If not, the resistor can burn open and unground the system. Two ratings that ensure safety are continuous and extended time. These are essentially the same, since the extended-time rating refers to a heat-dissipation ability for 90 days per year (*32*).

To provide a safety margin, the transformer-neutral side of the resistor (often called the hot side) must be insulated from ground at a level to withstand the line-to-line system voltage. Both resistor ends are at ground potential with normal operation but under a ground fault, the transformer end can approach line-to-neutral potential. To afford good insulation, it is recommended that the resistor frame be placed on porcelain insulators, not temporary supports such as wooden blocks. Furthermore, for wye-connected secondaries, the transformer-neutral bushing must be insulated to at least line-to-neutral voltage.

The last concern is the resistor connection. The grounding resistor is installed between the transformer neutral and the safety ground bed. In substations it is important to use insulated conductors, because bare conductors can easily compromise the required separation between the system and safety ground beds. Grounding conductors must extend from the ground-bed side of the resistor. Finally, to minimize resistor conductor lengths, the resistor must be located on the power-source end of distribution, as close as possible to the source power transformer. Distances greater than 100 ft are usually too long.

Grounding Transformers

Delta-wye, wye-delta, and delta-delta power transformers are extremely important in mine power distribution because they offer very high impedance to zero-sequence currents. As a result, a ground fault existing on the secondary will do no more than raise primary line current. However if the transformer has a delta secondary, there is no neutral point to which the grounding system can be connected. Another case where this occurs involves mines where the utility company owns the substation and supplies ungrounded delta power. For both these situations, a separate grounding transformer is needed to obtain an artificial neutral. The two types of grounding transformers in general use are the zig-zag and wye-delta, with the former being more popular.

As shown in figure 7.42, the zig-zag is a special three-phase transformer designed for deriving the neutral.

The transformer winding interconnections are such that a very high impedance is shown for positive-sequence and negative-sequence currents but a very low impedance is exhibited during zero-sequence flow.

A wye-delta grounding-transformer bank uses three identical single-phase transformers (fig. 7.43). The primary windings, rated at line-to-neutral voltage, are connected in wye among the power-transformer secondary terminals and the grounding-resistor hot side, and the secondaries are connected in delta. Any secondary voltage rating can be used. Normally, no secondary current will flow, but during a ground fault, current will circulate in the secondary. This will cause the ground to be shared by the three transformers such that the neutral point will remain at constant potential.

Grounding-transformer capacity only needs to be large enough to carry the maximum ground-fault current. Grounding transformers' primaries cannot be fused, as an open fuse will essentially unground the system, creating a dangerous situation.

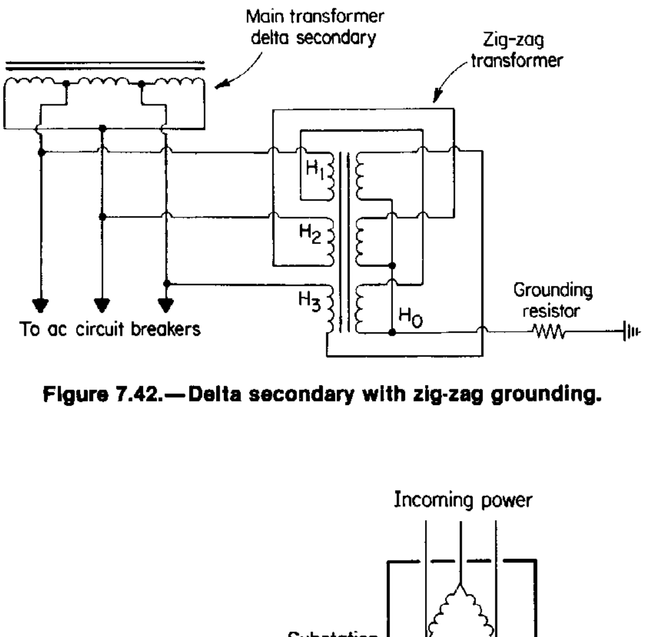


Figure 7.42.—Delta secondary with zig-zag grounding.

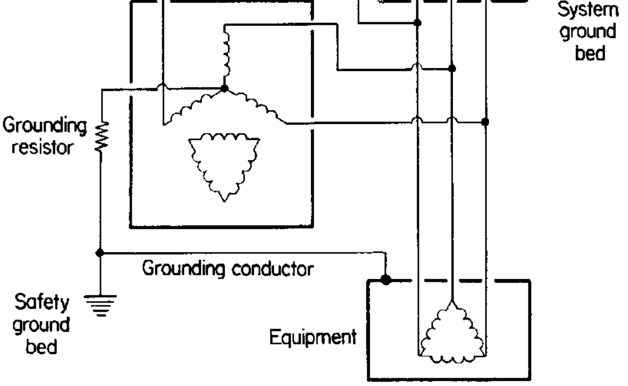


Figure 7.43.—Delta secondary with wye-delta grounding transformer.

SUMMARY

Several basic grounding methodologies exist, and each has its merits. The resistance-grounded neutral system is superior for mining applications involving portable or mobile equipment. The design of ground beds is a complex field, and many variables must be examined in an attempt to derive an optimum configuration. A low value of resistance is of primary importance so dangerous potentials are not developed on machine frames. High potential gradients in the ground-bed area must also be avoided to prevent injury to personnel. A study of electric shock and its effects on humans is helpful in further delineating this subject. Formulas have been presented that may be used to predict the earth resistance of a particular metallic array or to determine how much buried metal is needed to achieve a desired value. In order to verify the ground-bed earth resistance, a description of ground test instrumentation, its utilization, and data interpretation was also included. When designing a ground bed, corrosion effects and soil-heating phenomena, caused by current flow in the ground system, must be considered.

The resistivity of the soil in which the ground bed is immersed has a significant effect upon its earth resistance. Resistivity in turn is influenced by other factors such as earth composition, temperature, and moisture, and a thorough understanding of these relationships will be of use in metallic grounding-network design. Instrumentation was again discussed, as well as practical applications such as the determination of the best location for a ground bed. Chemical treatment of soils to increase conductivity and attenuate seasonal resistivity variations was reviewed.

Correct selection and coordination of protective circuitry is essential to gain the full benefits of a low-resistance ground bed. Protective circuitry must be installed to monitor current flow in the ground conductors or the potential drop across the neutral grounding resistor. When properly coordinated, this protective circuitry will quickly shut down faulty sections of the electrical system.

In the event of a fault or short circuit on a piece of mine machinery, its frame may become hot or elevated above ground potential. An unsuspecting miner could be seriously injured or killed if the machine is touched. Fast-acting relays and circuit breakers will minimize the length of time during which this shock hazard exists, and the bad circuit will be isolated from the remainder of the system. These protective devices form the subject of chapters 9 and 10. The grounding conductors that tie equipment frames to the safety ground bed are discussed in the next chapter, "Distribution."

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CHAPTER 8.—DISTRIBUTION[1]

The distribution system within a mine consists of various types of cables that connect equipment to power supply, the conductors that form the trolley system used in many underground mines, and the overhead lines that distribute power in some surface mines. The character of the mining operation imposes constraints on the distribution system unlike those of other industries and magnifies its importance within the overall power system. Mining is by definition constantly mobile; hence, the distribution system must be handled and extended frequently and can be susceptible to damage from mobile equipment. The mobility in turn necessitates efficient methods for joining cables and repairing them in order to minimize production downtime and operating costs. In all mines there is the potential for electric shock when handling distribution components. In the hazardous environment of an underground coal mine, damaged systems can be a potential fire and gas-ignition source. Proper installation and correct handling practices are essential if these hazards are to be minimized.

This chapter's purpose is to introduce the various distribution components used in mine power systems, as well as to discuss their construction, installation, and maintenance. Cable systems are covered first and comprise the majority of chapter content because of their uniqueness to mining. Typical trolley-system arrangements are then presented, and the chapter is concluded with a brief introduction to overhead lines.

NATURE OF CABLE DISTRIBUTION

It was shown in chapter 1 that cables can carry the electricity from the substation, where the power is taken from utility company lines, to the point of utilization by a mining machine, pump, conveyor belt, or other piece of equipment. There are many possible variations in mine distribution, and several types of cables can be put to a similar use. Only the most typical schemes are covered in this chapter, but some notable exceptions are included.

Representative systems are depicted for underground coal mines in figure 8.1 and for surface coal mines in figure 8.2. Obviously, the circuits shown in the figures are only simplified examples of actual mine systems. In practice, an underground coal mine would not have one longwall, one continuous mining section, and one conventional section, but several continuous mining sections or several conventional sections in addition to one or more longwall units. Surface mines would usually have more than one dragline and one stripping shovel, not necessarily all electrically powered.

As might be supposed, the kind of cable is tied to the application. Examination of figures 8.1 and 8.2 indicates that some cables remain in stationary locations for several years, while others are moved frequently. The cables that are connected to mining machines are termed *portable* by the Insulated Cable Engineers Association (ICEA) standards (*19–21*).[2] The Code of Federal Regulations uses the term *trailing cables* for the specific variety of portable cables used in a mine (*38*). Trailing cables are flame-resistant flexible cables or cords through which electrical energy is transmitted to a machine or accessory.

In underground mines, trailing cables are generally attached to the inby end (toward the face) of the power center or distribution box. The portable cables that feed the power center or are attached to the outby end (toward the portal or shaft) have to be moved when the power center is advanced and retreated (perhaps once every 2 weeks), but they are not moved as often as the trailing cables. The most stationary cables are those that bring power into the mine, for instance down the borehole and from the borehole to the portable switchhouses. These are the feeder cables. A special type, designated *mine power feeder*, can be used for installations that may not be moved for several years. However, the use of the word *feeder* here is to denote a cable type rather than a function in distribution. Both feeder and portable cables can be used for feeder applications, where the cable supplies two or more major loads (*38*).

[1] The author wishes to thank Robert H. King, who prepared original material for many sections of this chapter. Thanks are also extended to James N. Tomlinson, who assembled the original section on splicing, and to George Luxbacher, who assembled the original material on conductor ampacities and cable derating.

[2] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

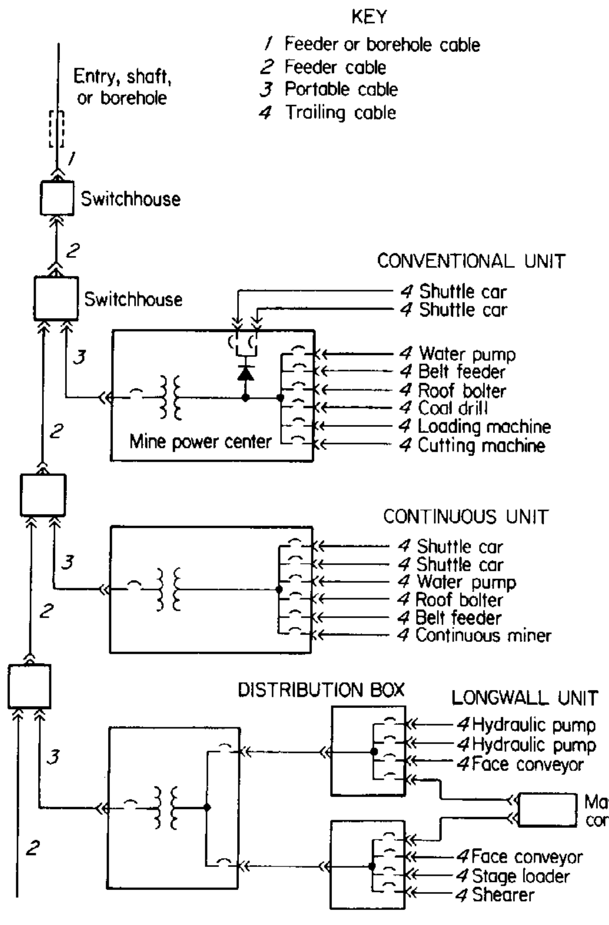


Figure 8.1.—Cable distribution in underground coal mines.

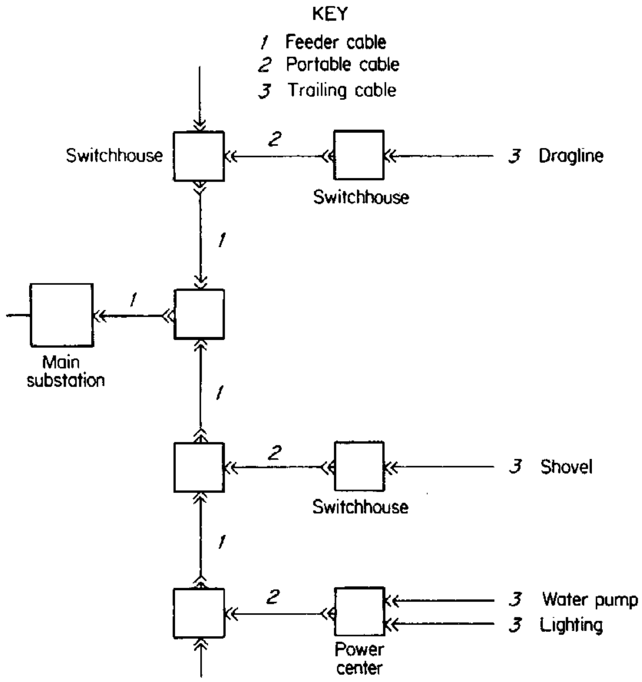


Figure 8.2.—Cable distribution in surface coal mines.

Similarly, in surface mines, the cables that feed from the switchhouses or unit substations to mobile equipment are trailing cables. Those moved only occasionally, which are not connected directly to a machine, are portable cables. Stationary (or near so) cables can be feeder or portable types.

Moving the cable is a constant task both under and above ground. Some trailing cables are placed on reels or spools to facilitate moving. Prime instances of reeled cables are cables associated with the reeling devices on board shuttle cars and with mobile cable reels used in conjunction with many draglines. Trailing cables without reels are usually termed drag cables. Regardless of the application, cables are heavy and cumbersome and must often be manipulated by hand. Although the most frequent personnel injuries are strains, bruises, and fractures, cable handling is always potentially hazardous, and investigations in mines have indicated that exposed "live" conductors are a too-common occurrence. Indeed, most fatalities in cable-handling accidents are a result of routine handling of unshielded cable (*25*).

Constant handling also imposes considerable stress on the cables. While cable life is rated by manufacturers at up to 20 yr for other industrial applications, in an underground mine the actual cable life does not even approach this. Mine personnel have estimated the life of continuous miner cables at 8 months; roof bolter cables at 7 months; and shuttle car cables at 3 months (*25*), and within this lifespan the cable usually requires frequent repair. It has been estimated, for example, that 75% of the total machine downtime for shuttle cars is cable related.

CABLE COMPONENTS

Cables are made up of three basic components: the conductor, the insulation, and the jacket, although there may also be fillers, binding, shielding and armor. In basic cable construction, the conductors are surrounded by insulation and the jacket covers the insulation. The design of these components is heavily dependent upon the physical stresses that the cable must withstand in the mine environment, including tension, heating, flexure, abrasion, and crushing. Hence, a discussion of typical stresses is helpful prior to describing component specifics, cable types (the various component assemblies into cables), and cable coding.

High cable tensions are characteristic of both drag and reeled cables. When combined with other stresses such as flexure and twisting, tension can be very harmful to cable life. Drag cables are pulled around pillar corners, through mud, and over jagged rocks where the drag resistance is high. Consequently, a considerable force can be required to drag the cable and, thus, high tensions can develop.

Machinery that utilizes cable-storage reels also frequently causes excessive cable tensions (*13*). For instance, the stored cable on the shuttle car is either payed out of the reel or spooled up into the reel as the machine is trammed. The tension required is dependent upon mine conditions, machine type, and cable size, but must be sufficiently high to prevent running over or pinching slack cable. However, if tensions become too high as a result of sudden jerks on the cable, cable and splice failures can become excessive. In addition, instantaneously high cable tensions can result in cable whipping. This is common with shuttle cars and also occurs on other machines that utilize cable reel storage devices, such as roof bolters, coal drills, and cutting machines. This whipping action is a hazard to mine personnel, who may be struck by the cables as they handle the cables or work nearby.

In addition to excessive cable tensions, high cable temperatures frequently occur on machinery that utilizes cable-storage reels (*6*). The cable is wound on the reel, layer upon layer. Such layering prevents the cooling action of circulating airflow, and heating occurs. Consequently, the cable jacket and insulation may become softened and more susceptible to damage from cutting, tearing, and abrasion. If excessive temperatures occur, the cable jacket and insulation can actually blister or crack, becoming brittle. Thus, the physical damage caused by heating poses another hazard to mine workers who must handle the cable, especially in a wet mine environment.

Another common cable stress prevalent in all mining cables is cable flexure. As with any material that is bent, internal tension and compression occur in flexed cables. These stresses cause relative movement of individual wire strands, abrading one wire against another and gradually deteriorating the conductors. Stresses fatigue the conductors, making them brittle and more susceptible to further damage.

Abrasion is also deleterious to cables and can have severe consequences. Cutting or tearing can occur when the cable becomes snagged or caught on rock, nails, and so on (*6*). Ripping or tearing of the cable jacket and insulation often results. Such damage can cause immediate cable failure, but more often than not, the damage goes unnoticed. In a wet environment, water penetration can create a current path to the outer surface of the cable. An individual could come in contact with the wet cable several feet from a damaged area and still receive a shock that might be fatal.

Another important cause of failure is cable crushing (*6*). This is usually the result of runovers or pinching the cable with a machine frame. Here, the conductors are compressed against one another or against the machine, causing the insulation and jacket to split, as well as damaging the conductors. Even if there is no immediate failure, line-to-line or line-to-neutral faults that result in nuisance tripping of the protective circuit breakers can occur later. Water penetrating into damaged areas of the jacket can eventually work into areas of damaged insulation causing short circuits or a safety hazard.

Conductors

Line and ground currents are carried by either copper or aluminum conductors, depending on the specific characteristics required. Copper has high conductivity, is heavier and more flexible, but also more expensive. Because of its greater flexibility, it is used in all portable mining cables.

Copper cable conductors are usually composed of many fine wires combined into strands. Varying numbers of strands form the conductor. At the cable manufacturing plant, a cold-drawing process is used in which the copper rod passes through successively smaller dies to reduce its diameter (*5*). This process hardens the copper and makes it less flexible, so that if a soft-temper copper (strength about 24,000 psi) is required, the wire must be annealed. Conductors that require a high tensile strength but are not bent frequently use medium- to hard-temper copper; medium-hard is rated at 40,000 psi.

Copper conductors can become annealed in service if they are used at high operating temperatures for long periods of time. In fact, copper can lose 5% of its original tensile strength in 10,000 h at 70°C (*5*). Cable manufacturers should always be consulted about the capability of their products to resist annealing when installed as borehole or high-tension overhead cables. To prevent corrosion by insulation vulcanizing agents, copper strands are usually coated or tinned with lead or tin alloys, though this reduces the surface conductivity.

Aluminum conductors are also used in mines. Aluminum is cheaper, lighter, and less flexible, and has lower conductivity than copper. Aluminum conductivity is 61% that of copper; therefore, an aluminum conductor must have a cross-sectional area 1.59 times that of copper to have an equivalent dc resistance. However, copper conductors weigh 3.3 times as much as aluminum; so even though the cross-sectional area of an aluminum conductor is greater, the total weight of an equivalent-resistance aluminum conductor is less. Poor flexibility eliminates the use of aluminum in trailing cables. Aluminum is sometimes used for feeder cables because of its lower cost, but problems can arise in jointing. An improperly constructed joint can allow the formation of aluminum oxides, which increase resistance and cause heating at the connection. Extreme care must also be taken to exclude moisture from any copper-to-aluminum joints because of the potential for electrolytic corrosion of the aluminum.

Conductor Sizes

The cross-sectional area of conductors is important for mechanical strength and is closely related to current-carrying capacity. Since the proper capacity is both a legal requirement and a desirable practice for safe operation, an understanding is needed of the methods commonly used to specify cross-sectional areas and ampacities. In the United States, both the American Wire Gauge (AWG) (or Brown and Sharpe Gauge) and circular-mil designations (MCM) are used (1 cmil is the area of a circle that is 1 mil in diameter). The AWG specifies 38 steps or sizes between No. 36, which is 0.0050 in. in diameter, and No. 4/0, which is 0.4600 in. in diameter (*5*). These sizes closely conform to the steps of the wire-drawing process. Table 8.1 specifies the cross-sectional areas and equivalent circular-mil sizes for some of the AWG designations. The 38 intermediate sizes are calculated in a geometric progression relating the ratio of any diameter to the next smaller or larger by:

$$\sqrt[39]{\frac{0.4600}{0.0050}} = 1.1229. \qquad (8.1)$$

Table 8.1.—Conductor sizes and cross-sectional areas

Conductor size	Cross-sectional area, cmil	Cross-sectional area, in²	Conductor size	Cross-sectional area, in²
AWG:			MCM:	
22	640	0.000502	250	0.197
20	1,020	.000804	300	.236
19	1,290	.00101	350	.274
18	1,620	.00127	400	.315
17	2,050	.00161	450	.353
16	2,580	.00203	500	.392
15	3,260	.00255	550	.432
14	4,110	.00322	600	.471
13	5,180	.00408	650	.510
12	6,530	.00513	700	.550
11	8,230	.00646	750	.589
10	10,380	.00808	800	.628
9	13,090	.0102	900	.707
8	16,510	.0129	1,000	.786
7	20,820	.0164	1,100	.863
6	26,240	.0206	1,200	.942
5	33,090	.0260	1,250	.981
4	41,740	.0328	1,300	1.02
3	52,620	.0413	1,400	1.10
2	66,360	.0521	1,500	1.18
1	83,690	.0657	1,600	1.26
1/0	105,600	.0829	1,700	1.33
2/0	133,100	.1045	1,750	1.37
3/0	167,800	.1318	1,800	1.41
4/0	211,600	.1662	1,900	1.49
			2,000	1.57

Short-cut conductor-size approximations can be made by applying some simple rules if a table is not available. For example, the diameter will be doubled or halved by moving six sizes up or down the table. The weight, area, and dc resistance is doubled or halved by moving three gauge sizes, and they are changed by a factor of 10 over 10 gauge sizes. A convenient reference point from which to apply these rules is the No. 10 wire since its diameter is about 0.1 in., its dc resistance is nearly 1 Ω per 1,000 ft, and it weighs 10π lb per 1,000 ft. In applying these rules, it should be remembered that the outer diameter and weight of conductors depends on the stranding configuration, which is described below.

Federal regulations require grounding conductors to have at least one-half of the cross-sectional area of the power conductors (*38*). When the power conductor is a No. 8 AWG or smaller, the grounding conductor should be the same size as the power conductor. The ground-check (pilot) conductor must not be smaller than a No. 10 AWG (*38*).

Conductor Stranding

In order to obtain the required flexibility, mining cable conductors are made with numerous small wires rather than a single solid copper rod. These small wires are wound or laid together in strands, which are wound together in a rope in specific patterns. In a shuttle car cable, 37 wires are wound or bunched together, then 7 of these strands are spiraled together to form the conductor. Consequently, the total number of wires in this case is 259 and equals the number of strands multiplied by the number of wires in each. The cross-sectional area of a stranded conductor is defined as the sum of the area of its component wires.

In the simplest terms, conductor flexibility is greatest when the largest number of small-diameter wires is used. However, a certain amount of tensile strength is also required in mining cable conductors, and the tensile strength is greatest when a small number of larger wires is used. The design of a specific cable must therefore optimize these opposing factors, while taking into account the effects of twisting and bunching. Different applications obviously necessitate different configurations. The engineer must examine cable stranding specifications carefully and select the one that best suits the application. Where historical information is not available, several types should be tried to find the best performer. Flexibility is also influenced by the method of insulating the power and ground-check conductors and applying the overall jacket.

Insulation

Insulation of mining cables is required to withstand stress from heat, voltage, and physical abuse. The insulation must be specially designed not only to protect mine personnel from electric shock, but also to separate power and grounding circuits effectively.

Heating affects insulating materials in different ways, depending on their chemical composition. Heating either softens insulation, causing it to lose physical strength, or causes it to age or become brittle. Consequently, heat can make insulation lose its original shape, tensile strength, cut resistance, elongation, and effectiveness as an insulator. The main sources of heat are the environment, related to the ambient temperature, and power (I^2R) loss in the cable conductors. Hence, cable heating is directly connected to the maximum current the conductors can carry safely.

Cable manufacturers usually prefer to use a thermosetting insulation. After being extended over the conductors, this insulation changes chemically by vulcanizing into a material that softens very little within the rated temperature range. The most common insulating compounds in this group are neoprene, styrene butadiene (SBR), ethylene propylene (EPR), and crosslinked polyethylene (XLP). These compounds are usually mixed with other materials to achieve improved physical and electrical properties.

SBR is used in 600-V trailing-cable insulation. It has a high modulus of elasticity, good flexibility, and a 75°C temperature rating, and resists damage by crushing from runovers and rock falls. EPR has replaced SBR in many trailing cables because it allows the cable rated voltage to be increased to 2,000 V and the temperature rating to 90°C, while maintaining the same insulation thickness as SBR and neoprene. The EPR emergency-overload rating is 130°C, and the short-circuit rating is 250°C. XLP is also rated at 90°C for normal operation and is used in high-voltage (>1,000 V) mine-feeder and portable strip-mining cables. XLP is a rather stiff material, however, and is not recommended for reeling applications.

The cable voltage rating is closely associated with the maximum anticipated operating voltage. The most common ratings for mining cables are 600 V, 2 kV, 5 kV, 8 kV, 15 kV, and 25 kV. The 5-kV, 8-kV, 15-kV, and 25-kV ratings are used primarily for stationary feeder cables and are generally not connected to mining equipment, except in surface mines. Usually, 4.16-kV distribution requires 5-kV rated cables, 7.2 kV requires 8 kV, and 12.47 kV and 13.2 kV require 15-kV ratings. The utilization voltages of 250 Vdc, 440 Vac, and 550 Vdc usually call for 600-V or 2-kV cables, and medium-voltage applications (661 to 1,000 V) need 2-kV insulation.

The voltage rating of an insulation is actually based on its ability to withstand a test voltage that is many times the anticipated operating voltage, for a specified period of time. The test procedure and specifications are published in ICEA standards (*19–21*). Insulating compounds have different voltage ratings, which are usually expressed as the amount of voltage they can withstand per mil of thickness. Consequently, higher voltages can be used with any compound by increasing its thickness. Insulation thicknesses are also specified by ICEA.

Insulation must resist damage from corona, particularly in high-voltage applications, as discussed in detail in chapter 17. The term *partial discharge* describes the type of corona stress imposed on cables. Partial discharges deteriorate insulation by ion bombardment and chemical action from ozone, nitrogen oxides, and nitric acid, which can occur in such voids as found between a stranded conductor surface and the insulation. Hence, insulation voids must be minimized and the insulation must resist the formation of this type of corona. ICEA standards specify corona-extinction voltage levels for insulation (*19–21*).

Ozone resistance is important for high-voltage cable insulation and sometimes for low voltage, and standards are again given by ICEA. Ozone is formed when electrical discharge is present in air, and it attacks compounds containing double carbon bonds, by splitting the carbon chain and deteriorating the material. Radiating cracks are a physical symptom of this occurrence.

Insulation must withstand cold temperatures as well as heat, particularly in surface operations: some of the open-pit iron mines in Minnesota and Michigan, for example, have experienced temperatures as low as −50°C. Cables stored on the surface at underground minesites are also exposed to extremely low temperatures. Most problems occur when a cold cable must withstand mechanical stress, such as bending or impact.

Cable Jacket

The main purpose of the jacket is to provide protection for the inner components and hold the assembly in the designed configuration. Jackets are not required to pass ICEA voltage withstand or insulation resistance tests, but tests for tensile strength, elongation, and aging are mandatory. Ozone and discharge-resisting jackets must also pass surface-resistivity and partial-discharge tests. Mining cable jackets must withstand an extensive temperature range, maintaining their physical properties throughout, and furthermore, they must not deteriorate when exposed to direct sunlight. Obviously, resistance to abrasion, crushing, tearing, and impact are extremely important. Cable jackets must also be resistant to the chemical action of acid or basic mine water and hydraulic fluids, and underground coal mine cable jackets must be flame resistant. Finally, jackets must exclude moisture and be very flexible.

One of the most commonly used materials for cable jackets is neoprene, a chloroprene polymer. Nitrile butadiene and polyvinyl chloride (NBR/PVC) is also used, particularly where jacket coloring is desired. Chlorosulfonated polyethylene (CSP) or Hypalon synthetic rubber is also used extensively, especially in combination with 90°C EPR insulation. EPR is used where extreme cold is encountered and flame resistance is not essential. Armored cables are used in some borehole applications. Here the jacket is a heavy metallic covering that affords extra protection to the conductors.

Cable Shielding

The ICEA defines the practice of shielding an electrical power cable as confining the electric field to the inside of the cable insulation or assembly with a grounded conducting medium called a shield (*19–21*). Two shield types are used in practice: the conductor shield and the insulation shield. Shown in figure 8.3, the conductor shield is placed between the conductor and the insulation, and the insulation shield surrounds the insulation.

Two distinct types of materials are employed in constructing cable shields: nonmetallic and metallic. Nonmetallic shields may consist of a conducting tape or a layer of extruded conducting compound. The tape may be made from conducting compound, be a conducting fibrous tape, or be a fibrous tape faced or filled with conducting compound. A typical conducting compound is carbon-impregnated rubber, which is commonly referred to as a *conductive-rubber, semiconducting,* or *semicon* shield. Metallic shields are nonmagnetic and may consist of a thin metal tape, wire-woven braid, or concentric serving of wires. Copper-braided shields may be made entirely of copper wires or have nylon twine in combination with copper wires. Nonmetallic and metallic elements may be juxtaposed to form the shield.

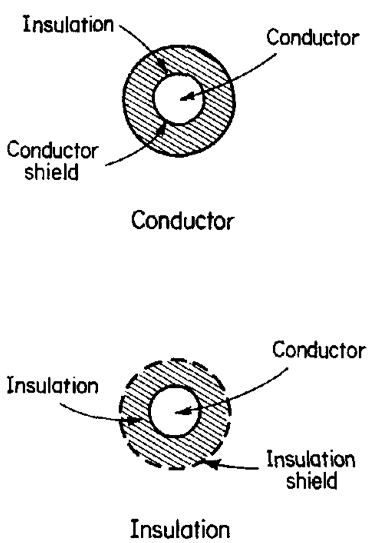


Figure 8.3.—Shield types.

Conductor shields are made of nonmetallic materials and are used only in high-voltage cable. The roles of this shield type are to eliminate air spaces or voids between the conductor and the insulation and to present a smooth electrode to the inner insulation surface. To be effective, it must adhere to or remain in intimate contact with the insulation under all conditions. This can substantially reduce the number of sites where partial discharge can form and helps reduce electrical stress on the insulation by uniformly distributing the electrical field about the conductor. The use of conductor shields becomes critical at higher operating voltages, especially 12.47 kV and above.

Insulation shields can perform three principal functions. If placed directly over individual conductor insulations, along with confining the electric field caused by conductor current within the insulation, the shield helps to maintain a symmetrical radial distribution of voltage stress within the dielectric. The possibility of partial discharges is minimized by precluding tangential and longitudinal stresses, and insulation is utilized to its greatest efficiency and in the direction of highest strength. This again becomes critical at higher operating voltages. Insulator shields also provide a continuous capacitance to ground for the conductor along its entire length. The uniformity is important in terms of transients on the power system, and this is discussed in chapter 11.

The third function of insulation shields is the most important for mining in view of the extensive handling of cables: reducing the hazard of electric shock. A major cause of electrical fatalities in mining has been workers' cutting into energized unshielded cables, for instance, during repair. Another source has been handling of energized unshielded cables with damaged jacketing and insulation or splices (the spot where a cable has been repaired). An insulation shield can be thought of as a safety barrier to penetrating metallic objects. If the percent of coverage of the shield over the insulation is high enough and its impedance is low enough, any metallic object compromising the conductor insulation will establish a fault between the power conductor and the grounded shield, with sufficient current to trip the ground-fault protective circuitry. Damage to insulation and jacketing, such as a pinhole, that would cause a handling danger to unshielded cable also creates a probable ground fault in cables with insulation shields. An individual touching the penetrating metallic object or handling the damaged shielded cable should be safe from electrocution.

Insulation shields are usually metallic. Recently, however, semicon insulation shields for trailing cables have found application in the United Kingdom, Australia, and to a lesser extent, the United States. This is to take advantage of semicon flexibility, especially in reeled-cable situations.

CABLE TYPES

An identifying code, related to standard specifications designated by ICEA, is embossed on the cable throughout its entire length. The code includes any approval number for flame resistance by the Mine Safety and Health Administration (MSHA) and approval by the Commonwealth of Pennsylvania (indicated by the letter P preceding the MSHA approval number). MSHA approval is mandated for cables in underground coal mines, and the

Pennsylvania approval is necessary for cables used in underground coal mines in that State.

The code includes the term *n/c* where n is the number of power conductors in the cable, an approved voltage designation, and letters describing the cable type. Table 8.2 summarizes the meaning of the letters used in the code, and table 8.3 presents the codes for typical cable types used in mining. Figures 8.4, 8.5, and 8.6 correspond to table 8.3 for unshielded round, unshielded flat, and shielded cable configurations, respectively, and detail the cable components as seen in cross section. Photographs of actual mining cables are provided in figures 8.7, 8.8, and 8.9 and show both side and cross-sectional views. Figures 8.10 and 8.11 are similar to figures 8.1. and 8.2 and show common applications of cable types in mine power systems.

Figure 8.7*A* is a single-conductor cable insulated for use at 600 V. This specific cable is not widely used. However, it has found application on twin-reel dc shuttle cars and small locomotives with reels; therefore, it must be highly flexible. Single-conductor cable similar to that shown is used extensively for connections inside power equipment, and a typical voltage rating is 15 kV for system voltages less than that level.

The most common dc shuttle-car cables are types W and G, figures 8.8*A* and 8.8*B*, respectively. The flat configuration is used since it allows an increased length on cable reels and is less susceptible to runover damage than round cables. The type W is used where diode grounding is allowed in lieu of a separate grounding conductor. Because shuttle car cables are damaged frequently, type W is preferred by some mine operators since it is easier to repair (splice).

Flat cable types employed for ac shuttle cars are shown in figure 8.8*C* and 8.8*D*. The three power conductors are separated by two grounding conductors in figure 8.8*C* and by one grounding and one ground-check conductor in figure 8.8*D*. These cables are also used on other equipment with reels, such as cutting machines and drills.

Table 8.2.—Letters used in alphabetic cable code

Code	Meaning	Comments
G	Contains uninsulated grounding conductor(s)	Common on low-voltage ac systems but used on dc systems where grounding conductors are needed.
W	Without uninsulated grounding conductor(s).	Typical on dc diode-grounded systems but 1 insulated power conductor may be used as a grounding conductor.
GC	Includes insulated ground-check (pilot) conductor.	Used where pilot-type ground-continuity monitoring is required, usually replaces 1 grounding conductor of type G cable.
SH	Shielded cable	None.
D	Multiple insulation shields	Shields surround each individual power-conductor insulation.
C	1 insulation shield	1 shield surrounds entire cable assembly just inside jacketing.
MP	Mine power feeder	None.

Table 8.3.—Codes for typical cables used in mining.

Code	Components	Comments
W	Contains 2, 3, or 4 insulated power conductors.	See table 8.2. Flat or round cross section.
G	Contains 2 or 3 insulated power conductors and 1 to 3 uninsulated grounding conductors.	Grounding conductors are placed in the interstices between the power conductors. Flat or round cross section.
G–GC	Contains 3 insulated power conductors, 1 or 2 uninsulated grounding conductors, and 1 insulated ground-check conductor.	Ground-check conductor replaces 1 grounding conductor of type G cable. Flat or round cross section. Presently, for ac systems only.[1]
G + GC	Contains 3 insulated power conductors, 3 uninsulated grounding conductors, and 1 insulated ground-check conductor.	Similar to round 3/C type G cable but has ground-check conductor in cable center.
SH–D	Contains 3 shielded insulated power conductors, 2 or 3 uninsulated grounding conductors.	Insulation shields about each individual conductor, grounding conductors contact shields. High-voltage cables usually have conductor shields. Round or flat cross sections. Presently for ac systems only.[1] A flexible portable cable.
SH–C	Contains 3 insulated power conductors, 2 or 3 uninsulated grounding conductors, assembly shielded.	Shielding encloses all conductors and is located just under jacketing. Grounding conductors should contact shield. Round or flat cross sections. Presently for ac systems only.[1] A flexible portable cable.
SHD–GC	Contains 3 shielded insulated power conductors, 1 or 2 uninsulated grounding conductors, and 1 insulated ground-check conductor.	Ground-check conductor replaces 1 grounding conductor of type SH–D cables. Round or flat cross section. Presently for ac systems only.[1] A flexible portable cable.
SHD + GC	Contains 3 shielded insulated power conductors, 3 uninsulated grounding conductors, and 1 insulated ground-check conductor.	Similar to round 3/C type SH–D cable but has ground-check conductor in cable center.
SHC–GC	Contains 3 insulated power conductors, 1 or 2 uninsulated grounding conductors, 1 ground-check conductor, assembly shielded.	Ground-check conductor replaces 1 grounding conductor of type SH–C cables. Round or flat cross section. Presently for ac systems only.[1] A flexible portable cable.
MPF	Contains 3 shielded insulated power conductors, 3 uninsulated grounding conductors.	Similar to round SH–D cable. Designed for relatively stationary high-voltage feeder applications.
MP–GC	Contains 3 shielded insulated power conductors, 2 uninsulated grounding conductors, and 1 ground-check conductor.	Similar to round SHD–GC cable. Designed for relatively stationary high-voltage feeder applications.

[1] Although not presently available, 2/C cable design for dc systems is possible.

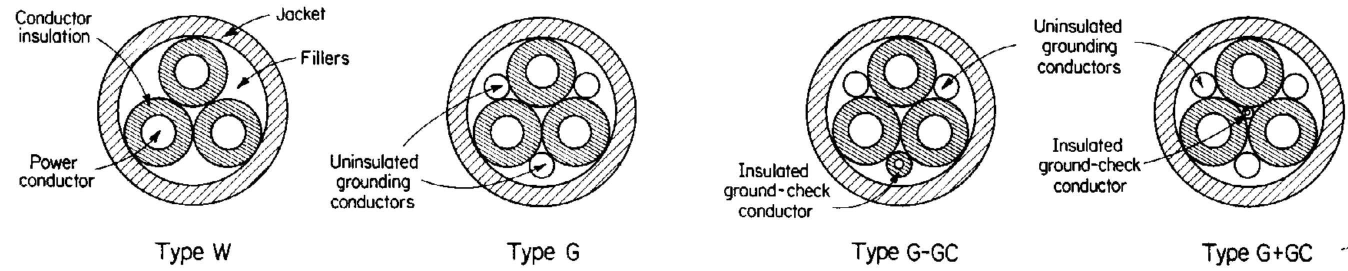


Figure 8.4.—Cross sections of round unshielded mining cables.

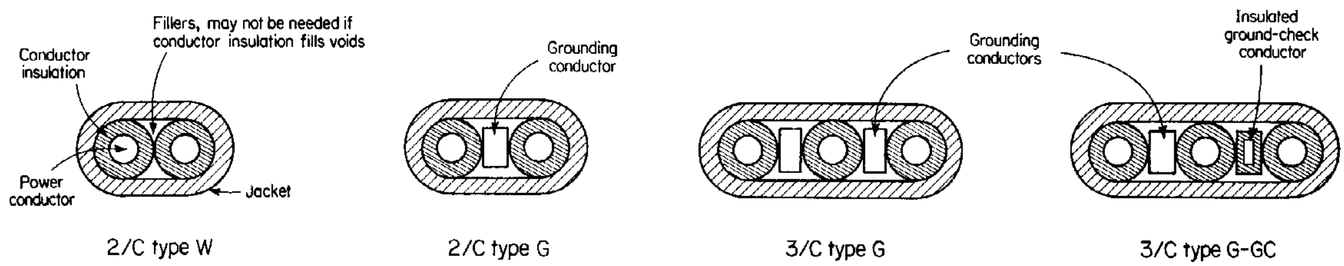


Figure 8.5.—Cross sections of flat unshielded mining cables.

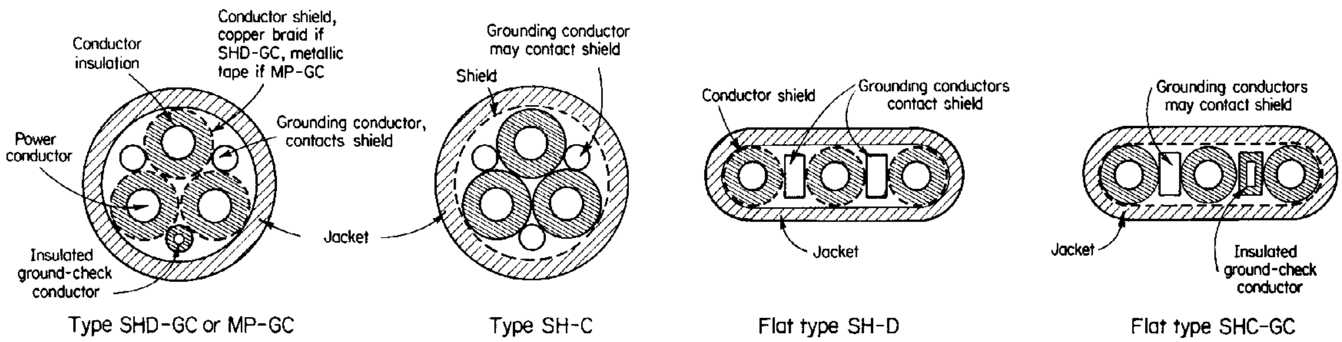
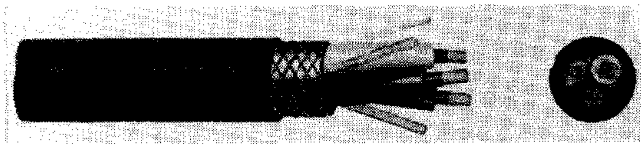
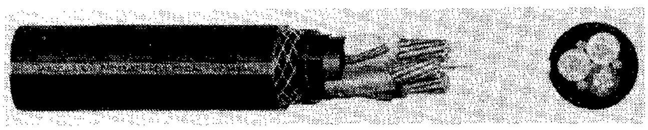


Figure 8.6.—Cross sections of some shielded mining cables.

(A) 1/C, 600 V



(B) 3/C type G-GC, 2,000 V

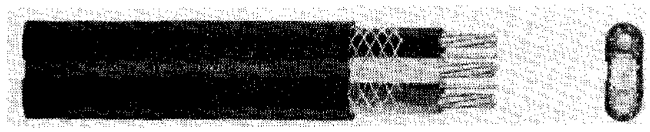


(C) 3/C type G+GC, 2,000 V

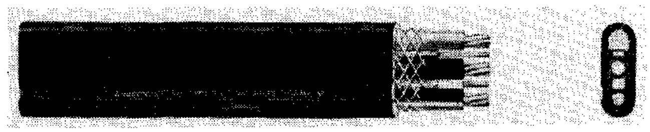
Figure 8.7.—Round unshielded mining cable. (Courtesy Anaconda Ericsson Co.)

(A) 2/C type W, 600 V

(B) 2/C type G, 600 V

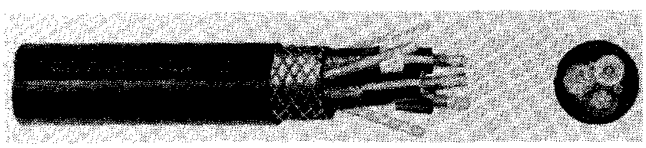


(C) 3/C type G, 600 V

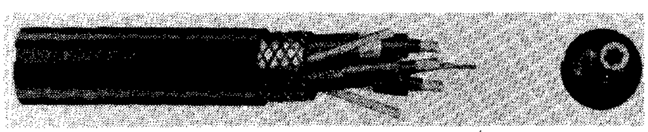


(D) 3/C type G-GC, 600 V

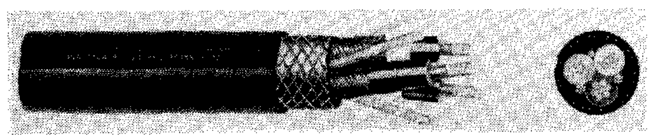
Figure 8.8.—Flat unshielded mining cables. (Courtesy Anaconda Ericsson Co.)



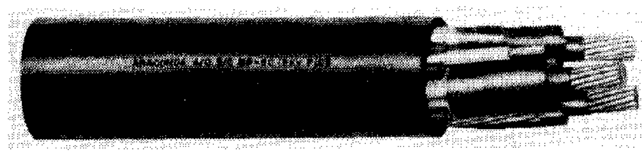
(A) 3/C type SHD-GC, 2,000 V



(B) 3/C type SHD+GC, 2,000 V



(C) 3/C type SHD-GC, 15 kV



(D) 3/C type MP-GC, 15 kV

Figure 8.9.—Round shielded mining cables. (Courtesy Anaconda Ericsson Co.)

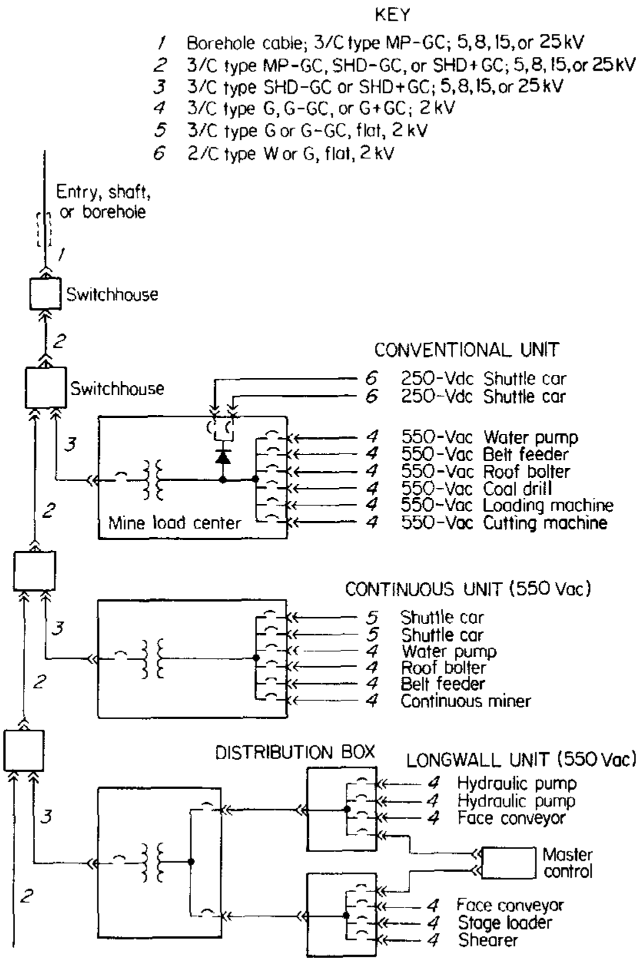


Figure 8.10.—Cable types for typical distribution systems in underground coal mines.

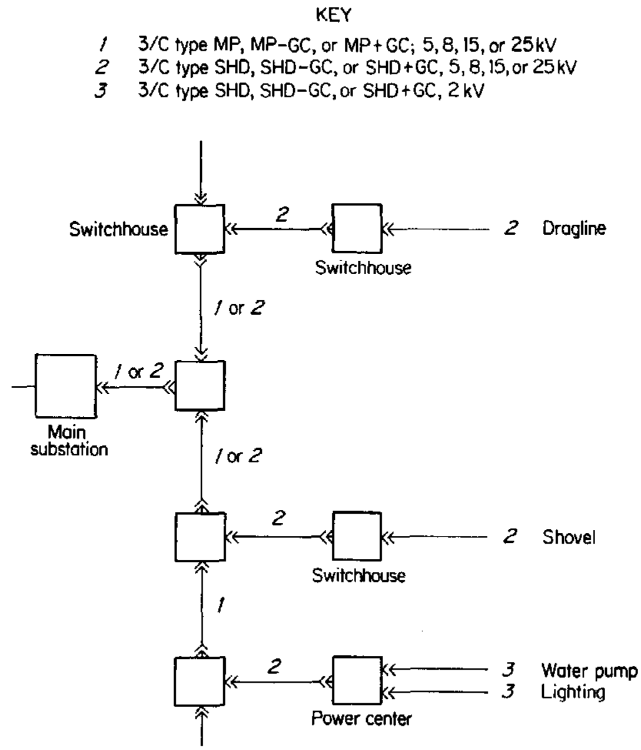


Figure 8.11.—Cable types for typical distribution systems in surface coal mines.

The ac shuttle cars also utilize round cables of type G or type G–GC. The grounding conductors are placed in the interstices between the power conductors in the type G, and a ground-check conductor replaces one of the grounding conductors in the type G–GC (fig. 8.7*B*). In addition to limited use on shuttle cars, the majority of longwall shearer, face-conveyor, stage-loader, roof-bolter, feeder, and continuous-miner cables are of this type. In some instances, the G–GC configuration can initiate induced voltages in the frame-grounding system (see chapter 17). Therefore, the G + GC type shown in figure 8.7*C* was constructed. Here the three grounding conductors are laid symmetrically in each interstice, and the ground-check conductor is placed in the center of the cable.

There are two basic configurations for shielded cables: the SH–D and the SH–C. As shown in figure 8.6, the shield of the SH–D cable surrounds each insulated conductor; in the SH–C cable, the shielding encloses all power conductors and grounding conductors. The SH–D shielding is preferred because the grounding conductor is in intimate contact with the shield, and line-to-line leakage current is detectable since the shield surrounds each individual power conductor. The SH–C shield, a single braid over the entire assembly, is sometimes found in low-voltage and medium-voltage portable cables. However, special designs are required to assure consistent, low-resistance contact between the shield and the grounding conductors.

In high-voltage cables, the insulation shield is generally comprised of two parts: an extruded layer or wrap of semiconducting material applied directly over the insulation, and a metallic cover applied over the semiconducting layer. The semiconductive material is considered to have a 100% coverage, but an associated high resistivity. If the metallic layer is composed entirely of copper braid, its coverage is 84% while the combination copper-nylon braid covers 60%. Shielding of unidirectional spirally wound wires, which gives 60% coverage, may also be used. The high-voltage insulation shield must be in intimate contact with the insulation under all conditions in order to be effective, and the metallic portion serves as a current-carrying medium for charging and leakage currents. Federal regulations require SH–D shielding for high-voltage cables in underground coal mines. Both SH–D and SH–C shielding are permitted for medium-voltage cables. Medium-voltage cables used on reels do not have to be shielded if the insulation is rated at 2 kV (*38*).

Two round shielded-cable configurations, SHD–GC and SHD + GC, are also used extensively for medium-voltage and high-voltage cables. The 2,000-V-rated SHD–GC cable, shown in figure 8.9*A*, and the SHD + GC cable in figure 8.9*B* are common on such equipment as 950-Vac continuous miners and longwall shearers, and on low-voltage surface coal mine equipment. Some high-voltage cables are required to be flexible, for example, surface mine shovel and dragline cables and underground mine distribution cables, which are connected to a portable power center. The SHD–GC cable shown in figure 8.9*C* is intended for this application. It is rated at 2, 5, 8, 15, or 25 kV depending on insulation thickness.

Stationary power cables are often mine power feeders of the MP–GC type as shown in figure 8.9*D* (see also tables 8.2 and 8.3). These cables can also be rated at 5, 8, 15, or 25 kV, but they are less flexible and have higher tensile strength than the SHD–GC type. Shielding is similar but uses different materials. MP–GC cables are also designed to be used in boreholes, aerial installations, ducts, and direct burial.

These are the basic power-cable types used currently in the mining industry. Other configurations are made for specific applications. For example, one double-drum shearing machine model requires a six-conductor cable with two ground-check conductors and a grounding conductor. Cable manufacturers are usually willing to produce these special cables, but they are not a part of normal product lines and the possible variations are too numerous to include here.

CABLE TERMINATIONS

The termination or end of any cable must encompass a means of sealing and protecting the cable from the weather above ground and contaminants such as dust below ground. It must often provide a means of electrical connection with other conductors. Particularly in the case of high-voltage cables, considerable stress occurs on the dielectric between the terminating point of the cable shield, which is at ground potential, and the end of the conductor, which is at line potential. These electrical stresses are ameliorated through use of a stress cone that forms part of the termination device.

The terminating device may take many forms, may be of varied complexity, and may be constructed from different insulating materials, depending on the cable type and the application. Taped terminations are very common, particularly at 15 kV and below. A simple sealing lug applied with insulating tape can be used on nonshielded cables, but where the cable is shielded, a stress-relief cone must be included. This may be preformed of rubber-like synthetic polymer and include an upper insulated cap, or may merely consist of lapped tape built up to the required cone shape. In either case, additional cover tapes are applied over the assembly and a rain hood or other protective housing may be added. An armor terminator provides a watertight grounding for armored cables and may be used in addition to a stress cone and insulation.

A *pothead* is a form of termination housing used frequently in surface mines and above ground at underground mines. The pothead is hermatically sealed and thus provides maximum cable protection from the environment. A typical pothead for a shielded cable is shown in figure 8.12. Note that with shielded cables the termination is taped prior to insertion in the pothead. In 15-kV applications and above, heated liquefied asphaltic or resinous material is then poured into the pothead cavity. The rate of cooling of this dielectric material must be controlled to prevent the formation of voids. The pothead may include a number of aerial and cable connectors.

Even though potheads are used, the standard termination-connector in mines is the coupler. An entire range of complex couplers has been developed specifically for the mining industry to accommodate the unique combination of environmental factors and operating procedures.

CABLE COUPLERS

Couplers are the complex sophisticated plugs and sockets used throughout the mine distribution system to connect mobile machinery to trailing cables, to connect cables with one another, and to connect cables to power centers, switchhouses, and substations. All couplers have certain common characteristics:

- They have either male contacts (plugs) or female contacts (sockets),
- They are either line mounted (at the end of a cable) or gear mounted (located on a piece of equipment),
- They are available in a wide voltage range, from high voltage (feeder cables) to low voltage (equipment related),
- They are available in a range of sizes to accommodate different types and ampacities of cable,
- They all have grounding contacts and may also include ground-check contacts,
- They all have sealing and locking devices and dust covers to protect the contacts when they are not in use.

The complexity of couplers is a direct result of the mine environment in which they are used; they must resist damage, be sturdy enough to withstand repeated use, prevent electrical hazards, be watertight, be dust proof, and withstand heat and cold. Some models are rated explosion proof. The plugging mechanism must be easy to use yet secure.

High-voltage couplers have been used in mine distribution for about 40 yr. Most of the initial problems encountered in 4,160- and 7,200-V systems have been resolved over the years through constantly improved design. Operating failures are no longer common at these levels. However, some problems are still found in the 15-kV class of couplers, and these have inhibited the switch to higher voltages by many mine operators. No ideal material has yet been developed for insulation; those

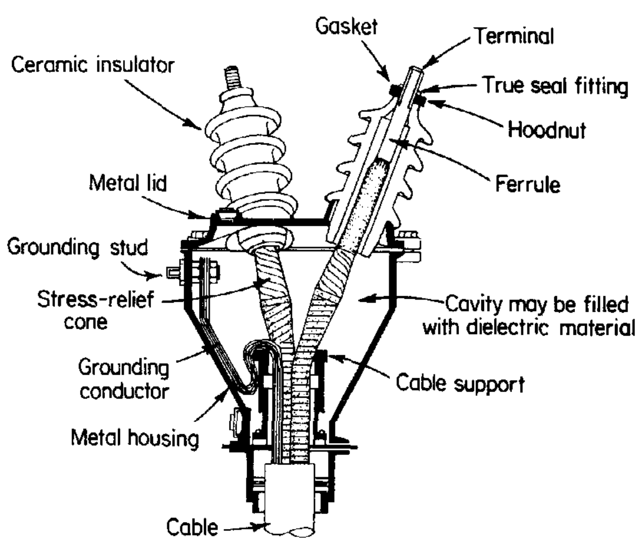


Figure 8.12.—Cable terminations for applications up to 15 kV (all or a portion of cable weight is supported by pothead).

with excellent electrical and chemical properties have been found to have mechanical inadequacies, and vice versa. The combination of dust and dirt with high humidity and moisture found in underground mines has posed many problems. In too many instances, these difficulties have been compounded by neglect, impatience, and total disregard for the purpose of a component by those who use them (7).

Coupler Contacts

The general requirements for coupler contacts are summarized as follows.

The coupler contact system should have

1. Adequate current-carrying capacity and low resistance,
2. The ability to withstand repeated coupling,
3. Protection from worker abuse,
4. A reliable and easy-to-make connection to the cable conductor,
5. Oxidation and corrosion resistance,
6. Uncoupling feature that allows a pilot or ground check to disengage first and the ground wires to uncouple last,
7. A guidance system to prevent misalignment and bending during coupling,
8. A feature to allow replacement of bent or damaged contacts.

It is important that the male and female pins that mate as the coupler is connected are of adequate size and have low contact resistance to prevent excessive heating when carrying current.

Frequent coupling and uncoupling can lead to a poor contact, particularly when a coupler is dropped, not an infrequent occurrence. Contacts can be bent and become dirty. Poor alignment during coupling and attempting to force a connection can also bend the contacts. In either case, the resulting high-resistance connection can lead to problems with overheating. Coupler manufacturers have attempted to reduce damage to contacts by recessing them in the housing and adding guidance systems to facilitate alignment when coupling must be carried out in restricted spaces.

Another possible failure point is the connection between the cable conductor and the contact. Set screws, soldering, thermit welding, and brazing are various methods for securing this connection. Extreme care must be taken when brazing or soldering these connections to remove excess flux, which can destroy coupler insulation. Severe vibration caused by dropping or by bouncing and bumping on a mobile machine such as a battery scoop can loosen a screw or crack a weld. The high-resistance broken connection then heats, which can cause insulation deterioration and a fault.

Electrical voids and protrusions caused by an improper mating have great significance at voltages greater than 8 kV because these localized nonconformities can become partial-discharge inception points. Hence, the insulation should be made of corona-resistant materials and the contact design should minimize the occurrence of voids and protrusions. Some low-voltage couplers, for example, have a "self-wiping" action to improve the contact; other, high-voltage contacts employ a Multilam band for the same purpose.

Coupler Insulation

The general requirements for coupler insulation are as follows.

The coupler insulation system should have

1. Adequate dielectric strength,
2. Adequate corona-extinction level,
3. Adequate tracking resistance,
4. Stress-relief feature,
5. Adequate impulse level,
6. Flame resistance,
7. Resistance to moisture penetration,
8. Insulators that align easily for coupling,
9. Resistance to cracking, chipping, and bending,
10. Resistance to heat deterioration,
11. The ability to withstand repeated coupling,
12. A feature that discourages phase reversal during mounting and coupling.

To ensure that coupler insulation does not break down in normal service, it should have a dielectric strength equal to or greater than that of the cable entering the connection. For high-voltage installations, the surface of the insulation should resist arc tracking, a process in which high-current arc discharges cross the insulator surface and carbonize the material, forming a conductive track. Keeping high-voltage insulators clean and dry will reduce the incidence of arc tracking. A common cause of moisture contamination is dropping the coupler on a wet mine floor.

Insulation, particularly if it has been weakened by partial discharges, is subject to breakdown by high-impulse voltages called transients, which usually occur during switching. Insulation materials must be able to withstand repeated occurrences of these high voltages.

Coupler Housing

Characteristics required for the housing are as follows.

The outer covering should have

1. A reliable easy-to-make ground wire connection,
2. A cable strain-relief mechanism,
3. A guidance system that improves the ease of alignment for coupling,
4. A durable material composition,
5. The ability to withstand repeated coupling,
6. Corrosion resistance,
7. Grommet or packing gland of the correct size for the cable used,
8. As little weight as possible,
9. A feature that facilitates ease in coupling and uncoupling.

If the coupler is classified as *explosion proof*, it incorporates a packing gland at the entrance to the housing that usually consists of asbestos fiber packed tightly between the cable and bushing. To be rated explosion proof by MSHA, an explosion that occurs inside the shell should not ignite any methane-air mixture surrounding the coupler. Explosion-proof couplers are allowed inby the last open crosscut in underground coal mines by all State and Federal regulations. Connectors without packing glands can be used inby the last open crosscut if they have a pilot or ground-check circuit that interrupts the power before the housing is opened. Instead of packing, non-explosion-proof couplers have a rubber grommet that allows cables of different diameters to fit into the same housing.

The cable strain-relief clamp is located on the outside of the cable entrance and prevents cable tension from pulling the conductors out of their connections. The clamp may be drawn down on the cable jacket by tightening a bolt on either side. If the bolts are not sufficiently tight, the clamp will not prevent tensile pullout, and if too tight, the clamp will damage the cable insulation.

Both packing glands and strain-relief clamps are made to fit a single cable jacket size or a small range of sizes. Thus knowledge of cable outer dimensions is necessary to match the coupler cable entrance to the cable. Tables 8.4 and 8.5 contain typical dimensions for round and flat cables, respectively (*38*). Variations in these values are allowed as long as the packing gland or strain relief is used.

High-Voltage Couplers

Couplers in the 15-kV, 500-A range are used as connections to switchhouses and power centers, to join high-voltage cables, and for high-voltage machines. A typical high-voltage coupler is shown in figure 8.13. In the following paragraphs, the numbers in parentheses refer to this diagram.

A high-voltage coupler accommodates the three power conductors (4), one or more grounding conductors (14), and one or more ground-check conductors (15). The contacts (8, 10, 11) are soldered and taped to the prepared conductor cables during installation. The contacts may be of copper, copper berylium, or in some cases, aluminum or brass. Male contacts have a split-pin design or incorporate a Multilam band of torsion-sprung louvres to improve the power contact.

Table 8.4.—Typical diameters for round portable power cables in inches, 601 to 5,000 V

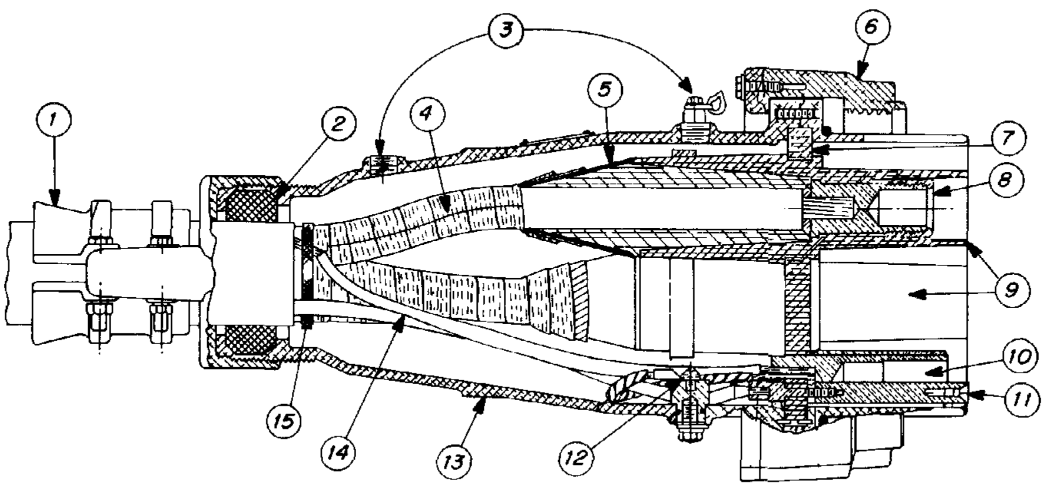
Conductor size	G–GC, 2 kV	SHC–GC, 2 kV	SHD–GC, <3 kV	SHD–GC, 3–5 kV
AWG:				
4	1.25	1.39	1.62	1.78
3	1.40	1.55	1.77	1.90
2	1.48	1.62	1.84	1.98
1	1.55	1.71	1.92	2.09
1/0	1.74	1.89	2.04	2.18
2/0	1.84	2.02	2.18	2.34
3/0	1.99	2.16	2.29	2.46
4/0	2.12	2.30	2.45	2.62
MCM:				
250	2.30	2.48	2.62	2.76
350	2.75	2.97	(1)	(1)

[1] Cable not made.

Table 8.5.—Typical dimensions for flat portable cables in inches, 600 V

Conductor size, AWG	2-conductor				3-conductor, G	
	W		G			
	Major axis	Minor axis	Major axis	Minor axis	Major axis	Minor axis
8	0.84	0.51	—	—	—	—
6	.93	.56	1.02	0.56	1.65	0.67
4	1.05	.61	1.15	.61	1.85	.75
3	1.14	.68	1.26	.68	1.99	.77
2	1.24	.73	1.35	.73	2.10	.81
1	1.40	.81	1.55	.81	2.43	.97
1/0	1.51	.93	1.67	.93	—	—
2/0	1.63	.99	1.85	.99	—	—
3/0	1.77	1.03	2.00	1.03	—	—
4/0	1.89	1.10	2.10	1.10	—	—

NOTE.—Dash indicates cable is not made.



KEY

1 Cable strain-relief clamp
2 Packing-gland bushing
3 Plugged holes for pouring potting compound
4 Power conductor (insulation wrapped) with shielding tape for high voltage
5 Molded stress-relief cone (for high voltage)
6 Shell engagement mechanism
7 Insulation mounting flange
8 Power-conductor contact
9 Power-conductor insulation tube
10 Grounding-conductor contact
11 Pilot-conductor contact
12 Metallic-shell grounding point
13 Coupler shell
14 Grounding conductor (from cable)
15 Ground-check conductor

Figure 8.13.—Coupler components.

The insulation materials and configuration vary according to the manufacturer, but they commonly have three main parts: a molded stress-relief cone (5), insulation tubes (9), and a flange (7). The molded stress-relief cone is now tending to replace hand taping as a method of providing termination stress relief. It combines the functions of a stress-relief cone and seal and also serves to position the conductors. The insulating tubes push onto the tapered cylinders of the molding and encase the contacts. Resistant rubber-like polymer tubes are now finding favor over flexible rubber tubes or cups that have a tendency to fold when a misaligned coupling is attempted. Both types replace an earlier polyester insulator that could crack and chip under the abuse almost inevitable when coupling in the confined spaces of an underground mine. The insulation tubes attach to a rigid insulation flange that positions the assembly correctly, seals the contact area from the rest of the coupler, and attaches it to the housing.

When the coupler assembly is complete, a potting compound may be poured into the coupler (3) to guard against the formation of moisture. The compound is derived from tung oil and sets to a gel-like consistency. Potting compound is not required for 15-kV couplers that use filler moldings, but is frequently used as an added precaution. Asphaltic compounds were originally used as coupler fillers but these were very difficult to remove if components were to be reused.

The coupler housing (13) is metal, usually a high-strength, light-weight, corrosion-resistant cast aluminum that resists physical abuse yet is portable. The coupler housing incorporates a threaded collar or lock ring (6) that secures one coupler to its mate. Some designs have a pin-and-slot mechanism to reduce the number of turns required to lock the connection and simplify alignment.

Low-Voltage Couplers

The standard sizes for low-voltage and medium-voltage couplers are 225, 400, 600, 800, and 1,200 A. Their primary use is to connect mobile equipment to power centers and junction boxes, and to connect cables in the 600- to 1,000-V range. Their construction is sturdy but less complex than that of high-voltage couplers. They have either a boxlike shape and are locked by a latch mechanism or a cylindrical lock ring similar to those on high-voltage couplers. They do not have stress-relief cones or packing compound. The packing gland is usually replaced by a rubber grommet seal, but these couplers do include a cable strain-relief clamp. Many different contact configurations are available to accommodate a wide range of equipment types. Lower powered couplers specialize in quick and easy connection and disconnection for equipment that must be changed out frequently.

CABLE SELECTION

The cable manufacturer can provide a proper cable to a mining company only if the exact operating conditions for the cable are specified. The purchaser has the responsibility for writing a purchasing specification that completely describes the operating environment. A revised ICEA listing of the information to be supplied by the purchaser, given below, will be used here to describe the step-by-step cable selection process.

1. System characteristics:
   a. Ac or dc.
   b. Grounding method (i.e., by grounding conductor or diode-grounding circuit).
   c. Normal operating voltage between lines or conductors (line-to-line voltage).
   d. Number of pilot or ground-check conductors and type of ground-check monitor.
   e. Minimum ambient temperature of cable storage and installation.
   f. Description of cable-installation area (surface mine, borehole, trailing cable, etc.).
   g. Environment of use (ambient temperature, amount of moisture, amount of sunlight, etc.).
   h. Maximum and normal operating current.
   i. Time schedule.
   j. Delivery point.
   k. Future changes in the system.

2. Cable characteristics:
   a. Cable length.
   b. Cable type, number of conductors, and flat or round configuration.
   c. Voltage rating.
   d. Type of conductor (copper or aluminum).
   e. Conductor size.
   f. Insulation type.
   g. Jacket type and color.
   h. Maximum outside diameter and tolerance.
   i. Method of conductor identification.
   j. Special markings (MSHA and P approval numbers, dating, etc.).
   k. End attachments (couplers), type of attachment, location of installer, and method of installation.

Many of the items in the system characteristics category are obviously designed to assist the purchaser in identifying a specific cable type. For example, the number of power conductors is determined when ac or dc is specified (1a). The need for one or more grounding conductors is noted when the grounding method (1b) is explained. Similarly, the normal operating voltage (1c) leads to the selection of a cable voltage rating that includes the operating voltage and the requirements for shielding. If the ground-continuity monitor requires a ground-check conductor, this should also be noted (1d). Any additional monitoring or remote-control systems may also require pilot conductors. Because cable jackets can crack during installation after being stored outside in extremely cold weather, the ambient temperatures of storage and use (1e) should be specified. The installation area, category (1f), explains special requirements such as high-tensile-strength conductors or a flame-resistant jacket for a borehole cable. Special environmental considerations (1g) that may affect cable life, such as an excessive exposure to sunlight in a surface mine, should be noted. Delivery time schedule (1i) and delivery location (1j) are obviously important considerations to be included so that a cable manufacturer can give the proper service. Finally, if changes to the electrical system (1k) are anticipated, they should be considered. Money can be saved by purchasing a cable that will accommodate both the present and future systems rather than replacing a cable after a short operating period.

Cable Length

The second section of specifications is concerned with the detailed description of a required cable. First the cable length (2a) must be specified. Many companies prefer to purchase a long length of cable, thereby receiving a price discount, and then cut the required lengths from this stock. For instance, high-voltage feeder cable is usually shipped to a shop where couplers are mounted onto the cable at 1,000-ft intervals before the cable, now in the desired lengths, is transported to the mine. However, other factors such as Government regulations and voltage drop must be considered. Table 8.6 gives relevant information for underground trailing cables longer than 500 ft, based on a 60°C-rated insulation (a table for 90°C insulation is not presently available) (*38*).

Table 8.6.—Specifications for trailing cables longer than 500 ft

Conductor size	Maximum allowable length, ft	Normal ampacity at 60°C copper temperature (40°C ambient), A	Resistance at 60°C copper temperature, Ω
AWG:			
6	550	50	0.512
4	600	70	.353
3	650	80	.302
2	700	85	.258
1	750	110	.220
1/0	800	130	.185
2/0	850	150	.157
3/0	900	175	.130
4/0	1,000	200	.116
MCM:			
250	1,000	220	.098
300	1,000	240	.082
350	1,000	260	.070
400	1,000	280	.061
450	1,000	300	.054
500	1,000	320	.050

Most of the remaining cable specifications have been discussed earlier in the chapter. Conductor size selection, however, is a complex topic that requires detailed analysis.

Conductor Selection

The selection of the conductor size (2e) is dependent on many parameters, such as ampacity, cable heating, voltage drop, length, breaking strength, weight, shielding, insulation, and conductor material; the cable application may place emphasis on specific parameters. The correct selection will allow the cable to carry current without overheating or physical damage, to withstand the rugged mine environment, and to limit the voltage drop between the power source and the machines.

Ampacity

The ampacity or normal continuous-current rating of a cable is the current-carrying ability of its power conductors. It is dependent upon the ability of the cable assembly to dissipate heat without damaging the insulation. The ampacity rating is usually based on the maximum conductor temperature rise, with the temperature limit chosen on the basis of the specified life expectancy of the cable insulation. The temperature class assigned to the material used for the conductor insulation describes the maximum allowable sustained conductor temperature in a specified ambient temperature. The popular temperature ratings are 75° and 90°C. Cable insulation with a 60°C rating can still be found, but this value is no longer used extensively in mining. An ambient temperature of 40°C is used for all ratings.

The heat generated in the cable is primarily caused by the I^2R power loss from current flow through the power-conductor resistance. The dissipation of this heat is a function of (*30*).

- The conductor diameter and the number of conductors in the cable;
- The thickness of the conductor insulation and the cable jacket;
- The cable configuration and outside dimensions;
- The heat-transfer properties of the cable components; and
- The type of conductor and cable outer jacket, and the ambient temperature.

A conductor size (cross-sectional area) within a specific insulation and cable configuration is given a current rating (its ampacity) through calculations using these parameters and the generated heat.

Cable ampacities are now designated in the United States by the National Electrical Code (NEC) (*2*) or by the ICEA for cables manufactured according to its design specifications. Parts 18, 75, and 77, 30 CFR, basically allow compliance with either the NEC or ICEA ratings (*38*). However, allowable ampacities for insulated conductors given in the NEC are broad in both scope and application, and the same current value can be specified for one, two, or three conductors in a raceway, cable, or buried directly in earth (*2*, table 310–16). The broad applicability of the NEC standards implies that a safety factor must be built into its ratings, and comparison shows that the NEC ampacities are approximately 25% higher than the ICEA ratings. While the NEC values are fine within the scope and objectives of that code, ICEA values are preferred for engineered systems. Tables 8.7 and 8.8 give the ICEA ampacities for the 90°C-rated cables preferred for mining. Table 8.6 includes ampacities for 60°C-rated cables as specified in 30 CFR 18, and these are similar to the NEC values.

The ampacity of a particular cable assumes that all splices, joints, and terminations in the cable are adequate in design and able to operate without restricting the loading on the cable. Considering the large number of splices made in mining cables, this assumption is a very important criterion for the cable rating.

The ambient air temperature for the ampacities given in tables 8.7 and 8.8 is 40°C. If the maximum ambient temperature is different from that specified, the ampacity correction factors shown in table 8.9 should be applied (*30*).

Table 8.7.—Ampacities[1] for portable power cables, amperes per conductor

Conductor size	Single conductor: 0–2,000 V unshielded	Single conductor: 2,001–8,000 V [2] shielded	Single conductor: 8,001–15,000 V [2] shielded	Single conductor: 15,001–25,000 V [2] shielded	2-conductor, round and flat, 0–2,000 V	3-conductor, round and flat, 0–5,000 V unshielded	3-conductor round: 0–8,000 V shielded	3-conductor round: 8,001–15,000 V shielded	3-conductor round: 15,001–25,000 V shielded	4-conductor, 0–2,000 V	5-conductor, 0–2,000 V	6-conductor, 0–2,000 V
AWG:												
8	83	—	—	—	72	59	—	—	—	54	50	48
6	109	112	—	—	95	79	93	—	—	72	68	64
4	145	148	—	—	127	104	122	—	—	93	88	83
3	167	171	—	—	145	120	140	—	—	106	100	95
2	192	195	195	—	167	138	159	164	178	122	116	110
1	223	225	225	222	191	161	184	187	191	143	136	129
1/0	258	260	259	255	217	186	211	215	218	165	—	—
2/0	298	299	298	293	250	215	243	246	249	192	—	—
3/0	345	345	343	337	286	249	279	283	286	221	—	—
4/0	400	400	397	389	328	287	321	325	327	255	—	—
MCM:												
250	445	444	440	430	363	320	355	359	360	280	—	—
300	500	496	491	480	400	357	398	—	—	310	—	—
350	552	549	543	529	436	394	435	—	—	335	—	—
400	600	596	590	572	470	430	470	—	—	356	—	—
450	650	640	633	615	497	460	503	—	—	377	—	—
500	695	688	678	659	524	487	536	—	—	395	—	—
550	737	732	—	—	—	—	—	—	—	—	—	—
600	780	779	—	—	—	—	—	—	—	—	—	—
650	820	817	—	—	—	—	—	—	—	—	—	—
700	855	845	—	—	—	—	—	—	—	—	—	—
750	898	889	—	—	—	—	—	—	—	—	—	—
800	925	925	—	—	—	—	—	—	—	—	—	—
900	1,010	998	—	—	—	—	—	—	—	—	—	—
1,000	1,076	1,061	—	—	—	—	—	—	—	—	—	—

[1] Based on a copper conductor temperature of 90°C and an ambient air temperature of 40°C.
[2] These ampacities are based on single isolated cable in air operated with open-circuited shield.

NOTE.—Dash indicates cable is not made.

Table 8.8.—Ampacities[1] for three-conductor mine power cables

Conductor size: Copper	Conductor size: Aluminum	2,001 to 8,000 V: Copper	2,001 to 8,000 V: Aluminum	8,001 to 15,000 V: Copper	8,001 to 15,000 V: Aluminum
AWG					
6	—	93	—	—	—
4	2	122	124	125	128
2	1/0	159	165	164	168
1	2/0	184	189	187	192
1/0	3/0	211	218	215	221
2/0	4/0	243	251	246	254
3/0	250	279	278	283	281
4/0	350	321	342	325	344
MCM					
250	400	355	360	359	367
300	450	398	395	401	393
350	500	435	425	438	424
400	—	470	—	473	—
450	—	502	—	504	—
500	—	536	—	536	—

[1] Based on ICEA values with an ambient temperature of 40°C and a conductor temperature of 90°C [taken from "Power Cable Ampacities" (*20*), v. 1 for copper conductors and v. 2 for aluminum conductors].

NOTE.—Dash indicates cable is not made.

Table 8.9.—Correction factors for ampacities at various ambient temperatures.

Ambient temperature, °C	*Correction factor*	*Ambient temperature, °C*	*Correction factor*
10	1.26	40	1.00
20	1.18	50	.90
30	1.10		

Cable Heating on Reels

A cable that is used in a confined space can become overheated with continuous-current flow at the ampacity rating. Perhaps the best example is a cable bound on a reel, either for storage purposes or to increase mining machine mobility. Investigations were conducted as early as 1931 to identify factors responsible for overheating of rubber-jacketed cables, with emphasis on increased temperatures occurring in reeled cables (*16*). A cable manufacturer manual published in 1940 was the first to contain a table of derating factors related to the number of layers wound on a reel to reduce the current-carrying capacity of the cable (*23*). These factors were included in ICEA specifications for 60°C-rated cables in 1946 and have remained a standard since that time. Table 8.10 presents the ICEA values presently required by Federal regulations for all cable insulations (*38*).

Research has been conducted since the publication of the ICEA derating factors to determine their applicability to the mining industry. McNiff and Shepherd (*23–24*) worked with cyclic currents, comparable to those experienced by shuttle cars in service, and steady-state loading at various percentages of cable ampacity, with both ac and dc power. Derating factors for 60°C-rated cables abstracted from these results are presented in table 8.10. An important contribution of their work, which cannot be shown in the table, is identification of the dependence of cable derating factors on the maximum-limit temperature permitted: at this temperature is increased or reduced, the derating factor changes accordingly. This was later verified by Woboditsch (*41*), and his values for a limit temperature of 60°C are also given in table 8.10.

Table 8.10.—Ampacity derating factors for 60°C-rated trailing cables operated on drums

Number of layers	ICEA	McNiff and Shepherd (23–24) [1] ac	McNiff and Shepherd (23–24) [1] dc	Woboditsch (41) [2]
1	0.85	[3]0.78	[3]0.82	0.78
2	.65	.54	.50	.62
3	.45	.42	.41	.48
4	.35	.36	[3].35	([4])
5	NA	.32	[3].34	([4])

NA Not available.
[1] Data for a 2-conductor, No. 4 AWG, type G cable at a maximum temperature of 60°C.
[2] Data for a 3-conductor, type NTSCE cable at a maximum temperature of 60°C.
[3] Values from extrapolated curves since data did not extend to this range.
[4] Cable not made.

Cable ampacity must be derated if the cable is used in a confined space. In view of the findings on limit temperature change, the ICEA values are probably adequate for 75°C-rated and 90°C-rated cables. It is significant that Australian mining companies have recently accepted the initial derating factors, but with qualification (*9*), as shown in table 8.11. The ICEA values are specified as pertaining only to round cable, while new values have been generated for flat cable (*8*). As flat cable usually occupies more volume on a reel than round cable, heat transfer for flat cables should be less, and the lower values appear reasonable.

Table 8.11.—Australian specifications for ampacity derating factors for trailing cables operated on drums

Number of layers	Circular cable	Flat cable
1	0.85	0.68
2	.65	.52
3	.45	.36
4	.35	.28

Current Calculations

Current and voltage regulation are the two major concerns in sizing a cable correctly for an intended application. The effective continuous current through the cable power conductors must be less than the cable ampacity, with correct derating factors applied. The voltage drop across the distribution and utilization systems must be such that voltage regulation is within the tolerances specified for the loads. For trailing cables serving machines, current is often the determining factor, since these cables are always short enough for voltage regulation not to be a problem. Feeder and portable cables serving many loads, however, are often so long that voltage drop becomes a principal concern. Even though the cable size may be found adequate in terms of ampacity and voltage drop, other factors may enter into the conductor sizing, such as tensile load, weight, and available short-circuit current.

There are three basic methods that can be used to find trailing-cable ampacity: a full-load current similar to that specified in the NEC, a 30-min effective current demand, and a load-factor approach. Regardless of the method used, the engineer should realize that the typical current requirements of mining machinery change continuously over time and may be described as unsteady in nature. The infinite variability of mining conditions makes it difficult to define current levels for any part of a given duty cycle with precision.

Calculation of cable ampacity requirements based on a 30-min effective current demand recognizes this variability and also that cable heating varies as the square of current. Here, line current measurements are taken from the machine, and an effective or rms value is found by weighting current with

$$I_{effective} = \left(\frac{\Sigma I^2 t}{\Sigma t}\right)^{1/2}, \tag{8.2}$$

where $I_{effective}$ = weighted current through cable, A,
I = current level for specific increment of time, A,
and t = time increment for current level I, s.

This method does account for the transient heating and cooling of the cable, which should be considered for matching the loading conditions found in mining with the specific limit temperature for the cable; in other words, the ampacity. Through this method, representative machines in typical mining conditions can be measured and a catalog of effective currents can be assembled for ampacity selection. However, actual measurements are not always possible, and the next two methods do not require them.

The full-load current approach is detailed by MSHA (*39*) and essentially follows the NEC requirements in sections 430–22, 430–23, and 430–24. Here the ampacity of a cable supplying a single motor must be not less than 125% of the motor full-load current rating. When two or more motors are supplied through one cable, the ampacity must be at least equal to the sum of the full-load current ratings of all the motors plus 25% of the highest rated motor in the group. Provisions are allowed in this approach for adjusting the current requirements of any motor used for intermittent or periodic duty, and for the 60-min-rated motors normally found in mining (*36*); that is, the ampacity may be reduced by 10% or 5%, respectively.

The third method uses the machine load factor and applies the average power formula (*32–33*). For ac machines,

$$I = \frac{P(LF)}{\sqrt{3}\,V\eta\,(pf)}, \tag{8.3}$$

and for dc equipment,

$$I = \frac{P(LF)}{V\eta}, \tag{8.4}$$

where I = machine line current, A,
P = (746) (hp) = rated average power of machine, W,
hp = rated machine horsepower,
$LF = \dfrac{\text{actual average power consumed}}{\text{rated average power}}$ = machine load factor,
V = line-to-line machine voltage, V,
pf = machine power factor,
and η = machine efficiency.

The formulas may be used for single motors or machines containing a complex of motors. Obviously, the load factor, power factor, and efficiency of a machine must be known in order to apply this method. With knowledge of typical operating conditions, these can be estimated. Values for many underground coal mining machines have been researched and may be found in references *28, 32,* and *33*. A summary of these and values extrapolated from representative underground mining conditions is given in table 8.12; 100% efficiency should be assumed when applying these values to the formulas. However, caution should be taken when using these parameters as they are only representative. If precise currents are necessary, power measurements should be taken to obtain load factors and power factors, and manufacturer specifications consulted for efficiencies. The formulas can also be employed directly for full-load current calculations by assuming that pf = 0.85, LF = 1 and η = 1 for ac induction machines, and pf = 1 and LF = 1 for dc motors.

Table 8.12.—Some estimated power factors and load factors for various underground coal mining equipment in good operating conditions

Machine	Power factor [1]	Load factor [2]
Battery chargers	1.0	0.8
Belt drives	.8	.7
Belt feeder	.8	.7
Belt feeder breaker	.7	.6
Continuous miners [3]	.6	.5
Cutting machines [3]	.7	.6
Drilling machines	.8	.7
Lighting	1.0	1.0
Loading machines [3]	.7	.6
Longwall shearing machines	.8	.7
Roof bolters	.6	.3
Section fans	.7	.6
Shuttle cars	.6	.4

[1] For ac equipment only.
[2] For ac or dc equipment.
[3] Values are for cutting and/or loading only. Values for other machines are an average over a typical duty cycle.

EXAMPLE 8.1

The difference between the last two methods can easily be seen through examples. First consider a 150-hp ac continuous belt-conveyor drive motor rated at 550 V and operating in 20° C ambient temperature. Using the NEC currents (*2*, table 430–150) and applying 125% for the full-load current approach, the current used to size the cable would be

$$I = (144)(1.25) = 180 \text{ A}.$$

The ICEA ampacities of Nos. 2 and 1 AWG 3/C unshielded round cable from table 8.7, corrected by the factors in table 8.9, are (138)(1.18) = 163 A and (161)(1.18) = 190 A, respectively. Hence the No. 1 AWG size would be indicated. Applying a load-factor calculation with table 8.12 data,

$$I = \frac{(150)(746)(0.7)}{\sqrt{3}\,(550)(1)(0.8)} = 103 \text{ A}.$$

This relates that a No. 4 AWG 3/C unshielded round cable is adequate with a corrected ampacity of (104)(1.18) = 123 A. The second method is probably more representative of actual conditions, since the NEC applies a 25% safety factor.

EXAMPLE 8.2

A cable size must be found for a 105-hp dc shuttle car. The machine is rated at 250 V, and it is assumed that the maximum ambient temperature is 20° C, and an average of two layers of cable will remain on the reel. The load-factor approach will be used.

From the information in table 8.12, a representative load factor for shuttle cars is 0.4. Applying equation 8.4 and assuming 100% efficiency,

$$I = \frac{(105)(746)(0.4)}{(250)(1)} = 125 \text{ A}.$$

Ampacities for two-conductor cables from table 8.7 corrected for a 20° C ambient temperature (table 8.9) are

- for No. 4 AWG, (127)(1.18) = 150 A,
- for No. 2 AWG, (167)(1.18) = 197 A,
- for No. 1 AWG, (191)(1.18) = 225 A.

This is a reeled application and these ampacities must be derated by the number of layers on the reel. Because of present Federal acceptance, the ICEA derating values from table 8.10 will be used. Thus for two layers, the ampacities must be reduced by 0.65, or

- for No. 4 AWG, (150)(0.65) = 97 A,
- for No. 2 AWG, (197)(0.65) = 128 A,
- for No. 1 AWG, (225)(0.65) = 147A.

Therefore, No. 4 AWG is too small, and No. 2 AWG would be selected.

It can be noted that No 3 AWG was not included in the example. The reason is that this cable is not popular and is not readily available from manufacturers.

EXAMPLE 8.3

Now consider a 550-Vac continuous miner that has five motors (two 50-hp gathering-head motors, two 175-hp cutter motors, and one 135-hp pump motor) for a total connected horsepower of 535 hp. Using the NEC currents (*2*, table 430–15), applying the intermittent-duty rating for the gathering head and cutter motors, and increasing the highest rated motor in the group by 25%,

- 50 hp, I = (52)(0.9) = 46.8, A
- 135 hp, I = (133)(1.0) = 133 A,
- 175 hp, I = (168)(0.9) = 151.2 A,
- 175 hp, I = (168)(0.9)(1.25) = 189 A.

Assuming the current phasor angles are such that a direct summation introduces only minor error, total current for ampacity selection would be about 520 A. Assuming the machine is operating in good mining condition, and using a load-factor calculation with table 8.12 values,

$$I = \frac{(535)(746)(0.6)}{\sqrt{3}\,(550)(1)(0.6)} = 419 \text{ A}.$$

Continuous miners of this size commonly use unshielded 4/0 trailing cables with 90° C-rated insulation. If the ambient is 20° C, the ICEA ampacity from table 8.7 corrected with table 8.9 data is (287)(1.18) = 339 A. This is considerably below the calculated values of 520 and 431 A. Actual visits to underground mines using continuous miners of the same size (535 hp) showed that the 4/0 cable jackets were not warm to the touch, implying cable-conductor temperatures well below the 90° C limit temperature (*32*). Furthermore, the load-factor calculation is based on data from machine cutting and loading, and since a continuous miner does not cut and load continuously, the current would be biased toward a worst case situation. Including the other machine operations (tramming, idle, etc.) would lower the load factor and the calculated current, probably below the ICEA ampacity. Regardless, the load-factor approach reflects this utilization environment more accurately than the NEC approach. It should be obvious that the effective current demand method would be more precise than either of these approaches.

Intermittent Duty Ratings

A major problem implied in the preceding example is that intermittent, fluctuating, or cyclic current through a cable has a different effect on cable heating than continuous loading. The full-load current or NEC approach for conductor sizing basically assumes continuous loading, but true continuous operation of most mining machinery would be a rare occurrence. Mining is inherently cyclic in nature. The Institute of Electrical and Electronics Engineers (IEEE) (*17*) does publish guidelines for rating electrical equipment under various operating conditions, durations, and time sequences of duty. Even though these terms have been used previously in this text, it is beneficial to define them here:

- *Continuous duty.* Operation at a substantially constant load for an indefinitely long time.
- *Short-time duty*. Operation at a substantially constant load for a short and definite specified time.
- *Intermittent duty.* Operation for alternate intervals of load and no-load as definitely specified.
- *Varying duty.* Operation where the amount of load and the length of time the load is applied are subject to considerable variation.

In an endeavor to overcome the problem of mining duty cycles, the United Kingdom and Australian mining laws permit intermittent-duty ratings for mining trailing cables (*9, 37*). These ratings for several popular cable sizes are given in table 8.13. It can be noted that in both United Kingdom and Australian practice, the rating criteria are

Table 8.13.—Intermittent-duty ratings for trailing cables

Cable size, mm ^{2}	Approximate U.S. cable equivalent, AWG	Continuous current rating, A	Intermittent current rating, A	Increase, %
		UNITED KINGDOM [1]		
16	5	85	90	6
25	3	110	120	9
35	2	131	145	11
50	1/0	168	190	13
70	2/0	205	235	15
95	4/0	247	290	18
		AUSTRALIA [2]		
21	4	70	95	36
33	2	90	125	39

[1] Criteria: full-load current for 40 min, no-load current for 10–15 min, 1/2 full-load current for 40 min, no-load current for 10–15 min; ambient at 25°C.
[2] Criteria: full-load current for 30 min, no-load current for 30 min.

independent of the cable size. An attempt to match or classify the duty of mining machines with the well-defined IEEE categories, however, results in only one conclusion: the typical mining duty is equivalent to a varying-duty classification. Although mining sequences through given events regularly, distances constantly change; hence, equipment utilization changes. In such cases, the IEEE recommends the use of standard application methods to offset the problems of a nonconstant load, and suggests the use of load-factor and rms current calculations. These should be applicable to electrical equipment, such as cables, which are "sufficiently standardized both in performance and construction" (*17*).

Voltage Calculations

The major concern for voltage calculations is that adequate voltage must be at the machine terminals for proper starting and operation. As stated in chapter 6, the allowable voltage tolerance on all rotating machines is ±10% for normal load conditions. Maintaining adequate voltage is one of the more difficult problems in mining, and is often the main constraint on mine expansion from a point of power delivery to the operation.

As mentioned earlier, the voltage drop across trailing cables that have been properly selected by current calculations is usually not a problem because of length constraints in mining. This is especially true in underground coal mining, where the maximum length is restricted by the cable size used (as shown in table 8.6). One problem here, however, is that the maximum practical trailing-cable size that can be used is also constrained by the maximum weight that workers can physically handle. For three-conductor cables, this is considered to be 4/0 AWG, but use of 4/0 AWG can cause voltage-regulation restrictions on high-horsepowered machinery. Trailing-cable voltage drop may also be a concern in surface mines where utilization is at distribution voltage levels.

Using the allowable voltage tolerance as a guide, good practice calls for limiting the maximum voltage drop under normal load conditions to not more than 10% of the nominal system voltage for each voltage level. For surface mines where machines operate at the distribution voltage, this would be equivalent to a maximum voltage drop from the substation secondary to the machines. In underground or surface mines containing power centers or a unit substation, this is not so apparent. Again, the maximum voltage drop must be restrained to 10%, but such a drop can occur across the trailing cable alone. Consequently, the power-center or unit-substation primary must be maintained as close to its normal voltage rating as practical. To obtain this objective in practice can be a very difficult task, because power centers, for example, are usually at the extreme end of the distribution system. However, most mine power-center transformers are designed with two 2.5% taps above and below the rated primary voltage. Therefore, when voltage taps are available, the maximum allowable voltage drop under normal load conditions in the distribution system (from the substation to the power centers or unit substations) is 10%.

It is interesting to compare the 10% allowance with other electrical applications. For lighting, the NEC recommends 1.0% (*2*). Industries other than mining consider 2.0% as good-to-excellent regulation and 4.0% as satisfactory.

For a thorough voltage-regulation study of a mine, the impedances of the source, the transformers, and all cables must be known. Tables 8.14 and 8.15 provide typical resistance and 60-Hz reactance values for popular mining cables (*5*); the missing parameters in these tables imply the cable is not popular or not considered suitable for mining usage. Manufacturer, power-equipment, and utility specifications must be consulted for other information. If cable sizes are not known, an assumption has to be made in order to carry out the calculations. Obviously, the loads on the power system must also be known. A circuit diagram must then be prepared and calculations performed to see if there will be adequate voltage levels at the loads. If calculated voltages are below those tolerated, system impedance must be reduced: the most convenient way is to increase cable sizes. Calculations are again performed to check for the desired result. In other words, the process is basically trial and error. It must be performed for normal load conditions; however, it is also recommended that calculations be made to ensure that critical motors can be started under worst case conditions.

Even with a small system using the per-unit method, the computations can become so involved that accurate hand calculations are extremely time consuming or nearly impossible to obtain. Consequently, load-flow computer programs are the only answer; these are discussed further in chapter 10. However, there are some simple hand-calculation procedures that may be used for initial cable sizing, or for quick verification of voltage conditions in an existing system. These methods will be explored in the next example.

Table 8.14.—Resistance and reactance of portable power cable

Conductor size	R (ac),[1] Ω/Mft		X_L (60 Hz),[2] Ω/Mft					
	75°C	90°C	G–GC G+GC, 2 kV	SHD–GC, 2 kV	SHD–GC, 5 kV	SHD–GC, 8 kV	SHD–GC, 15 kV	SHD–GC, 25 kV
AWG:								
8	0.838	0.878	0.034	—	—	—	—	—
7	.665	.696	.033	—	—	—	—	—
6	.528	.552	.032	0.038	0.043	—	—	—
5	.418	.438	.031	.036	.042	—	—	—
4	.332	.347	.031	.035	.040	0.043	—	—
3	.263	.275	.031	.034	.039	.042	—	—
2	.209	.218	.029	.033	.038	.040	0.044	—
1	.165	.173	[3].030	.033	.036	.039	.042	0.046
1/0	.128	.134	.029	.032	.035	.037	.040	.044
2/0	.102	.107	.029	.031	.034	.036	.039	.043
3/0	.081	.085	.028	.030	.033	.035	.038	.041
4/0	.065	.068	.027	.029	.032	.034	.036	.040
MCM:								
250	.055	.057	.028	.030	.031	.033	.036	.039
300	.046	.048	.027	.029	.031	.032	.035	.038
350	.039	.041	.027	.029	.030	.032	.034	.037
400	.035	.036	.027	.028	.030	.031	.033	.036
500	.028	.029	.026	.028	.029	.030	.032	.035
600	.023	.024	.026	.027	.028	.030	.032	.034
700	.020	.021	.026	.027	.028	.029	.031	.033
800	.018	.019	.025	.026	.028	.029	.030	.033
900	.016	.017	.025	.026	.027	.028	.030	.032
1,000	.014	.015	.025	.026	.027	.028	.030	.032

[1] Criteria: a. Sizes 8 to 1 based on tinned copper 94.16% conductivity.
b. Sizes 1/0 AWG and larger based on tinned copper 96.16% conductivity.
c. Resistance increased by increments per ASTM B–172, Note 7 (*3*), to compensate for stranding factor.
d. Skin effect calculated according to Arnold's Table, National Bureau of Standards Monograph 125 (*29*).
e. Nominal cross-sectional areas.

[2] Criteria: a. Based on conductor dimensions given for class-H rope-lay conductors in table 2.5 of ICEA S–19–81 (*21*).
b. Extruded-strand shield thickness, 0.015 in.
c. Insulation thickness according to nominals given in Interim Standard 6 to ICEA S–68–516 (*19*).
d. Diameter adder of 0.075 in to allow for semiconducting tape and metal-braid shield.

[3] Deviation from normal progression due to changes in insulation.

NOTE.—Dash indicates cable is not made.

Table 8.15.—Resistance and reactance of mine-power-feeder cable

Conductor size	R (ac), [1] Ω/Mft, 90°C	X_L (60 Hz), [2] Ω/Mft MP–GC, 5 kV	MP–GC, 8 kV	MP–GC, 15 kV
AWG:				
6	0.510	0.041	0.044	—
5	.404	.040	.042	—
4	.321	.038	.041	—
3	.254	.037	.039	—
2	.201	.036	.038	0.042
1	.160	.035	.037	.041
1/0	.127	.034	.035	.039
2/0	.101	.033	.034	.038
3/0	.080	.032	.033	.036
4/0	.063	.031	.032	.035
MCM:				
250	.054	.030	.031	.034
300	.045	.029	.031	.034
350	.039	.029	.039	.033
400	.034	.029	.030	.032
500	.027	.028	.029	.031
600	.023	.028	.029	.031
700	.020	.027	.028	.030
800	.017	.027	.028	.030
900	.016	.027	.027	.029
1,000	.014	.026	.027	.029

[1] Criteria: a. Based on bare copper 100% conductivity.
b. Nominal cross-sectional areas.
c. Resistance increased by increments per ASTM B–8, Note 3, to compensate for stranding factor.
d. Skin effect calculated according to Arnold's Table, National Bureau of Standards Monograph 125 (*29*).

[2] Criteria: a. Based on conductor dimensions given for class B concentric stranded conductors in table 2.2 of ICEA S–19–81 (*21*).
b. Extruded strand shield thickness, 0.015 in.
c. Insulation thickness according to nominals given in Interim Standard 5 to ICEA S–68–516 (*19*).
d. Diameter adder of 0.033 in to allow for semiconducting tape and copper-tape shield.

NOTE.—Dash indicates cable is not made.

EXAMPLE 8.4

Distribution cables for a segment of an underground coal mine must be sized. A sketch of the situation is provided in figure 8.14 where the loads are two continuous mining sections. Voltages given are line to line. In-mine measurements and analysis of identical section equipment working in similar conditions have shown an effective current demand of 58 A with 0.8 lagging power factor at the power-center primary, when the continuous miner is cutting and loading. Maximum ambient temperature is 20°C. In a detailed study, the substation transformer impedance must be included. For the sake of demonstration, however, the 7,200-V line-to-line voltage at the substation secondary will be assumed constant. The recommendation for allowable voltage drop is 10% across the distribution system. As the impedances of the feeder and portable cables must be known to make the calculation, a good place to start is to estimate line currents and make an initial cable selection by ampacity. From the given information,

$$I_1 = I_2 = 53 \text{ A}.$$

I_3 is related to I_1 and I_2 but is not necessarily equal to their sum, because of the diversity of mining

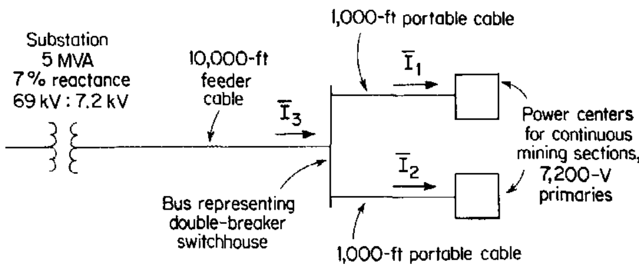


Figure 8.14.—Simplified one-line diagram for situation described in example 8.4.

operations. Chapter 4 presented the concept of demand factor (DF) where using a value from 0.7 to 0.8 is considered reasonable for mining sections: 0.8 corresponding to two sections and 0.7 to four or more sections. Therefore,

$$I_3 = \text{DF}(I_1 + I_2) \qquad (8.5)$$

or
$$I_3 = (0.8)(53 + 53) = 84.8 \text{ A}.$$

A 7,200-V system requires the use of 8-kV shielded cables, and the corrected ampacity for No. 6 AWG from table 8.7 or 8.8 and table 8.9 is

$$\text{ampacity} = (93)(1.18) = 110 \text{ A}.$$

This means that on a current basis the size is adequate for all distribution cables. Considering the preference of the coal mining industry for using only portable cables for flexibility, ground-check conductors for ground-continuity monitoring, and 90°C insulation, an SHD–GC cable is indicated. Table 8.14 can be consulted for its impedance. It can be seen in the table that No. 4 AWG is the smallest 8-kV SHD–GC portable cable readily available. Hence, a No. 4 AWG will be tried. Its impedance per 1,000 ft is

$$\begin{aligned} Z_{\text{cable}} &= 0.347 + j0.043\ \Omega \\ &= 0.35\ \underline{|7.1^\circ}\ \Omega. \end{aligned}$$

Referring to figure 8.14, the voltage drop across the distribution line conductors to either power center is (taking the power-center voltage as the reference phasor):

$$V_d = \bar{I}_3\,[10(Z_{\text{cable}})] + \bar{I}_1\,[1(Z_{\text{cable}})]$$

$$\begin{aligned} \bar{V}_d &= (84.8\underline{|-36.9^\circ})(3.5\underline{|7.1^\circ}) \\ &\quad + (53\underline{|-36.9^\circ})(0.35\underline{|7.1^\circ}) \end{aligned}$$

or
$$\begin{aligned} \bar{V}_d &= 296.8\underline{|-29.8^\circ} + 18.6\ \underline{|-29.8^\circ} \\ &= 315\underline{|-29.8^\circ}\text{ V}. \end{aligned}$$

As per-phase analysis is required to compare this drop with that allowed, the line-to-neutral voltage of the distribution system is used, or

$$V_{1n} = \frac{7{,}200}{\sqrt{3}} = 4{,}160 \text{ V}.$$

The allowable voltage drop is

$$V_d \text{ allowable} = 0.1(4{,}160) = 416 \text{ V}.$$

Therefore, the 315-V drop using No. 4 AWG SHD-GC cables is tolerable. If the voltage drop were not acceptable, an increase in cable size would lower the impedance and the drop.

This simple example had equal cable lengths to the loads, and currents operating at the same phase angle. It should be noted that typical mining systems have many more loads, varying cable length, varying load power factors, and so forth, and the complexity of hand calculations will increase substantially. Per-unit techniques are a tremendous help, but computer analysis is a much more efficient way to solve such problems. Nonetheless, the techniques shown here are useful for partial sizing or spot-checking distribution cables.

Cable Mechanical Strength

The tensile load on the cable should be determined from measurements in the mine, bearing in mind the problems discussed at the beginning of this chapter. The power-conductor breaking-strength data in table 8.16 should then be consulted to assure that the conductor size is large enough to carry the tensile load (*5*). Two things must be considered when using this table. First, grounding and ground-check conductors should not support any of the tensile load, so the overall cable breaking strength should include only the sum of the power-conductor values. Second, the working tension, especially in reeling applications, should not exceed 10% of the breaking strength because copper begins to elongate at that point. Federal regulations acknowledge the problem of exceeding the cable mechanical strength and mandate a minimum trailing-cable size for underground coal mine face equipment: No. 4 AWG for two-conductor dc cables and No. 6 AWG for three-conductor ac cables (*38*).

Table 8.16.—Solid-wire breaking strength

Conductor size, AWG	Hard— 65,000 psi		Medium— 55,000 psi		Soft— 40,000 psi	
	lb	kg	lb	kg	lb	kg
4/0	8,143	3,693.6	6,980	3,166.1	5,983	2,713.8
3/0	6,720	3,048.1	5,666	2,570.1	4,744	2,151.8
2/0	5,519	2,503.4	4,599	2,086.1	3,763	1,706.9
1/0	4,518	2,049.3	3,731	1,692.4	2,985	1,354.0
1	3,688	1,672.8	3,024	1,371.7	2,432	1,103.1
2	3,002	1,361.7	2,450	1,111.3	1,928	874.5
3	2,439	1,106.3	1,984	899.9	1,529	693.5
4	1,970	893.6	1,584	718.5	1,213	550.0
5	1,590	721.2	1,265	573.8	961.5	436.1
6	1,280	580.6	1,010	458.1	762.6	345.9
7	1,030	467.2	806.7	365.9	605.1	274.5
8	826.1	374.7	644.0	292.1	479.8	217.6
9	660.9	299.8	513.9	233.1	380.3	172.5
10	529.3	240.1	410.5	186.2	314.0	142.4
11	423	191.9	327	148.3	249	112.9
12	337	152.9	262	118.8	197	89.4
13	268	121.6	209	94.8	157	71.2
14	214	97.1	167	75.7	124	56.2
15	170	77.1	133	60.3	98.6	44.7
16	135	61.2	106	48.1	78.0	35.4
17	108	49.0	84.9	38.5	62.1	28.2
18	85.5	38.8	67.6	30.7	49.1	22.3
19	68.0	30.8	54.0	24.5	39.0	17.7
20	54.2	24.6	43.2	19.6	31.0	14.1
21	43.2	19.6	34.4	15.6	24.6	11.2
22	34.1	15.5	27.3	12.4	19.4	8.80
23	27.3	12.4	21.9	9.93	15.4	6.99
24	21.7	9.84	17.5	7.94	12.7	5.76
25	17.3	7.85	13.9	6.30	10.1	4.58
26	13.7	6.21	11.1	5.03	7.94	3.60
27	10.9	4.94	8.87	4.02	6.33	2.87
28	8.64	3.92	7.02	3.18	4.99	2.26
29	6.97	3.16	5.68	2.58	4.01	1.82
30	5.47	2.48	4.48	2.03	3.14	1.42

Short-Circuit Currents

The emergency-overload currents that copper conductors can withstand without serious insulation damage are shown in the graph in figure 8.15 (*5*). If the anticipated short-circuit currents are greater than those shown in the graph for the initial selection of conductor size, a larger conductor or a better grade of insulation should be chosen. Chapter 10 covers the calculation methods.

CABLE INSTALLATION AND HANDLING

Cables must be installed and handled correctly in order to minimize damage from tension, bending, twisting, physical wear, cold, heat, and chemical reaction. Cable maintenance costs can be reduced, cable life improved, and safety enhanced by proper installation and handling. In other words, the considerable amount of

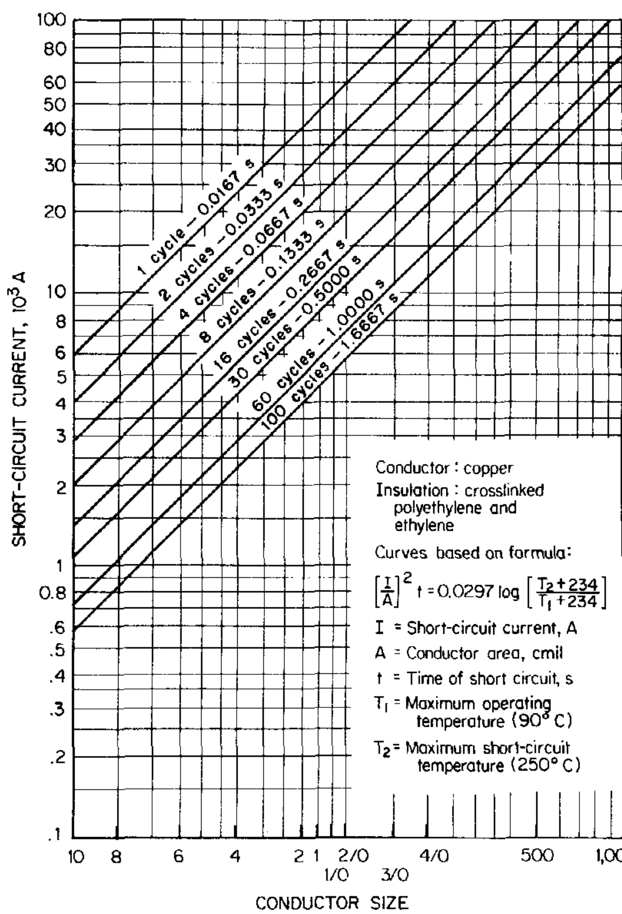


Figure 8.15.—Allowable short-circuit currents for insulated copper conductors.

engineering expertise expended in the design, manufacture, and selection processes can be wasted if the cable is not utilized properly at the mine.

Borehole Cables

The mining or electrical engineer may not have to plan and supervise the installation of a borehole cable frequently; however, since this may be the main power-supply cable for the entire or a large part of an underground mine, safety and production are highly dependent on use of the correct techniques. The term *borehole cable* comes from the common practice of installing a cable in the vertical borehole that has been drilled into an underground mine for the purpose of power entry. However, it applies to any cable that is vertically suspended into a mine, regardless of the opening in which it is placed. The typical location other than the borehole is a shaft.

Considerable tension is imposed on borehole cables, depending on the weight of the cable and the depth of the mine. Proper conductor selection, installation procedure, and suspension method are necessary to assure that the cable provides trouble-free service for the life of the mine. Shaft cables are also subject to damage from moving skips and spillage. An extremely wet environment is often encountered, which may cause corrosion and icing problems. In addition, safety precautions must be taken to keep the cable from breaking loose and falling into the opening during installation. If the power conductors have enough strength to support the weight of the cable during and after installation, messenger wires (wire ropes) with cable-gripping clamps are not necessary. Otherwise, a messenger-wire suspension method or a metallic-armored cable must be employed.

If an unarmored cable such as an MP–GC is used, the following formula can be used to calculate the safety factor for the tensile strength:

$$SF = \frac{0.80\ AT}{W}, \tag{8.6}$$

where, A = total area of power conductors, in ^{2},
T = tensile strength of conductors, psi (24,000 psi for soft drawn copper and 40,000 for medium-hard drawn copper),
and W = weight of length of cable to be suspended, lb.

If the safety factor is greater than 7, an end suspension as shown in figure 8.16 may be used without messengers (*18*). Equal tensioning of the conductors is imperative.

If messengers are needed, the wire ropes must be made from a corrosion-resistant material such as stainless steel. The typical system uses clamps or wire-type cable grips at specified intervals to secure the cable to individual messengers. The cross-sectional area and tensile strength of each messenger must be such that it can support the total weight of itself, the clamp, and at least a cable portion. Proven and tested clamps of the best quality should be used, or they will become the weak link in the installation. The high gripping force necessary on the cable jacket should be spread over a large area so the jacket is not damaged by pinching. The clamps are often vulcanized to the jacket to prevent this. In addition, a jacketing material that is not subject to cold-flowing should be utilized to prevent the clamps from loosening. A useful formula for determining the cable-clamp spacing is

$$S = \frac{9DL}{W}, \tag{8.7}$$

where S = distance between clamps, ft,
D = cable diameter, in,
L = clamp length, in,
and W = weight of cable, lb/ft.

Generally, clamp spacing is greater than 25 ft and should not be more than 100 ft.

An armored borehole cable is used where depth or location precludes the use of messengers. The armor usually consists of a serving of steel or aluminum alloy wire typically placed over the cable jacket. If this type of cable is chosen, the armor carries the tensile load, and the tension safety factor can be determined by

$$SF = \frac{0.60\ (BS)}{W}, \tag{8.8}$$

where SF = safety factor,
BS = breaking strength of each wire in armor multiplied by number of wires, lb,
and W = weight of cable length to be supported, lb.

The minimum safety factor for armored cable is 5. Armored cable may be necessary in shaft installations as

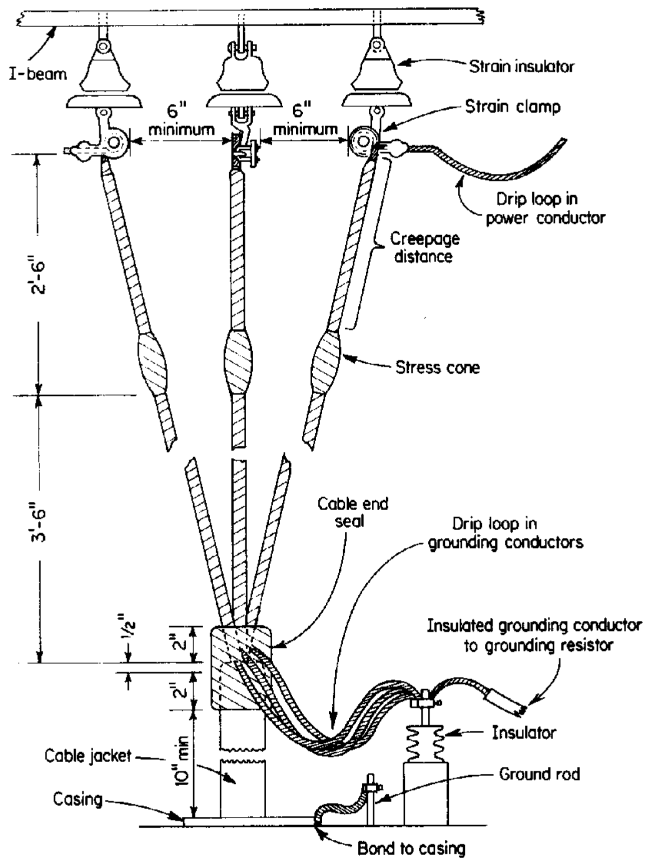


Figure 8.16.—Representative end-suspension termination for borehole cable.

protection against jacket damage from skips, cages, and spillage.

Cables can be installed either by raising or lowering. Messenger-supported cables are usually lowered into position as each messenger must be clamped at the top. Raising is often preferred for self-supported cables because of the need to have a brake on the surface as well as a pulling force at the bottom when a cable is lowered. In either case, the location should be straight and free of obstacles. It is also important to locate the cable in an area protected from any ground movement that may result from the mining operation. When a structure is used at the top to support the cable weight, it should not only be strong enough but also be placed on a substantial concrete base. Any sheave wheels utilized during the installation should be larger then the minimum bending radius specified. Rollers should be used to prevent jacket damage when the cable is dragged on rough surfaces and to minimize the pulling force by reducing friction. Crews working at the top and bottom should have a good communication system, and the personnel working at the bottom should be adequately protected from injury should the cable break loose and fall.

Feeder Cable Installation

The power-feeder cable must be located in an area that is protected from damage by mobile equipment. In underground mines, it is supported from the roof in regularly inspected fresh-air courses and haulageways on properly spaced insulated hangers, which may be supported by a messenger wire. Messenger supports are usually installed at 20-ft intervals, and cable support clips are placed on a 3/8-in messenger wire at 4-ft spacings as shown in figure 8.17 (*31*). The recommended static load per clip is 100 lb. A 1-1/4-in-diameter hole is drilled in the roof to place a 6-in-long expansion shell bolt for the messenger-wire support. The cable must not come into contact with any combustible material. In underground coal mines, the cable must be guarded in any location where miners regularly work or pass under it, unless it is 6-1/2 ft above the floor or rail. Extra lengths of cable should be stored in large figure 8 configurations in a well-ventilated area. The bending radii recommended by ICEA, shown in tables 8.17 and 8.18, should be observed for both mine power-feeder and portable cables when they are being installed (*19–21*). During installation, care must be taken not to twist the cable; that is, the reel should be turned so the cable is unrolled rather than pulled from the end of the reel. Finally, damage can be averted if a cable that has been stored on the surface during the winter is brought into the mine or a workshop to warm before being flexed.

Recommended Handling Practices

After cables are installed, proper cable-handling practices can increase personnel safety and cable life. Otherwise, damage can easily occur, especially to trailing cables, such as machine runovers, cutting by sharp edges of machine frames and stress clamps, and abrasion from sharp rocks and mine openings. Research has produced numerous recommendations to minimize this damage (*10, 14*). These are presented in the following section and are divided between those directly applicable to underground mining and those for surface mining. It should be noted, however, that some recommendations apply to all mines.

Table 8.17.—Recommended minimum bending radius, unshielded or unarmored cables, as a multiple of cable diameter

Conductor insulation thickness, mils	1.0-in diam and less	1.001- to 2.000-in diam	2.001-in diam and over
155 and less	4	5	6
170 to 310	5	6	7
325 and over	NA	7	8

NA Not available.

NOTE.—These limits do not apply to bending around curved surfaces in tension during installation. Larger bends are required for such installations.

Table 8.18.—Recommended minimum bending radius, shielded and armored cables, as a multiple of cable diameter

Cable type	*Minimum bending radius*
Armored:	
Flat tape and wire	12 times the overall diameter.
Interlocked	7 times the overall diameter, except for tape-shielded cables and where a larger radius is specified for unshielded cables.
Shielded:	
Tape	12 times the overall diameter.
Wire	Same as for portable cables unless the cable is flat-tape or wire armored.
Portable	6 times the overall diameter for round cables or the minor dimension for flat cables for insulations rated at less than 5,001 V. The minimum is 8 times the diameter for cables rated over 5,001 V.

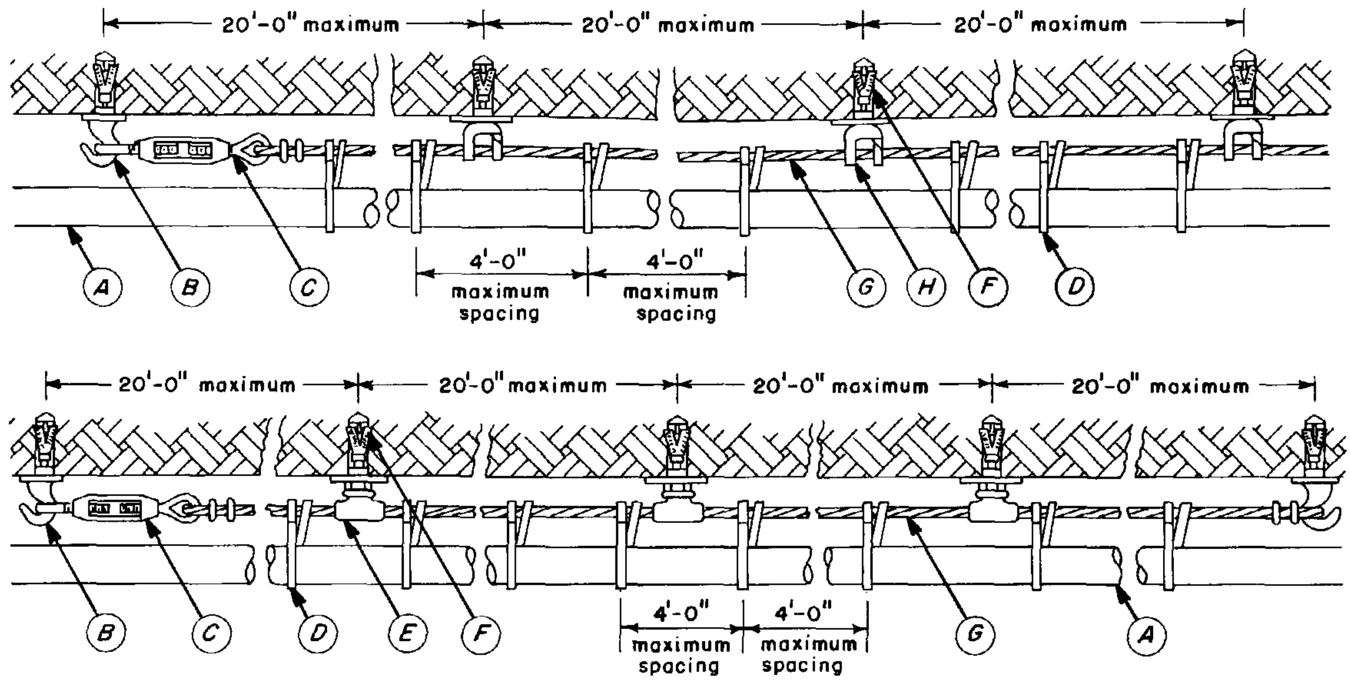
NOTE.—These limits do not apply to bending around curved surfaces in tension during installation. Larger bends are required for such installations.

Cables in Underground Mines

For reeled-cable applications, such as on shuttle cars, the cables must be anchored separately from the power equipment serving as the power source. The cable anchor points should be constructed so as to prevent personnel injury should the tie point pull out of position. When more than one reeled cable is at the tie point location, separate anchor points should be used for each cable. This will ensure that a cable will not whip dangerously should one of the anchor points fail. This precaution will also prevent subsequent cable damage.

A shock absorber should be used between the reeled cable and the anchor point to reduce instantaneous cable tensions (jerking). The use of a rubber-tire shock absorber is adequate, provided that a cable clamp is employed rather than tying the cable to the tire. However, other types of shock absorbers may be more effective. Hydraulic pressure for the machine reel should be checked periodically and set to manufacturer specifications to minimize instantaneous cable tensions.

Backspooling is the process of moving a reeled-cable vehicle in a direction opposite from that for which it was primarily designed, for example, where a shuttle car dump point is beyond the tie point in a direction opposite to (outby) the mining face (inby). Research has found that the highest cable tensions occur during backspooling, resulting from the sudden change in reel rotation as the shuttle car passes the tie point (*14*). Backspooling should be avoided, but if it is necessary, the cable anchor point should be located as far away from the travel entry as practical. This allows more time for the cable reel to change the rotation direction, and thus, cable tension will be less.



KEY

A	Feeder cable	*E*	Bulldog clamp
B	Dead-end hook	*F*	Expansion shell
C	Turnbuckle	*G*	Messenger wire
D	Cable clip	*H*	Sister hook

Figure 8.17.—Messenger wire supports for mine power-feeder cable.

Minimizing the number of cable friction points between the tie point and the face will ensure the most effective use of a cable shock absorber located at the tie point. Friction points prevent the tensions from being transferred back to the tie point. When slack cable is reeled in, every precaution should be taken to minimize reel momentum to prevent jerking the cable when the slack cable supply is exhausted. Reeling in slack cable slowly and cautiously will help minimize the possibility of whipping the cable. Maintaining a smooth mine bottom, especially in the vicinity of the tie point, will help minimize instantaneously high cable tensions resulting from the shuttle car's bouncing over an uneven mine bottom. Minimizing shuttle car speed when rounding pillar corners and passing the tie point will help prevent fast changes in reel momentum. Consequently, instantaneous cable tensions will be less severe.

Minimizing the amount of excess cable stored on a reel will prevent heat buildup in the cable. Cable abrasion on the shuttle car can be reduced by assuring that all contact points are smooth and rounded. If possible, install rollers or sheave wheels at contact points between cables and shuttle cars to reduce abrasion and cable flexure. Avoid severely bending and twisting the cable at the tie point and elsewhere. A clamp should be used to limit cable bending at the tie point to 90°. Cable twisting between the machine and the anchor point can also be minimized by locating the tie point a maximum distance away from the machine travel entry. If possible, locate repairs to the shuttle car cable outby the tie point, where cable stresses are less severe.

Recommendations for drag-cable installations are not as extensive as those for reeled cables but are just as important. First, the length of drag cable that is pulled should be minimized in order to reduce tension. Pillar corner edges should be rounded to prevent cutting or tearing of the cables. Precautions should be taken not to pull the cable over jagged rocks, timber, or other sharp objects that might damage the cable.

There are some general practices that should be followed for handling all cables. Insulated gloves should always be worn, particularly when cables are energized. All cables should be stored in a warm environment during cold winter months. If storage facilities are limited, cold cables should be placed in a warm location for at least 24 h prior to use. Small-gauge uninsulated wire must not be used to suspend cables from the roof, as it has a tendency to cut the cable jacket. All cable routes should be located in entries where they are safe from runovers. All cables should be checked periodically for damaged areas and electrical deterioration. Cables should be prevented from coming into contact with various oils, greases, or other contaminants that may deteriorate the cable jacket. When purchasing cables, make certain that they comply with all Federal and State regulations. In terms of jacket outer dimensions, this precaution will ensure effective use of packing glands and cable-laying devices.

Cables in Surface Mines

Various types of equipment are available to assist with cable handling in surface mines, from insulated long-handled hooks to elaborate hydraulic reels and aerial crossover bridges. Despite this, considerable haulage, dragging, and hand-loading of cables onto sleds and trucks is still required in many surface mines. Superficial cable damage from abrasion is a common problem, as is cable crushing by mobile equipment.

The following cable-handling recommendations for surface mines were detailed in a 1981 report to the IEEE (*10*). Systems should be developed for clearly marking cable lines along roadways and in pit areas. Suitable crossovers should be provided; in heavy traffic areas, these should be elevated. Sleds, skids, reels, and so on should be utilized rather than dragging the cable. Nylon rope or any device that can kink the cable should be avoided. Strain relief should be provided where cables are attached to equipment; rope or wire cable should not be used for this purpose. Insulated gloves are in poor condition at many minesites and provide inadequate protection for cable handling. In addition, personnel tend to place the cable across the body, negating any protection afforded by the gloves. Tools designed for cable handling should be clean and in a good state of repair. They should always have insulated handles.

Conroy and Mertain (*10*) have made a very important statement about cable handling that is applicable to all mines: "A training program for all persons engaged in cable handling should be mandatory. This should cover both electrical precautions and procedures—particularly de-energization and lockout—and physical methods. Cable handling tools and devices should be made available to all concerned, and their use should be mandated. Mechanized cable handling equipment should be considered from both a safety and an economic viewpoint; and it may occur that an actual cost saving can be demonstrated for its use."

CABLE FAILURES AND REPAIRS

Most electrical cables used in mining are designed to have a minimum life expectancy of 20 yr, with a safety factor of about 2. The life expectancy is controlled primarily by the service life of the insulating jacketing materials, which, as noted earlier, are temperature related. Where specified operating temperatures are exceeded, deterioration of the insulating materials is accelerated and the useful service life is shortened accordingly. Temperature is the main factor in the deterioration of nonportable cables that are fixed in place for extended periods of time, provided that proper installation practices are followed using good techniques. Portable cables, on the other hand, are frequently exposed to both excess generated temperature and mechanical abuse. As a result, portable cables can experience repeated failures at frequencies directly related to the proximity of the cable to the active mining area, the general mining conditions, and maintenance and cable-handling practices. For portable cables, the design life of 20 yr can easily deteriorate to 1 or 2 yr of actual in-service use.

Cable deterioration due to overheating is a time-dependent function and can go unnoticed in routine mining operations. The main indication is that the cable becomes uncomfortably hot to the touch or, in more severe cases, produces smoke or steam in wet conditions. Excess cable on a reel, created, for instance, by not taking into account cable derating factors, is the most probable contributing cause of cable overheating.

Mechanical wear can also be a time-dependent factor in cable failures, as, for example, repeated abrasion on a sheave support or spooling eye on a shuttle car. The most likely causes of failure, however, are those abuses associated with immediate or nearly immediate power interruptions. A prime example is the case where a shuttle car operator exceeds the length of the car umbilical, and the cable is tensioned to the point of failure. Similarly, a shuttle car might run over its own cable, pulling it apart or crushing the conductors and insulations. One machine running over the cable that powers another machine is also a common abuse that eventually, if not immediately, takes its toll. Obviously, special care and consideration are needed to adapt such a relatively vulnerable item as a power cable to the mining environment. Unfortunately, once a cable has been damaged to the point of requiring a repair, it becomes more vulnerable than ever, since it is almost impossible to restore its original performance characteristics.

Cable Testing

Although cables are often not tested routinely at a mine, there are instances where testing is recommended. Manufacturers test the components used in manufacturing and do carry out limited testing of completed cables as prescribed by ICEA standards (*19–21*). When couplers are added to cables at a cable repair shop, further testing can be done and the person responsible should ensure that these tests are performed effectively. The mine should require every cable removed from service to be tested before re-installation. If this were done, many costly in-service failures, production losses, and safety hazards could be prevented.

Visual observation of cable condition is an important and simple task that can be carried out even when the cable is in service. It is important to require machine operators to walk to and from their equipment along the cable and visually examine the jacket for damaged areas. Outside diameter and hardness can also be determined on in-service cables. Any significant reduction in the overall diameter is an indication of excessive tension, while increased hardness results from excessive temperature or bending.

More extensive evaluations can be made when the cable is out of service and the conductor ends are accessible. Obviously, an ohmmeter can be used to test for broken conductors; however, more sophisticated equipment is necessary to locate an open circuit. Insulation damage may be detected by using a megohmmeter or a high-potential tester (hipot), each of which can give an indication of the ability of the insulation to withstand the operating voltage without allowing excessive line-to-line or line-to-ground leakage currents. Portable megohmmeters and dc hipots can be used in the field, and ac hipots are sometimes available at cable repair shops. In order to test nonshielded cables completely, they must be surrounded by a grounding medium such as a water bath; otherwise, only the insulation directly between power conductors and between power and grounding conductors can be examined. If a shield or armor is present, either can be used as a grounding medium for the test.

Two basic types of insulation testing can be accomplished with these methods: acceptance and maintenance.

For acceptance testing, the ICEA standard procedures and voltage levels should be followed *(19–21)*. Maintenance testing requires lower voltage levels to avoid damaging the cable during the test. In both tests, the voltage level and duration of test should be adequate to ensure that the cable will perform safely in the intended service. As a general rule-of-thumb, maintenance test voltage is at 50% to 70% of the ICEA acceptance test values and should be at least as high as the cable rating. Insulation resistance values from megohmmeter testing and leakage currents at specified test voltages, obtained from dc hipot testing, can be used for preventive maintenance scheduling. If records are maintained, these tests can be used to indicate replacement schedules and prevent in-service breakdowns.

Failure Location

Failure location, often termed *fault location*, is another type of testing that is extremely important because of the susceptibility of mine cables to damage. It is less time consuming to repair or splice a cable in the mine than to replace it, and it is essential to have quick and accurate methods for locating cable failures in order to minimize the loss of production time. The Bureau of Mines has evaluated several methods, some of which follow *(11)*.

Some faults are low-resistance short circuits that can be found by visual inspection. Nonvisible short circuits can be blown by applying a high-energy power source to make them visible. However, this practice is not recommended within mines because of the potential safety hazards of fire and personnel injury. When there are faults in more than one place or when they cause low-resistance open circuits or high-resistance short circuits, they are extremely difficult to locate.

A thumper or capacitance-discharge fault locator has been used successfully in surface mines and in cable repair shops; however, associated safety hazards restrict its use in underground mines. A capacitor is charged until a spark gap breaks down, sending a pulse along the cable. If the resistance is low enough, the pulse will discharge across a short and return. The pulse will not propagate across a high-resistance open circuit at the same intensity as it was transmitted, and an acoustic sensor can be used to locate the area where the signal caused by the pulse became diminished.

The time-domain reflectometer (TDR) is another fairly successful method for locating failures. It works on the principle of a reflected pulse that either reinforces or reduces the original signal, depending on whether the discontinuity is an open or a short. The time of arrival of the echo is proportional to the distance to the failure, and the distance is then visually displayed on a meter. An accessory probe is necessary for exact failure location when a TDR is used, since the precise measurement of distance along a cable is difficult in a mine. A tone transmitter can be used in conjunction with an audio probe to locate the failure precisely. An infrared probe can also be used to locate faults where temperature increases are evident. Probes sensitive to 1° or 2° F are available; however, a current source must be attached to the cable end.

Splicing

Once a cable is damaged and made unsafe or inoperable, the damage must be repaired so that the machine might be put back into service with the least delay. In U.S. mines, repairs of this type can be made on the spot, whereas in some countries, such as the United Kingdom, the cable is replaced in its entirety and transported to a cable repair shop. The Code of Federal Regulations *(38)* states that "temporary splices in trailing cables or portable cables shall be made in a workmanlike manner and shall be mechanically strong and well insulated." It further states that "when permanent splices in trailing cables are made, they should be:

- Mechanically strong with adequate electrical conductivity,
- Entirely insulated and sealed so as to exclude moisture,
- Vulcanized or otherwise made with suitable materials to provide good bonding to the outer jacket."

By Federal regulations, only one temporary splice is permitted in any one cable at any given time, and this must be removed or repaired within 24 h. A permanent splice, as the name suggests, can remain in place indefinitely so long as it is safe and effective. The number of permanent splices in a cable is not limited, except by Pennsylvania law where no more than four permanent splices are permitted along with one temporary splice. In other words, a trailing cable may contain five splices but only for a maximum time of 24 h.

By law, specially approved splice kits or materials must be used when making a permanent splice repair. These kits and materials are tested and approved by MSHA and given an approval number similar to the approval number for cables. As with cables, a P is added to the MSHA number to signify approval for use in Pennsylvania. Depending on their basic components and outer coverings, splice kits are generally classified as tape splices, cold-sleeve splices or heat-shrink splices. Variations of these three types depend on the manufacturer.

Tape splices use tape for the conductor insulation components as well as for the outer jacket replacement materials. In some cases, slit insulation tubes might be used with the insulating tape and in other designs, blanket-type wraps might be applied in combination with moisture sealing tapes to provide an overall covering prior to applying the final layers of tape that form the jacket replacement.

Cold-sleeve splices include a variety of conductor insulation materials and usually consist of tapes that are used alone or in combination with slit-tube insulations. The main differences lie in the method of applying the outer jacket replacement. In all cases, a sleeve is slid onto the cable prior to making the conductor connections. In some designs, the splice area is built up using insulating tapes, and then a generous amount of adhesive is applied over the tape. The adhesive also serves as a lubricant and so the tubular covering must be moved into place immediately to cover the splice. This covering is designed to be slightly undersized so that it stretches as it is placed over the bulky taped area. The sleeve must be pushed into place or grasped at the end for pulling, otherwise additional drag is produced in grasping the sleeve, and it might be almost impossible to position it properly.

Some of the cold-sleeve splice coverings are prestretched during the manufacturing packaging process. When it is time to place the sleeve, it is merely moved into place over the splice area and the restraining device is removed, thus allowing the sleeve to shrink (recover) onto the splice area, which has a smaller cross section. One such device holds the sleeve in an expanded position by use of an inner plastic core, which is progressively collapsed along the length of the sleeve. In another design, the sleeve is held expanded by an adhesive bond between it and a rigid external concentric tube. A solvent is applied between the sleeve and the tube when it is time to allow the sleeve to recover onto the splice area. In still another design, the covering comes with both ends prerolled toward the middle in a toroidal fashion. The covering remains in this configuration until it is unrolled over the splice area. All of these special cold-splice coverings are designed to facilitate placement of the covering. The adhesive bonding and moisture sealing vary from manufacturer to manufacturer, with the necessary components included in the kit.

The heat-shrink splice coverings are also prestretched but in a different sense. The sleeves are made of special cross-linked polymers, which stretch readily when warm. In the manufacturing process, they are heated to 270°F and expanded radially to a given oversized dimension and then cooled. While at room temperature, they retain this oversized dimension, which easily accommodates placement over the spliced cable. When reheated to 250°F, the sleeve will shrink onto the splice area. A factory-applied thermal-melt adhesive on the inside surface of the sleeve softens with the applied heat and forms a moisture seal and adhesive bond between the sleeve and the original cable jacket. Similar smaller sleeves are used for conductor insulation over the individual power conductors.

The packaged splicing kits contain all the materials necessary to reinsulate and rejacket the splice area, together with special illustrated instructions. Cleaning materials, a cloth, and a can of solvent might also be included, together with an emery cloth (nonconducting) or scraper, which is used to prepare the surface of the cable jacketing for improving the adhesive bond. The connectors used to rejoin the power, grounding, and ground-check conductors may or may not be included in the kits, depending mainly upon customer specifications.

The Bureau of Mines has sponsored research into the causes and prevention of splice failures, with emphasis on shuttle car cables (*26, 34–35*). Since this research has had a positive influence on the splice kits and insulation procedures used in the mining industry, a brief overview of the results is presented.

Deenergizing Procedures

In the interest of safety, it is essential to follow strict lockout procedures before cutting into any cable that has been put into use. Improper lockout prior to splicing cables has been a major source of electrocutions in the mining industry. The individual making the repair must go to the power-source end of the cable, disconnect the cable, and tag (danger off) and lock out the disconnecting device, which is usually a coupler. This step must never be left for someone else to do.

Cable Preparation

After the cable has been properly disconnected from the power source, the next step is to remove the damaged area and prepare the conductors for splicing. The preparation procedures vary slightly depending on the types of connections used, and a representative procedure is presented in figure 8.18. A guide or template is recommended for marking the cable for cutting and removing the insulation and jacketing. Such a guide is included in some kits but can be easily fabricated from light-colored material for repeated use. Use of a marking guide can help to standardize procedures and increase speed and accuracy.

Once the cable pieces are properly marked, the next step is to remove the unwanted insulation. An effective method is presented in figure 8.19. The key here is to use a sharp knife and to take care not to cut all the way into the conductors. Nicking the conductor strands will minimize their performance. The conductor connections are usually staggered to help reduce bulkiness (fig. 8.20). The marking guide maintains good positioning, and before the insulation is actually cut into, the guide allows an immediate check point to determine that the power conductors are properly registered; that is, black to black, white to white, and so forth.

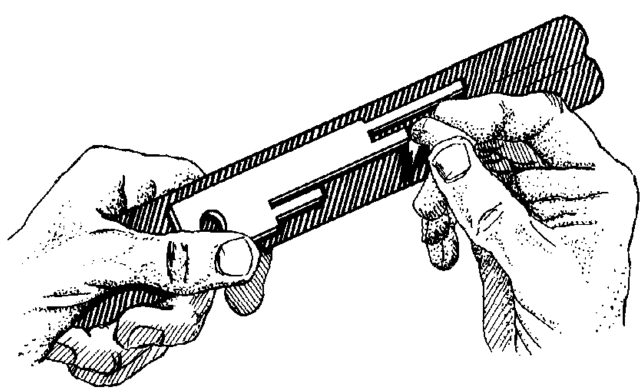


Figure 8.18.—Splice layout using template for staggered connections.

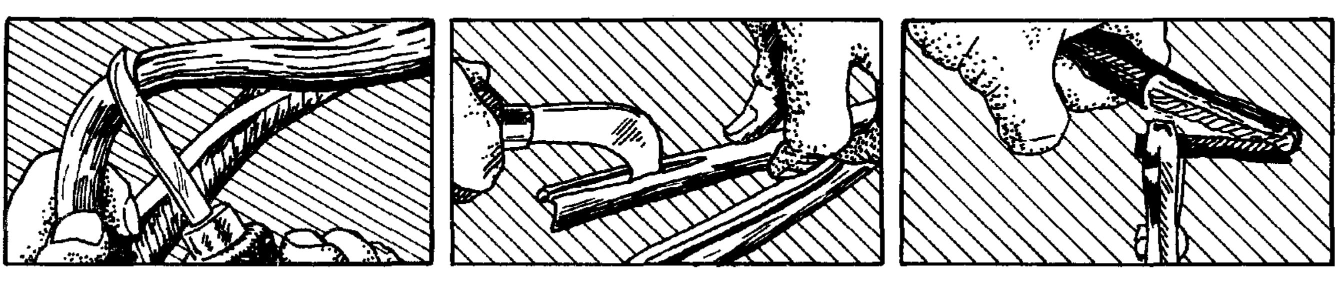


Figure 8.19.—Effective method for removing unwanted insulation.

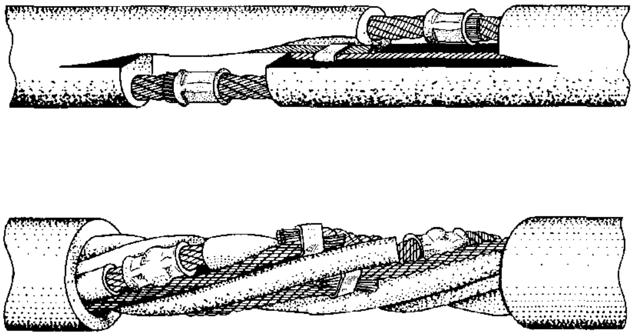


Figure 8.20.—Staggering splice connections.

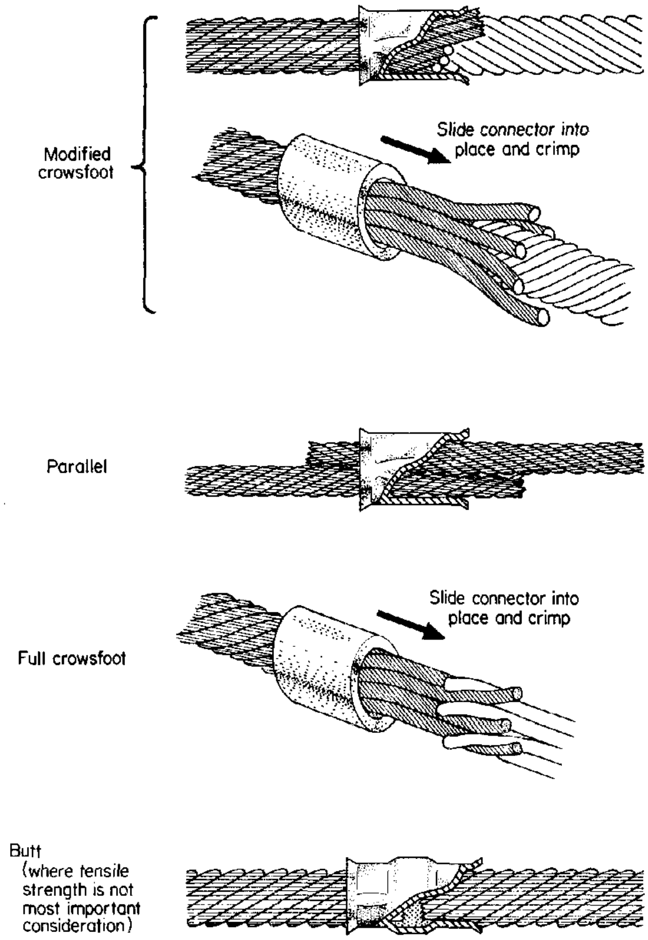


Figure 8.21.—Examples of popular connectors and connections used in splices.

Connectors

A variety of connectors (fig. 8.21) and connector crimping tools are available. It is generally recommended that lapped-joint connections be used where maximum tensile strength is desirable, as in shuttle car cables. Research has shown that the modified crowsfoot connection, when properly installed, can restore 80% to 100% of the original tensile strength for Nos. 6, 4, and 2 AWG conductors, the smaller conductors being easier to restore. The modified crowsfoot connection offers additional advantages of axial symmetry (no mechanical couple) and a small profile (an important consideration with multiconductor cables).

The lap joints, being shorter than butt joints, are better for reeling applications since repeated flexure on a long connection might accelerate fatigue failures. The lap joints generally outperform butt connections in tensile strength, and Bureau-of-Mines-sponsored research has shown that restoring tensile strength is probably more important than restoring high flexibility to shuttle car cables. Either way, the lap connections are superior.

A major consideration in obtaining high tensile strength is the use of the proper crimping tool for a given connector. Furthermore, tools that reduce or eliminate operator judgment tend to provide the best repeatability, since overcrimping as well as undercrimping can reduce tensile values.

The lap connection has also been recommended for restoring the grounding conductors. In this case, it has been suggested that the connection be a little forgiving and allow the grounding conductors to slip slightly inside the connector should the cable undergo excess tension. This would cause the power conductors to take all the tension and would perhaps prevent the grounding conductors from being tensioned, so that they would be the last to fail. Although this concept has not been verified, it may have some merit (assuming of course that a good electrical connection is maintained and that otherwise the grounding conductor might not extend sufficiently under tension).

An important consideration in selecting and installing connectors in reeled cables is awareness of the connector profile after installation. Bulky connectors with abrupt edges are more difficult to insulate effectively, simply because they tend to cut through the insulation materials with repeated cycling under normal operations. These connectors can also cause excess pressure and fatigue on adjacent grounding conductors, which are uninsulated and somewhat less protected from mechanical abuse.

Although generally unsatisfactory for related applications, the butt connection is effective for larger portable cables such as those used for continuous miners, because it offers the least bulk. Here it does not need to withstand the repetitious flexing so often experienced by the smaller size cables.

Reinsulating

Because of the repeated bending stresses, reinsulating procedures require special attention in portable cables. The key is to provide a flexible joint and seal where the new insulation contacts the original cable insulation. As shown in figure 8.22, this is best accomplished using soft rubber tape that completely fills the volume and laps over the original insulating material. The lap is important since a tape fill that only butts to the insulation is almost sure to separate after very little flexing. Where it is desirable to use slit tubes as part of the reinsulating procedure, soft tape is recommended underneath and over the tubing.

Soft rubber tape alone will not hold up under repeated cable flexing. Therefore it is further recommended that tougher vinyl tape be applied over the rubber tape. The vinyl tape accomplishes two objectives: it restrains the soft tape, thus preventing it from squeezing and extruding from its intended area, and it allows the reinsulated connections to slide relative to one another and the

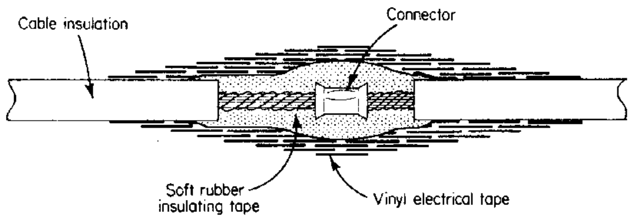


Figure 8.22.—Reinsulating power conductors with soft rubber tape.

grounding conductors with minimum wear. The vinyl tape can also be used to bind the multiple conductors together for maintaining positioning and limiting excess relative motion. A single-width wrap of tape near the middle of the splice area is generally sufficient. Care should be taken not to use too much vinyl tape over the splice area, since the final splice covering is generally intended to bond to the inner parts and the vinyl can in some cases make the subsequent adhesive bond less effective.

In the case of heat-shrink splices, the conductor insulations are also made of heat-shrinkable tubing, and the tubes must be slipped onto the conductors before the connector is applied. When shrinking the tubes with a heat source, care must be taken to avoid overheating or rupturing the insulation on the sharp connector edge, and so forth. After heating, the installer should inspect the work to ensure that the adhesive has sealed the sleeve to the original insulation material. This is especially important for flat cables where the insulation cross sections are not always smoothly continuous. The heat-shrink insulation tubes provide a generous lap over the original insulation and are usually tough and resistant to rubbing wear inside the splice.

Shielded cables require complete shield replacement over the conductor insulation. This process is similar for all cables but requires more care in high-voltage splices and will be covered later.

Rejacketing

The outer splice covering provides protection for the more delicate inner splice components and serves basically the same purpose as the rugged cable jacketing. It is important that it be tough and flexible and at the same time maintain an acceptable bond to the original jacketing material. Of principal concern is a splice condition generally termed *end lipping*, the result of the splice-covering ends' pulling away from the cable jacket. When this occurs, contaminants such as fine solids and water can enter the splice and contribute to failure or an unsafe condition. The causes of end lipping are combinations of poor adhesive bonds, discontinuities and dissimilar materials, or simply physical wear as a result of the normal mining process. The amount of end lipping will vary depending on the types of covering used and the conditions to which it is exposed.

Various attempts have been made to provide splicing products that resist end lipping, with varying degrees of success. The general recommendation is to prevent occurrence by making every effort to clean the cable surface where the adhesive bond is to be made. As a minimum, any soiled surfaces should be wiped with a suitable solvent and abraided with nonconductive emery material to reveal a fresh bonding surface. It should be noted that newer cable jackets can be more difficult to bond simply because waxes from the manufacturing process are often on the jacket surface.

In general, the heat-shrink sleeves are good abrasion-resistant coverings. However, it should be noted that they are usually stiffer and sometimes require more attention to obtain a good and lasting bond to the cable jacket. Furthermore, a heat-shrink sleeve can take on a thermal set, for example, if it is allowed to cool in a curved position on a reel and then is later unreeled while still cool. The cold splices are generally quite flexible, but end lipping can result from bending and scuffing on various machine parts. Major cable and splice wear usually occurs during contact with the machine and its spooling mechanism when the relative motion is at a maximum.

It is normal practice to tape down the ends of the splice coverings. This can help to reduce end lipping and can also prevent foreign matter from entering an already lipped end. Regular inspection and renewal of the end taping is a must, since abrasive wear and cutting on machine parts can quickly destroy even well-applied end tapes. The use of exposed soft rubber tapes is considered poor cable repair practice. The softer rubber tapes can provide good moisture seals but should be protected with an overcovering of tough vinyl tape. This vinyl tape will help contain the rubber tape, and the lower friction will give better wear characteristics.

High-Voltage Cable Splices

When splices are required on high-voltage cables in underground mines or in surface mines, problems are introduced by the presence of shields and semiconducting layers. The high voltage means that care must be taken to achieve an excellent splice, that is, one that closely approximates the qualities of the original cable.

The splicing procedure is basically the same as that just covered, but the cable insulation and jacket are usually tapered as shown in figure 8.23. Tapering is performed to improve the bond, increase the leakage path length, and lessen the chance of a direct vertical path to a ground plane. Extra care and skill are necessary as any damage to the insulation during splicing, such as a small cut, will cause more rapid dielectric failure at higher voltages. In the same context, a small protrusion such as a sharp edged connector or loose wire will be a more noticeable failure initiator as the voltage increases.

The presence of semiconductive tape and braid or tape shielding in cables requires extra caution. The shielding system must be separated from the conductor insulation in such a way that residue on the insulation from the semiconducting tape is completely removed before the conductor insulation is reapplied. In addition, the wires from a braid shield must not protrude into the insulation. The shield must be replaced completely, and the grounding conductor must be placed in intimate contact with the shield.

Splice Inspection

A recommendation for improving splice performance is to inspect splices on a regular basis and use the information to institute new procedures or even new splice kit designs. An opportune time for doing this is before shipping an extensively damaged cable to a repair shop for vulcanized repairs. When the cable is idle and quite

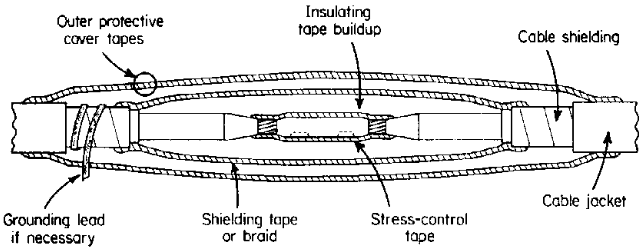


Figure 8.23.—Typical taped splice in high-voltage shielded cable.

accessible, perhaps stored in a supply yard, old splices can be cut out and scrutinized. Just the simple process of slitting the old splice lengthwise using a sharp linoleum knife can provide good information regarding insulation procedures, wear characteristics, effectiveness of bonds, and so forth. Electrical tests and tensile evaluations can be made on the insulations. Samplings of this type can readily provide extensive data on splices with varying amounts of in-service time.

TROLLEY SYSTEMS

The conductors that provide power for electric track haulage systems form a major part of the power-distribution system in many underground mines. The trolley system is a potential hazard for fires, ignition of methane, and shock since it utilizes uninsulated conductors. The danger in underground coal mines is greater than that in surface mines because of limited space and the presence of methane, However, all mines that utilize trolley conductors can benefit from proper design, selection, and installation of the system components.

Several conductors are used in the trolley circuit: trolley wire, feeder cable, rail-bond cable and steel track rails. The trolley wire supplies power directly to a rail-mounted vehicle, such as a mine locomotive, through a collector called a shoe or harp. The trolley wire and collector connection can cause frequent severe arcing, which may damage either part and cause an obvious ignition hazard. Proper positioning of the trolley wire, particularly at curves and switches, correct holding force on the collector, and the required amount of lubrication are necessary to minimize arcing.

A feeder cable supplies power to the trolley wire. Consequently, both must be sized properly to provide enough current-carrying capacity yet minimize heating and voltage drop. In addition, rectifiers must be positioned at adequate intervals to supply the proper voltage to the feeder.

The current return path utilizes the steel rails, which must have adequate conductivity to minimize the total system resistance. Rails are laid in segments, and the connections between them can loosen or the rails could break; hence, rail-bond cable is installed to maintain continuity. Rail-bond cable is attached at each rail joint, and as a further precaution, between the two rails at specified intervals (cross bonds).

Trolley Wire

The trolley-wire conductor used in mines is hard-drawn copper, but brass is available for high-speed surface transportation. Round, grooved, figure 8 and figure 9 (deep-section) wire shapes, shown in figure 8.24, are available (*31*). At one time, round wire was prevalent, but the clamps necessary to support it caused the collector to jump and arc, so it was replaced with the figure 8 shape. Additional problems occurred with the figure 8 because it twisted and kinked when being reeled and unreeled during installation, and it frequently pulled out of hangers on curves. Consequently, the grooved type was developed and, together with the figure 9, has almost completely replaced the round and figure 8 shapes. Figure 9 and deep-grooved shapes are almost mandatory with a 350-MCM size and above, because these sizes require large splices and fittings and the widths are too large for proper tracking of the collector.

The upper section of the wire, to which the support clamp attaches, has the same width dimension whether the wire is grooved, figure 8 or figure 9. Table 8.19 provides the necessary specifications for correct wire size selection (*31*). The most common wire is 350 or 400 MCM (both often called 6/0) figure 9.

Trolley Feeder

In order to reduce voltage drop and supply the necessary current, a feeder cable, which is uninsulated and stranded, is hung alongside the trolley wire. Both aluminum and copper feeders are used, and their size depends on the load drawn by the track vehicles and the voltage regulation desired. Common sizes are 1,000 MCM copper or 1,590 MCM aluminum. Tables 8.20 and 8.21 specify copper feeder data (*31*). As noted in the tables, feeder can be purchased with a weatherproof jacket.

Supports, Lubrication, and Turnouts

As shown in figure 8.25, the feeder cable and trolley wire can be hung side by side to gain additional support clearance. The feeder can also be used as a messenger to increase the support-bracket spacing, as shown in figure 8.26. In this configuration, a cushioning effect is provided for the trolley wire since the wire is free to flex under pressure.

Typical brackets for supporting trolley and feeder are shown in figure 8.25 and 8.26. The amount of deflection or sag between supports can be calculated by

$$D = \frac{3WL^2}{2T}, \tag{8.9}$$

where D = sag, in,
W = weight, lb/ft,
L = distance between supports, ft,
and T = tension, lb.

Since the figure 9 350-MCM conductor has a breaking strength of 12,000 lb, it can safely be tensioned to 1,200 lb, which is 10% of the breaking strength. This will reduce sag and keep the wire straight and level. The dead-end hooks and turnbuckles shown in figure 8.27 are used to install tension in the wire.

The maximum spacing recommended for roof-mounted support for a semicatenary installation (fig. 8.26) is 20 ft. Direct suspension (fig. 8.25) spacing should be less than 15 ft. Table 8.22 gives support spacings on curves (*31*). When selecting proper support types and spacings,

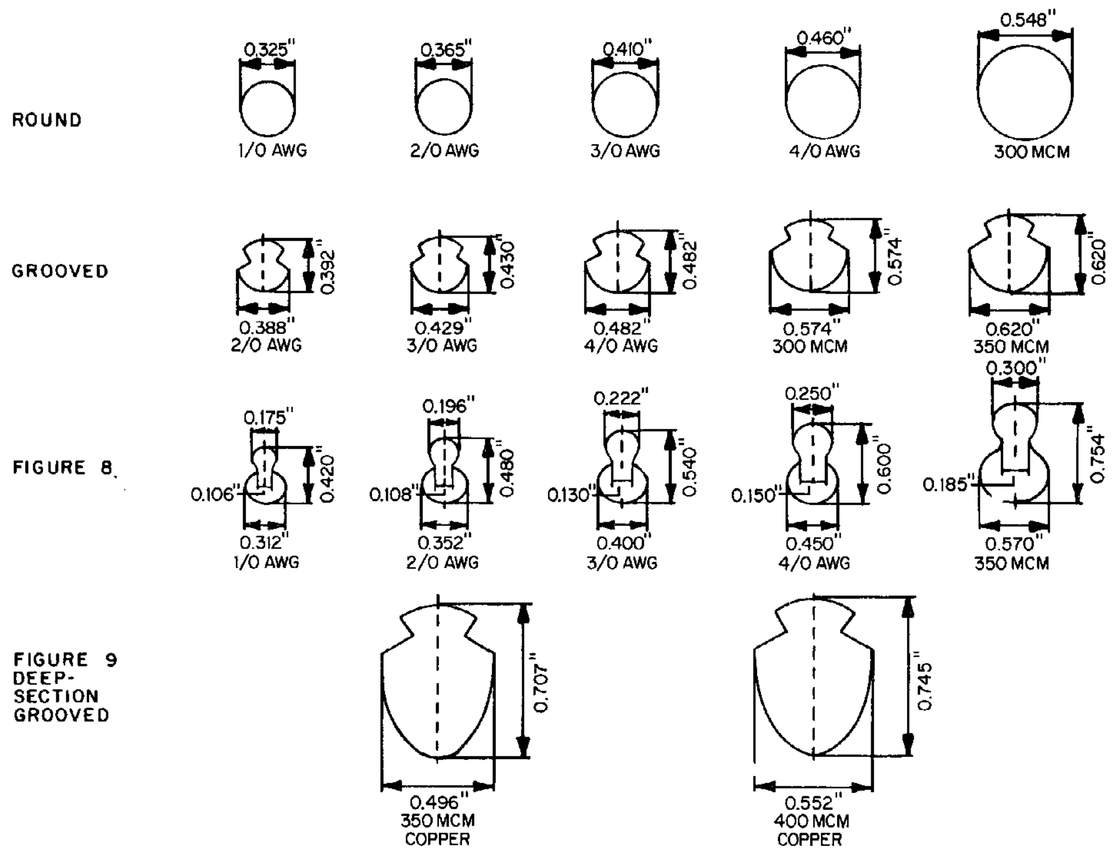


Figure 8.24.—Trolley-wire cross sections.

Table 8.19.—Trolley-wire specifications

Type of wire	Nominal size, AWG or MCM	Cross-sectional area: Nominal MCM	Cross-sectional area: Actual MCM	Cross-sectional area: Actual in ^{2}	Weight lb/Mft	Weight lb/mi	dc resistance (volts drop per amp at 20°C) Ω or V /Mft	dc resistance Ω or V /mi	Minimum tensile strength, psi	Minimum breaking load, lb	Elongation within 10 in, %
Round, hard-drawn copper, 97.16% conductivity	1/0	105.6	105.6	0.0829	319.5	1,687	0.1011	0.5339	54,500	4,518	2.40
	2/0	133.1	133.1	.1045	402.8	2,127	.08021	.4235	52,800	5,519	2.80
	3/0	167.8	167.8	.1318	507.8	2,681	.06362	.3359	51,000	6,720	3.25
	4/0	211.6	211.6	.1662	640.5	3,382	.05045	.2664	49,000	8,143	3.75
	300	300.0	300.0	.2356	908.0	4,794	.03558	.1879	46,400	10,930	4.50
Grooved, hard-drawn copper, 97.16% conductivity.............	2/0	133.1	137.9	.1083	417.6	2,205	.07741	.4087	50,200	5,437	2.80
	3/0	167.8	167.3	.1314	506.4	2,674	.06380	.3369	48,500	6,373	3.25
	4/0	211.6	212.0	.1665	641.9	3,389	.05035	.2659	46,600	7,759	3.75
	300	300.0	299.8	.2355	907.6	4,792	.03560	.1880	44,200	10,410	4.50
	350	350.0	351.2	.2758	1063.0	5,612	.03040	.1605	42,800	11,800	4.50
Figure 8, hard-drawn copper, 97.16% conductivity.............	1/0	105.6	105.6	.0329	319.5	1,687	.1011	.5340	51,800	4,294	2.40
	2/0	133.1	133.1	.1045	402.8	2,127	.08021	.4235	50,200	5,246	2.80
	3/0	167.8	167.8	.1318	508.0	2,682	.06361	.3359	48,500	6,392	3.25
	4/0	211.6	211.6	.1662	640.5	3,382	.05044	.2663	46,600	7,745	3.75
	350	350.0	350.1	.2750	1,060.0	5,597	.03049	.1610	42,800	11,770	4.50
Figure 9 deep- section, hard-drawn copper, 97.16% conductivity..........................	350	350.0	348.9	.2740	1,056.0	5,576	.03060	.1616	42,800	11,730	4.50
	400	400.0	397.2	.3120	1,202.0	6,347	.02687	.1419	41,300	12,890	4.50

Table 8.20.—Characteristic data for solid copper feeder cable

Conductor size, AWG	Section area			Overall diameter, in		Weight, lb/Mft		Bare wire breaking strength, lb	
	cmil	in²	m²	Bare	Weatherproof	Bare	Insulated	Hard drawn	Annealed
0000	211,600	0.1662	107	0.4600	0.6163	641	767	8,143	5,320
000	167,800	.1318	85.0	.4096	.5659	508	629	6,722	4,220
00	133,100	.1045	67.4	.3648	.5211	403	502	5,519	3,340
0	105,500	.08289	53.5	.3249	.4812	320	407	4,517	2,650
1	83,690	.06573	42.4	.2893	.4456	253	316	3,688	2,100
2	66,370	.05213	33.6	.2576	.3826	210	260	3,003	1,670
3	52,640	.04134	26.7	.2294	.3544	159	199	2,439	1,325
4	41,740	.03278	21.2	.2043	.3293	126	164	1,970	1,050
5	33,100	.02600	16.8	.1819	.3069	100	135	1,591	880
6	26,250	.02062	13.3	.1620	.2870	79	112	1,280	700
7	20,870	.01635	10.6	.1443	.2693	63	NA	1,030	550
8	16,510	.01297	8.37	.1285	.2535	50	75	826	440

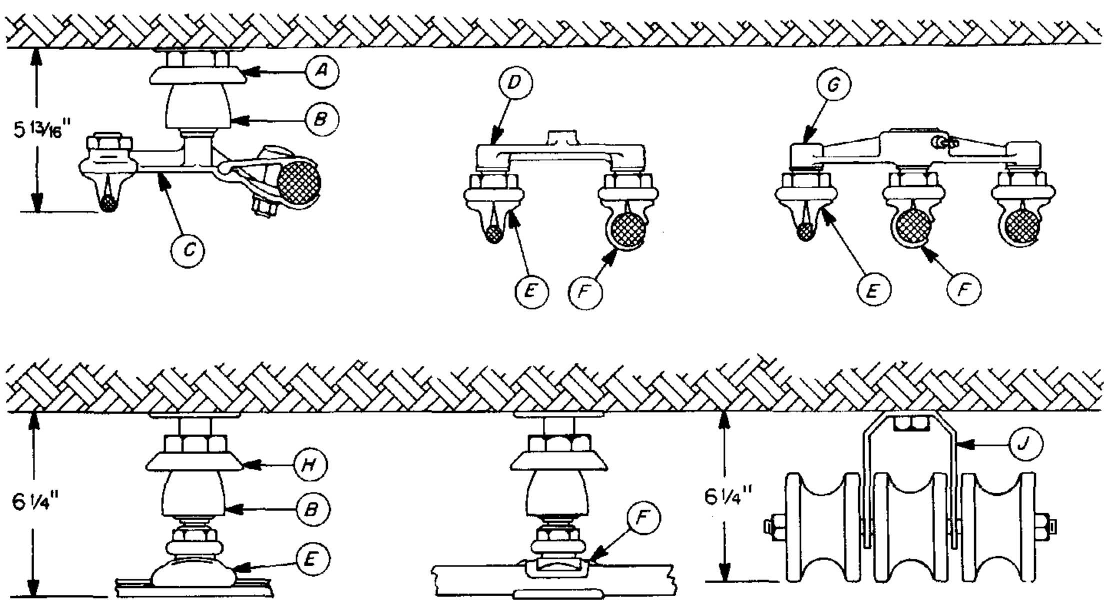
NA Not available.

Table 8.21.—Characteristic data for stranded copper feeder cable

Conductor size	Cross-sectional area			Number of wires in strand	Overall diameter, in		Weight, lb/Mft		Bare wire breaking strength, lb		Resistance, Ω/Mft at 20°C, standard annealed
	cmil	in²	m²		Bare	Weatherproof	Bare	Insulated	Hard drawn	Soft annealed	
MCM:											
2,000	2,000,000	1.571	1,014	91	1.630	1.880	6,175	7,008	87,790	43,830	0.005289
1,750	1,750,000	1.374	887	91	1.526	1.776	5,403	6,193	77,930	38,350	.006045
1,500	1,500,000	1.178	760	61	1.411	1.661	4,631	5,380	65,840	32,870	.007052
1,250	1,250,000	.9817	633	61	1.288	1.538	3,859	4,508	55,670	27,390	.008463
1,000	1,000,000	.7854	507	61	1.152	1.402	3,088	3,674	45,030	21,910	.010578
900	900,000	.7069	456	61	1.094	1.313	2,779	3,332	40,520	19,270	.011753
800	800,000	.6283	405	61	1.031	1.250	2,470	2,992	36,020	17,530	.013223
750	750,000	.5890	380	61	.998	1.217	2,316	2,822	34,090	16,430	.014104
700	700,000	.5498	355	61	.964	1.183	2,161	2,650	31,820	15,340	.015112
600	600,000	.4712	304	37	.893	1.112	1,853	2,235	27,020	13,150	.017631
500	500,000	.3927	253	37	.813	1.001	1,544	1,894	22,510	10,960	.021157
450	450,000	.3534	228	37	.772	.960	1,389	1,724	20,450	9,860	.023508
400	400,000	.3142	203	19	.726	.914	1,235	1,553	17,560	8,765	.026447
350	350,000	.2749	177	19	.679	.867	1,081	1,345	15,590	7,669	.030225
300	300,000	.2356	152	19	.629	.817	926	1,174	13,510	6,574	.035262
250	250,000	.1963	127	19	.574	.762	772	985	11,260	5,478	.042315
AWG:											
0000	211,600	.1662	107	[1]7,19	.528	.684	653	800	9,617	4,637	.04999
000	167,800	.1318	85.0	[1]7,19	.470	.626	518	653	7,366	3,677	.06304
00	133,100	.1045	67.4	7	.414	.570	411	522	5,926	2,916	.07949
0	105,500	.08289	53.5	7	.368	.524	326	424	4,752	2,312	.10024
1	83,690	.06573	42.4	7	.328	.484	258	328	3,804	1,834	.1264
2	66,370	.05213	33.6	7	.292	.417	205	270	3,045	1,525	.1594
3	52,640	.04134	26.7	7	.260	.385	163	206	2,433	1,209	.2009
4	41,740	.03278	21.2	7	.232	.357	129	170	1,938	959	.2535
5	33,100	.02600	16.8	7	.206	.331	102	140	1,542	761	.3195
6	26,250	.02062	13.3	7	.184	.309	81	115	1,228	603	.4029
7	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	.5080
8	16,510	.01297	8.37	7	.146	.271	51	78	777	379	.6406

NA Not available.

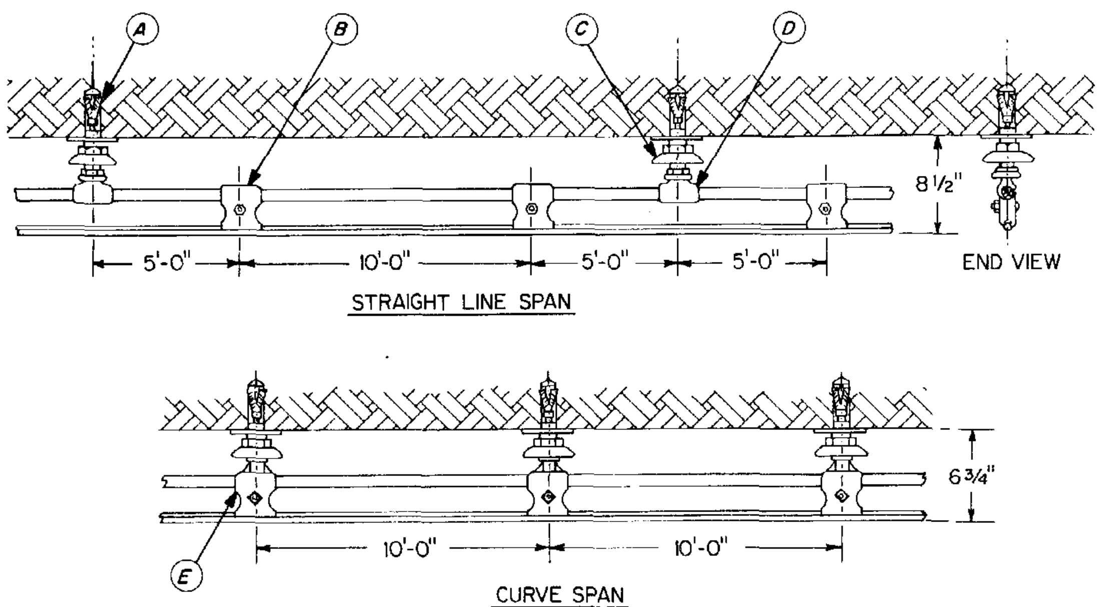
[1] Sizes AWG 0000 and 000 cable are usually made of 7 strands when bare and 19 strands when insulated.



KEY

- *A* Mine hanger
- *B* Dirigo spool insulator
- *C* Dual suspension clamp
- *D* Double yoke
- *E* Bulldog clamp
- *F* Bulldog feeder sling
- *G* Triple yoke
- *H* Mine hanger
- *J* Triple horizontal insulator

Figure 8.25.—Typical trolley-wire and feeder-cable supports.



KEY

- *A* Expansion bolt
- *B* Combination clamp without boss
- *C* Mine hanger
- *D* Bulldog feeder sling
- *E* Combination clamp with boss

Figure 8.26.—Trolly-wire semicatenary suspension.

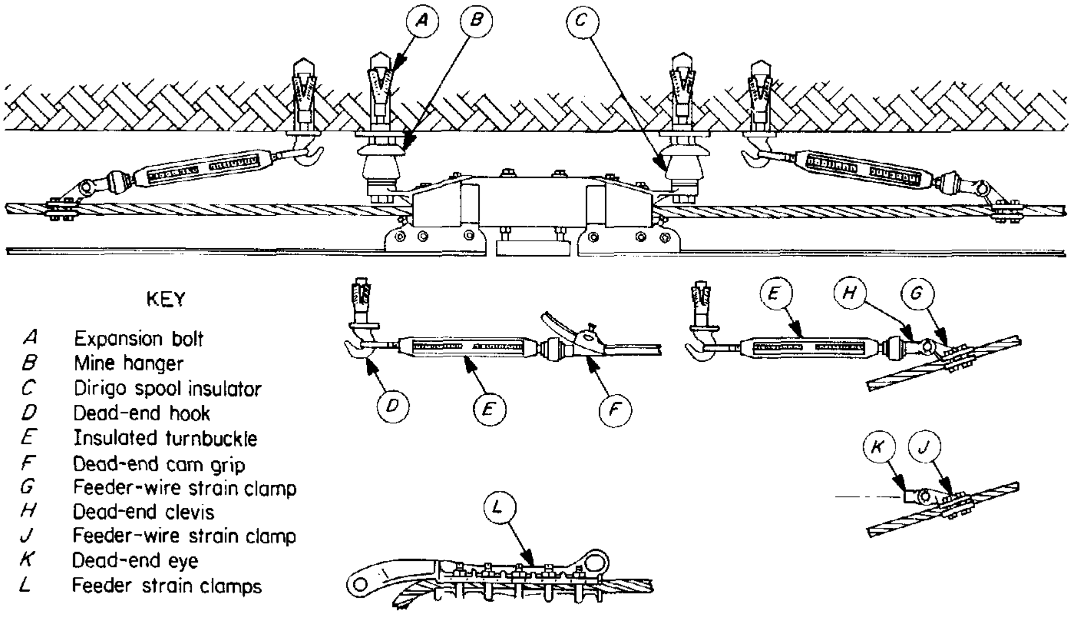


Figure 8.27.—Trolley system accessories.

Table 8.22.—Trolley-wire support spacings on curves

Radius of curve, ft	*Maximum spacing, ft, with deflection angle of 5°*	*Radius of curve, ft*	*Maximum spacing, ft, with deflection angle of 5°*
350 and over...	[1]30	150................	13
300................	26	120................	10
250................	22	100................	9
200................	17	80..................	7

[1] On straight lines, spacing can be increased to 30 ft where wire is 5 ft or more above rail. If wire is less than 5 ft above rail, the limit on inside construction is 20 ft.

the weight of the trolley guards must be included. Underground coal mine regulations require guards to be positioned wherever personnel normally work or pass under the uninsulated trolley and feeder wires (*38*). When the roof is uneven or too high for the trolley wire, pole extension brackets can be used from either the roof or rib.

For trolley haulage outside the mine, catenary or direct support can be used. Simple catenary support, suspending the trolley wire from a messenger, works best for long haulage distances since 100-ft spans are possible on straight track. Compound catenaries, using two messenger wires and subspan catenaries are also employed. Semicatenary or direct suspension can be used on the surface, employing the same components as shown in figures 8.25 and 8.26 but mounting the hangers on wooden or metal poles or structures, and the spacing may be increased to 25 or 30 ft. The deflection formula (equation 8.9) may be used for more precise calculations.

A properly installed trolley wire looks level and straight, without bends, kinks, or sags. The rubbing surface of the metal should look polished and smooth, not scraped or burned bright. A graphite-based lubricant should be used periodically to form a smooth contact surface and maintain the smooth polished-brown appearance. An unlubricated, uneven trolley wire wears out quickly, is unsafe because of arcing, reduces power efficiency, and also wears out the collector.

At track switches or turnouts, trolley frogs must be installed at the proper location to assure that the collector will pass on to the correct wire. Normally, a standard 10° trolley frog is used for any degree of track turnout. The frog must be positioned far enough beyond the track turnout that sufficient side force exists to guide the collector on to the correct wire. Too much side force will cause the collector to be pulled off the wire. The location for frog angles is found by determining the position where the collector pole exerts adequate side force on the collector. This point occurs where the pole angle is equal to the track-frog angle plus 10°.

Rails and Bonds

Another important consideration of the trolley system is the resistance of the current return path: the rails and bonds. The specific resistance of high-carbon steel rails is 118 Ω/cmil-ft at 20° C, about 12 times that of copper. In table 8.23, this value is used to provide resistance values for mine track per rail.

Table 8.23.—Resistance of steel rail at 20°C

Rail weight, lb/yd	*Resistance, Ω/Mft*	*Rail weight, lb/Mft*	*Resistance, Ω/Mft*
40..................	0.0250	80..................	.0124
50..................	.0200	90..................	.0111
60..................	.0166	100................	.0100
70..................	.0143		

A major concern with rail resistance occurs at switch contacts and bolted rail joints. These points can have very large resistances, sometimes approaching an open circuit. Rail bonds are used to ensure a low-resistance contact or joint. With inadequate bonding, the return current from trolley system loads can stray into the mine floor or earth, and the stray current can cause electrical problems throughout the mine, including nuisance tripping of protective circuitry.

Rail-bond cable is soft, stranded copper in sizes from 2/0 to 500 MCM. It is attached with stud terminals or welds across every rail joint. Welded terminals are preferred for permanent application on main-line haulage; however, the amount of heat used must be controlled in order to prevent steel rail recrystallization. Thermite welds have also been used successfully in this operation. Cross bonds are applied at least every 200 ft along the track, so that if one rail or a bond along a rail breaks, the current return path can be completed through the other rail. This practice also halves the resistance in the current return circuit by paralleling the two rails. Cross bonds are also recommended at all rail turnouts in conjunction with the switch points.

All bonds are susceptible to damage by the wheels of derailed mine cars and hence should be located next to ties and secured to the tie side for protection. If possible, joint bonds should be placed under the rail-connecting plates.

Bond size is usually determined by voltage drop rather than by current-carrying capacity. It is a general rule that bare conductors will carry 1 A per 5,000 cmil of area without excessive heating. Five times this amount can be carried for brief periods, and short bonds on heavy rail can carry 150% of the normal load current. Bond-cable resistances are shown in table 8.24 along with the usual sizes according to rail weights. The terminal resistance is negligible at 1 $\mu\Omega$, and cross bonds are normally equal in length on the track gauge, typically 42 in. Joint bonds are usually 10 in long. The added resistance of joints, expressed in feet of rail, can be found from the chart in figure 8.28 (*22*).

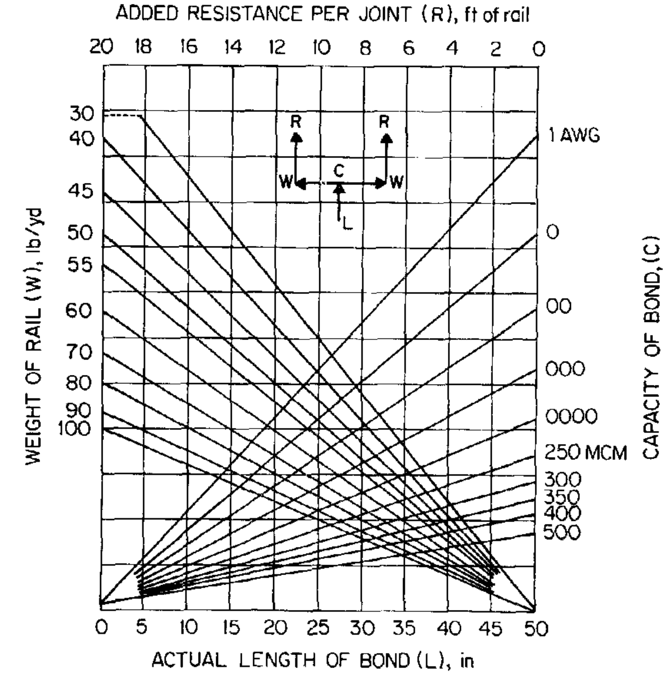


Figure 8.28.—Theoretical resistance of bonded joint. The proper moves to make in using this chart have been indicated by the small diagram in the upper central portion. Starting with the length (L) of the bond, move vertically to the bond capacity (C), then horizontally either right or left to the rail weight (W), then vertically to the equivalent feet of rail.

Table 8.24.—Data for rail-bond cable

Rail-bond size	Resistance of 1 ft at 20°C, $10^{-6}\ \Omega$	Used with rail size, lb/yd
AWG:		
2/0	79.35	20– 40
4/0	49.97	20– 80
MCM:		
300	35.31	80–100
500	21.16	80–100

OVERHEAD LINES

The most common method used for electric power transmission and distribution is overhead conductors. Although their size and detailed construction can vary widely, overhead powerlines normally consist of bare metallic conductors supported by insulators from some elevated structure. The conductors use air space for insulation over most of their length, while their elevation protects them from contact with ground objects.

Overhead-line installations use numerous types of conductor arrangements and support structures in various combinations. Utility systems range from single wooden poles, carrying conductors at low voltages, to self-supporting steel towers bearing major transmission lines. Wooden polelines with or without crossarms, for example, may be part of a single-phase or three-phase distribution system with voltages of 2.3 to 35 kV. By contrast, steel towers often carry lines transmitting large amounts of power at 115 kV and up, connecting major load centers of a utility company grid (*12, 40*). Utility-owned lines are commonly classified by function, which is related to voltage. There are no utility-wide standards for voltage classification, but the system that is typically used differs from the classification used in the mining industry.

Overhead conductors are arranged in various configurations to reduce line-to-line contacts due to wind, ice loading, or sudden loss of ice load, and may include different combinations of power, neutral, and static conductors. Aluminum conductors with steel reinforcement (ACSR) are commonly used because of their strength and relatively low price, but special applications may call for other materials such as copper (*12, 40*).

The types of overhead-line installations used for mining applications are similar to those in utility distribution systems. Typical are polelines to supply equipment in surface mining and lines feeding surface facilities related to mining. These lines are normally installed on single wooden poles and may carry only two conductors, as in single-phase supplies, or have up to six conductors, including three power, one neutral, one ground-check (pilot), and one static. The polelines may be relatively permanent installations such as those feeding plants, shops, and other surface facilities, and long-term pit baselines or ring mains. Chapter 1 includes a discussion of the poleline application in strip and open pit mining operations. Sometimes, temporary poles are mounted in portable bases (such as concrete-filled tires) for ease of relocation, and these are commonly used in open pit mining operations to carry power into the pit. Conductors are again usually ACSR, but hard-drawn copper is used where blast damage is a problem (*31*).

If these lines are not installed properly, failures from conductor breakage, arcing between phases, and structure collapse can occur. Obviously, serious safety hazards and costly power outages can result; therefore, proper design and installation are important. In-depth treatment of overhead-line design is provided in such texts as Fink and Carroll (*12*) and Westinghouse (*40*), which give excellent detailed summaries of design, installation, and repair practices in overhead distribution.

This section is intended to be a brief introduction to overhead-line design combined with some details of the wooden pole structures that are the main type found in mines. Overhead lines, unfortunately, have been a major cause of electrocutions in the mining industry; thus, an extensive discussion of injury prevention is also included.

Overhead-Line Design

The design of surface overhead lines relies as much on a knowledge of structure and mechanics as it does on electricity. The design is concerned with obtaining the correct size and placement of the structures that support the power and grounding conductors and keep them from damage. Obviously, the vertical weight of several single conductors or one multiconductor cable must be supported. Additional vertical loading can be caused by ice accumulation. The height of the structures must be adequate to provide the required ground clearance considering the amount of line sag. At the same time, the structures must be planted firmly enough to counteract the force placed on them by the conductors on a steep slope.

Tension is applied in order to install the conductors with the correct amount of sag, and this results in stretch. The stretch causes creep or elongation in the conductors over a period of time, which must be accommodated in the design. Aluminum conductors are particularly susceptible to creep. Another factor that must be considered is the effect of weather. Temperature changes cause expansion and contraction, which affect the amount of deflection. Wind causes the conductors to vibrate vertically and imparts a horizontal force to the structure. Calculation of horizontal forces is particularly important at angle points in the line. Additional large horizontal forces exist when a conductor breaks, and this factor too must be incorporated into the design. The vertical spacing between conductors must be large enough so that arcing does not occur during high winds or when a large accumulation of ice falls off a line.

Extremely tedious calculations for catenary spans can result from attempting to take all of these factors into account. Fink and Carroll explain graphic solutions and describe Thomas Charts that assist in the computation process (*12*). The National Electrical Safety Code (NESC) (*2*) gives design information for ice loadings, temperature variations, and wind velocities for different areas in the United States.

As mentioned earlier, several different types of structures are used to support the conductors. Selection of a specific type depends on the terrain, accessibility requirements, right-of-way availability, distribution voltage, span length, number of circuits, conductor size, weather, life of installation, availability of material, and economics. The types commonly used are self-supporting and guyed-steel or aluminum towers, steel or aluminum poles, concrete poles, wooden H-frames, and wooden poles. Steel and aluminum structures are used for high-voltage distribution where long service life and long spans are necessary but are only used in some mine power systems within substations. Wooden poles are the most prevalent overhead-line support in the mining industry, so a few details of their design will be presented.

Wooden poles are usually constructed from fully treated pine or butt-treated cedar. They are classified by their circumferential dimension measured at a point 6 ft from the butt. Consequently, the nominal ultimate strength is the same for all lengths and species of the same class. Wooden poles, classed 1 through 7 with this system, have the capability of withstanding the ultimate loads shown in figure 8.29 (*12*). The correct setting depth for various lengths is also noted in the figure. The setting depth is important to prevent butt "kickout," since the pole is primarily a cantilever column.

GUYED POLE

f = Safe bending stress, psi

t = Taper, in/ft

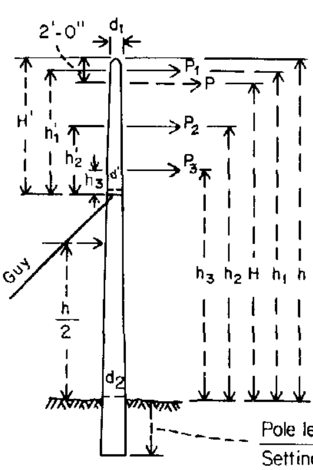
Moment of load = $P_1 h_1' + P_2 h_2' + P_3 h_3'$

Safe moment on pole = $\frac{1}{40} f (d')^3$

$d' = d_1 + tH'$

Wind on pole (W) = $\frac{13h(d_1+d_2)}{2}$

Variety	Ultimate fiber stress bending	Taper, inches circum per ft length	Average top diam
Western cedar	5,600	0.38	d_2-0.38h/3.14
Pine	7,400	.25	d_2- .25h/3.14
Northern cedar	3,600	.63	d_2- .63h/3.14
Chestnut	6,000	.40	d_2- .40h/3.14



UNGUYED POLE

P = Safe load 2 in from top

P_1 P_2 P_3 = Wind on wires, lb/ft

M_L = Moment of load

M_P = Safe moment on pole

$M_L = P_1 h_1 + P_2 h_2 + P_3 h_3$

$M_P = PH - W\frac{h}{2}$

$\frac{M_P}{M_L}$ = 1/2 max sum of adjacent spans

Class	Ultimate load 2 ft from top, lb
1	4,500
2	3,700
3	3,000
4	2,400
5	1,900
6	1,500
7	1,200

Pole length, ft	30	35	40	45	50	55	60	65
Setting depth, ft	5.5	6.0	6.0	6.5	7.0	7.0	7.5	8.0

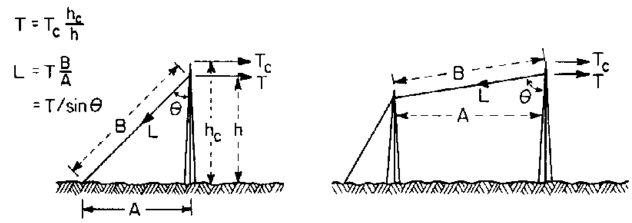
Figure 8.29.—Pole strength calculations.

Guy wires are used where necessary to assist in supporting the horizontal loads. Usually, 3/8-in galvanized steel wire tied to a log-anchored rod is used, as shown in figure 8.30 (*12*). The ratio of the guy load (L) to the conductor tension (T) is the same as the ratio of guy length (B) to the distance away from the pole base (A).

Both wooden and steel crossarms are used on wooden poles. Steel crossarms give better protection from lightning strokes, but they are more expensive than equivalent-strength wooden arms. Treated yellow pine or untreated Douglas fir are commonly used for crossarms. Their length ranges from 10 to 25 ft depending on the required conductor spacing. In size they range from 5 by 6 in to 6 by 10 in. Two planks, 3 by 8 in, can be mounted on either side of the poles to form a crossarm for heavy conductors.

Typical conductor arrangements and spacing for pin insulators are shown in figure 8.31. The dimension B is often determined by the span length and calculated according to the method given by Fink and Carroll (*12*). Because the sleet-jump failure experience, the dimension E should be at least 1 ft. Swing-type insulators require additional spacing to give a clearance for the swing. The clearance between the conductor and any grounded structure must be at least 0.75 times the dry-flashover distance of the insulator or the "tight-string" distance under an 8-lb wind at 60° F.

In addition to the conductors positioned at the insulators, overhead ground conductors or static wires are strung above the conductors to give lightning protection. Lightning protection is discussed in detail in chapter 11.



DESIGN DATA FOR GUYS

Guy	Ultimate strength, lb	Rod	Net area, in²	Ultimate strength, lb
3/8 in s.m.	6,950	5/8 in	0.202	11,000
3/8 in h.s.	10,800	3/4 in	.302	16,500
7/16 in s.m.	9,350	1 in	.551	30,500
7/16 in h.s.	14,500			

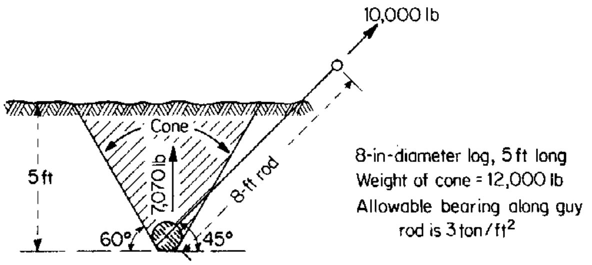


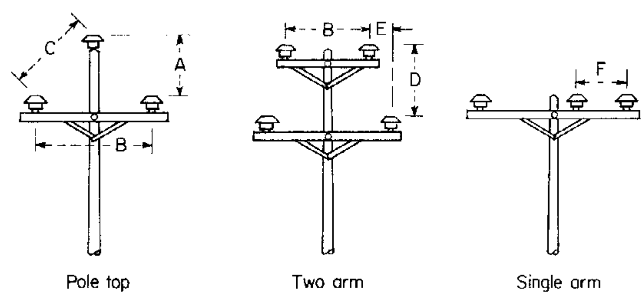
Figure 8.30.—Guy and log-anchor calculations.

Overhead-Line Electrocutions

Overhead lines, whether utility transmission and distribution lines or part of the minesite electrical system, present a serious electrical shock hazard to mining personnel. Overhead lines in and near mining operations can be exposed to many types of mobile equipment and even handheld tools. Metallic frames of such equipment, upon contact with energized overhead lines, can become elevated above earth potential, and simultaneous contact of the hot frame and ground by an individual can create a path through the body for lethal levels of line-to-ground fault current. Personnel are therefore exposed to a shock hazard through indirect contact with overhead powerlines. Although this mode of electrocution seems (at least outwardly) straightforward, it has been very difficult to find effective means of prevention.

Examination of mining industry statistics since 1970 reveals that one-third of surface coal mine electrical fatalities and approximately one-third of all electrical fatalities in metal-nonmetal operations are directly attributable to the indirect contact of overhead lines (*25*). The majority of these accidents involved mobile equipment, and the hazard can exist anywhere that high-reaching equipment operates near overhead lines (*1*).

Trucks commonly involved in overhead-line contacts are highway-legal end-dump tandems, triaxles, and tractor trailers. They can contact overhead lines, and their frames become subsequently energized, through their beds being raised or driven into lines. Victims normally bridge lethal potentials when stepping from the cab onto the ground or by operating external controls. Mobile cranes, which present a substantial line-contact hazard in other industries, find various uses around a mine site. They range from large, solid-boom, high construction cranes to smaller hydraulically powered units with retractable booms. Lines can be contacted by the boom or hoisting cable, and in both cases workers around the crane have the greatest shock-hazard exposure. Mobile drilling rigs are susceptible to overhead-line contact because of their masts, which can be raised or driven into the lines. Operators are the most likely victims, bridging potential gradients while operating drill controls.



Voltage	Spacing A	Spacing B	Spacing D	Spacing F
23 kV	3'-0"	5'-0"	2'-6"	3'-0"
34.5 kV	3'-4"	5'-0"	3'-0"	3'-4"
46 kV	4'-0"	7'-0"	5'-0"	5'-0"
69 kV	6'-0"	8'-0"	6'-0"	

Figure 8.31.—Typical arrangements and pin-insulator spacings on wooden poles.

In response to the problem, a detailed investigation has been made into these mining accidents, into prevention methods used by utility companies and other industries, and into various additional methods that might reduce electrocutions from indirect contact with overhead lines (*27*). From this effort, typical hazardous mining locations with overhead lines were identified and several recommendations were established to reduce the associated hazards.

The listing of areas and situations that pose the greatest overhead-line hazard is important since it shows the target areas for application of recommended solutions. These locations can be divided into two groups: mining surface facilities and active excavations in surface mines.

Loading and dumping facilities, including stockpiles, loading bins or hoppers, material transfer points, and adjacent areas, yards and roads, are hazardous overhead-line locations for truck operation. Some factors contributing to the risk are operator unfamiliarity with the dump, use of a temporary dump point, and fluctuations in the edge or height of a stockpile. Trucks and cranes can easily be exposed to line hazards near various mine plant areas such as mineral processing, storage, handling installations, refuse dumps, and settling ponds. Construction sites may or may not be near permanent mining facilities but often present hazards involving construction cranes and preexisting overhead lines.

Overhead lines traversing active surface mine workings present potentially dangerous situations. The fatalities that have occurred in these areas were from lines other than pit power distribution. Hazards exist primarily over mine benches as well as access and haulage roads. Although not responsible for electrocutions in the past, pit power distribution can create a hazard when overhead lines are used, such as for strip-mine base lines.

The recommendations to reduce these hazardous situations include isolating overhead lines from mobile equipment, modification of overhead lines, use of protective devices, and safe work practices, each of which will be discussed in the following paragraphs.

Overhead-Line Isolation

It is the responsibility of the power engineer in a surface mine to assess the overhead distribution system with regard to the movement of mobile equipment and to ensure that wherever possible overhead lines are isolated from travel routes. This may seem an obvious course of action, but previous accidents have shown that correctable hazardous situations are often allowed to exist at mining operations.

Where there is frequent dump-bed truck traffic, lines must be restricted from dump sites and approach or exit roads. A safety margin of at least 100 ft should be allowed outside normal truck routes. This would allow for limited truck movement beyond the route to account for mechanical problems, bed cleaning, backups and temporary dump sites. Roads leading away from dump locations should not be crossed by lines for at least 250 ft beyond the dump site, since beds may not be completely down as trucks leave the area. This distance would give additional time for the bed to lower or for the driver to recognize the condition.

Construction cranes that remain stationary while operating at a project site can be positioned so that line contact cannot occur at any position. Cranes that travel during operation will require barriers around hazardous areas. When a safe distance from overhead lines is being determined, contact by hoist cables and swinging loads should be considered.

One situation is which line isolation may not be feasible is where the lines supply power to a surface facility or a nearby installation. In order to eliminate bare overhead conductors in these situations, some alternate method must be used to supply power. One alternative for permanent installations is underground cable. Cables in conduit or directly buried are suitable for lines entering plants, dump facilities, shops, supply yards, and support buildings. Cables similar to those found in mine power distribution, such as MPF and SHD types, are used for buried applications.

Cables present a safe alternative to bare overhead conductors in areas where high-reaching equipment must travel. Underground service removes line exposure completely, but overhead cable with pole support may be preferable because of cost, ground conditions, or expected installation life. In either method, the cable should completely span the hazardous area, or its purpose is defeated. These cable runs should continue for a short distance beyond the hazard area to allow for equipment extensions protruding beyond area limits.

Overhead lines traversing active surface mine workings present a hazard to high-reaching equipment. Whether they are preexisting utility lines or part of mine power distribution, hazards can result for trucks and drills on benches or on haulage and access roads. The removal of these lines from the work area is the most direct solution. This may involve the permanent relocation of a utility line over a proposed open pit or a temporary rerouting of a line about a strip operation. Elimination of overhead lines in a pit power-distribution system would probably involve replacement of cable. Operations such as strip mines can and commonly do use all-cable distribution with good results, provided that proper cable-handling techniques and equipment are used (*27*). Open pit operations normally use overhead distribution to switchhouses in the pit and shielded trailing cables to mobile equipment. However, none of the fatal accidents examined were due to contact of these overhead distribution lines. In large open pit mines, overhead distribution is the most practical because of the long distances and cable protection requirements, but where frequent equipment operation poses a contact hazard, cable may be more desirable.

When rerouting lines around surface mine work areas, all aspects of the operation should be considered, including surface clearing, reclamation, access roads, and haulage roads, as well as actual mining activities. A safety margin should again be provided beyond normal work areas to account for occasional abnormal truck traffic, excavator booms, and similar situations.

Contact with overhead lines can also be avoided by removing the equipment operation from the hazardous area instead of moving the lines. Although this should be a very effective method, sometimes equipment movement is necessary: for instance, access cannot be restricted for cranes in supply yards or trucks in dump areas. However, where lines traverse active surface mine workings, equipment could be kept out of any contact-hazard area. Limiting access to lines can be the only economic alternative for a very small strip operation, which may be unable to sustain the cost of relocating even a small overhead distribution line. Nevertheless, any efforts to restrict mobile equipment must be carefully planned and implemented so as not to hamper normal operations or antagonize the work force.

It may be possible to restrict high-reaching equipment from some permanent surface facilities. Where this is possible, it provides an effective and less costly alternative to relocating overhead lines, so long as normal operations are not hindered. Restriction can be accomplished by posting the area, or using barriers such as steel crossbars, which allow only low vehicles (cars and small trucks) into the area. Provisions can easily be included to allow occasional entrance of higher equipment.

One option for the operation is to avoid the hazard by leaving the overhead-line right-of-way undisturbed. However, this option can result in a loss of resource as well as a disruption in the continuity of mining. The right-of-way may involve forfeiting only a single pass, as in a contour strip operation, but may seriously affect the mine layout if a large-area strip mine is traversed by a major transmission line. In order for a contour mining operation with an overhead powerline across the projected path to continue through the right-of-way but not mine below the lines, the towers or poles beyond the pit width limits would have to be guyed. The cables could then be removed or lowered into trenches, and all large equipment would be trammed or walked over the right-of-way. The lines would then be replaced, and mining operations would resume on the far side of the overhead lines.

Exploration drilling commonly requires operation in unfamiliar surroundings, often under minimal supervision. However, drill sites may usually be relocated to avoid overhead-line hazards.

Overhead-Line Modification

Solutions discussed prior to this point isolate overhead lines from mobile equipment to reduce the change of contact. There are modifications to existing overhead lines that can substantially reduce hazards without resorting to the extreme measures stated earlier. Such techniques are important because many cases will arise where an operator cannot eliminate overhead-line hazards nor limit access to them.

Overhead-line heights must never be less than the minimum mandated by Federal regulations (*38*). These heights are extracted from the NESC for driveways, haulageways, and railroads, and 15 ft is stipulated as the minimum height for any high-voltage power line (*2*). Table 8.25 lists the NESC standards that cover most overhead lines in mining, while table 8.26 provides the required minimum distances for higher voltages.

Some hazards can be reduced by raising some overhead lines above the NESC minimums. Where dump-bed truck traffic is a concern, lines over roadways could be raised to clear most dump-bed units without extensive support structure. A line height of 45 ft would place lines above most highway-legal dump-bed trucks, even with their beds fully raised, and would also clear most drills and cranes when they are in transit with their booms and masts lowered. If necessary, it is possible to raise lines to more than 65 ft using single wooden-pole supports. However, the line heights attainable depend upon line spans, cable sag, and surrounding terrain, but in most cases 45 ft is an achievable height.

Another line modification that lends itself to road crossings is the guarding of power conductors by effectively grounded conductors. If it can be ensured that any accidental contact with power conductors will be simultaneous contact with grounded conductors, a line-to-ground current will probably be provided. This reduces current flow through an equipment-ground contact and increases the chance of rapid fault clearing by circuit protective devices. However, several grounded conductors will be necessary to ensure simultaneous contact and may make this method impractical because of cost. Under these circumstances, rubber guarding may be used on overhead lines at hazardous crossings.

Utilities will often supply electricity to a mining facility substation by running a branch overhead line from their lines. If the branch line creates a contact hazard on or around the mining property, a disconnect switch should be provided external to the utility system and upstream from any contact-hazard area. Should the need arise to work in close proximity to these lines, power could be cut with no disturbance to other utility customers. Disconnects that are quickly accessible from mine work areas would also encourage deenergization prior to work about lines, but this depends upon ownership of the lines, availability of qualified personnel to cut power, and utility policy.

Protective Devices

Devices exist that attempt to reduce overhead-line hazards either by insulation from line potentials or warning of overhead-line proximity. Representative of the insulation method are insulated boom cages and insulating load hook links; proximity warning devices are intended to indicate the presence of energized conductors. Most devices are directed primarily toward protection of mobile cranes but do have other applications.

An insulated boom cage is an enclosure or guard mounted on and electrically isolated from the boom or mast to be protected. If the boom is moved into an energized overhead line, the insulated cage makes initial

Table 8.25.—Minimum vertical conductor clearances as specified by the NESC, applicable to mining and mining-related operations

Criteria	Nature of surface underneath wires, conductors, or cables	Open supply line conductors, ft	
		750 V to 15 kV	15 to 50 kV
Locations where wires, conductors or cables cross over	Track rails of railroads (except electrified railroads using overhead trolley conductors).	28	30
	Roads, streets, alleys, parking lots subject to truck traffic	20	22
	Other land traversed by vehicles, such as cultivated, grazing, forest, orchard, etc.	20	22
Locations where wires, conductors, or cables run along and within the limits of highways or other road right-of-ways but do not overhang the roadway.	Roads in rural districts	18	20

Table 8.26.—Minimum distances from overhead lines for equipment booms and masts (*38*)

Nominal powerline voltage, kV	*Minimum distance, ft*	*Nominal powerline voltage, kV*	*Minimum distance, ft*
69 to 114	12	345 to 499	25
115 to 229	15	500 and up	35
230 to 344	20		

contact and prevents the boom from becoming energized. The device only protects covered areas and cannot easily guard hoisting ropes, and its effectiveness also depends on the integrity and surface condition of the insulators used (*1*).

During crane hoisting operations, workers steadying or directing a load from the ground are in an extremely hazardous situation should an overhead line be contacted, as they are commonly in contact with both the ground and load. Insulating links can be used to isolate loads from the crane hoisting rope and are placed between the load hook and the hoist rope. The links are constructed of a dielectric such as resin-impregnated fiberglass.

A proximity warning device indicates by a visual or audible alarm the proximity of equipment extensions to energized overhead powerlines (*15*). Unlike cages and links, these devices attempt to prevent equipment-line contact, and the operation is theoretically independent of human judgment, at least so far as indication of powerline presence is concerned. Ideally, such devices alert an operator if the protected equipment extension enters a predetermined zone about a power conductor. Several types of proximity warning devices are available in the United States, all operating on the principle of electrostatic-field detection (*15*). The electrostatic field about a group of overhead conductors is primarily a function of their voltage and geometry. The units generally operate by monitoring 60-Hz electrostatic fields, amplifying, rectifying, and measuring the signal, and then activating an alarm at some preset signal level. The sensor used may be short and effectively a *point sensor*, which will create a spherical detection area, or a *distributed sensor* spanning the length of a protected extension. The type, number, and location of these sensors greatly affect the operation of a proximity warning device. Proximity warning devices operate as intended under many circumstances, but their reliability can be compromised by a complex array of factors. These limitations can be grouped into those arising from operational principles of electrostatic-field detection and those which are due to the design of individual devices.

The concept of a device to alert equipment operators to possible overhead-line contacts has great merit, but given the inconsistent operation of currently available devices, they should only be applied with full recognition of their limitations. Dangerous conditions can exist where workers place too much faith in a warning device or ignore it due to previous unreliable operation. Proximity warning devices are best applied only as a supplement to other overhead-line contact safety measures. Boom cages and insulating load links also have sound theories of operation but problems in implementation, and major drawbacks stem from flashover due to insulator surface conditions. The use of any warning or insulating technique does not relieve the operator from the responsibility of maintaining the minimum line-equipment clearances stated earlier.

Safe Work Practices

Any attempt to reduce overhead contact hazards at a mining operation must also involve the development of safety awareness within the work force. Training of personnel in safe operation of mobile equipment near overhead lines will complement any other safety method and, in some cases, may be the only effort necessary for preventing indirect-contact electrocutions. The following recommendations include guidelines for work near overhead lines, some passive-warning techniques, and safety training of personnel.

Before work is done near high-voltage overhead lines, the areas in question should be thoroughly examined by supervisory personnel and workers to determine the presence of any overhead-line hazards. All overhead lines should be considered energized unless an authorized representative of the line owner indicates otherwise. If the lines are utility owned, the utility should be contacted for assistance with planning safe operating procedures for the project. Equipment should be operated only by a competent, experienced, qualified operator, and the operations should be observed by a reliable worker, watching for maintenance of minimum clearances and unsafe conditions. This observation should be the worker's designated and only task. Another competent worker should be designated to direct the equipment operator, and only this worker should give directions. Standard signals should be agreed upon and used. Booms, masts, beds, and so forth should be in a lowered position when equipment is in transit, and minimum legal clearances should be maintained. If minimum clearance cannot be provided, the overhead lines in question should be deenergized and visibly grounded.

The following procedures should be followed if an energized overhead line is contacted. If contact was momentary and no lines are down, a calm and experienced crew member should be certain that the equipment is no longer in contact and should then assign members of the crew to check for injuries among the work party, to administer first aid if necessary, such as basic life support and cardiopulmonary resuscitation, and to send for an ambulance immediately, to notify supervisory personnel, to check for dangerous equipment damage, and to secure the area for possible accident investigation. If contact is made and maintained, a calm and experienced crew member should instruct personnel aboard the equipment to remain in place and not to contact the ground, then have the operator move equipment out of contact if possible. Crew members should be assigned to keep all other personnel clear of the area, including equipment, hoisted loads, and fallen lines, to notify appropriate mine supervisory personnel or utility to have lines deenergized, and to send for an ambulance if needed. The crew should not contact any victims still in contact with energized equipment. When victims can be rescued safely, the crew should administer first aid, move equipment to a safe position, check for damage, and secure the area for possible accident investigation.

Investigation of past fatalities shows clearly how essential it is for workers to be familiar with these procedures, and the importance of regular training in cardiopulmonary resuscitation (CPR).

Passive-warning techniques, including signs, stickers, posters, and line indicators, should be highly visible and in color to draw worker attention. They should be to the point and simple to understand. Signs in hazardous areas should be large enough to be easily read from approaching equipment and should warn operators well in advance of the danger.

Paragraphs 48.25 through 48.28, and 48.31, 30 CFR, mandate the initial training and periodic retraining of mine personnel with respect to the occupational hazards of mining. High-voltage overhead-line safety should be included in this training. New employees at surface operations are often laborers assisting on or about mobile equipment, and in their initial training they must be alerted to the danger presented by overhead lines. Hazards specific to the mining facility in question should be brought out in initial training and retraining, as well as in the hazard training required for workers assigned to new jobs, particularly new equipment operators. Frequent reviews of safe practices regarding overhead lines would be advisable for all operators of high-reaching equipment, regardless of the minimum legally required training. Particularly important is the review of safety guidelines with crews about to begin operations with exposure to overhead lines. Familiarizing supervisory personnel with safety guidelines and company policies is also essential if they are to be competent in directing the work force under hazardous conditions.

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CHAPTER 9.—PROTECTIVE EQUIPMENT AND RELAYING

Even the best designed electrical systems occasionally experience faults and overloads, or disturbances that cause abnormally high currents. These currents can exist in the ground system or in the phase conductors. Wherever the occurrence, the situation is likely to precipitate a hazard to either equipment or personnel.

Of the basic design criteria that underlie all mine power systems, three are of critical importance in protective equipment and relaying: adequate interrupting capacity, current-limiting capacity, and selective system operation. The first two provide protection to the system during a disturbance, while the third is designed to locate the problem, then minimize its effect. In chapter 7, current limiting and selective relaying were designated as two prime purposes of grounding. It was shown that ground-fault currents can be limited by inserting a resistance in series with the neutral conductor. However, not much has been presented about selective system operation, other than its need. Protective circuitry and protective relaying are the tools behind selective system operation and are the main topics of this chapter.

The protective circuitry associated with the power system consists of transducers, relays, and switching apparatus. Its role of safeguarding personnel and equipment can be effected manually or automatically. An instance of manual utilization would be removing power from a system portion for maintenance. An example of automatic operation would be a situation in which protective circuitry first senses then clears each hazardous current resulting from a disturbance. As might be expected, the process of clearing is disconnecting the affected circuit from the power source safely and as quickly as possible, with minimum interference to the system balance. In other words, protective circuitry must isolate a malfunction at a given location with minimum damage to circuits and equipment and minimum operation downtime. The function of protective circuitry to provide detection and isolation is termed *selective relaying.*

All the devices that comprise the protective circuitry in the mine power system thus play a vital role in safety. In fact, protective circuitry is probably the most important component of the power system and forms a major portion of all power equipment. For example, a switchhouse, which has the principal function of protection, is simply a complex of protective devices.

The basic concepts of overloads and faults are introduced in chapter 4. Although the removal of destructive overloads is important, the main concern is the clearing of faults, since their occurrence can be catastrophic.

Because of the preponderance of cables, cable shielding, and grounded equipment in mine power systems, line-to-neutral faults are the most common, and most of these are arcing with relatively short length and controlled distance. Ground-fault current is predominantly limited by neutral grounding resistors, whereas in other industrial applications, ground-fault currents are often limited by fault impedance.

Line-to-line and three-phase faults can also occur, as when a mobile machine severs a cable during a runover. Extremely large line currents can result, which can be limited in the mine system only by transformer and line-conductor impedances. System components, such as couplers, cables, transformers, bus bars, and disconnect switches, must be capable of withstanding the momentary mechanical and thermal stresses created by the flow of fault current through them. Interrupting devices, such as circuit breakers, must be able not only to withstand these momentary *fault-through* stresses, but to interrupt or terminate these anomalous currents.

The maximum magnitude of possible fault currents existing in line conductors must be known in order to select adequate ratings of protective equipment. Indeed, this knowledge is required to coordinate protective-circuitry operation for the entire complex. It may also be necessary to know the minimum sustained fault current that is available in the system in order to determine the sensitivity requirements of the current-responsive protective devices. These fault magnitudes, both maximum and minimum, are usually estimated by calculation, and the equipment is selected using the calculated results.

Because of the many hazards that can occur, the system must be capable of detecting overloads, short circuits (line faults), undervoltage, and ground faults, as well as any compromise in grounding-conductor continuity. With the use of resistance grounding in mine power systems, the protective relaying or sensing device associated with ground faults or zero-sequence currents is usually handled separately from that for line faults causing only anomalous positive-sequence or negative-sequence currents. In addition, the relaying for overloads may be separate from that for faults. Except for fuse applications, the sensing devices for each function will normally cause the activation, or tripping, of the same circuit-interrupting device no matter what the protection requirements are for an individual location. The sensing devices may be an integral part of the interrupting apparatus or be separated from it and connected only through control wiring.

This chapter builds upon the material covered in chapter 4, beginning with the main protection components, switching apparatus and sensing devices. Basic relay connections, relay terminology, and different kinds of protection follow. Finally, typical assemblies and combinations of protective circuitry are discussed. Essentially, this chapter sets the stage for chapter 10, where fault calculations, device sizing, and coordination are outlined.

SWITCHING APPARATUS

A switching apparatus is defined as a device for making (closing), breaking (opening), or changing connections (*6*).[1] There are three basic types of apparatus in this classification: switches, circuit breakers, and fuses.

All switching devices are given certain design ratings, which are a measure of the electrical stresses they can withstand (*6*). Obviously, the ratings must be correlated with the intended use or duty. A listing and definition of these ratings follows but is restricted to those terms having direct application to the development of the topic in this and the subsequent chapter. Further concepts will be added in the discussion of transients and overvoltages in chapter 11.

1. *Voltage.* The maximum nominal system voltage at a specified frequency (usually line-to-line for ac devices) on which the device may be installed.

[1] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

2. *Continuous current.* The maximum continuous current that the apparatus may carry.

3. *Short-circuit current.* Usually, the maximum current the device is capable of interrupting. This may be further qualified by an interrupting-current or interrupting-capacity rating.

4. *Close-and-latch or momentary current.* The maximum short-circuit current that the device can withstand during the few cycles after the fault occurs without experiencing severe mechanical damage.

The ratings of switching apparatus are based on the maximum possible values of fault currents. To help visualize the importance of these ratings, consider that a three-phase fault has occurred on a power system. Figure 9.1 illustrates the resulting line current versus time, created by the flow of energy from the source or sources to the fault (*7*). This asymmetrical waveform is made up of two components: dc and a symmetrical ac. At any instant after the fault occurs, the total fault current equals the sum of the two. The dc component decays to zero in a short time, with the total current gradually changing from asymmetrical to symmetrical.

Switching-apparatus ratings, as a measure of the stresses involved during faulting, are based on the symmetrical rms value. Asymmetry is accounted for by taking the basic symmetrical value and applying multiplying factors. These concepts are presented in detail in chapter 10. However, figure 9.1 does provide a useful visualization of rating magnitudes, and these will be discussed in following sections, along with each switching device.

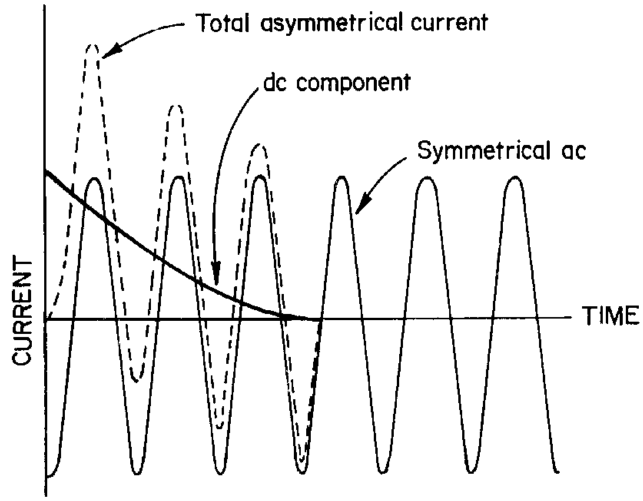


Figure 9.1.—Typical system fault current.

ARCS AND CIRCUIT INTERRUPTION

After a switching apparatus receives a message that circuit current is to be interrupted, the device proceeds through definite steps to terminate the current (*4*). These are illustrated in figure 9.2. Under normal operation, the contacts of the apparatus are closed, current flows through the interface, and the outgoing circuit is thus energized. To terminate current, the contacts begin to separate and an electric arc is drawn.

The arc is composed of free electron and free positive-ion flow, as shown in figure 9.3. To initiate this arc, free electrons and/or free positive ions must exist between the contacts. Their availability depends upon the following environmental conditions:

- In air or gas, the conductive elements are generated prior to the initiation of the arc by radiation and cosmic rays, which knock off electrons from neutral gas molecules.
- In a liquid such as oil, the conductive elements exist as impurities.
- In a vacuum, they can be emitted from the cathode by a high-strength electric field with the process known as high field emission.

The last case can add free electrons to any environment. Even though the voltage between the cathode and anode is low immediately after separation, the free electrons are attracted to the anode, and the positive ions toward the cathode. The electron flow accounts for about 90% of arc current (*4*).

If the voltage across the arc remains large enough, the movement of charge between the contacts initiates the mechanisms that can increase and sustain the arc. This

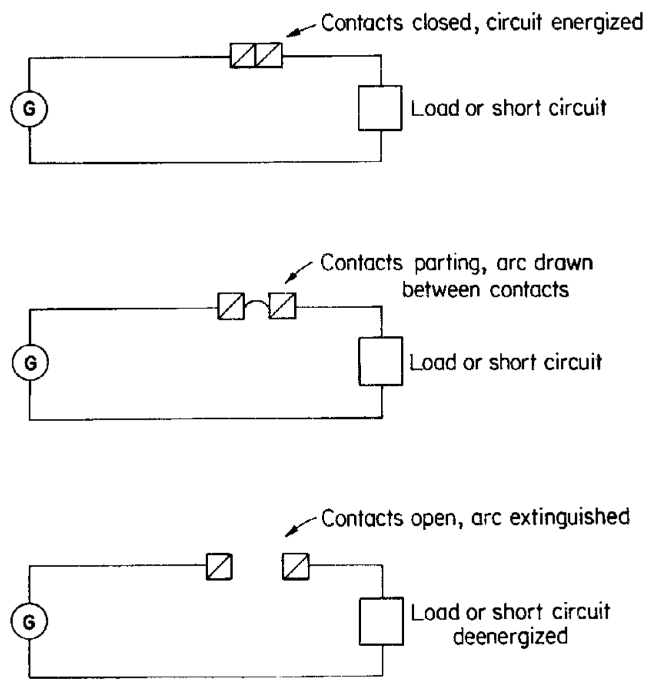


Figure 9.2.—Steps in circuit interruption.

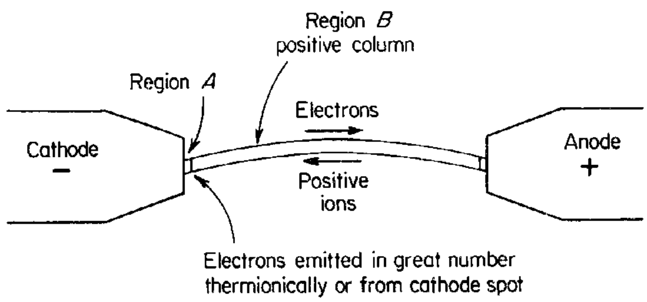


Figure 9.3.—Arc between two contacts.

again depends upon the environment. In a gas, the free electrons can collide with neutral gas molecules, producing additional free electrons and positive ions, termed *ionization by collision.* In any atmosphere, the collision of heavy positive ions on the cathode produces heat, which can augment field emission in low-melting-point materials such as copper, creating intense electron discharge from a small area, called a cathode spot, and can cause thermionic emission in high-melting-point substances, such as carbon, where electrons are boiled out by high temperature.

Once the arc is established, processes must be brought into play to extinguish it. In general, the greater the arc current and the higher the voltage of the circuit, the more difficult is the problem of arc extinction. The situation is easier in ac systems than in dc systems because the current waveform passes through zero in ac systems. However, the arc can restrike when the voltage rises again if the ionic conditions across the contacts permit. For dc, the arc is readily maintained because a normal currentzero does not exist.

Whatever the extinction process, the switching device can open the circuit successfully, provided that the current to be interrupted is within the rated value. However, if the switching device is required to terminate a current well above the design value, the arc between the parting contact may not extinguish or may continue to restrike, and the apparatus could be destroyed by the gas pressure built up within it (*5*).

When a device is designed to interrupt fault current, it is often called an interrupting device; otherwise, it is commonly called a *switch* and is designed only to open and close a circuit. While some switching apparatus are intended to serve only one of these functions, others can do both.

SWITCHES

A switch has exactly the same definition as switching apparatus, with the qualification that it is a manual device (*6*); in other words, its operation is a normal or intended occurrence on the power system. The switch types common in mine systems are the disconnect and the load interrupter. Both have the prime function of isolating outgoing circuits from the power source.

A disconnect switch is not intended to interrupt circuit current and can be operated only after the circuit power has been removed. Interlocks must be provided to prevent manual operation under load, and latches may be needed to prevent opening from the stresses resulting from fault circuits. Consequently, disconnect switches do not have an interrupting rating; but beyond a continuous-current rating, they may need a momentary-duty or close-and-latch rating for handling fault-through currents.

An interrupter or load-break switch differs from a disconnect in that it has an interrupting rating. The device has the capability of terminating currents that do not exceed the continuous-current rating, although this is not its normal operation. Interrupter switches usually have a quick-make, quick-break mechanism, which provides a fast switch-operation speed independent of the handle speed. The illustration in figure 9.4 shows a three-pole device; the mechanism on the right side of the connecting shaft provides the fast operation. Some units can be motor driven, thus allowing remote or automatic operation. In most mining applications, load-break switches need a close-and-latch rating. Where interlocks are not employed, this rating indicates the margin of safety when the switch is closed into a faulted circuit.

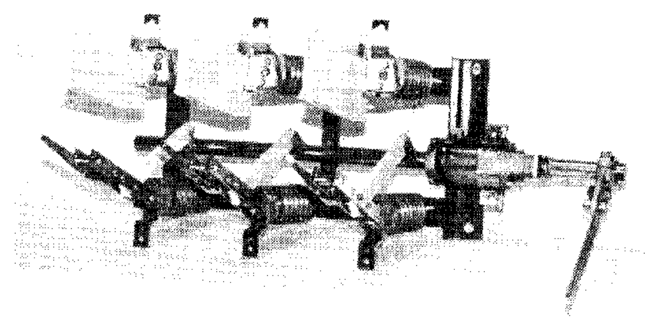


Figure 9.4.—Load-break switch. (Courtesy Line Power Manufacturing Corp.)

Switches are normally used as disconnects in mining systems regardless of their ratings; in fact, some States require load-break switches with interlocks for all disconnecting applications. These interlocks cause interruption of source power prior to contact separation, and the operation is usually performed through the ground-monitoring circuitry. Load-break switches, used in conjunction with fuses, are employed as interrupters in certain circumstances.

CIRCUIT BREAKERS

A circuit breaker is primarily an interrupting device, but in some cases it is also used as a switch (*6*). A circuit breaker can be defined as a device designed to open and close a circuit manually and to open the circuit automatically at a specific current level without injury to itself when properly applied. It is available as a single pole, double pole, or triple pole. Manual operation, be it mechanically or electrically actuated, is again intended where the circuit current is not in excess of rated continuous current. Automatic operation is dictated by a system abnormality, such as a fault or an overload. In this case, the device may be called upon to interrupt current in excess of the rated continuous current.

Circuit breakers in the role of interrupting devices must be used with sensing devices to perform their intended function. In medium-voltage and low-voltage mining applications, the operation may be internally controlled by self-contained current-responsive elements, external protective relays, or a combination of both. In high-voltage situations, the sensing devices are always separate, with interconnections only through control wiring.

Circuit breakers can generally be broken into two classifications: those intended for systems over 1,000 V, and those for 1,000 V and below. Devices in the first class are called power circuit breakers, while the second class is divided into power circuit breakers and molded-case circuit breakers. Following mining standards, circuit breakers for systems below 661 V are called low voltage; for 661 to 1,000 V, medium voltage; and above 1,000 V, high voltage. It should be noted that IEEE Standards define above 1,000 V to 72,500 V as medium voltage and below 1,000 V as low voltage. Low-voltage and medium-voltage circuit breakers are usually considered together and can find ac and dc service. High-voltage breakers involve only ac circuits. The next paragraphs look at typical apparatus and operation.

CIRCUIT BREAKERS FOR LOW AND MEDIUM VOLTAGE

The term *air circuit breaker* is often used when referring to molded-case and power circuit breakers designed for low-voltage and medium-voltage systems (*7*). Air circuit breakers employ the simplest method of interrupting current: extinguishing the arc in normal atmosphere by increasing its length (*4*). Several different processes can be used to force the arc to lengthen.

To illustrate one arc-lengthening technique, consider figure 9.5, where two circuit breaker contacts, a and b, have just separated. The horn-like arrangement of the contacts shown in the figure can be considered an arc chute, which is a barrier that confines, cools, and extinguishes the arc (*5*). By the ionization of the air between the contacts, an arc is drawn and heat is generated. The arc extinction action then commences; this is also called deionization because it serves to reduce the free electrons and positive ions in the gas (*4*). Air currents, created by the heat and confined by the arc chute, force the arc upward to form a loop. Electromagnetic forces within the loop further encourage the lengthening. As a result of cooling by radiation or convection, the longer arc requires a higher arc voltage to sustain current flow, and thus, the arc is extinguished.

As noted earlier, arc interruption in an ac circuit occurs much more easily than in a dc circuit. All voltages and currents in an ac system go through cyclic changes, and consequently, ion-producing effects for the arc are variable too: falling as current becomes smaller, ceasing at current zero (*4*). Deionizing effects in the arc chute remain steady. To take advantage of this situation, circuit breakers for ac systems are often designed around the minimum voltage required to establish a cathode spot. Because there is no natural current zero in dc systems, the circuit breaker must force the current to zero. For this to happen, the arc voltage must be greater than the system voltage (*14*). An enormous amount of heat can be generated in all circuit breakers while the arc exists, and an important function of the circuit breaker assembly is to dissipate this heat safely.

The foregoing simple arc-lengthening technique works well for 240-Vac applications. Conventional practice is to use a single-pole breaker for 120 Vac and a double-pole breaker for 240-Vac single-phase circuits. The latter employs one pole of the circuit breaker in series with each power conductor.

For circuits 250 V and above, the direct arc-lengthening approach is not enough; special arc chutes, quenchers, or deionizing chambers are needed to assist in arc termination (*5*). Figure 9.6 illustrates one approach, where the arc is forced into metallic barriers by magnetic attraction and broken into a series of smaller arcs. Each of these arcs is subjected to lengthening, cooling, and the problem of reestablishing a cathode spot if low-melting-point materials are used (*4*). Another approach is depicted in figure 9.7. Because the arc establishes its own electromagnetic field, an external magnetic field can enhance arc lengthening. The process is termed *magnetic blowout*, and breakers using this principle are called *air magnetic*. Coils carrying the circuit current in series with the arc can provide the

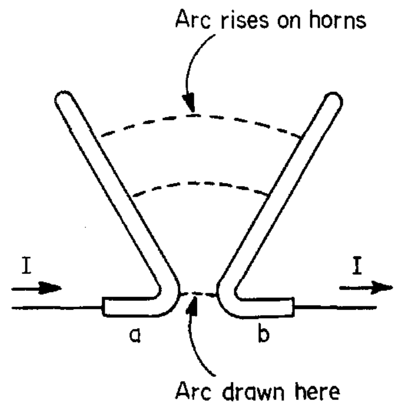


Figure 9.5.—Extinguishing arc by increasing the length.

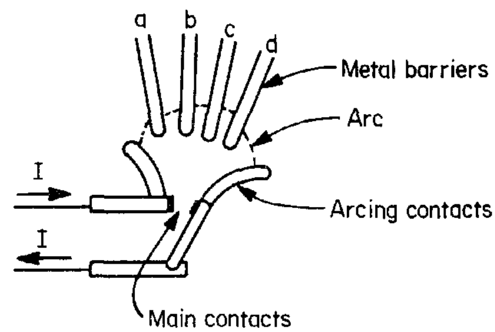


Figure 9.6.—Metal-barrier arc chute assists in arc deionization.

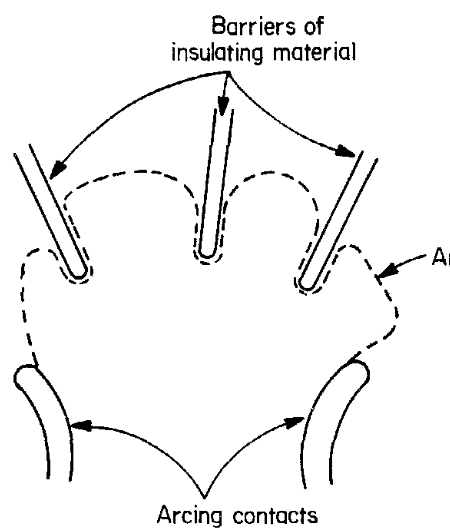


Figure 9.7.—Insulated-barrier arc chute used with magnetic field.

magnetic field. As shown in the figure, the magnetic field forces the arc into insulated barriers or fins, creating further lengthening; recombination and cooling at the barrier surfaces accelerates deionization (*4*).

In dc mine power circuits below 660 V, air-magnetic breakers are used extensively, especially on trolley systems. With very few exceptions, molded-case breakers are employed for ac circuits below 1,000 V. In addition, molded-case units are often used to protect low-voltage dc face equipment.

Molded-Case Circuit Breakers

The molded-case circuit breaker is the most explicit example of interrupting apparatus with self-contained current-responsive elements. It is defined as a breaker that is assembled as an integral unit in a supporting and enclosing housing of insulating material (*5*). Depending upon the amount of protection desired, these devices can sense internally and then clear undervoltage, overcurrent, and short-circuit conditions. Some tripping elements, that is, the actual components that cause the contacts to start separating, are also externally accessible through control wiring. Hence, other circuit protection can be added. Except for some power circuit breakers of low-voltage and medium-voltage design, all the circuit breakers that will be discussed in this chapter rely solely on outside information to perform their prime function. Molded-case apparatus will be presented first so that many important terms can be introduced.

The application of molded-case circuit breakers in mining began in the 1950's with the conversion from low-voltage dc power distribution to ac power distribution and face rectification, expanding further with the trend toward ac face equipment. In fact, Wood and Smith (*21*) have attributed the introduction of low-height, solid-state rectifier units in underground mines (which permitted the use of ac distribution) to molded-case circuit breakers, citing the lack of high-speed dc circuit breakers of the proper height as the previous limitation. Molded-case breakers placed between the transformer and the rectifying bridge lowered the height limitation to that of the transformer, allowing a unit design complementary with the mining environment.

The largest mining application is trailing-cable protection in underground face areas. The breakers are located in power centers and provide cable protection on each outgoing circuit, as required by 30 CFR 75.900, in addition to functioning as switching devices. The typical molded-case breakers, however, are not designed for repetitive switching. Mining use subjects them to many more operations than found in other industries, and regular or standard circuit breakers generally cannot hold up to the stress. Several manufacturers, recognizing this problem, have produced a special line of mine-duty molded-case breakers, which have stronger construction to withstand the punishment of mine use.

Except for external adjustments, molded-case devices sometimes do not allow field maintenance; many are sealed to prevent tampering. Although some manufacturers offer a complete line of replacement components, repairs other than an exchange of easily removable parts, such as arc chutes or trip units, should be made only by qualified repair facilities. This is critical, given the importance of the molded-case circuit breaker in personnel protection.

All component parts of these circuit breakers are built into one insulated housing, the molded case. These parts are the operating mechanism, arc extinguishers (arc chutes), contacts, trip elements, and terminal connectors, as shown in figure 9.8 (*19*). Additional accessories may be included.

The molded case is made of a glass polyester or similar synthetic material that combines ruggedness and high dielectric strength with a compact design. Each type and size of molded case is assigned a frame size or designation for easy identification. This coding, loosely based on an old Underwriters' Laboratories standard, refers to a number of breaker characteristics, including maximum allowable system voltage, maximum allowable continuous current, interrupting capacity, and the physical dimensions of the molded case. Several trip units may be available for a particular frame size, so a specific assembled breaker may have a lower continuous-current rating than the current designation of the frame. Table 9.1 lists the continuous ratings considered to be standard for mining service. The currents in parentheses are the lower current settings available in that frame size from certain manufacturers. Unfortunately, manufacturers have varying design criteria and hence size their units to dissimilar

Table 9.1.—Ratings for mining-service molded-case circuit breakers

Frame size,[1] *A*	*Continuous-current ratings,*[2] *A*
100	100 (70, 50, 30)
225	225 (175, 150, 125, 100)
400	400 (300, 225, 175, 150, 125, 100)
600	600 (500, 400)
800	800 (600)
1,200	1,200 (1,000, 800, 600)

[1] Regular-duty breakers also available in 1,600-, 2,000-, and 2,500-A frames.
[2] Currents in parentheses are lower settings available in the frame size.

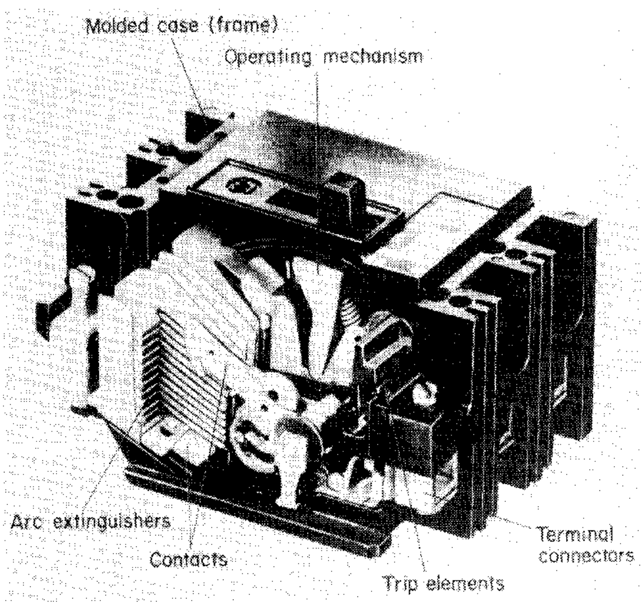


Figure 9.8.—Molded-case circuit breaker components. (Courtesy Westinghouse Electric Corp.)

specifications. For example, a 225-A, 600-V breaker supplied from two separate manufacturers may have different physical dimensions so that direct interchanging is difficult, if not impossible.

The circuit breakers rated in table 9.1 are generally available as two-pole or three-pole units at 600 Vac or 300 Vdc, but only as three-pole devices at 1,000 Vac. The two-pole breakers are intended for dc face equipment or single phase ac applications. By convention, one pole is used for each ungrounded conductor in a circuit (*5*).

The arc chutes define the interrupting-current capacity of the assembly in conjunction with the insulating and heat-dissipation properties of the molded case. The chutes assist arc deionization by the principle discussed for figure 9.6. They are also termed *arc extinguishers* or *arc quenchers* by some manufacturers. The breaker case must be mounted vertically with the arc chutes at the top for correct arc-extinction operation.

Circuit breakers designed for 1,000 V and below are capable of clearing a fault faster than those constructed for high voltage (*6*). The contacts often begin to part during the first cycle of fault current, and consequently, the breaker must be capable of interrupting the maximum allowable first-cycle asymmetrical current. Thus, for lower voltage breakers, the close-and-latch and interrupting ratings are usually the same, a characteristic not found with high-voltage breakers. The rating of these units is carried out on a symmetrical basis, so multipliers accounting for the dc offset need not be applied as long as the system X/R ratio does not exceed 6.6 (*6*) (see chapter 10). Table 9.2 lists typical interrupting ratings versus the system voltages for mine-duty circuit breakers; the ac system values are based on the symmetrical rating. Some manufacturers offer both standard-duty and high-interrupting-capacity breakers for mining service. The table values presented parenthetically indicate the superior construction, which incorporates sturdier contacts and mechanism plus a special high-impact molded casing.

Table 9.2 shows that typical molded-case circuit breakers constructed for 1,000-Vac mine systems have only a 10,000-A symmetrical interrupting rating. This presents a concern, as available short-circuit currents on high-power 1,000-Vac systems can be greater. Instances include longwall mining equipment, which needs a power-center capacity of 1,500 kVA or more. To overcome the problem, a manufacturer has introduced molded-case breakers with a 24,000-A asymmetrical interrupting rating at 1,000 Vac and continuous-current ratings of 600, 800, 1,000, or 1,200 A. The asymmetrical rating is used to provide more flexibility for designing the breaker into power systems.

The function of the operating mechanism of a typical molded-case circuit breaker is to provide a means of opening and closing. It is a toggle mechanism of the quick-make, quick-break type, meaning that the contacts snap open or closed independent of the speed of handle movement. The breaker is also trip-free; that is, it cannot be prevented from tripping by holding the breaker handle in the *ON* position during a fault condition. In addition to indicating whether the breaker is *ON* or *OFF*, the operating-mechanism handle indicates when the breaker is tripped by moving midway between these positions. To reactivate the tripped breaker, the handle must first be moved from the central position to *OFF*, which resets the mechanism, and then to *ON*. This distinct trip point is particularly advantageous where molded-case breakers are grouped, as in a power center, because it clearly indicates any faulty circuits.

The function of the trip elements is to trip the operating mechanism in the event of prolonged overload or short-circuit current. Two common types of trip elements are used in mining, magnetic and thermal magnetic. When the circuit being protected involves portable or trailing cables, the thermal-magnetic combination is strongly recommended and is mandated by some States.

The magnetic trip protects against short circuits, and an electromagnet in series with the load current provides the trip action (*19*). This type of short circuit is actually a line-to-line or three-phase fault on ac, or a line-to-line fault on dc systems. When a short occurs, the high fault current causes the electromagnet in the breaker to attract the armature, initiating an unlatching action, which in turn causes the circuit to open (fig. 9.9). The action takes place within 1/2 s (usually within 1 cycle or 16 ms), instantaneously tripping the breaker. Since tripping takes place with no intentional delay, the magnetic trip is often called the instantaneous-trip element. Screwdriver slots, located on the front of the trip unit, are used in adjusting the sensitivity (fig. 9.10*A*). By law, the maximum setting is established by the protection of the minimum conductor size in the circuit (*16–17*). Table 9.3 lists these maximum settings applied to trailing cables. Figure 9.10*B* illustrates a family of time-current curves resulting from the adjustable range; to the left or below each curve, the breaker will not be tripped magnetically. Typical instantaneous-trip ranges versus frame sizes for mining-service breakers are given in table 9.4. Note that this is not a rigorous listing, since some manufacturers will provide any desired trip range with most frame sizes upon request.

The other common molded-case breaker type is the thermal-magnetic variety. In addition to providing short-circuit protection, the thermal-magnetic breaker also guards against long-term current overloads existing longer than roughly 10 s, by incorporating thermal trip elements (fig. 9.11). The thermal action is accomplished through use of a bimetal strip heated by load current (*19*). The strip consists of two pieces of metal bonded together, each with a different coefficient of thermal expansion. A sustained overload causes excessive heating of the strip, resulting in deflection of the bimetal, which in turn causes the operating mechanism to trip the breaker. Because the

Table 9.2.—Interrupting-current ratings[1] versus system voltage, amperes

Frame size, A	240 Vac	480 Vac	600 Vac	1,000 Vac	300 Vdc[2]
100	18,000 (65,000)	14,000 (25,000)	14,000 (18,000)	10,000	10,000 (20,000)
225	25,000 (65,000)	22,000 (35,000)	22,000 (25,000)	10,000	10,000 (20,000)
400	42,000 (65,000)	30,000 (35,000)	22,000 (25,000)	10,000	10,000 (20,000)
600	42,000 (65,000)	30,000 (35,000)	22,000 (25,000)	10,000	10,000 (20,000)
800	42,000 (65,000)	30,000 (35,000)	30,000 (25,000)	10,000	10,000 (20,000)
1,200	42,000 (65,000)	30,000 (35,000)	30,000 (25,000)	10,000	10,000 (20,000)

[1]Parenthetical ratings are for typical premium-duty circuit breakers.
[2]Actual dc interrupting current dependent upon system inductance.

Table 9.3.—Maximum instantaneous-trip settings

Conductor size	*Maximum allowable instantaneous setting, A*	*Conductor size*	*Maximum allowable instantaneous setting, A*
AWG:		AWG:	
14	50	1	1,000
12	75	1/0	1,250
10	150	2/0	1,500
8	200	3/0	2,000
6	300	4/0	2,500
4	500	MCM:	
3	600	250 to 500	2,500
2	800		

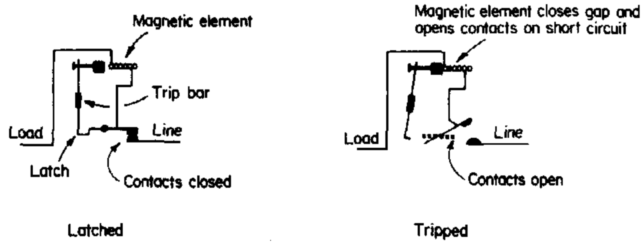


Figure 9.9.—Magnetic-trip relay.

Table 9.4.—Commonly available magnetic-trip ranges for mining-service molded-case breakers

Frame size, A	Magnetic-trip range, A	Range of allowable conductor sizes
100	50– 180	14–10 AWG
	150– 500	10–4 AWG
225	300– 700	6–3 AWG
	500–1,000	4–1 AWG
	300–1,000	6–1 AWG
400	500–1,000	4–1 AWG
	800–1,600	2–2/0 AWG
	500–1,500	4–2/0 AWG
600	900–3,000	1 AWG - 500 MCM
800	750–1,500	2–2/0 AWG
	1,000–2,000	1–3/0 AWG
	1,500–3,000	2/0 AWG - 500 MCM
	2,000–4,000	3/0 AWG - 500 MCM
1,200	1,500–3,000	2/0 AWG - 500 MCM
	2,000–4,000	3/0 AWG - 500 MCM
	2,500–5,000	4/0 AWG - 500 MCM

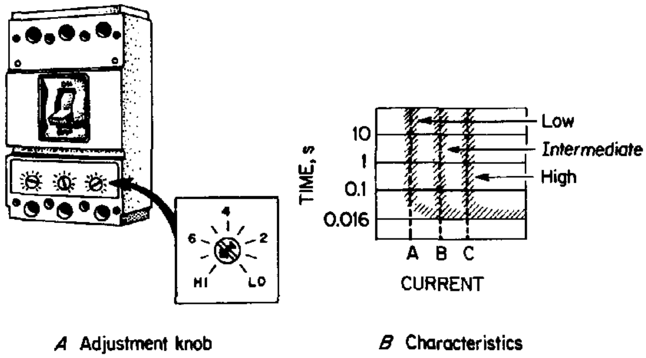


Figure 9.10.—Adjustable instantaneous setting.

bimetal deflection is dependent upon current and time, the thermal unit provides a long-time delay for light overloads and a fast response for heavy overloads. A representative current-time curve for the thermal unit alone is shown in figure 9.12*A*; later in this chapter, it will be described as an inverse-time characteristic. In comparison, figure 9.12*B* shows the circuit breaker response when both thermal and magnetic trip elements are incorporated. The shaded area for each curve represents a tolerance between the minimum and maximum total clearing time.

The thermal-magnetic unit shown in figure 9.11 is ambient-temperature sensitive. Assuming the circuit breaker, cable, and equipment being protected are in the same ambient temperature, the circuit breaker trips at a lower current as the ambient temperature rises in correspondence to safe cable and equipment loadings, which vary inversely with ambient temperature (*19*). Thermal-magnetic trip elements are available that automatically compensate for ambient-temperature variations. The ambient compensation is obtained through an additional bimetal strip, which counteracts the overload bimetal. Such trip units are recommended whenever the protected conductors and the circuit breakers are in different ambient temperatures (*19*).

Most mining-service molded-case breakers with 225-A frame sizes and above have interchangeable trip units. For straight magnetic elements these allow different instantaneous-trip ranges per frame size. However, thermal-magnetic units can be used to establish a lower continuous-current limit for the breaker. The National Electrical Code (*13*) is used as a guide to define the current

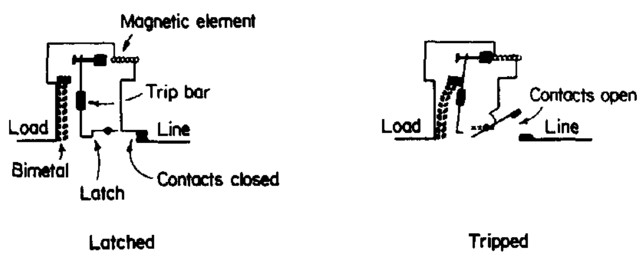


Figure 9.11.—Thermal-magnetic action of molded-case circuit breaker.

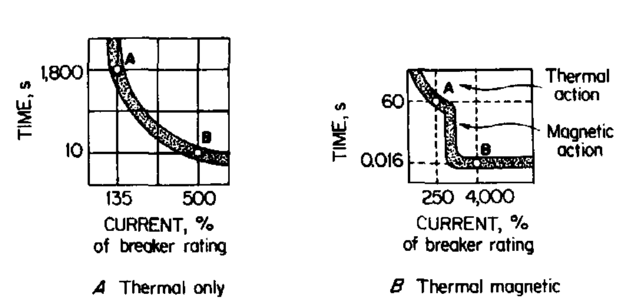


Figure 9.12.—Time-current characteristics for thermal-magnetic molded-case circuit breakers.

at which the long-time-delay thermal element must initiate the circuit-clearing operation and specifies a point that is 125% of the rated equipment or conductor ampacity. As seen in figure 9.12*A*, the circuit breaker will take no action below this current. Hence, the thermal portion defines the continuous-current rating of the breaker, specified as 100% at 40° C for conventional (non-compensating) thermal-magnetic elements. Obviously, the thermal element current rating cannot exceed the frame rating. Because of the connection, some manufacturers recommend that the continuous current through the breaker be limited to 80% of the frame size. This topic will be continued in chapter 10.

Electromechanical magnetic and thermal-magnetic trip elements have been replaced by solid-state components in some molded-case breakers. Although the solid-state counterparts may become popular in the future, they have not yet achieved wide acceptance in the mining industry. Nevertheless, these breakers are discussed in chapter 14.

The last basic breaker components are the terminal connectors. Their function is to connect the circuit breaker to a desired power source and load. They are usually made of copper and must be constructed so that each conductor can be tightened without removing another. The terminal connectors shown in figure 9.8 are for direct connection of one cable connector per terminal. Many molded-case breakers also have provisions for threaded-stud terminals. These studs can be used not only for connection of more than one conductor per terminal, but also for breaker mounting. It should be noted that the type of terminal used on a breaker may change its heat dissipation properties and thus lower its interrupting rating.

In addition to the basic components, several accessories are available, of which the most common are the terminal shield, shunt trip, and undervoltage release (UVR). Terminal shields protect personnel from accidental contact with energized terminal connections and are simply plates that shield (guard) the terminals. The other two accessories are used to trip the operating mechanism.

A shunt trip is employed to trip a circuit breaker electrically from a remote location. It consists of a momentary-rated solenoid tripping device mounted inside the molded case that activates when control power is applied across the solenoid coil. The magnetic field created by the solenoid moves a plunger, which in turn activates a trip bar. At the same time, a series cutoff switch removes power to the solenoid coil, preventing it from burning up under continuous load. A typical shunt-trip assembly is shown in figure 9.13. The shunt trip can remotely trip the breaker but cannot remotely operate it. To reclose the breaker, the handle must first be moved to the reset position, then to the *ON* position.

The purpose of the UVR is to trip the breaker whenever control voltage to the UVR falls below a predetermined level, usually 35% to 70%. This device is also mounted inside the breaker frame and consists of a spring and a solenoid. The spring is cocked or precharged by the operating mechanism when the breaker is closed and is held in the cocked position by the solenoid after closure. If the voltage drops below the required level, the solenoid releases the spring, causing the circuit breaker to trip. The breaker cannot be turned on again until the voltage returns to 80% of normal.

The importance of the shunt trip and UVR is far ranging, as they allow the protection capabilities of circuit breakers to be extended. The molded-case breaker alone can provide overload and short-circuit protection in an outgoing circuit. The UVR adds undervoltage protection; in fact, undervoltage protection is normally required at most breaker locations. Note that undervoltage protection is required for all equipment, but it is not required on all circuit breakers as long as all equipment downstream from the breaker has undervoltage protection. The undervoltage protection provided by a UVR is actually "loss-of-voltage" protection since the dropout level is well outside the recommended operating range of most motors (see chapter 6). Through a specific combination of relays and sensing devices, additional types of protection can be applied through shunt or UVR tripping. With a shunt trip, the relay completes the circuit between the control-power source and the solenoid coil. When a UVR is used, the relay removes the control voltage across the solenoid coil. This circuitry will be discussed in detail later in the chapter.

The molded-case circuit breaker is the most widely used breaker in mining, even though its employment is restricted to low-voltage and medium-voltage systems. The principal application is on ac, where it provides high

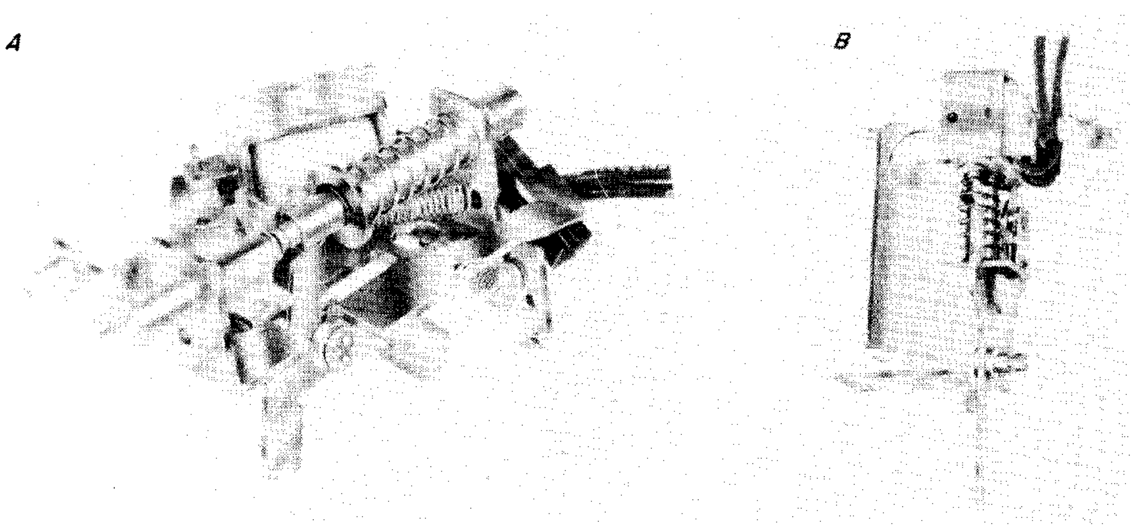


Figure 9.13.—Shunt-trip (*A*) and undervoltage-release (*B*) accessories. (Courtesy General Electric Co.)

interrupting capacity for short circuits in minimum space. On ac or dc systems, it is often the first protection device called upon to handle electrical problems existing on trailing cables and mining machinery. A clear understanding of the construction and rating of these breakers is required to assure adequate protection. The operating characteristics must be closely matched with those of the trailing cable to minimize hazards to personnel.

Power Circuit Breakers

Some mining-industry engineers have found that molded-case circuit breakers cannot handle the available short-circuit currents in certain low-voltage applications, such as the outgoing dc circuits of trolley rectifiers and dc face equipment. The low-voltage power circuit breaker provides an alternative in these cases.

Power circuit breakers for applications of 1,000 V and below are of open construction assembly with metal frames. They are designed to be field maintained under planned periodic inspection, and all parts are accessible for ease of maintenance, repair, and replacement (*6–7*). The design enables higher endurance ratings and greater repetitive-duty capabilities than are available from molded-case devices. However, power circuit breakers are intended only for service inside enclosures with "dead-front" construction, that is, not accessible to unauthorized personnel.

Electromechanical units are available for long-time tripping, but mechanical-displacement dashpot types are normally used for this function and provide the same overcurrent protection as does the bimetal thermal tripping in molded-case breakers. Although long-time characteristics are not adjustable with bimetal strips, the dashpots allow the long-time-delay "pickup" current and operation time to be changed. This extends the capabilities of the power circuit breaker over the molded case by providing not only short-circuit but also overload tripping adjustments, thereby allowing a broader range of applications (*7*). Low-voltage power circuit breakers are available with or without direct-acting instantaneous units and with or without long-time-delay units. Furthermore, most manufacturers offer three different separately adjustable long-time-delay operation bands as well as three different short-time-delay operation bands. As with molded-case breakers, power breakers are available with either shunt-tripping or UVR units or both. Solid-state devices are also manufactured for all tripping arrangements.

Some typical ratings for low-voltage power circuit breakers are provided in table 9.5 (*7*). In addition to these

Table 9.5.—Some typical ratings for low-voltage power circuit breakers

Ac system nominal voltage, V	Rated maximum voltage, V	Frame size, A	3-phase short-circuit current rating, symmetrical, A	Range of trip-device current ratings, A
600	635	225	14,000	40– 225
		600	20,000	40– 600
		1,600	42,000	200–1,600
		2,000	42,000	200–2,000
		3,000	65,000	2,000–3,000
		4,000	85,000	4,000
480	508	225	22,000	40– 225
		600	30,000	100– 600
		1,600	50,000	400–1,600
		2,000	50,000	400–2,000
		3,000	65,000	2,000–3,000
		4,000	85,000	4,000

listed values, frame sizes are available up to 6,000-A continuous ac current and 12,000-A continuous dc current (*5*). These frame sizes are rated to carry 100% of the continuous-current rating inside enclosures at 40°C. In power breakers with low current ratings, arc interruption can utilize arc-chute arrangements similar to those used in molded-case breakers. The full air-magnetic arrangements described for figure 9.7 are employed for high-current-interruption power breakers.

HIGH-VOLTAGE CIRCUIT BREAKERS

The power circuit breakers used in high-voltage mining applications include air-magnetic, oil, minimum-oil, and vacuum types. Vacuum circuit breakers or VCB's are by far the most popular because of their small size and high efficiency. Oil circuit breakers or OCB's once were the most common, but their use has dropped substantially in recent years, since the interrupting sizes needed for mining are not available. Air-magnetic types are normally limited to surface installations. The next few paragraphs will examine typical apparatus ratings, and then the operation of oil, minimum-oil, and vacuum types will be described; air-magnetic breakers are excluded as their operation is the same as that presented previously for lower voltage breakers.

Typical Ratings

The typical nominal voltage ratings corresponding to nominal system voltages are 4,160, 7,200, and 13,800 V, with 23,000 V used in some strip mines. The system portions of interest are obviously ac. Common continuous-current ratings are 400, 600, 800, 1,200, and 2,000 A. The majority of mine systems do not call for current greater than 600-A continuous, which has become the most used rating.

Interrupting and close-and-latch ratings are very important high-voltage parameters (*6*). For low-voltage and medium-voltage circuit breakers, the two ratings are usually the same. As high-voltage circuit breakers rarely terminate current flow until a few cycles after the first-cycle peak, the close-and-latch rating must be higher than the interrupting rating. A typical interrupting rating for high-voltage circuit breakers found in mining is 12,000-A rms symmetrical, while the typical close-and-latch rating is 20,000-A rms asymmetrical. The asymmetrical close-and-latch rating is often found by multiplying the symmetrical interrupting rating by 1.6 (see chapter 10) (*6*).

High-voltage circuit breakers can also be given an interrupting-capacity class, which is an identifying grouping rather than a rating. It is expressed in megavoltamperes, such as 250, 350, 500, and 750 MVA. The interrupting capacity is related to the interrupting-current rating by (*5*)

$$\mathrm{MVA} = \sqrt{3}\,\mathrm{kV}_{\mathrm{rated}}\mathrm{kA}_{\mathrm{rated}} \tag{9.1}$$

where MVA = interrupting capacity, MVA,
$\mathrm{kV}_{\mathrm{rated}}$ = rated system voltage, kV,
and $\mathrm{kA}_{\mathrm{rated}}$ = rated rms interrupting current, kA.

Oil Circuit Breakers

Even though their popularity has been dropping, OCB's are still used extensively in surface installations, especially substations. The common type of construction is the dead tank, shown in figure 9.14*A*. This steel tank is partly filled with oil and has a cover with porcelain or other composition bushings or insulators through which the conductors are carried (*4–5*). The breaker contacts are located below the bushings and are bridged by a conducting crosshead supported by a lift rod. In most designs, two contacts and the crosshead provide two interruptions per pole. The majority of OCB's in mining have three such poles in one tank. The tank has an insulated liner to prevent the arc from striking the tank walls. The entire assembly is oiltight; a vent with oil-separating properties permits the escape of any gases generated but prevents the escape of entrained oil.

Arc interruption in high-voltage circuit breakers employs the cathode-spot phenomenon combined with arc lengthening and deionization of the arc path. In the case of the OCB, oil is vaporized as an arc is established between the parting contacts, and this produces a bubble around the arc. The gases within the bubble are generally not conducive to ionization, but in most modern OCB's, an oil-filled insulating chamber surrounds the parting contacts (fig. 9.14*B*). When the moving contact is lowered, the gas generated by the arc portion within the chamber forces oil out through the chamber throat (*4*). The blast of oil comes into intimate contact with the arc, accelerates the cooling and ion recombination process (fig. 9.14*C*), and carries away available ions. A different arc-chamber approach is shown in figure 9.15. Here the chamber throat is made of laminations so that during interruption, the oil can move radially into the arc path. This is sometimes termed a *turbo* action. In high-interrupting capacities, the gases developed within the chamber can be used to blast oil horizontally across the arc path. Whatever the specific design, the chambers are intended to contain the developed high gas pressures and reduce any pressure on the main oil tank (*5*). After being effectively cooled, the generated gases are allowed to pass through the vent into open air.

The result of OCB construction and operation is a very effective arc interrupter. However, beyond availability, there are inherent disadvantages that discourage use of OCB's (*4–5*). The oil presents a fire hazard, particularly if the tank is ruptured because of unexpected pressure; this has led some States to prohibit OCB application in underground coal systems above 10,000 V. The oil is bothersome to handle and creates maintenance problems including cleanliness problems. Finally, the inertia of the heavy operating mechanism severely limits operational speed, causing a time delay in opening the arc. Despite these problems, other advantages, which are discussed in chapter 11, still make the OCB desirable to many industry engineers.

When used underground, the physical size of three-pole units usually limits the interrupting capacity to 100 MVA or less, with continuous-current ratings of 400 A. The operating mechanism on these small OCB's is typically spring-gravity and manual-reset; a handle-driven mechanism (quick break, quick make) is used to close the breaker manually while at the same time automatically tensioning an opening spring. With the breaker engaged, the spring becomes armed, allowing a shunt-trip or UVR device to trigger the breaker opening by releasing the spring. A motor-driven system is also available to close the breaker, but the tripping method is the same. The motor-driven OCB's can thus be electrically engaged as well as tripped. Larger OCB's such as those used in substations are typically motor driven.

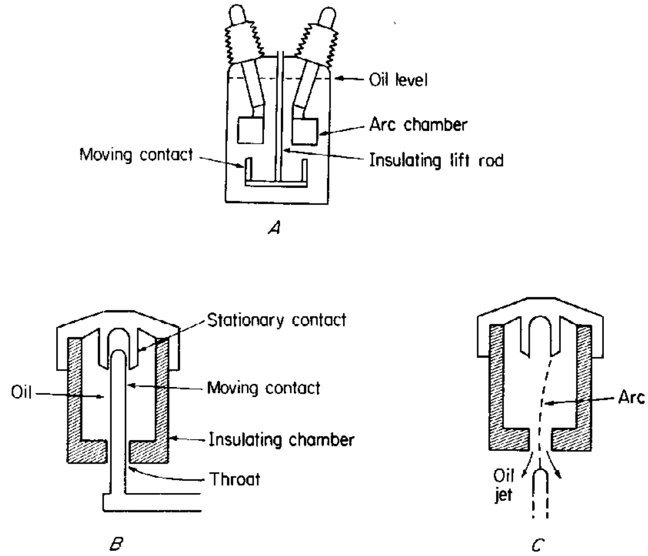


Figure 9.14.—Construction and operation of dead-tank OCB.

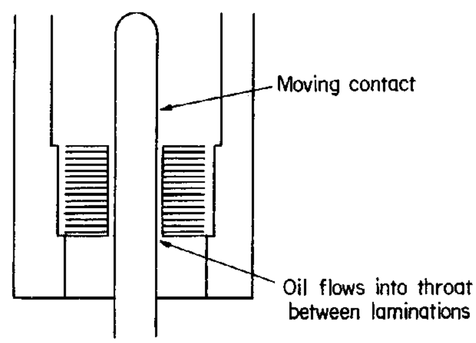


Figure 9.15.—Turboaction arc chamber for OCB's.

Minimum-Oil Circuit Breakers

Minimum-oil circuit breakers, also termed *low-volume oil* or *live tank*, enclose each pole in its own small-diameter tank (*5*). In modern versions, the tank is made of insulated high-strength, high-resistance material, and the top and bottom covers are high-dielectric-strength insulators (fig. 9.16). Contacts consist of a movable vertical rod and a stationary contact in the tank bottom. Oil volume is about 1 L, and the top surface of the oil is at atmospheric pressure. Arc extinguishing is assisted by oil blast, and resulting gases are vented to outside air. The operating mechanism can be either manual-reset and spring-trip or motor-reset and spring-trip. Some typical ratings of these breakers are listed in table 9.6.

The arrangement of a three-pole minimum-oil unit with moving contacts mechanically interconnected results in a smaller overall package than comparable dead-tank breakers. The smaller mass of moving parts (operating

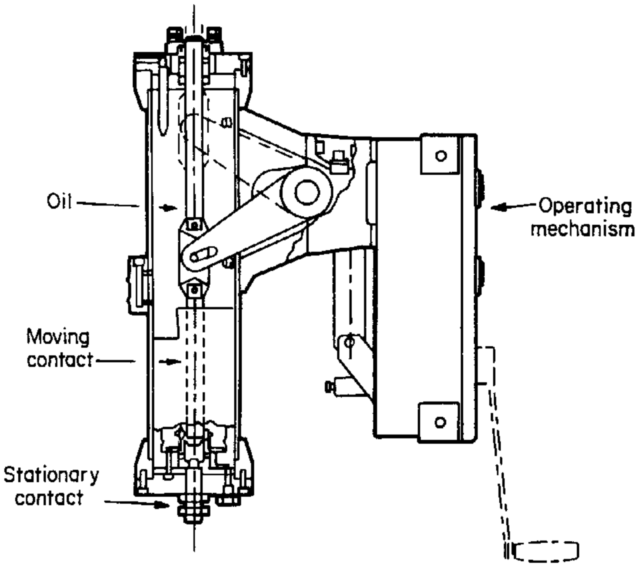


Figure 9.16.—Cross section of minimum-oil breaker.

Table 9.6.—Typical minimum-oil circuit breaker ratings

Rated voltage, V	Interrupting capacity, MVA	Continuous current, A
5,000	173	1,000
15,000	478	1,000
25,000	680	630

mechanism and rods) enables higher operating speeds, while the advantages of oil interruption are maintained. However, the low volume of oil is such that after about five operations, the oil level must be checked. Even though oil-level indicators are available, this can create a maintenance problem in mining.

Vacuum Circuit Breakers

With all the circuit breaker types covered so far, a gaseous atmosphere exists between the parting contacts. The gas is ionized by many processes and thus provides free electrons, which move to the anode, and positive ions, which are attracted to the cathode (*4*). As the positive ions arrive at the cathode, they can cause thermionic or high-field emission of electrons, which has a negative effect on arc interruption. Almost all these phenomena cease to exist if the gas between the breaker contacts is removed; in other words, if the arc is drawn in a vacuum. For this reason, vacuum is considered an extremely good medium for switching, and circuit breakers have been developed to take advantage of this feature. Figure 9.17 shows a sketch of a VCB, again with one pole. The assembly is sometimes called a bottle.

The main advantages of VCB's are

- Interruption usually occurs at the first zero current;
- There are no blind spots in their interrupting range;
- They have extraordinarily long life;
- They are relatively maintenance free; and
- Recovery of dielectric strength (between the parting contacts) following interrupting is extremely fast.

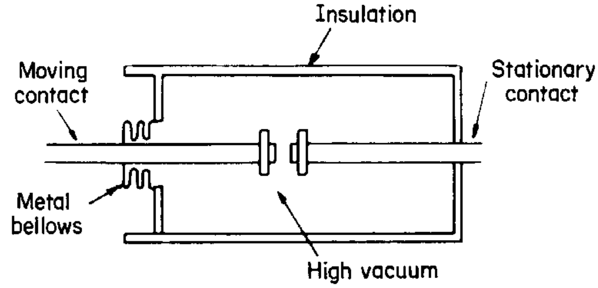


Figure 9.17.—Cross section of VCB.

These all result from the fact that the vacuum totally discourages ionization.

An important aspect of VCB's is in the long service. For instance, if a unit fails to clear a short circuit beyond its interrupting range, but another unit down the line does, the exceeded VCB can be employed again up to the full rating without difficulty. Because of their efficient ratio of size to capacity, they are extremely well suited to underground mining use. Their interrupting capacity for large currents is such that they can be utilized anywhere on high-voltage distribution, usually without reservation. This flexibility has made the VCB the most popular high-voltage interrupter for distribution systems in mining today.

Added to these advantages is the fact that the VCB does not have any physical orientation problems. This is a considerable constraint with OCB's, where the tanks must always be vertical. Vertical placement of VCB bottles is sometimes necessary, however, to minimize dust accumulation.

Ironically, the high efficiency of vacuum interrupters, which has favored their wide application, is the same property that can lead to severe transients. If care is not taken with VCB installation, switching transient-related problems can occur throughout the mine electrical complex. A detailed discussion of this important problem is deferred until chapter 11 because of related phenomena.

The operating mechanism, which includes the mounting structure for the vacuum bottles, is an important factor in proper VCB operation. As a result of the small contact travel distance, usually on the order of 1/4 in, four criteria are mandatory:

1. Rugged construction to withstand the shock and stress of equipment movement;
2. A firm, smooth closure motion to prevent contact bounce;
3. Forceful opening of contacts in the case of contact welding; and
4. Clean, smooth opening motion to prevent contact bounce and subsequent arc restriking.

In most cases, manufacturers rely on a spring-reset and spring-trip mechanism to meet items 2 through 4, and figure 9.18 illustrates one approach. The closing operation, also termed *resetting* or *reclosing*, may be manually or motor driven. The trip solenoid can be a shunt-trip or UVR device, and in some cases, both are used.

In VCB applications, the compact size of the operating mechanism and mounting structure has made possible a substantial reduction in overall power-equipment dimensions. Manufacturers have even incorporated a disconnect

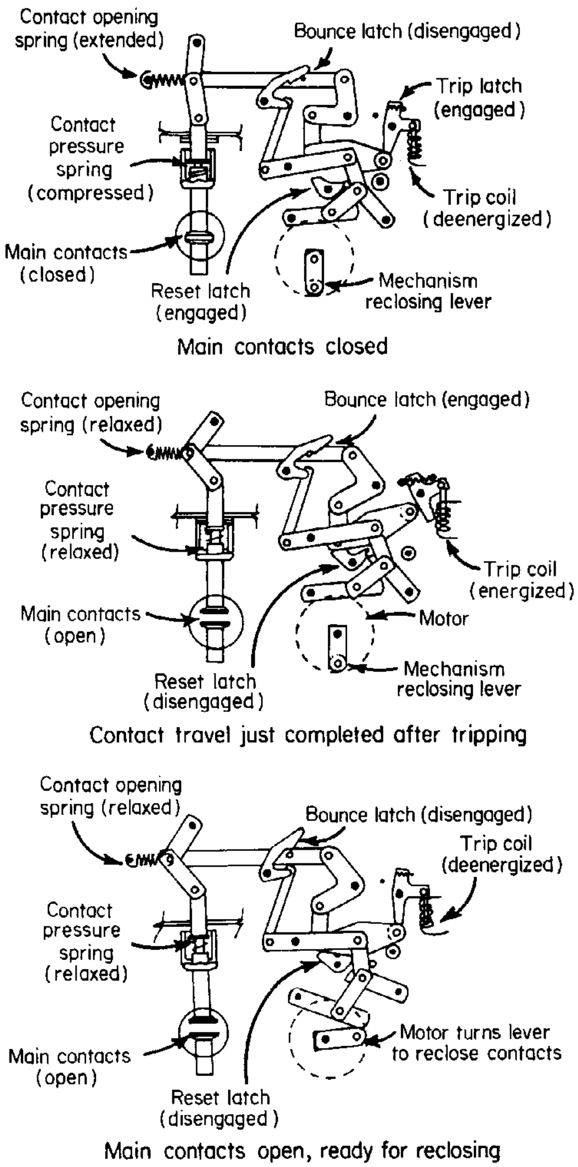


Figure 9.18.—Operating mechanism for vacuum interrupter. (Courtesy McGraw Edison)

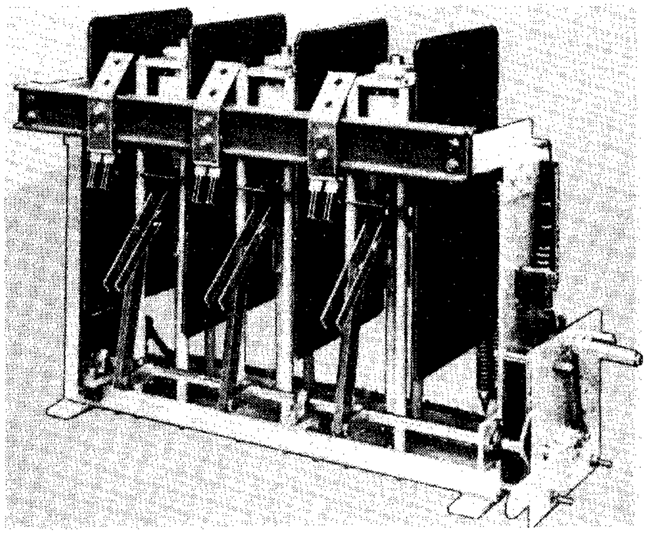


Figure 9.19.—VCB assembly incorporating a load-break switch. (Courtesy Ensign Electric)

switch in their designs (fig. 9.19). The operating mechanism for the switch is mechanically interlocked with the circuit breaker mechanism. If the switch is opened when the breaker is closed, the interlock trips the circuit breaker prior to switch-contact parting.

FUSES

The fuse is the simplest and oldest device for interrupting an electrical circuit under short-circuit or excessive-overload current (*5, 7*). Fuses are installed in series with the protected circuit and operate by melting a fusible link. The response is such that the greater the current, the shorter the time to circuit opening, that is, an inverse-time characteristic. Fuses may be used in ac or dc circuits, and there is such variation in their time-current characteristics that they are suitable for many special purposes. While circuit breaker contacts rely on external sensing, the fuse acts as both the sensing device and the interrupting device. Unlike circuit breakers, fuses are "one-shot," as their fusible element is destroyed in the circuit-protection process. Fuses are available with interrupting-current ratings up to 200,000-A symmetrical rms, much higher than the capacity of circuit breakers. Fuses are also available with current-limiting abilities to provide maximum protection for all circuit components.

Fuses are normally classified as *low voltage* or *high voltage*: The low-voltage types are intended for service in systems 600 V and below, while the high-voltage varieties are suitable for installations 2.3 to 161 kV (*7*).

LOW-VOLTAGE FUSES

Plug fuses and cartridge fuses are the two principal categories of standard low-voltage fuses, and they are classified as *non-time-delay, time-delay, dual-element,* or *current-limiting* (*13*). There are also miscellaneous and nonstandard fuse classes.

As with circuit breakers, there are three general fuse ratings (*7*);

1. *Current.* The maximum dc or rms ac, in amperes, which the fuses will carry without exceeding a specified temperature rise limit (available range: milliamperes to 6,000 A).
2. *Voltage.* The maximum ac or dc voltage at which the fuse is designed to operate (usual low-voltage ratings are 600, 300, 250, or 125 V ac or dc or both).
3. *Interrupting.* The assigned maximum short-circuit current that the fuse will safely interrupt (typical ratings are 10,000-, 50,000-, 100,000-, or 200,000-A symmetrical rms).

Special ratings are also given to current-limiting fuses to specify the maximum current and energy the device will let through to the protected circuit when clearing a fault (*7*).

Plug fuses are rated at 125 V and are available with current ratings up to 30 A. Their use is thus limited to circuits with this voltage rating or less, except that they may be employed on systems having a grounded neutral where the maximum potential to ground of any conductor does not exceed 150 V (7). As a result, plug fuses have limited application in mine power systems (although an extensive popularity still exists for homes). Cartridge fuse applications, on the contrary, are widespread, to the point where mention of a fuse implies a cartridge. Figure 9.20 shows the three standard low-voltage cartridge-type fuses (7).

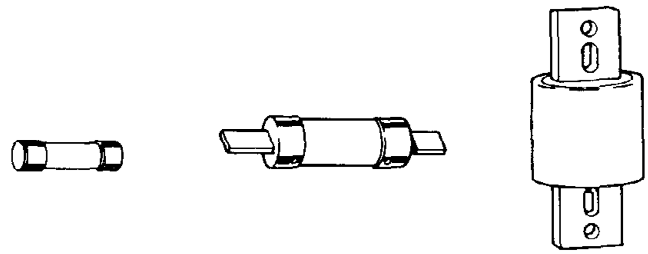


Figure 9.20.—Common cartridge fuses.

Non-Time-Delay Fuses

As the name implies, these have no intentional built-in delay. They have a very simple construction, consisting of two end terminals joined together by a copper or zinc fusible element. The link is more current sensitive to melting than to time. Non-time-delay fuses are available as one-shot (or nonrenewable) and renewable; the former is the oldest cartridge fuse type in use today (7). With the one-shot, the link is in a sealed enclosure and the entire cartridge must be replaced after interruption. The renewable fuse can be disassembled, and the link replaced. The lack of intentional time delay and a limited interrupting rating of around 10,000 A have substantially reduced the popularity of these fuses in recent years.

Figure 9.21.—Inside view of dual-element fuse.

Time-Delay Fuses

The metal alloy used in time-delay fusible links is not only sensitive to current but also to the time period involved. In other words, a specific current existing for a specified time period is necessary to cause the heat-melting energy of the alloy. Such an arrangement permits harmless high-magnitude, short-duration currents to exist, which are oftentimes necessary for proper system operation, as in motor starting.

Dual-Element Fuse

Originally designed primarily for motor-circuit protection, the dual-element fuse (fig. 9.21) combines the features of non-time-delay and time-delay units. The time-delay or thermal cutout is provided for overload protection, while two fuse link elements give short-circuit protection, blowing in a fraction of a cycle on heavy currents. The thermal cutout will allow the passage of currents as high as five times its continuous rating for up to 10 s. Hence, these fuses may be matched closely to protect the actual motor running current and at the same time be sized to protect wiring and other equipment, and provide both these functions without nuisance blowing. In fact, properly sized dual-element fuses are required on all fuse-protected trailing cables. They are available with up to a 200,000-A symmetrical rms interrupting-current rating, and for further protection, most dual-element fuses also have a current-limiting feature.

Current-Limiting Fuses

Short-circuit protection requires that a fuse limit the energy delivered by the short circuit to a faulted component. Obviously, the energy any interrupting device lets through under fault conditions cannot exceed the protected components withstand rating. Current-limiting fuses provide this protection by restricting or cutting off fault currents before damaging peaks are reached. With very high fault currents, they are extremely fast, limiting current in less than one-quarter cycle, with current interruption occurring within the first one-half cycle. Only a portion of the destructive short-circuit energy that is available is let through. By this, the current-limiting fuse allows the use of lower momentary and interrupting ratings by cutting off current within equipment ratings (7). Figure 9.22 illustrates how the fuse operates: the large waveform represents the available short-circuit current on a faulted system, and the performance of the fuse is superimposed.

Restricting energy is a means of limiting the mechanical and thermal stress imposed on equipment that is carrying fault current. To illustrate this energy, consider figures 9.22 and 9.23 and the peak let-through current, I_p. It has been found that the magnetic forces during a fault vary as the square of fault current, I_p^2 (7). These forces translate to mechanical stress, which could damage transformer frames, bus structures, or cable supports. The let-through energy, I^2t, represents a measure of the heating effect or thermal energy of the fault with or without the fuse (with the fuse, the value is I_p^2). I^2t actually equals $\int i^2 dt$, the time integral of the current squared for the time under consideration (8). Both I_p^2 and I_p^2t can be considerably reduced when current-limiting fuses are used (7). Furthermore, equipment with an I^2t withstand rating can be matched with the energy let-through limit of the fuse.

Standard Fuses

As implied by the foregoing, cartridge fuses come in a wide range of types, sizes, and ratings. Various classes for

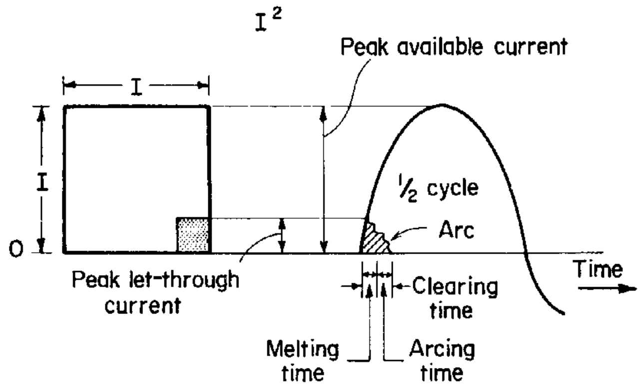


Figure 9.22.—Current-limiting action of fuses.

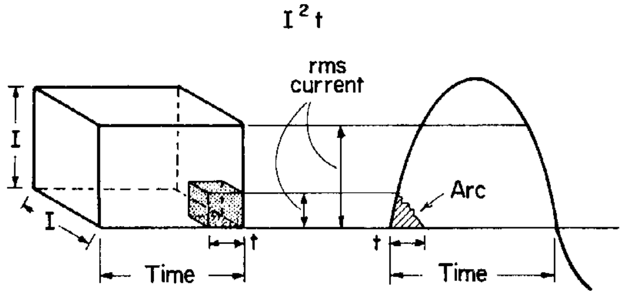


Figure 9.23.—Energy-limiting action of fuses.

low-voltage units have been standardized *(15)*, and a listing of general-purpose fuses follows (the first value listed is the range of continuous currents):

Class G: 0 to 60 A, 300 V to ground maximum, 100,000-A symmetrical rms interrupting, current limiting, fit only class G fuse holders.

Class H: 0 to 600 A, 250 and 600 V, interrupting capacity up to 10,000 A, either one-time or renewable construction, commonly termed the "old NEC fuse."

Class J: 0 to 600 A, 600 V, 200,000-A symmetrical rms interrupting, current limiting, fit only a class J fuse holders.

Class K: 0 to 600 A, 250 and 600 V, 50,000-, 100,000-, or 200,000-A symmetrical rms interrupting, have the greatest current-limiting effect of all low-voltage fuses (available as straight current limiting, dual-element current limiting, and dual-element time-delay current limiting), fit class H fuse holders.

Class L: 601 to 6,000 A, 600 V, 200,000-A symmetrical rms interrupting, current limiting, bolt-in mounting.

Class R: 0 to 600 A, 250 and 600 V, 200,000-A symmetrical rms interrupting, current limiting similar to class K level 5 fuse, fit only class R fuse holders.

Class T: 0 to 600 A, 250 and 600 V, 200,000-A symmetrical rms interrupting, current limiting but effect less than class J fuses, fit only class T fuse holders.

Nonstandard Fuses

Nonstandard fuses receive their name because of their special dimensions or use in special applications; they are not general-purpose fuses *(7)*. Of the many available, four have important applications in mining:

Cable Limiters. These fuses are for use in multicable circuits (paralleled cables) and are placed in series with each cable in parallel. They are designed to provide short-circuit protection to each cable, removing it from power in case of failure. Cable limiters are rated according to cable size (AWG 4/0 and so forth).

Semiconductor Fuses. These devices are available in two types: semiconductor-protection fuses or semiconductor-isolation fuses. Both are used in series with the application. Protection fuses are employed where solid-state devices are to be protected rather than isolated after a failure; they have lower let-through characteristics than other current-limiting fuses. A specific application is protecting a rectifier or thyristor in case of an overload current. Isolation types are high-speed fuses, used to isolate a defective solid-state device in case of its failure. These are mandatory fuses for individual power diodes paralleled in large rectifier banks.

Capacitor Fuses. Capacitor fuses are applied in series with power-factor (pf) correction (or other type) capacitors and are used to isolate a failed component by clearing short-circuit current before excessive gas is generated in the capacitor.

Welding Fuses. These are current-limiting fuses for use in welder circuits only. The time-current characteristics are such that these fuses allow a longer intermittent overload than general-purpose fuses, but still provide short-circuit protection.

HIGH-VOLTAGE FUSES

High-voltage fuses provide usable protection for 2.3- to 161-kV systems and fall into two general categories: distribution fuse cutouts and power fuses *(7)*. Distribution fuse cutouts were designed for overhead distribution circuits, such as the protection of residential distribution transformers. Even though their employment in utility-type systems is extensive, their use in mining is limited and in some cases restricted. Power fuses are another matter, as certain types offer extremely practical protection in mine power systems. They can be applied to substation, distribution, and potential transformers (in series with the primary) and occasionally to distribution circuit conductors. For surface mine systems, the fuses are often equipped with contacts, arranged so that the fuse and its mounting act as a disconnect switch (fig. 9.24). There are two basic power fuses, expulsion and current-limiting types, and the next few paragraphs will discuss their operation, ratings, and application.

Expulsion Types

As with low-voltage fuses, high-voltage types start the current-interruption process by the melting of a fusible link, but as might be expected, deionization of the attendant arc becomes the most substantial part of current termination. To help the process, as shown in figure 9.25, the link is held under tension by a coil spring; upon melting, the spring pulls the contacts apart, lengthening the arc *(4)*. In expulsion fuses, gases are liberated from the lining of the current-interrupting chamber by the heat generated from the arc. Both the earliest form of expulsion

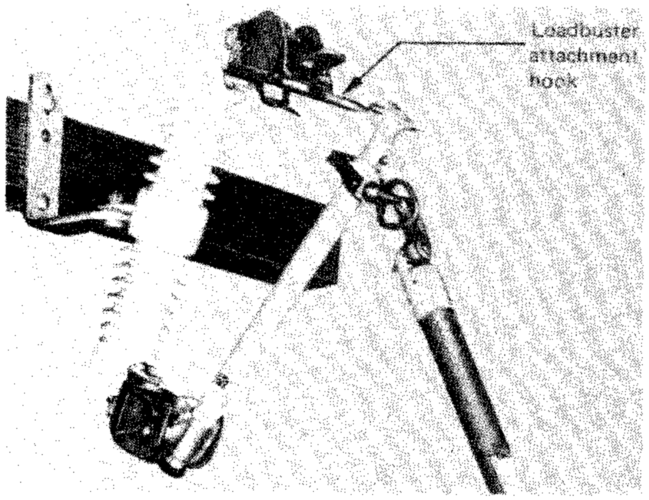


Figure 9.24.—High-voltage power fuse and support. (Courtesy S&C Electric Co.)

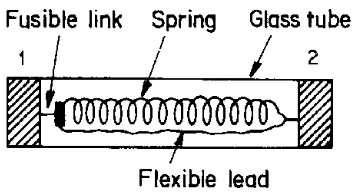


Figure 9.25.—Fusible element under spring tension in high-voltage fuse.

fuse and distribution fuse cutouts use a liner of organic material to deionize the generated gases by expelling them from the fuse holder tube to the surrounding air. The problem with this operation is the attendant flame expulsion and loud noise. Hence, expulsion fuses are suitable only for outdoor usage, generally in substations remotely located from human habitation (7).

The limited interrupting capacity (table 9.7) and unsuitability for indoor use of early expulsion fuses led to the development of the boric acid or solid-material fuse (7). Here, the interrupting chamber is made of solid boric acid. When exposed to arc heat, the material liberates steam, which can be readily condensed to liquid by venting the gas into a cooling device. The result is an operation with negligible or harmless flame and gas emissions and noise levels. The range of voltage, continuous current, and interrupting ratings is also greatly expanded.

High-voltage boric acid fuses are manufactured in two styles (7): the fuse unit (nonrenewable), where the fusible unit, interrupting element, and operating element are all combined in an insulated tube; and the refill unit or fuseholder (renewable), where only the refill unit is replaced after interruption. Figure 9.26 shows the internal components of a refill unit, while figure 9.27 illustrates the construction of the entire fuse. Table 9.7 provides a list of typical ratings for both styles. The fuse-unit style is intended for outdoor use at system voltages of 34.5 to 138 kV, while the refill unit can be used indoors or outdoors on the surface at 2.4 to 34.5 kV.

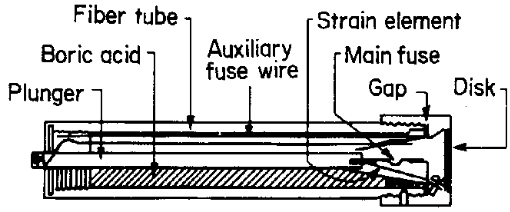
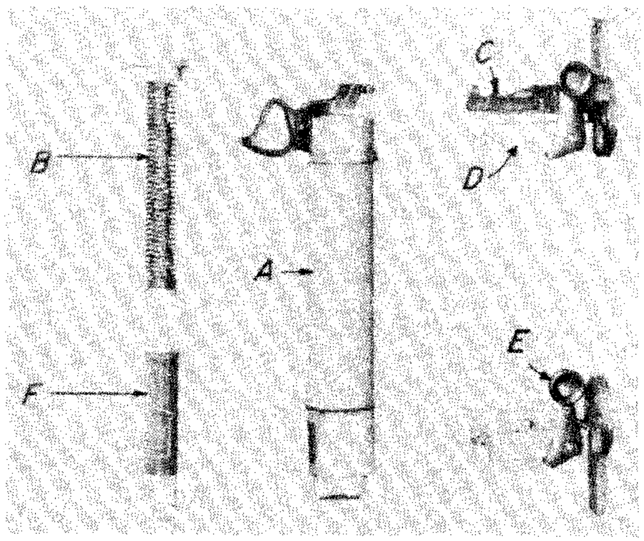


Figure 9.26.—Cross section of boric acid power fuse refill.



KEY

A Fuseholder
B Spring-and-cable assembly (copper cable carries load current)
C,D Fuseholder upper contacts and latch
E Fuseholder lower contacts and latch
F Refill unit

Figure 9.27.—Disassembled refill unit for boric acid fuse. (Courtesy S&C Electric Co.)

Current-Limiting High-Voltage Fuses

High-voltage current-limiting or silver-sand fuses have the same advantages as previously discussed for low-voltage fuses and are of two different forms (7): those to be used with high-voltage motor starters for high-capacity distribution circuits at 2,400 and 4,160 V and those for use with potential, distribution, and small power transformers from 2.4 to 34.5 kV. The operation of either form is such that the arc established by the melting of the fusible element is subjected to mechanical restriction by a powder or sand filler surrounding the fusible element. The technique provides three important features:

- Current is interrupted quickly without arc-product or gas expulsion. This allows use indoors or in small-size enclosures on the surface or underground. There is no noise from the operation, and since there is no gas or flame discharge, only normal electrical clearances need by met.

Table 9.7.—Ratings of high-voltage power fuses

Nominal rating, kV	Expulsion-type fuse		Boric acid fuse, 1-shot type		Boric acid fuse, renewable		Current-limiting fuse	
	Maximum continuous current, A	Maximum interrupting rating, MVA[1]	Maximum continuous current, A	Maximum interrupting rating, MVA[1]	Maximum continuous current, A	Maximum interrupting rating, MVA[1]	Maximum continuous current, A	Maximum interrupting rating, MVA[1]
2.4	—	—	—	—	200, 400, 720	155	100, 200, 450	155, 210, 360
4.16	—	—	—	—	200, 400, 720	270	450	360
4.8	—	—	—	—	—	—	100, 200, 300, 400	310
7.2	100, 200, 300, 400	162	—	—	200, 400, 720	325	100, 200	620
14.4	100, 200, 300, 400	406	—	—	200, 400, 720	620	50, 100, 175, 200	780–2,950
23	100, 200, 300, 400	785	—	—	200, 300	750	50, 100	750–1,740
34.5	100, 200, 300, 400	1,174	100, 200, 300	2,000	200, 300	1,000	40, 80	750–2,600
46	100, 200, 300, 400	1,988	100, 200, 300	2,500	—	—	—	—
69	100, 200, 300, 400	2,350	100, 200, 300	2,000	—	—	—	—
115	100, 200	3,110	100, 250	2,000	—	—	—	—
138	100, 200	2,980	100, 250	2,000	—	—	—	—
161	100, 200	3,480	—	—	—	—	—	—

[1]3-phase symmetrical rating.

NOTE.—Dashes indicate that standard fuses are not available in the specific voltage rating.

- Very high interrupting ratings are available so these fuses can be applied on systems with very high short-circuit capacity (within their voltage rating).
- All of the advantages of current-limiting action are available for high voltage.

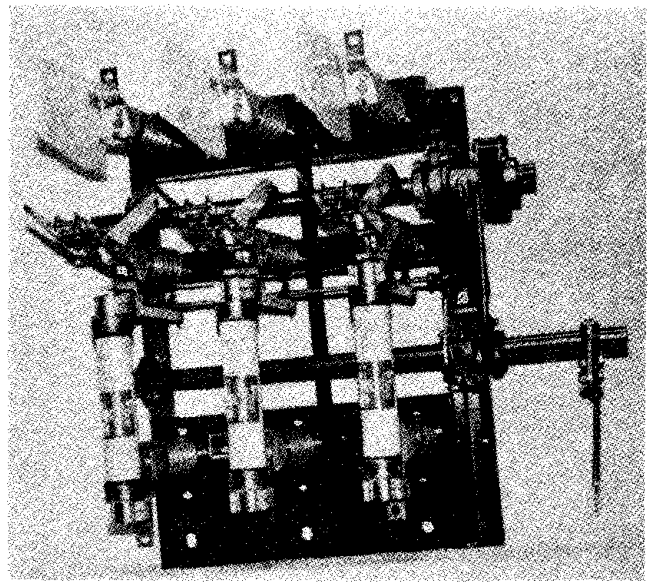
Table 9.7 provides a listing of typical ratings for current-limiting fuses. Instead of being rated by current, these fuses can also be "E-rated" (for instance, 100 E instead of 100 A), "C-rated," or "R-rated." The specifications for E and C ratings are as follows:

- *E-rated fuses*: 100 E and below, open in 300 s at an rms current within the range of 200% to 240% of the continuous rating of the fuse element; above 100 E, open in 600 s at an rms current within the range of 220% to 264% of the continuous (or E) rating;
- *C-rated fuses*: open in 1,000 s at an rms current within the range of 170% and 240% of the C ratings.

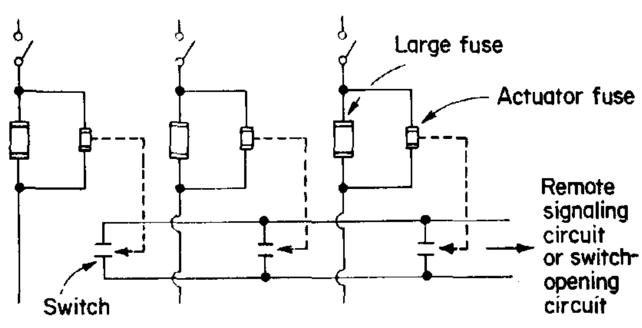
E-rated fuses are considered as general-purpose or backup fuses, while R-rated devices are intended for use with high-voltage motor starters (7).

Load-Break Switches

It is possible that after the occurrence of a short circuit on a fuse-protected three-phase system, only one of the three fuses could open. Here current through the remaining two fuses might be reduced so that they do not open. The system then becomes single phased, which can cause serious damage to equipment. In a low-voltage circuit, dual-element fuses that are closely matched to the overcurrent point can usually handle the situation. On high-voltage systems, the problem is much more difficult when protection is by fuses alone. However, to take advantage of the lower cost of fuses and load-break switches versus the cost of a high-voltage circuit breaker, some manufacturers produce load-break switches with incorporated high-voltage fuseholders. An example is shown in figure 9.28 where the fuses are interlocked to trip the operating mechanism of the switch if one or more of the fuses fail. Interlocking is usually accomplished with special high-voltage fuses that contain a spring-loaded plunger. Fuse activation releases the plunger, which trips



Fused load-break switch



Schematic showing interlocks

Figure 9.28.—Load-break switch with interlocked high-voltage fuses. *(Courtesy Line Power Manufacturing Corp.)*

of time and is determined by the heater rating. The trip setting is commonly based on a 40° C ambient temperature, but the relay may be ambient or nonambient compensating. Most relays of this type must be manually reset after tripping.

An electromechanical-thermal device not using bimetallics is the melting-alloy or eutectic-alloy relay, figure 9.31*B*. Being shock resistant and having high contact force, this is considered one of the most reliable thermal relays available, but because of its cost, it is not nearly as popular as the bimetallic type. The alloy melting point is extremely precise and is again related to a specific current-time characteristic. The relay can be reset after tripping and alloy resolidification.

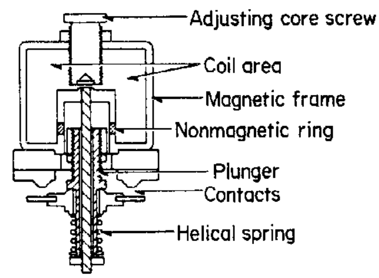
Two other thermal devices, resistance or thermistor types and thermocouples, operate with associated electronic equipment to provide very precise temperature sensing and relaying. Here, for example, a probe can be inserted or embedded in a transformer or a motor winding to provide a spot temperature response. This type of device is very popular especially where large horsepower or capacity is involved.

Electromagnetic-Attraction Relays

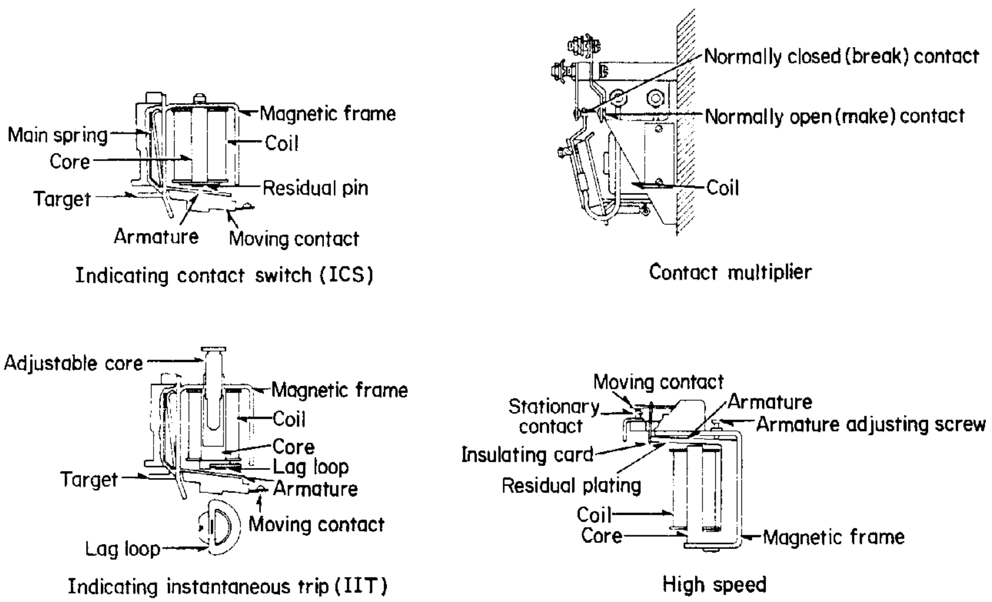
There are three electromagnetic-attraction relays in common use: the solenoid, the clapper, and the polar (*20*). Although their operational speed might vary, all are considered instantaneous relays, since there is no built-in delay for pickup or reset. The solenoid and clapper types are available for ac or dc and are voltage or current actuated. Coil impedance is high for voltage and low for current. Polar units are dc sensing only, but may be used on ac circuits through rectification. All electromagnetic relays are available with *NO* contacts, *NC* contacts, or both.

In solenoid units, the relay contact movement is initiated by a plunger being drawn into a cylindrical solenoid coil. Typical operating times are 5 to 50 ms, with the longer times associated with operation near the minimum pickup value (*20*). A cross-sectional sketch of a solenoid relay is given in figure 9.32*A*.

Four different clapper relays are shown in figure 9.32*B*. These have a magnetic frame with a movable armature and operate by the attraction of the armature to



A Solenoid-type relay



B Clapper-type relay

Figure 9.32.—Solenoid and clapper relays. (Courtesy Westinghouse Electric Corp.)

an electromagnetic pole (*20*). The armature controls the pickup or reset of contacts.

As illustrated in figure 9.33, polar relays have a hinged armature in the center of the magnetic structure, which is here shown as an electromagnet but may be a permanent magnet. The relays operate when dc is applied to the actuating coil, and the polarity of the actuating source determines armature action, be it stationary or movement in either direction (*10*). In some units there is no retaining spring, and through a combination of contacts, the relays can sense actuating current through the coil in either direction (*20*).

The pickup and reset values of clapper units are less precise than those of solenoid and polar relays; thus, clapper relays are used often as auxiliary or *go, no-go* devices (*20*). A common use for polar relays is in dc circuit protection where the actuating source is obtained from a shunt or directly from the circuit (*10*).

A characteristic that should be considered when applying any electromagnetic-attraction relay is the large difference that can exist between pickup and reset values. When an attraction relay picks up, the air gap is shortened, and a smaller coil current is needed to retain pickup. Thus, the reset current may be much lower than the pickup current. The disparity is usually expressed as a percent ratio of reset current to pickup current, and is less pronounced in ac than dc relays. The ac relays can have a reset up to 90% or 95% of pickup, but dc ratios range from 60% to 90% (*10*). This is no problem in overcurrent applications where relay coil current drops to zero after pickup, but it is a concern where reset values are important.

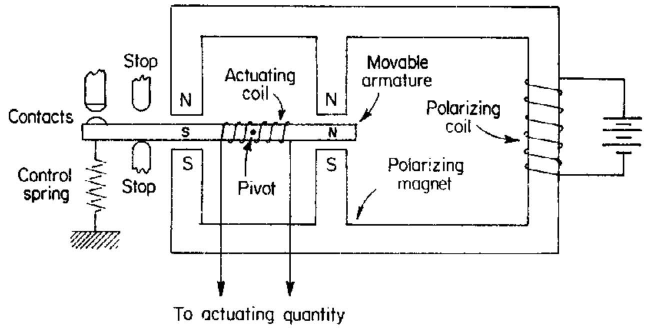


Figure 9.33.—Polar relay.

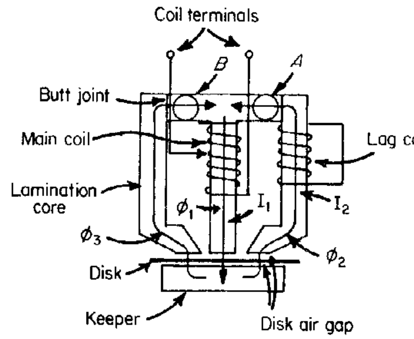


Figure 9.34.—Common induction-disk relay.

Electromagnetic-Induction Relays

Electromagnetic-induction relays are of two general types: induction disk and cylinder (*20*). Depending on the design, the induction-disk unit can be either a single-quantity or directional relay, whereas cylinder relays are intended to be directional. A single-quantity relay, as might be supposed, is actuated by and compares two sources (*10*). The most commonly used time-delay relays for system protection employ the induction-disk principle (*7*).

Single Quantity

Single-quantity time-delay relays of the induction-disk type use the same principle of operation that was described for induction motors in chapter 6, but the physical construction is quite different (*20*). A sketch of an elementary induction-type device is shown in figure 9.34, and most time-delay relays in use today have this arrangement. The disk, made of aluminum, is mounted on a rotating shaft restrained by a spring, and a moving contact is attached to the shaft (fig. 9.35). On one side of the disk is a three-pole electromagnet; the other side has a common permanent magnet or keeper. The operating torque on the disk is produced by the electromagnet, and the keeper provides a damping action or restraint after the disk starts to rotate. The retarding effect of the keeper creates the time delay or desired time characteristic of the relay. Figure 9.35 is a front-view illustration of an actual induction-disk relay removed from its drawout case; all important components are indicated. The unit pictured is for overcurrent, but overvoltage and undervoltage relays are also available and are identical in construction except for the electromagnet coil rating.

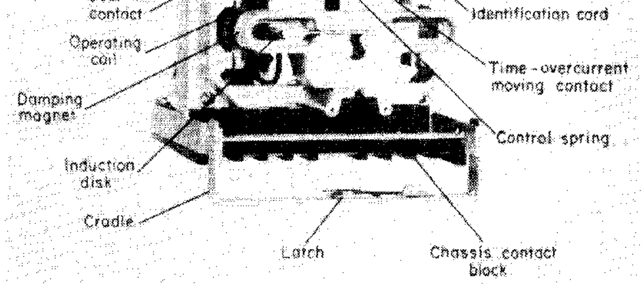


Figure 9.35.—Front view of induction-disk relay removed from case. (Courtesy General Electric Co.)

The control spring carries current for the moving contact. If the actuating quantity driving the electromagnet is of sufficient magnitude and is sustained for enough time, the disk will rotate until the moving contact touches the stationary contact. (Some relays use a lever on the moving disk that forces a pair of stationary contacts to close, so that no current flows through the control spring and disk.) Pickup of these main contacts triggers the seal-in or time-delay element, which is an electromagnetic-attraction relay with its coil in series and contacts in parallel with the main contacts. When activated, this relay picks up and seals in, thus lightening the current-carrying duty of the main contacts as well as operating a target indicator. After pickup, it usually must be reset manually.

The tap block at the top of figure 9.35 is to allow different tap settings on the electromagnet coil. Table 9.8 lists the tap settings generally available in overcurrent relays (*7*), but some relays have wider ranges than those shown. Each range represents a different operating coil. Voltage relays have a narrower range of adjustment, because they are usually expected to operate within a limited change from the normal magnitude of the actuating quantity (*10*). Be it a voltage or current relay, the coil and its tap settings are normally selected with respect to the ratios of the potential or current transformer used.

Table 9.8.—Common current ratings of induction-disk overcurrent relays

Time-delay elements		Typical instantaneous elements,[2] adjustment range, A
Coil range, A	Tap settings,[1] A	
0.5 to 2.5	0.5, 0.6, 0.8, 1.0, 1.2, 1.5, 2.0, 2.5	0.5– 4.0 2 – 16
1.5 to 6.0	1.5, 2, 2.5, 3.0, 3.5, 4.0, 5.0, 6.0	10 – 80 20 –160
4.0 to 16	4.0, 5.0, 6.0, 7.0, 8.0, 10, 12, 16	([3])

[1]Tap settings will vary slightly according to manufacturer.
[2]Additional units are available for each time-delay range.
[3]Not adjustable.

As shown in figure 9.35, overcurrent disk relays often have a second (auxiliary) ac-operated instantaneous element, which is a clapper-type relay (*7*). The unit is continuously adjustable over a calibrated range, and table 9.8 lists some of these representative values. This relay operates in series with the time-delay operating coil and is usually set to operate instantaneously at a current pickup value higher than that of the time-delay element. However, since the same actuating source drives both elements, the instantaneous-relay setting must be coordinated not only with the same source but also with the timed element. The instantaneous contacts can be in parallel with the time-delay contacts or can be connected to separate terminals. The unit also has a target indicator, which normally requires manual reset after tripping.

The operational characteristic produced by the induction-disk principle is termed *inverse time*. Although mentioned earlier in this chapter, the inverse response is illustrated again in figure 9.36 to emphasize that the operating time becomes less as the magnitude of the actuating quantity is increased (*10*). The more pronounced this effect becomes, the more inverse the curve is said to be. All relay time curves are actually inverse, with the exception of a theoretical definite-time response. By definition, definite-time characteristics imply that the operating time of the relay is unaffected by the magnitude of actuating quantity. In reality, an actual definite-time curve is very slightly inverse (fig. 9.36). Regardless, the term *definite time* is normally applied to all fixed-time relays that approach this response.

The control-spring tension, the damping magnet, and the magnetic plugs (*A* and *B* of figure 9.34) provide separate and relatively independent adjustment of the relay inverse-time characteristics. They are preset by the manufacturer, and the common responses are "inverse," "very inverse," "extremely inverse," "short time," and "long time," the first three being the most popular in mining. A comparison of these responses is given in figure 9.37. The need for a specific response depends upon the application, and a few thoughts in terms of overcurrent relays follow (*6*).

When the available fault-current magnitudes vary considerably, faster overall protection is usually gained with an inverse-time response. Very inverse curves provide the best overall protection where fault current remains

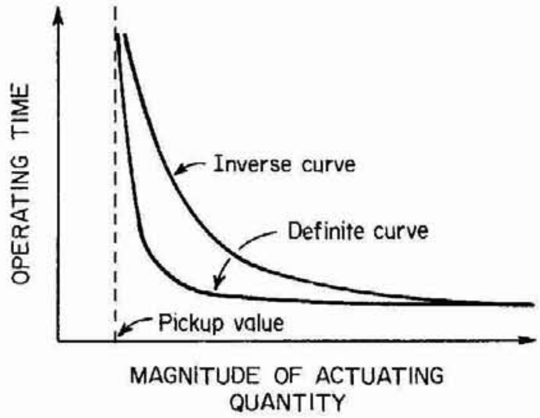


Figure 9.36.—Inverse-time curve compared with definite-time curve.

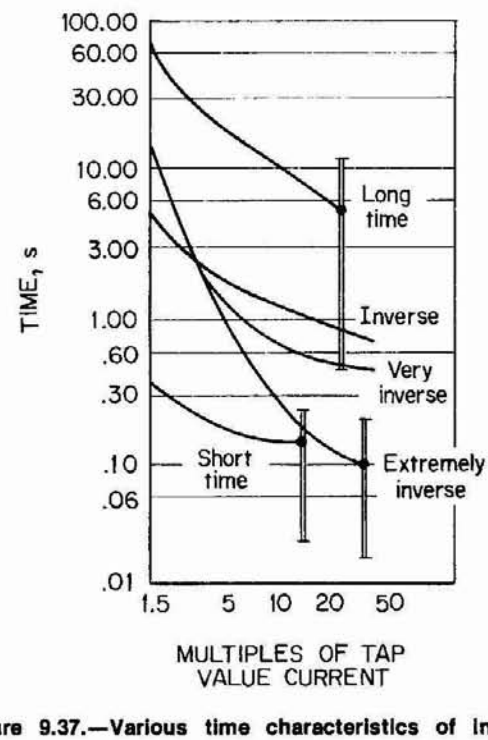


Figure 9.37.—Various time characteristics of induction units.

constant (detection of the fault, as seen by the relay, is mainly a function of fault location). Extremely inverse relays are designed to coordinate rather closely with power fuses and distribution cutouts and are also used in systems that have large inrush currents. The actual application of these characteristics in the mine is given in chapter 13.

The operating time of an induction relay can usually be adjusted by selecting the distance of rotor travel from the reset to the pickup position (*10*). This is accomplished by adjusting the rest position of the moving-contact stop. The time dial, with evenly divided markings, facilitates positioning. When the response of the relay for different time dial settings is plotted, the result is a family of curves, an example of which is shown in figure 9.38. Current is plotted in terms of multiples of pickup, which enables the curves for a specific relay to be used with any tap setting.

Directional

The basic ac directional electromagnetic-induction relay or cylinder unit in common use is sketched in figure 9.39. Its operation is similar to that of an induction motor that has salient poles for the stator, except that here the rotor iron is stationary and only the rotor conductor is free to rotate (*10, 20*). The rotor conductor is a thin-walled aluminum cylinder, and the two actuating quantities, causing I_1 and I_2, independently produce torque on the cylinder. The cylinder drives a moving contact whose travel is restricted to a few degrees by the stationary contact and stops. Reset torque is established by a spiral spring.

The ac directional relays are used to distinguish between current supplied in one direction or the other in an ac circuit, by recognizing phase-angle differences between the two actuating quantities (*10*). (Conversely, a dc directional relay, or polar unit, recognizes differences in polarity.) To perform the ac comparison, one actuating value is used as a reference or polarizing quantity. Therefore, the polarizing quantity phase angle must remain fixed while the phase angle of the other fluctuates widely. One application of this technique is in power relays where the unit is polarized by circuit voltage, with circuit current being the other actuating value. Through this, the cylinder detects power flow in one direction or the other. Another important application is an ac directional relay combined with an overcurrent relay, as shown in figure 9.40. Here, tripping occurs only when the current has a specific relationship to the voltage, and power flow is in the tripping direction.

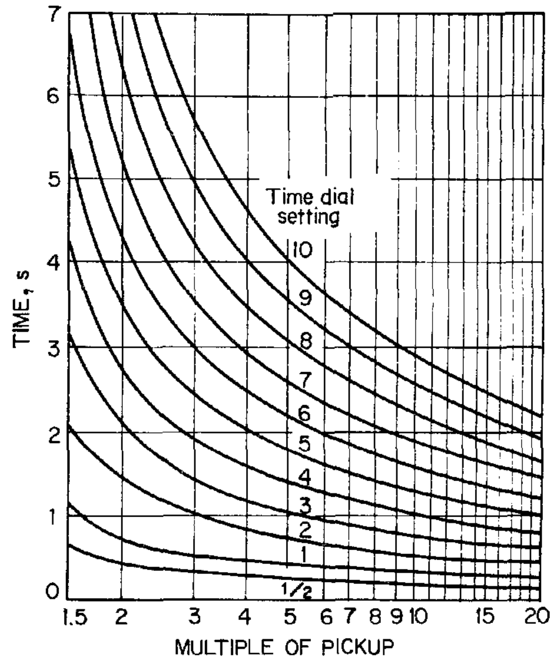


Figure 9.38.—Family of inverse-time characteristics.

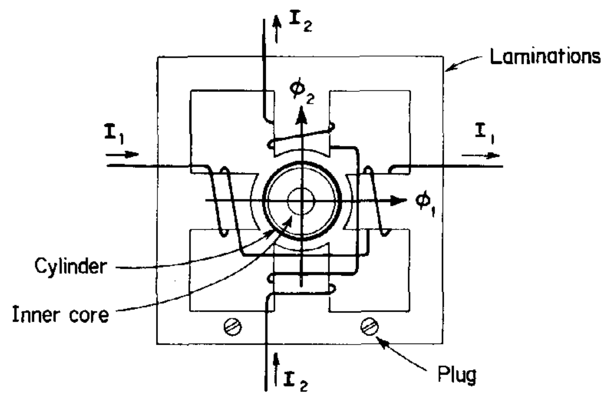


Figure 9.39.—Cylinder directional relay.

BASIC RELAY CONNECTIONS

In order to sense a malfunction and then supply tripping energy to the appropriate circuit breaker, a relay must be attached in some manner to the power system. Circuit connections for protective relaying are basically not too different from those discussed for instrumentation in chapter 5. Here, however, the relay coil receives the input information, and its contacts pick up or reset, thus affecting the control power to the circuit breaker. Direct relay connections to the monitored circuit are often restricted to low-voltage, low-power circuits because most relay current or voltage coils are designed to operate in the vicinity of 5 A or 120 V (*4*). Obviously, if power-system values exceed these levels, some interface is needed between the monitored circuit and the relays. Again, instrument transformers for ac and resistors for dc are used, a subject also introduced in chapter 5.

There are five basic relay connections used for protective relaying in the mining industry. For ac systems, these are direct, potential, and differential; and for dc work, direct and potential are used. Differential relaying is also available for dc, but the circuitry is not considered basic. Although some of the techniques are employed much more frequently than others, this section serves to introduce all these connections.

Alternating Current Direct Relaying

Direct relaying is used to sense the magnitude of current flow. As shown in figure 9.41*A*, its simplest form consists of a current transformer (CT) secondary connected to a relay operating coil. Relay pickup current is thus a function of line current. For instance, consider that the transformer ampere-turns ratio or current rating is 50/5 A or 10/1 and the relay pickup setting is at 0.5 A. This relay would theoretically pick up its contacts when line current is (10)(0.5) or 5 A. The purpose of this connection is therefore to provide protective relaying for current in any conductor.

The important items to consider in direct relaying are concerned with matching the performance of the CT with that of the relay. IEEE standards provide most of these (7).

1. *Ratios.* As an obvious starting point after the foregoing example, standard ratios are listed below:

Single-ratio CT, amperes:

10/5	200/5	2,000/5
15/5	300/5	3,000/5
25/5	400/5	4,000/5
40/5	600/5	5,000/5
50/5	800/5	6,000/5
75/5	1,200/5	8,000/5
100/5	1,500/5	12,000/5

Double-ratio CT with centered-tapped secondary, amperes:

25/50/5	400/800/5
50/100/5	600/1,200/5
100/200/5	1,000/2,000/5
200/400/5	1,500/3,000/5
300/600/5	2,000/4,000/5

Multiratio CT with multitapped secondary, amperes (current ratings higher than those shown are also available):

Rating	*Taps*
600/5	50/5
	100/5
	150/5
	200/5
	250/5
	300/5
	400/5
	450/5
	500/5
	600/5
1,200/5	100/5
	200/5
	300/5
	400/5
	500/5
	600/5
	800/5
	900/5
	1,000/5
	1,200/5
2,000/5	300/5
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	1,600/5
	2,000/5

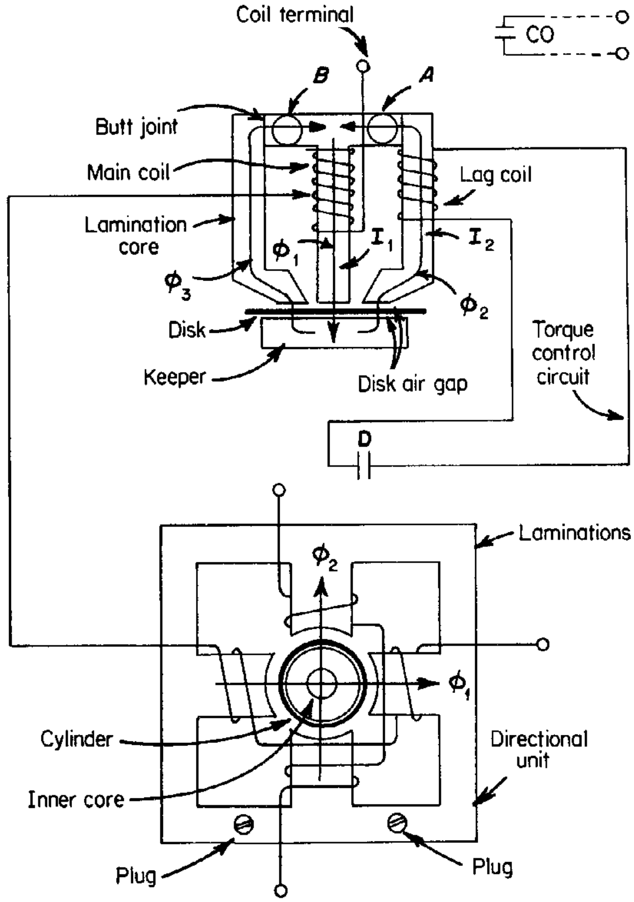

Figure 9.40.—Directional overcurrent relay using induction-disk relay and cylinder relay.

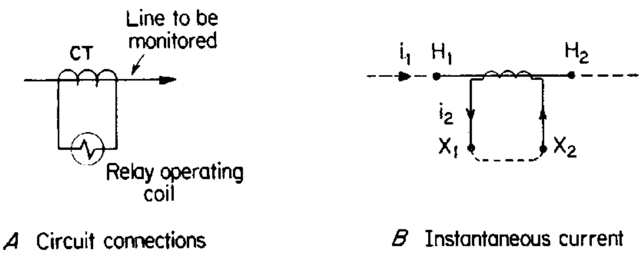

Figure 9.41.—Direct relaying in ac systems.

The double-ratio and multiratio types provide flexibility through secondary taps. These values are for bushing-type or window-type CT's, which are the most popular in the industry. All these have the standard 5-A-rated secondary current.

2. *Secondary Current.* The continuous-current rating of the secondary should be at least equal to the actual drain, but a full-load secondary current of 3 to 4 A is normal practice. An oversized CT is bad practice, as the percent error is much greater than with a correctly rated CT.

3. *Short-Time Ratings.* Both thermal and mechanical ratings should be considered. The thermal short-time value relates to the maximum symmetrical rms primary

current that the CT can carry for 1.0 s without exceeding its maximum specified winding temperature. The mechanical rating refers to the maximum asymmetrical rms current the CT can withstand without damage. In both cases, the rating is made with the secondary short-circuited.

4. *Voltage Rating.* Standard voltage ratings are 600, 2,500, 5,000, 8,700, and 15,000 V, and are the same as insulation classes found in mine systems. The CT will operate continuously at 10% above rated voltage without insulation failure.

5. *Burden.* As defined in chapter 5, burden is the load connected to the CT secondary; expressions used are volt-amperes at a given power factor or an impedance with a power factor. The power factor is that of the burden. Table 9.9 lists standard values for CT's at 60 Hz. Relay burdens are so varied they cannot be listed, but chapter 10 shows how CT burden and relay burden can be compared.

6. *Accuracy.* Accuracy of a CT relates to its transformation ability. In protective-relaying applications, accuracy is not only important at normal circuit currents but also at fault-current levels. The problem in CT's is that core saturation leads to poor accuracy or ratio errors. Accuracy class designations use a C or T identifying letter followed by a classification number. C states that percent ratio error can be calculated, whereas T means that the value has been found by testing. The classification number relates to a standard secondary voltage of 10, 20, 50, 100, 200, 400, or 800 V. At this voltage, the CT will deliver to a standard burden, 20 times normal secondary current with 10% ratio error or less, and it will not exceed 10% with any current from 1 to 20 times rated current with a lesser burden. (For example, C200 relates that for a 2.0-Ω burden, (20)(5) or 100 A can be delivered from the CT without exceeding 10% error. This error can also be calculated.)

7. *Polarity.* Polarity relates to the correct phasing of primary and secondary currents, and figure 9.41*B* shows the relative instantaneous directions of current as per standard markings. This allows correct connections when more than one transformer is used, which is imperative in three-phase systems.

As can be seen in the foregoing listings, actual manufacturer specifications should always be consulted before attempting to match CT's with relays for direct-relaying applications.

Alternating Current Potential Relaying

Potential relaying is as simple as direct relaying and enables circuit voltage to be monitored. Figures 9.42*A* and 9.42*B* show two applications: sensing voltage across a resistance and between two conductors. There is a potential transformer (PT) between the circuit and the relay. Figure 9.42*C* gives the polarity correspondence of instantaneous voltages between the primary and secondary windings as well as conventional transformer markings. Standard PT's are single-phase, two-winding units constructed so that the primary and secondary voltages always have a fixed relationship (7).

To visualize the operation, consider figure 9.42*A*. The transformer is rated 2,400/120 V or has a 20/1 ratio, an overvoltage relay is used, and the relay coil is rated at 120 V with the contacts set to pick up at 80% of rated. The contacts will therefore pick up when 1,920 V exists across the resistor.

IEEE standards also provide guidelines for PT utilization, and a summary of these follows (7). In general, they are less rigorous than those for CT's.

1. *Voltage.* Standard voltage ratios are available in table 9.10. When applied to sense voltage between two conductors, the nominal system voltage should be within ±10% of the transformer nameplate rating. When used in three-phase mining systems supplying portable or mobile equipment, primary connections must be line to line. Special ratings, providing other than the standard 120-V secondary, are usually available.

Table 9.9.—Standard burden for current transformers

Standard burden designation[1]	General characteristics		Characteristics for 60 Hz and 5-A secondary current[2]		
	Resistance (R), Ω	Inductance (L), mH	Impedance (Z), Ω	Apparent power (S), VA	pf
B–0.1	0.09	0.116	0.1	2.5	0.9
B–0.2	.18	.232	.2	5.0	.9
B–0.5	.45	.580	.5	12.5	.9
B–1	.5	2.3	1.0	25	.5
B–2	1.0	4.6	2.0	50	.5
B–3	2.0	9.2	4.0	100	.5
B–4	4.0	18.4	8.0	200	.5

[1]B–0.1, B–0.2, and B–0.5 are usually applied for metering purposes; B–1 through B–4 are usually applied for relaying.
[2]At 5 A, $S = I^2Z$; for example, for B–2, $S = 5^2 2 = 50$ VA.

Table 9.10.—Standard ratings for potential transformers

(Secondary, 120 V)

Primary, V	*Ratio*	*Primary, V*	*Ratio*
120	1/1	8,400	70/1
240	2/1	12,000	100/1
480	4/1	14,400	120/1
600	5/1	24,000	200/1
2,400	20/1	36,000	300/1
4,200	35/1	48,000	400/1
4,800	40/1	72,000	600/1
7,200	60/1		

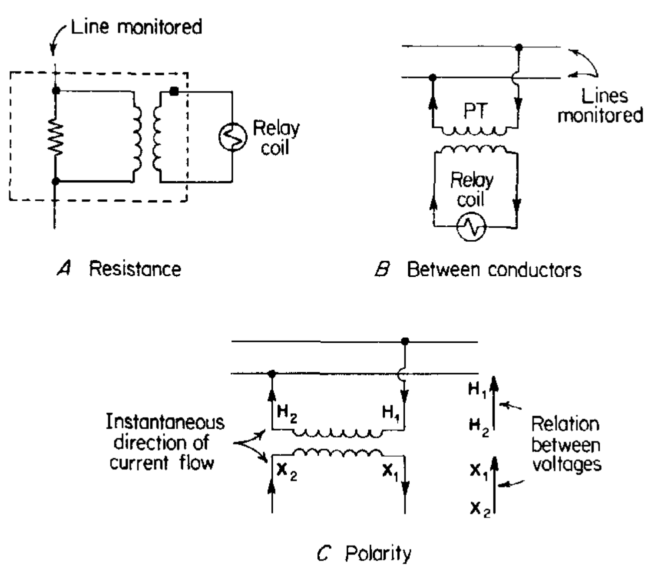

Figure 9.42.—Potential-relaying connections.

2. *Accuracy.* Ratio and phase-angle errors of standard PT's are usually so small they can be neglected, and any standard transformer is satisfactory as long as it is used within its thermal and voltage limits. If the transformer load is within rated burden, the transformer is suitable over the range from zero to 110% of rated voltage. Regardless, standard accuracy classes do exist for PT's, ranging from 0.3 to 1.2. These values represent the percent ratio corrections to obtain true ratio.

3. *Burden.* The burden of a PT, or thermal burden limit, is expressed in voltamperes. It is usually sufficient to add the voltampere ratings of parallel loads arithmetically to obtain a total voltampere burden. Accuracy is usually satisfactory at burdens well below rated, but the transformer voltampere rating should not be exceeded.

4. *Fusing.* In some instances, fusing the primary of a PT is not advisable, especially when the protective-circuitry's function is to sense a critical overvoltage condition (for instance, monitoring the voltage across a grounding resistor). Yet when the PT is connected line to line, it must be protected in case of PT failure or secondary conditions that will lead to failure. General practice is to use current-limiting fuses, sized to the transformer full-load rating and installed in the primary circuit between each ungrounded conductor of the system. Fuses are preferred over circuit breaker primary protection because the latter is accessible for manual tripping. A major use for PT's in mine systems is to supply control power to protective circuitry; secondary protection in this case is unnecessary. For other loads such as branch circuits for 120-V convenience outlets, the additional branches should be fused or protected by molded-case circuit breakers, the latter being general practice.

Alternating Current Differential Relaying

In *differential* relaying, a relay is operated by the vector difference of two or more actuating quantities, and relay pickup is determined by a difference threshold (*10*). Most applications of this scheme are of the current differential type. A basic circuit is shown in figure 9.43*A*, where the dashed portion represents the area to be protected. Two matched CT's are interconnected, and an overcurrent relay is inserted between them. Under normal conditions, or even when a fault occurs outside the protected zone, the CT secondary currents will circulate and not flow through the relay coil. However, if the current in both CT primaries becomes unbalanced, current will flow through the relay in proportion to the vector difference of the current entering and leaving the protected circuit (fig 9.43*B*).

A problem with this basic circuit is that CT's are very difficult to match; on identical units, the same primary current will not always give the same secondary current (*7*). Thus, the relay must be set so that it does not pick up on maximum error current between the CT's. An approach that usually overcomes the mismatch problem is the percentage differential connection. As illustrated in figure 9.43*C*, the main change is that the relay is now an overcurrent current-balance type (*10*). The differential current required to operate the relay is a variable quantity because of the relay restraining coil, and it offsets errors in the actuating sources.

Direct Current Connections

In addition to their popularity with ac systems, direct relaying and potential relaying are also the two most used protective relaying connections for dc systems. Direct relaying (fig. 9.44) consists of a dc overcurrent relay connected to a resistance shunt. The relay operating coil is matched to the shunt voltage at the desired pickup level (shunt full-load current rating usually gives 50 or 100 mV across the shunt). For low-current applications such as sensing dc current in a grounding conductor, current-relay operating coils are sometimes inserted in-line with the monitored conductor. Potential relays are also directly connected with the coils between the conductors of interest. Resistive dividers are at times employed to drop the dc system voltage down to the coil rating (as discussed in chapter 5).

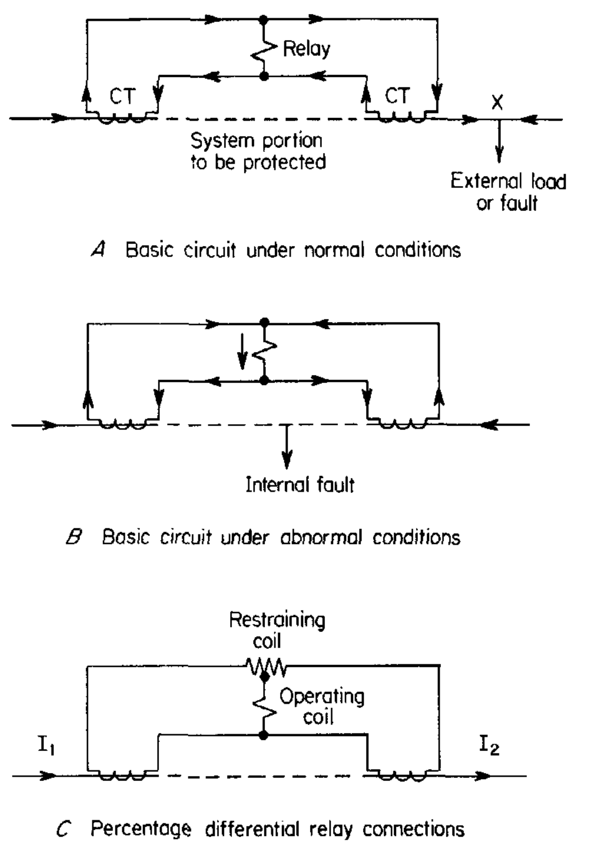

Figure 9.43.—Differential-relaying connections.

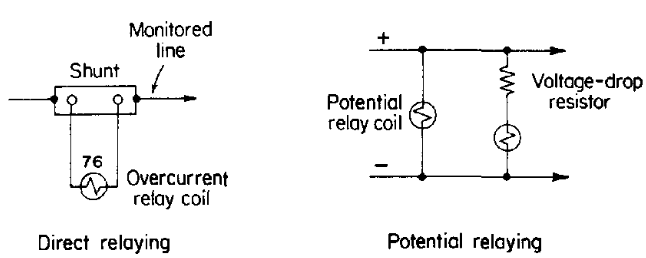

Figure 9.44.—Dc direct-relaying connections.

the switch mechanism. Precautions must be observed when using or considering these devices, and these are discussed in chapters 12 and 13.

RELAYS

Relays perform a major role in power-system protection, where their purpose is to detect voltage and current anomalies. They normally receive information about system conditions through transformers or resistors, which reduce system parameters down to levels that the relays can handle. Upon detection of a problem, a relay operates to supply or remove control power to the shunt or UVR tripping elements of the switching apparatus.

Because of their function, relays are sometimes called sensing devices. While transformers might also be considered sensing devices, their function in protective relaying is solely as transducers.

There are four basic relay types: thermal, electromagnetic attraction, electromagnetic induction, and static. (D'Arsonval movements are actually considered another relay type, but their operation is completely covered in chapter 5). The first three are electromechanical devices, and the following paragraphs will present their operation. Static or solid-state relays are discussed in chapter 12, because of related content.

Relay Terminology and Types

When a relay operates, it is said to close or open its contacts (9). Most relays are restrained by spring control and assume a specific position, either open or closed, when deenergized: hence there is a normally closed or *NC* contact and a normally open or *NO* contact. Symbols for both situations are shown in figure 9.29.

When a relay operates to open *NC* contacts or close *NO* contacts, it is said to *pick up* the contacts, and the smallest actuating quantity to cause contact operation is referred to as the *pickup value*. When a relay operates to close *NC* contacts or open *NO* contacts, it is said to *reset* or drop out, and similarly, the largest actuating quantity to cause reset is the *reset value* of the relay. When the relay is deenergized to reset, the reset value is almost always greater than zero and is often specified as a percentage of normal operation. Most relays have adjustments or tap settings to adapt them to as wide an operating range as possible.

The word describing relay operation has a formal meaning; for example, *overvoltage* relays, *overcurrent* relays, *overtemperature* relays, and so forth. Here the suffix refers to the actuating source (voltage, current, etc.), and the prefix "over" means that the relay picks up to close a set of *NO* contacts (or open *NC* contacts) when the actuating quantity exceeds the magnitude at which the relay is adjusted to operate. Similarly, undervoltage, undercurrent, and undertemperature relays reset to close *NC* contacts (or open *NO* contacts) when the actuating quantity decreases below a predetermined level. Some relays have both "over" and "under" functions (*7, 10*).

Even with these definite meanings, common usage of relay terminology is rather straightforward. *Pickup* is used to refer to the point where the relay changes from its normal state to indicate a malfunction, while *reset* implies that the relay returns to its normal position. The normal position may occur when the relay is energized or deenergized and depends on the application.

Relays designed for protective circuits are usually provided with some means of visual indication that a specific relay has operated to trip a circuit breaker. These operation indicators or targets are often brightly colored and are operated mechanically or electrically.

Specific relay types have been developed to meet special or general system-protection needs. Thermal relays serve directly or indirectly to measure power-system temperatures. Electromagnetic-attraction relays are used to instantaneously detect voltage and current changes. Electromagnetic-induction relays allow a time delay between relay detection and contact action. Directional relays can sense the direction of current flow.

Thermal Relays

Thermal relays most commonly employ bimetallic-driven contacts with an operation similar to that described for the molded-case circuit breakers. Another approach is to use ambient temperature, as in the temperature-monitoring protector shown in figure 9.30. This is a sealed bimetallic thermostat that opens or closes at a specific temperature; it can be used, for example, to sense motor overtemperature if mounted against the end turns of a motor winding.

Yet another bimetallic approach is to employ a heater element within the relay enclosures, connected in series with the circuit under consideration, as illustrated in figure 9.31*A*. The relay trip point for opening or closing the contacts is expressed in amperes, but is also a function

Figure 9.29.—Relay contact symbols.

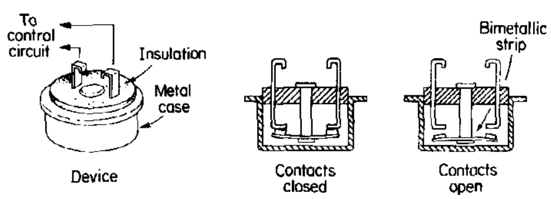

Figure 9.30.—Temperature-monitoring protector.

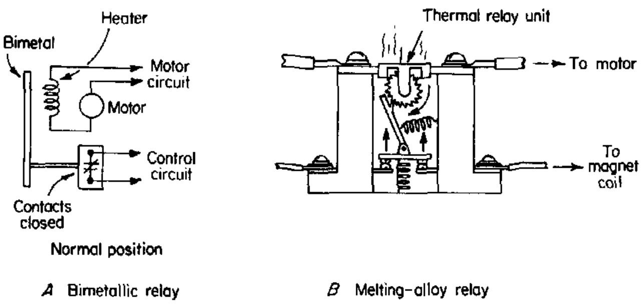

Figure 9.31.—Electromechanical-thermal relays.

KINDS OF PROTECTION

Several relaying terms describe the protection required in many mine power systems:

1. Undervoltage,
2. Overload (sometimes called overcurrent),
3. Short circuit,
4. Ground overcurrent (or ground fault),
5. Ground continuity, and
6. Overtemperature.

Classifications such as these are known formally as *kinds of protection*. The first five are necessary protection on all portable and mobile mining equipment, although exceptions are provided within Federal regulations (*17*). This section expands the basic relaying material by describing how each kind of protection is used in the mine power system. The content is mainly pointed at high-voltage, three-phase ac mining systems and in general is restricted to relaying external from circuit breakers. Accordingly, these kinds of protection imply the following parameters:

- Line-to-line voltages for undervoltage,
- Line overcurrent for overload,
- Three-phase or line-to-line faults for short circuit,
- Faults causing zero-sequence current for ground overcurrent, and
- Grounding-conductor resistance for ground continuity.

Even though overtemperature is listed as item 6, it is usually applied to protect a specific component; thus, it will be discussed in chapters 12 and 13.

Control Wiring

Figures 9.45 and 9.46 show simplified diagrams of typical control wiring interconnections among the power source, relay contacts, and circuit breaker tripping elements. In both diagrams, a potential transformer supplies 120 Vac with its fused primary connected line to line.

Figure 9.45 illustrates cases where the tripping element is a UVR. The contacts can either reset to remove power from the coil (contacts in series with coil) or close to short it out (contacts parallel the coil). In the latter case, it can be seen that a resistance is placed in series with the contacts. In fact, the UVR itself will trip the breaker if control voltage is decreased in the range of 40% to 60%.

Basic shunt-tripping connections are given in figure 9.46, where power is supplied to the element to cause tripping. Here the contacts for the various protective relays are paralleled, and the combination is in series with the trip coil; closure of any contact trips the breaker. It should be obvious in figure 9.46*A* that the power causing tripping is ac, while in figure 9.46*B*, it is dc. The capacitor in the dc circuit is employed for energy storage to augment tripping if there is a drop in the PT primary voltage when a relay contact closes.

Phase Protection

Phase protection by protective relaying can be overload, short circuit, or both, depending upon the relays used. (Molded-case circuit breakers afford this same flexibility depending upon the internal tripping element used.) Time-delay relays are employed for overload, with instantaneous units for short circuit. Figure 9.47 illustrates the combined protection for three line conductors, using an

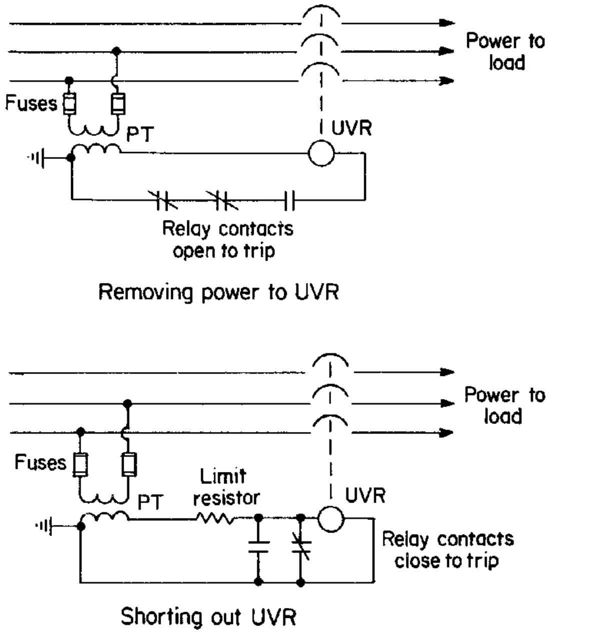

Figure 9.45.—Typical control wiring for UVR.

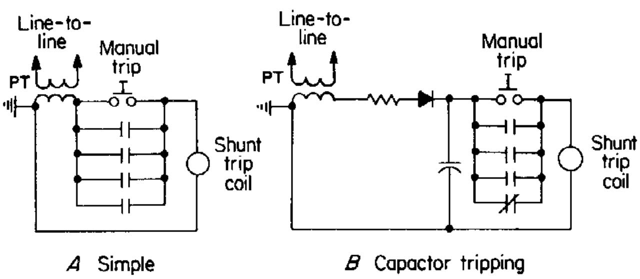

Figure 9.46.—Typical control wiring for shunt-tripping element.

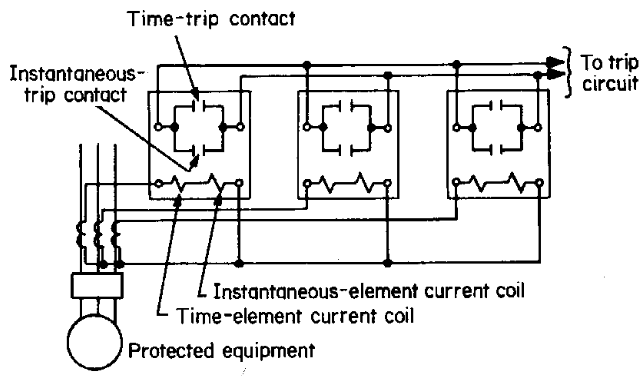

Figure 9.47.—Three-phase overcurrent and short-circuit connections.

induction-disk relay (7). The three current transformers are placed in wye, driving the wye-connected operating coils. The time-delay element is set on as low a tap setting as practical, enabling protection for sustained moderate overloads. The instantaneous units, however, are set to pick up on a current value slightly higher than the maximum peak load, thereby affording protection against short circuits or enormous overloads.

The device numbers that were presented in chapter 4 are used extensively to describe the relay function. The number 51 signifies time-delay relays for ac overcurrent, and 50 is used for instantaneous devices. A combination instantaneous and time-delay ac overcurrent relay is often noted by 50/51.

If the connections are as shown in figure 9.47 and transformer phase-angle errors are ignored, the secondary currents of each CT are in phase with the primary currents, and each relay responds to abnormal conditions for its respective line (*10*). This also applies to figure 9.48*A* where the 50 elements are omitted. If line currents are approximately balanced, short-circuit protection for all three lines can also be provided with an open-delta connection, as in figure 9.48*B* (*10*). As might be expected, this approach is not as precise as the straight wye connection, but a third overcurrent relay may be inserted in the CT common connection for backup protection (see chapter 5 for a similar discussion on instrumentation). An advantage of the wye connection that is lost when the open-delta approach is use is the ability to sense zero-sequence currents through residual relaying. Thus, two current transformers are rarely applied as the only means of circuit protection.

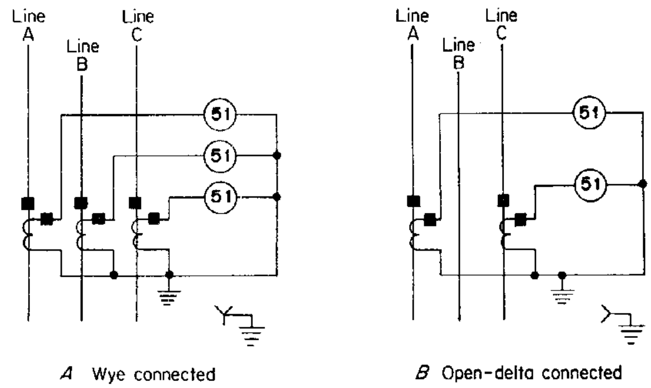

Figure 9.48.—Two CT approaches.

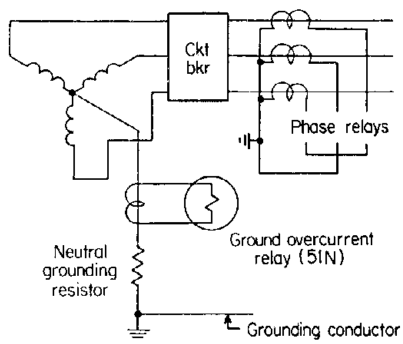

Figure 9.49.—Neutral-resistor current-relaying scheme.

Ground Overcurrent

To this point, the chapter has basically considered power-conductor protective relaying, and the extremely important subject of ground-fault protection has received only terse reference. Various relay configurations may be utilized to provide ground-overcurrent protection, some of which are quite elaborate. However, nearly all these techniques fall into one of five broad classifications (7): direct relaying, potential relaying, residual connection, zero sequence, and broken delta. Direct relaying, potential relaying and zero sequence are frequently used in resistance-grounded mine power systems, with zero-sequence relaying being the most popular. Unless otherwise noted, the following discussion will assume that the system is resistance grounded.

The point of application for direct and potential ground-fault relaying is usually restricted to the system neutral point or grounding resistor, whereas the other three techniques can provide protection anywhere in the system. Usually a combination is needed for complete assurance of clearing all ground faults.

Direct Relaying

The simplest form of ac ground-fault protection is direct or neutral relaying. A current transformer is placed about the grounding conductor and located between the neutral point of the source transformer and the grounding resistor, as shown in figure 9.49. The grounding conductor acts as the primary winding of the CT, while the secondary winding is connected to the ground-overcurrent relay (51N). If the current through the grounding conductor exceeds a predetermined value, the relay acts to trip the circuit breaker.

In many situations, some ground-current flow is normal, due to system unbalance, capacitive-charging currents, or inductive-coupling effects, and so the circuitry must be adjusted to pick up only when the normal level is exceeded. As will be seen, the pickup point should always be less than the system current level.

The major disadvantage with this direct relaying method is that, should the grounding resistor or the grounding conductors become open, it will never detect any ground-current flow. The system will continue to operate with no abnormal indication, and then the system can become essentially ungrounded, posing a personnel hazard especially where resistance grounding is mandatory (*12, 14, 18*). Accordingly, although the technique does find application on some portions of the ground system, some States do not allow its use on substation grounding resistors, even for a second line of defense.

Potential Relaying

Potential relaying, as shown in figure 9.50, is often used as a sole means of ground-fault protection at the surface substation and can also be used as a backup to other protection schemes at a unit substation or power center. With this method, the primary winding of a PT is connected across the neutral grounding resistor, while the secondary winding is connected to a voltage-sensing ground-trip relay (59G). If current flows through the grounding conductor, a voltage is developed across the grounding resistor. When the voltage rises above a preset level, the ground-trip relay causes the circuit breaker to trip.

Unlike direct relaying, potential relaying has the advantage of being able to detect a ground fault with the neutral grounding resistor in an open mode of failure. However, if the grounding resistor fails in a shorted mode, potential relaying is rendered inoperable.

Zero-Sequence Relay

Zero-sequence relaying, also termed *balance-flux* relaying, is the most reliable first defense against ground faults in mine power systems. As shown in figure 9.51, the circuitry consists of a single window-type CT; the three line conductors are passed through the transformer core, forming the CT primary (*6*). On four-wire systems that have line-to-neutral loading, the neutral conductor must pass through the window but the grounding conductor must not. However, such loading is not allowed in mining, so only the line conductors can be used.

In a symmetrical phase set, the vector sum of the three currents in the primary circuit will be zero, and no current will flow in the secondary. During a line-to-neutral fault, the phase unbalance will induce a current flow in the CT secondary, proportional to zero-sequence current. If the secondary current exceeds the relay pickup setting, circuit breaker tripping will be initiated.

This phenomenon can be easily demonstrated. In terms of symmetrical components, the three phase currents can be written as

$$I_a = I_{a1} + I_{a2} + I_{a0},$$
$$I_b = I_{b1} + I_{b2} + I_{b0},$$
$$I_c = I_{c1} + I_{c2} + I_{c0}.$$

The primary current, I_{prim}, for the CT can be considered as the vector sum of the three line currents, which is also the current flowing through the grounding conductor (or that external to the CT window). Therefore,

$$\begin{aligned} I_{prim} &= I_a + I_b + I_c \\ &= (a^2 + a + 1)I_{a1} + (a^2 + a + 1)I_{a2} + 3I_{a0}. \end{aligned}$$

Because $a^2 + a + 1 = 0$,

$$I_{prim} = 3I_{a0}.$$

Thus, for an unfaulted or balanced condition,

$$I_{prim} = 3I_{a0} = 3I_{b0} = 3I_{c0} = 0,$$

and no current is induced in the CT secondary. However, for a line-to-neutral fault involving phase a, the primary current equals the ground-fault current, I_f, or

$$I_{prim} = I_f = 3I_{a0},$$

and a current is induced in the CT secondary to initiate tripping.

Zero-sequence relaying is not affected by CT error, and therefore gives very sensitive tripping. The scheme is widely applied in mining at all voltages.

Residual Relaying

Residual relaying (fig. 9.52) is used in conjunction with CT's placed about the phase conductors. This relaying technique is used primarily on high-voltage distribution circuits that require CT's and inverse-time relays for overcurrent protection. The CT's and the phase-overcurrent relays are both connected in a wye configuration. The ground-fault or residual relay is connected between the neutral points of the CT's and the relays.

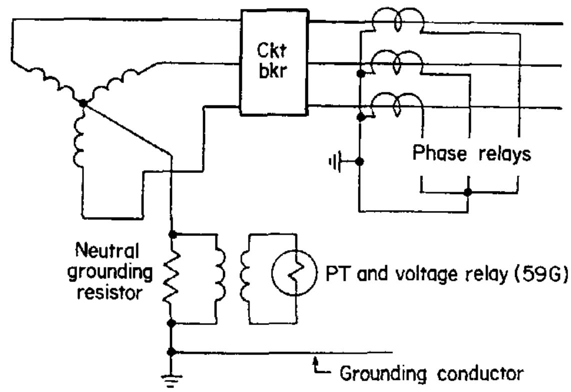

Figure 9.50.—Neutral-resistor potential-relaying scheme.

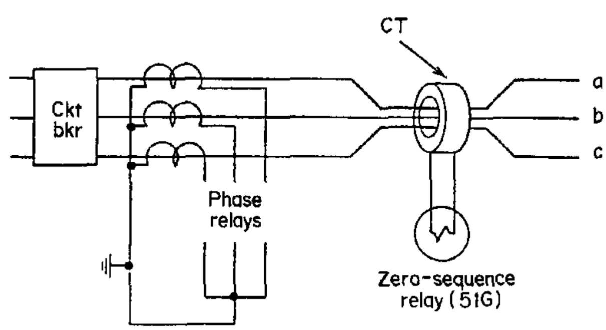

Figure 9.51.—Zero-sequence ground relay connections.

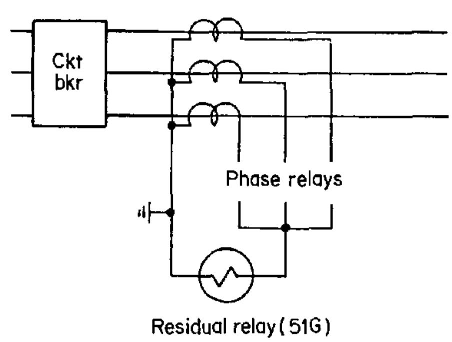

Figure 9.52.—Ground relay in residual connection.

As the current flowing through the residual relay is in proportion to the sum of the line current, the principle of operation of the residual method is similar to that of zero-sequence relaying. However, because of errors due to CT saturation and unmatched characteristics, residually connected relays are often subjected to nuisance tripping. Hence, they cannot have sensitive or low pickup settings. This arrangement will not always provide consistent repetitive tripping at the required tripping levels for mine power systems.

Broken-Delta Relaying

Broken-delta ground-fault protection is somewhat similar to the residual method, except that the three CT's are wired in series, as shown in figure 9.53. The resulting output voltages from the transformers form a closed delta if the load is balanced (fig. 9.53*A*). An unbalanced condition, such as a line-to-neutral fault, will cause the formation of an open delta (fig. 9.53*B*), and the resulting voltage causes current through the relay operating coil. The broken delta is sensitive to any unbalance, but the zero-sequence relay operates only on faults causing ground-current flow (*6*).

Ground-Check Monitoring

The effectiveness of all the ground-relaying methods depends upon the integrity of the grounding system. A ground-check monitor is the device used to continuously monitor the grounding connections to verify continuity (*2–3, 9*). If conductivity is inadequate, it is the function of the monitor to trip the circuit breaker that feeds power to the system experiencing defective grounding.

As shown in chapter 7, the grounding conductor is not essential for mining machine operation, but it is imperative for personnel safety. The ground-check monitor enhances safety by making sure, via the ground connections, that the equipment frames are at near-neutral potential. Again, the maximum allowable frame potential to earth is 40 V on low and medium voltage and 100 V for high-voltage systems.

Ground-check monitoring is an extensive subject, which can only be outlined here; the references listed at the end of the chapter can be consulted for more detail, particularly references *2–3*, *9*, and *11*.

Although there are potentially numerous ways of monitoring ground continuity, only a few are considered practical to construct or have the required high reliability (*9*). These techniques can be divided into two general classifications: pilot monitors and pilotless monitors. Monitors in the mining industry use these techniques but are also referred to by different names, which will be discussed later.

Pilot

Pilot monitors use a pilot or ground-check conductor (see chapter 8) to perform the task and are of three general kinds: series loop, transmitter loop, and bridge (*9*).

In the most common series loop circuit, a power supply, the relay operating coil (instantaneous contacts or a minimal time delay), the pilot conductor, and grounding conductors are connected as shown in figure 9.54 (*9*). If the pilot or grounding conductor breaks the loop or if the power supply fails, the relay contacts will reset. The circuit can be either ac, using 60-Hz line frequency, or dc.

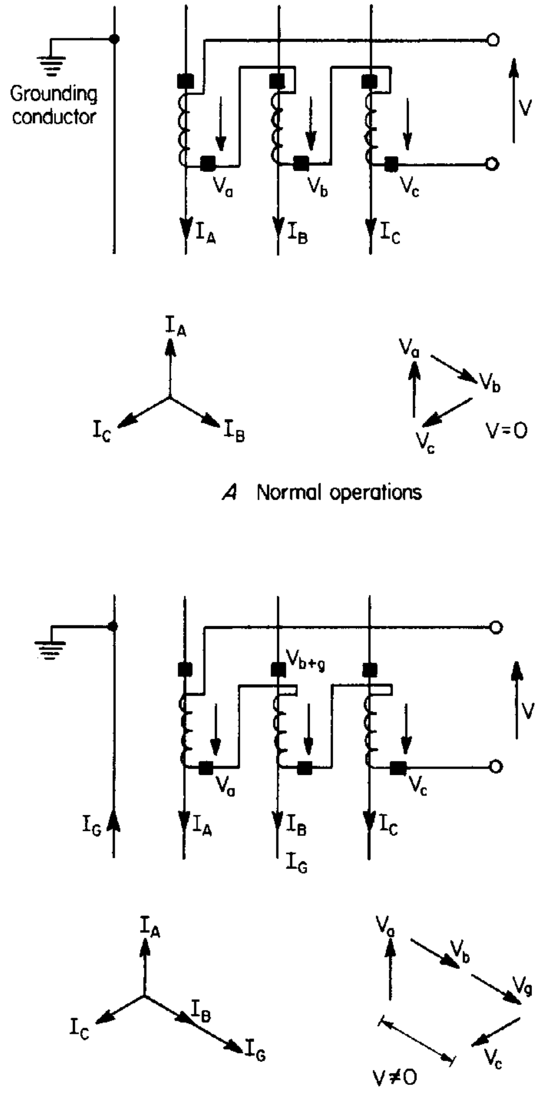

Figure 9.53.—Broken-delta protection.

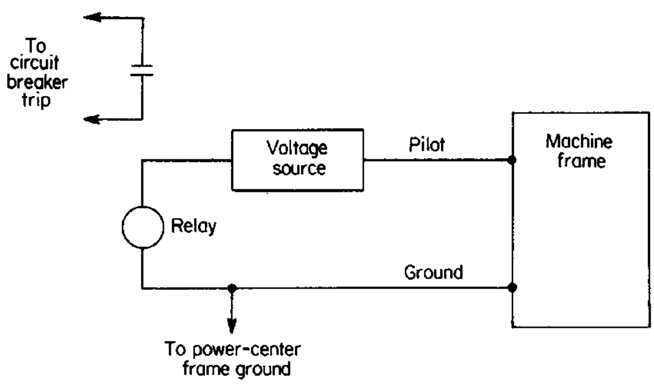

Figure 9.54.—Series loop ground-check monitor.

The transmitter loop concept is basically the same as the series loop, except the voltage source is installed in the machine (fig. 9.55) (*9*). Here, the source must receive its power from the machine and the relay cannot pick up until the circuit is energized; therefore, the monitor must be temporarily bypassed in order to close the circuit breaker.

Bridge-type monitors use the series combination of the pilot and grounding conductors as one leg of a Wheatstone bridge. Figure 9.56 shows a general circuit, where Z_3 is used to balance the bridge for a specific pilot and grounding-conductor impedance (*9*). Bridge output is sometimes amplified, but with or without amplification, the relay resets if a preset impedance level is exceeded. Bridge input can be 60-Hz ac, dc, or an audio frequency such as 5,000, 2,500, or 900 Hz (*2, 9*).

Pilotless

Pilotless monitors, as the name implies, do not use the pilot conductor. Instead, as shown in figure 9.57, an audio signal is placed on the phase conductors through a filter and removed at the machine through filters, completing its path back to the source in the grounding conductor (*9*). Instead of the filters, some models use coils similar to CT's to send and receive the audio signal. Between the grounding conductor and the power-center frame is a saturable reactor, which shows high impedance to the monitoring frequency. Its purpose is to restrict the monitoring signal to the intended path. This presents problems in coupler grounding, as the coupler metallic shell is commonly grounded to the grounding conductor as well as physically connected through its receptacle to the power-center frame. The grounding conductor must be isolated from the shell ground so that the reactor will not be bypassed.

Problems and Requirements

All of the basic techniques are plagued by some disadvantages, and the attempt to achieve a reliable monitor has been a perplexing experience for the mining industry. The reason is tied to the basic character of the mine power system: figure 9.58 provides a conceptual view of some difficulties that can arise (*2*). One of the more pronounced problems is the parallel ground paths established by contact through the mine floor or through grounding conductors on other machines. The alternate ground paths may have a resistance as low as the grounding conductor, but in the majority of cases these are very temporary in nature and thus cannot be relied upon. Stray ac and dc and induced ac are an ever-present problem in many mines. With dc rail haulage, for example, substantial direct current can stray from the rail when a poor bond is present and end up flowing in the ac ground system. If the cable has a G–GC or SHD–GC configuration (chapter 8) or the system current is unsymmetrical, ac can be induced in the grounding conductors. At times, this current can be of significant magnitude, not only on underground loads but especially on surface excavating machinery. Another problem results from trailing-cable deterioration; in a splice, for instance, there is a chance the ground-check conductor could short to the grounding conductors. Power-system transients, occurring from lightning or switching surges and wiper contacts on reeled units, present additional problems, and all of these situations can affect ground-check monitors.

MSHA has established several guidelines for low-voltage and medium-voltage monitors that must be met before a monitoring device is approved. In these regulations, two monitors are recognized: a continuity type and an impedance type. A continuity monitor is one that meets the general requirements of 30 CFR 75.902 (*17*). It monitors only the grounding-conductor continuity and does not measure impedance; pilotless techniques fall into this

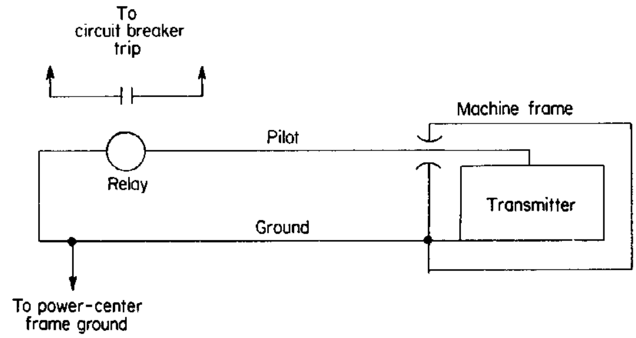

Figure 9.55.—Transmitter loop ground-check monitor.

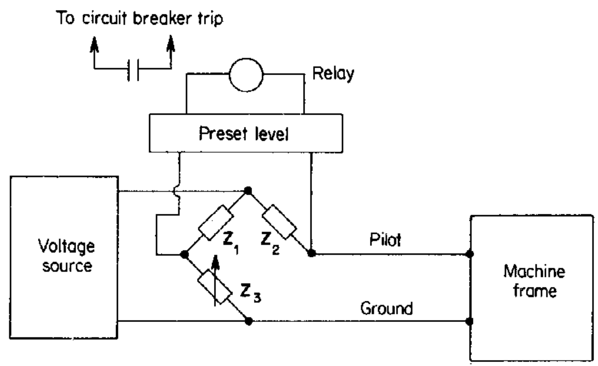

Figure 9.56.—Bridge-type ground-check monitor.

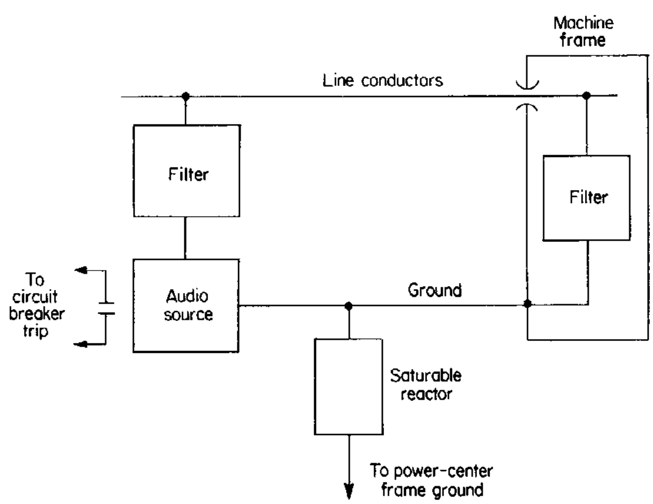

Figure 9.57.—Pilotless ground-check monitor.

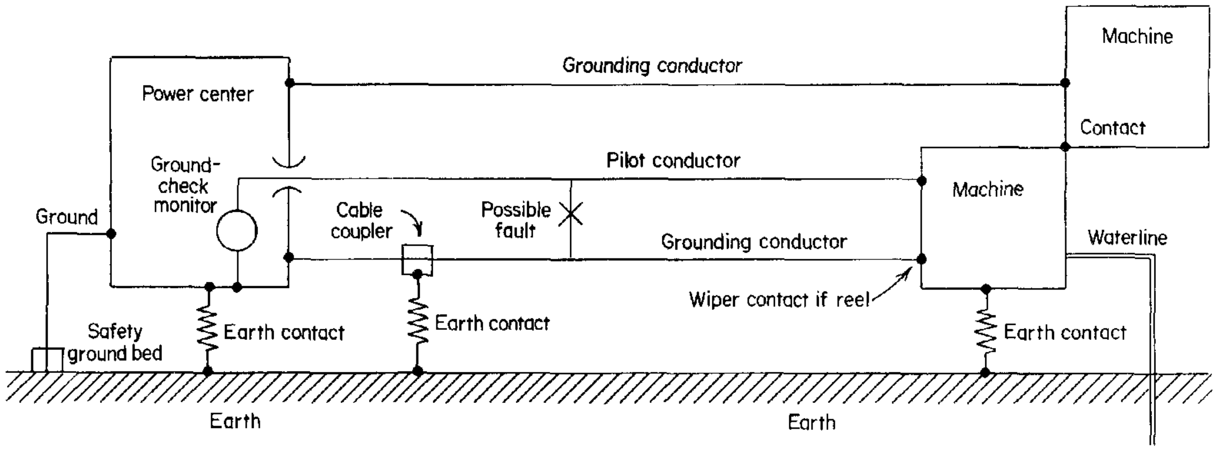

Figure 9.58.—Some difficulties associated with ground-check monitoring in mining.

class. An impedance monitor requires a pilot conductor and monitors any change in the impedance of the loop formed by the pilot and grounding conductors. The bridge technique is therefore an impedance monitor. The relevant requirements for both monitor types are as follows (*2, 9, 11*):

1. The monitor must be "fail-safe"; in other words, the failure of any component, other than the relay contacts, must make the trip-circuit contacts reset. (The relay must pick up its contacts when in normal operation.)
2. The monitor must not trip when (a) input voltage is varied by +15% or −20%, or (b) 5.0 V minimum to 25 V maximum, 60 Hz, or 10 A dc is introduced in the grounding circuit. These conditions are intended to ensure that the unit stays operational, even when under the influence of power-line fluctuations, stray currents, or induced currents.
3. The *open-circuit monitor voltage* cannot exceed 40 V rms.
4. When detecting grounding-conductor continuity, (a) continuity monitors must trip the circuit breaker if the grounding conductor is broken at any point regardless of low-impedance parallel paths (75 Ω is considered an open connection), and (b) impedance monitors must trip the circuit breaker if the impedance of the grounding circuit, external to the grounding resistor, increases to cause a 40-V drop under fault conditions (or, by Ohm's law, 1.6 Ω for a 25-A limit).
5. Filters must not cause a personnel hazard during normal operation or when the grounding conductor is opened.
6. The maximum time delay for contact reset, after an inadequate ground is detected, cannot exceed 250 ms.
7. When two or more monitors are operated in parallel, no interference can occur to cause incorrect tripping.

At this writing, similar guidelines for high-voltage ground-check monitors have yet to be established. However, by 30 CFR 75 and 77 (*17*), the maximum open-circuit monitor voltage is established at 96 V rms. Furthermore, *continuity monitors* must adhere to item 4a above, with impedance monitors conceivably tripping if the grounding circuit impedance causes a 100-V drop.

Advantages and Disadvantages

As mentioned, each basic ground-check technique has inherent advantages and disadvantages. A listing of these is rather informative (*2–3, 9*).

Other than simplicity, the advantages of series loop circuits are minimal. Designs using a dc source are immune to stray ac but not dc. Further, ac monitoring can be subject to *nuisance tripping* by stray dc current that offsets the signal current. However, when the relay operating coil is isolated by a blocking capacitor, an immunity to stray dc is gained. In any case, the relay coil must have a very low impedance to be sensitive to grounding-conductor impedance. Two disadvantages of the circuit are substantial: Parallel paths and grounded pilot conductors easily negate its operation.

The advantages and disadvantages of the transmitter loop technique are basically the same as for series loop, except the circuit can detect pilot-to-ground shorts.

Simple bridge monitors have the same problems as series loop models, but they are very sensitive to changes in grounding-conductor and pilot-conductor impedance. The more elaborate designs can be made immune to ac and dc stray currents, yet even these sometimes cannot distinguish between a sound grounding conductor and an illegal parallel path.

In general, pilotless monitors are superior to pilot designs, except they are obviously more expensive. Because the ground-check conductor is not needed, all associated problems are removed. The most elaborate models can distinguish parallel paths and are immune to stray currents; however, the simple designs are vulnerable to both. Some pilotless designs also have the advantage of being adaptable to pilotless or pilot use, depending on the need.

Pilot-conductor ground-check monitors can serve the very important function of safety interlocking. This feature is required on many portions of the mine power system and is used on almost all high-voltage systems. An example of interlocking is shown in figure 9.59. The loop circuit sensed by the monitor not only includes the pilot and grounding conductors, but can also involve a series of contacts and switches. Whenever one of these is opened,

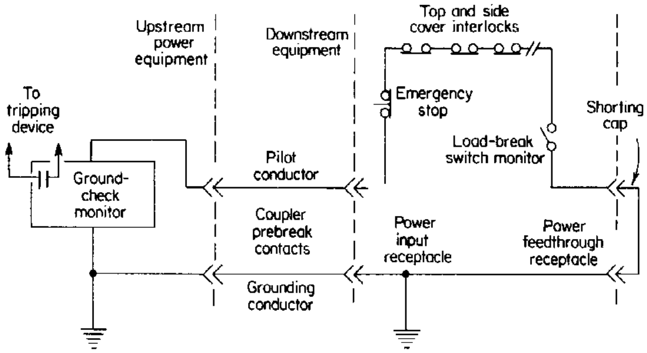

Figure 9.59.—Pilot interlocking circuit using ground-check monitor.

the continuity of the ground-check circuit is compromised, the monitor resets its rely, and the circuit breaker trips the power to that section. Safety devices incorporated in the interlocks include top and side cover switches, which open if a cover is removed, emergency-stop switches, the prebreak contacts on couplers, and pilot-break monitors on disconnect switches.

When such sophisticated interlocking is required, pilot-type monitors must almost invariably be used. For instance, pilotless models cannot be employed when in-line cable couplers are on the circuit. When there are gear-mounted couplers on the power equipment in the monitored circuit, pilotless monitors may be used if they are wired into the ground-check and grounding contacts of the couplers, so that they will trip the circuit breaker prior to plug removal. This applies to almost all low-voltage and medium-voltage monitored circuits in mine distribution.

ARRANGEMENTS FOR MINING

There are two groups of protective-relaying equipment within a typical power system: primary and backup. Primary relaying has the goal of clearing all faults and overloads, and aims to isolate offending power-system segments with minimum interruption to the system balance. Backup relaying operates only in the event of a primary relaying failure; its action is only for uncleared faults. In mining, be it overload, short-circuit, or ground-fault relaying, both groups are used extensively to the point of redundance.

Zones of Protection

Protection to the entire system is principally related to primary relaying and is accomplished by establishing zones of protection. Each zone has an associated circuit breaker and fusible disconnect, or fuses with the required sensing devices, and adjacent zones overlap (*10*). If a failure occurs within an individual zone, only the switching apparatus within that zone should open. If there were no overlap, there could be an unprotected region in the system in some situations; a failure within this area would produce no safety tripping. Although such a situation seems unlikely in theory, in practice it does occur and can be caused either by oversight or ignorance on the part of an engineer. It results in broad outages to the system, rather than restraining the problem within a specific zone.

The coordination of protective relaying between the zones is extremely important. The aim is to isolate faults downstream from the power source without disturbing upstream zones. Unfortunately, obtaining this coordination is perhaps the most outstanding problem of relaying.

By introducing the general arrangements of protective relaying in resistance-grounded mine power systems, this section actually serves as a transition between the basic relaying principles and chapter 10, where there is a more detailed and specific analysis. The objective here is to show how primary and backup relaying, zones of protection, and coordination are utilized in both surface and underground mines.

At all levels of the mine power system, protection against short circuits, overloads, and ground faults holds priority. Because the majority of failures in mining involve line-to-neutral faults, ground-fault protection commands special interest, but this does not negate the need to establish adequate line-conductor relaying.

Coordination

The objective of coordination is to determine the optimum characteristics, ratings, and settings for the protective-relaying devices (*7*); consequently, fault analysis of the system must be involved. The two common coordination schemes that are utilized are pickup setting and time. With the first, relay pickup settings for a specific actuating quantity are set at progressively higher values from the loads to the power source, such that a higher level of the actuating quantity is required to trip the circuit breaker in an upstream zone. For the time coordination of a specific actuating quantity, pickup settings throughout the system will generally be the same, but the operating times to achieve contact closure at or above pickup are set progressively longer toward the source. One technique or the other may be applied to provide coordination between zones in the mine power system; at times, a combination might be needed. The design of this protective-relaying system can be a substantial problem, as exemplified by the fact that many engineers consider protective relaying more an art than a science.

Ground-Fault Protection

Except for capacitive ground current (see chapter 11), ground-fault current magnitudes are limited by grounding resistors. As mentioned in chapter 7, the maximum current is 25 A at low and medium voltages, but the level is seldom limited below 15 A, whereas it is very rare for a 50-A limit to be exceeded on high-voltage grounding systems, with 25 A seen on many systems. Thus, ground-fault current is probably close to constant throughout the system. The use of delta-wye, delta-delta, or wye-delta transformers enhances this nearly constant current situation. At each transformer step, a new or separately derived ground system is produced, each with its own ground resistor. Because the transformer configuration blocks zero-sequence components, a ground fault on circuits connected to the secondary raises primary current, but the vectorial sum of the line currents is zero (*1*).

It can thus be seen that coordination of ground-fault relaying across transformers is unnecessary, and both primary and backup protection must be established for each derived grounding system. Selective coordination at each voltage level by pickup setting alone is normally impossible, and time settings must be relied upon when multistage protection is used (*1*).

It must be remembered that resistance grounding is used to reduce fault energy, frame potentials, or system potentials. If the first ground fault is not cleared, an alternate ground fault in another machine will be in a ground condition and not be limited by the grounding resistor (*1*). The result is the possibility of dangerous frame potentials and powerful intermachine arcing. Consequently, ground-fault relays must always be arranged to trip circuit breakers; fuses cannot be used.

Overloads and Short Circuits

The occurrence of a line-to-line or three-phase fault anywhere in the system can cause substantial outages if it is not cleared downstream. Because they are not restricted like ground faults, the anomalous positive-sequence and negative-sequence currents can be passed across transformers to higher levels of the system. The resulting wide-range problems are especially evident on radial distribution. System protection is usually coordinated by using instantaneous relays to adjust pickup, and the effort could involve fault currents from the machines to the substation. A maximum setting within an individual zone would be the minimum fault current at which thermal or mechanical damage to a protected device could occur.

The problems resulting from inadequate overload protection are just as widespread. In most cases this protection must be applied to the specific zone that the fuse or circuit breaker is protecting. For example, the pickup setting of a time-delay relay could be determined by the ampacity rating of the smallest conductor within the zone of the switching apparatus, whereas time settings might be employed to coordinate between zones. Overload protection can be critical in large underground mines with numerous sections; when several machines are operating simultaneously, the maximum current demand could overload an upstream conductor.

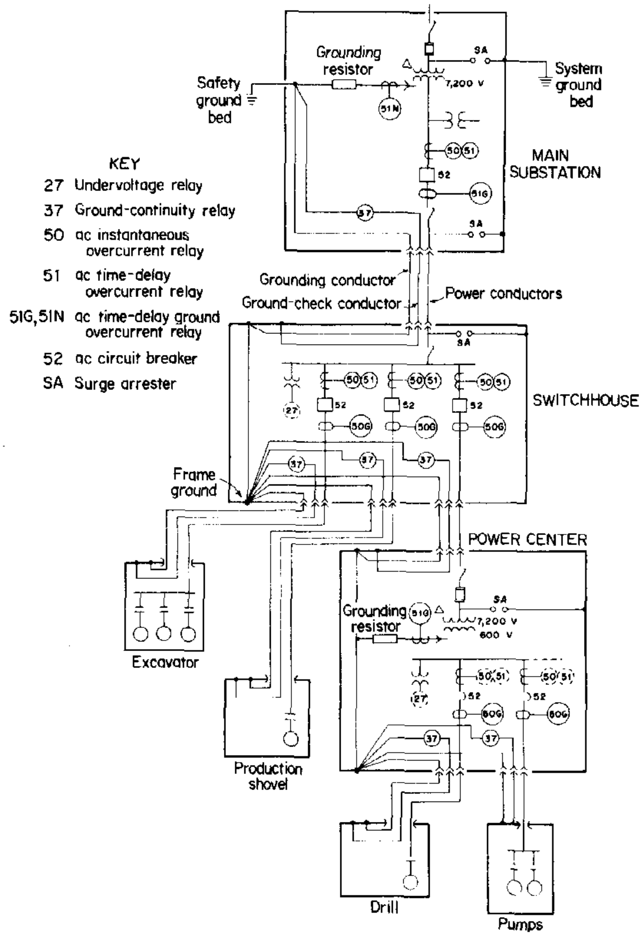

Figure 9.60.—Simple surface mine power system illustrating protective relaying.

Surface Mines

To illustrate the application of these general considerations, consider figure 9.60, a one-line diagram (including grounding) of a simple surface mine power system (*1*). The diagram is drawn to show the various combinations of protective circuitry that can be found in surface operations. Each protective device is installed to trip the closest circuit breaker (52).

In terms of ground-fault protection for the low-voltage machines, zero-sequence relaying (50G, usually instantaneous) establishes primary protection. Backup protection could be placed here by adding timed direct relaying (51N) about the grounding conductor between the grounding resistor and the transformer neutral. For high voltage, remembering that the transformer configuration blocks zero-sequence current, primary ground-fault protection is provided in the switchhouse, again by zero-sequence relaying (50G, instantaneous or minimum-time dial setting), establishing a zone of protection for each outgoing circuit (the excavator, loader, and ac power center). The time-delay zero-sequence relay (51G) in the substation can be considered to give both backup protection for the downstream 50G relaying and primary ground-fault protection for the zone between its location and the switchhouse. In any event, backup protection is the principal duty of the 51N direct relay about the neutral conductor of the substation. (N is used to signify neutral relaying, but G can be used.) All this relaying is normally coordinated by time settings, the pickup setting being specified as a percentage of the ground-current limit.

The use of the grounding-conductor direct relaying, shown in figure 9.59, is restricted to situations where the main circuit breaker that the relay trips sees total ground current. In the ac power center, this means that the backup relaying must cause tripping of both circuit breakers. Because here there is no main breaker, some States require potential relaying of the grounding resistor for backup. Potential relaying does give more safety than direct relaying, as previously mentioned.

Ground-check monitoring is illustrated in its most extensive form for surface mining in figure 9.59, where every grounding conductor in the system is measured. Ground-check conductors are shown connected to each monitor. In instances where pilotless devices are applied, ground-check conductors are not needed. In cases where conductors from the substation form an overhead ring bus, such as in some open pit operations, ground-check monitors are often not used because of the circular nature of the distribution. In any overhead distribution arrangement, monitors can experience extensive failures because of lightning strokes.

Overload and short-circuit protection in the high-voltage portion is provided by line-conductor CT's usually connected in wye to time-delay relays that are normally induction-disk types with both instantaneous (device 50) and time-delay (51) elements. Each 50/51 combination establishes a zone of protection downstream from its location. As shown in the power center, the same kinds of protection can be provided at utilization by low-voltage power circuit breakers in conjunction with external CT's and relays. In figure 9.60, these protection devices are indicated as numbers within dashed circles. Normally, they would be protected by molded-case units. Note that all 50/51 phase protection could also be done by fuses or fusible disconnects, but circuit breakers or power-driven load-break switches still are needed for ground-fault protection.

Figure 9.61 shows a three-line diagram of a typical molded-case arrangement. These components replace the 50, 51, 50G, and 27 devices associated with each low-voltage breaker in figure 9.60. The molded-case circuit breaker provides short-circuit protection through its magnetic trip units (not shown), with overload tripping given by the internal thermal elements. (Overload protection may not be required on low- and medium-voltage circuits; see chapter 10 for discussion.) The external protective circuitry is commonly a zero-sequence relay and pilot-type ground-check monitor, and the contacts of both trip the undervoltage release. Notice that here the ground-fault relay shunts the trip coil, while the ground-trip relay is in series with the trip coil. The UVR itself provides undervoltage protection. In this way all the essential kinds of protection are available for a resistance-grounded trailing-cable installation.

An undervoltage relay (device 27) is also shown in the switchhouse of figure 9.60, but this would not be required at this or any location provided that all equipment downstream has undervoltage protection.

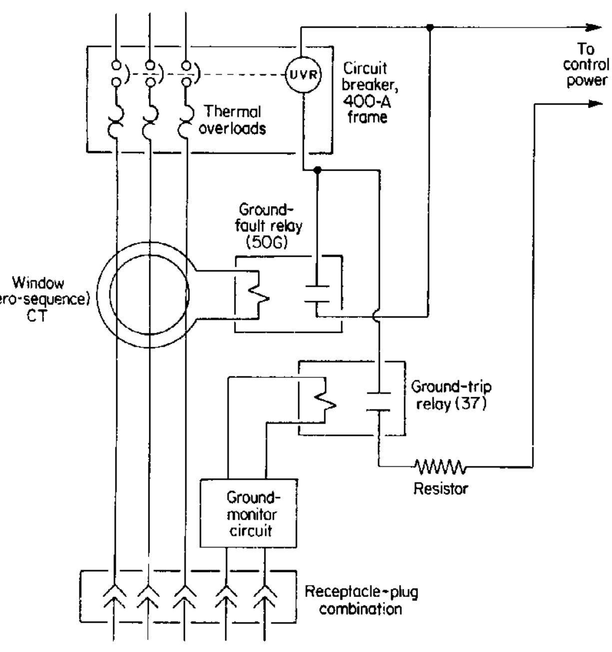

Figure 9.61.—Typical schematic for three-phase molded-case circuit breaker with ground-overcurrent and ground-check protection.

Underground Mines

For comparison, figure 9.62 illustrates a simple one-line diagram of an underground mine power system. The ac portion is very similar to the surface circuitry of figure 9.60, and the reasoning and arguments behind the protective devices are the same. There are differences, however, in the disconnect switch, the relaying in the substation, and possible extra outgoing circuits from the substation, but again, all these features are also possible in surface mines.

By law, some means of visible disconnect is required within 500 ft of the point at which power enters the underground workings, and a separate switch is shown in the diagram for this purpose. Actually, it is also advisable, if not required, to place visible disconnect switches on incoming high-voltage (distribution) circuits within all power equipment. (These have not been included in figures 9.60 and 9.62 merely to maintain clarity.) Consequently, it is not uncommon to find the first disconnect in the mine as part of a switchhouse.

Ground-fault backup relaying located in this substation is mainly by the potential technique. Here, the relay operating coil is placed directly across the resistor, and in addition to relaying, the transformer is used for resistor impedance matching for current limit. Although not shown, the outgoing circuit to the surface equipment has basically the same relaying as for underground (see chapter 13 for the implications of such loads).

Direct Current

Only one form of relaying for dc equipment is provided in figure 9.62; this is short circuit for rail haulage systems and consists of a dc overcurrent relay (device 76) driven by the voltage drop across an in-line shunt. For off-track dc machinery, the protective-relaying arrangement is directly tied to the power source. Five basic systems of dc ground-fault protection are presently being used in the United States:

- Diode grounding,
- Basic grounding conductor,
- Relayed grounding conductor,
- Neutral shift, and
- Differential current.

The grounding philosophy for all but the last was introduced in chapter 7. The neutral-shift and differential-current systems can be used only when the machine is powered from the output of a rectifier, such as that contained in a section power center, whereas the first three systems are more commonly employed when a trolley system is the dc source.

When dc equipment is powered from the trolley system, short-circuit and overload protection is normally provided by a dual-element fuse. The device is mounted in a holder or nip, which is clipped to the trolley wire. The other trailing-cable power conductor is connected to the rail. Fuses are rarely applied to trailing-cable protection in load-center rectifiers. Here, air-magnetic power breakers, molded-case devices, or dc contactors (see chapter 12) are employed. In practice, these are usually tripped only for short-circuit conditions, with overload protection not used.

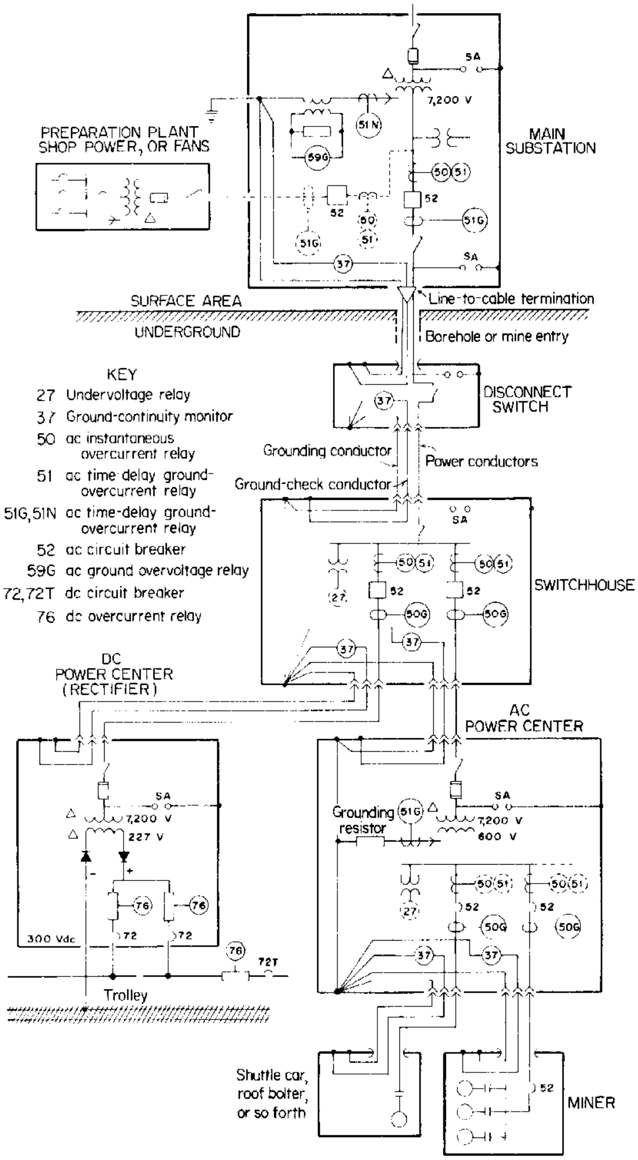

Figure 9.62.—One-line diagram of simple underground mine power system illustrating protective circuitry.

Diode-Grounded

The diode-grounded system is often found with dc vehicles that employ cable reels, because it permits a two-conductor cable to be used, which is less expensive and takes less space on the cable spool than type G cables. These features make the system attractive, but the diode-grounded system has deficiencies from a safety standpoint.

A simplified diagram of the system is shown in figure 9.63. The machine frame is tied to the grounded negative conductor by means of the grounding diode (D_1). The grounded conductor is connected to the power-center frame or trolley system rail, depending on the power source. In series with the diode lead is a ground-fault device, G, a mercury-magnetic switch or the operating coil of an electromagnetic attraction relay. The pickup setting of this device can be no greater than 25% of the forward-current rating of the diode.

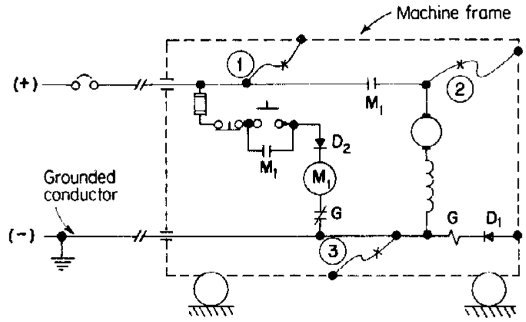

Figure 9.63.—Diode-grounded system with possible fault indicated.

If a positive-conductor-to-machine-frame fault occurs (location 2 of figure 9.63), current flows through the grounding diode and the ground-fault relay. When the fault current exceeds the pickup setting of the relay, the fault is then isolated by deenergizing the machine contactor (M_1) with opening of the G contacts. Actually, this sequence of events occurs only if the fault exists between the load side of the contactor and the motor. This leads to some of the basic drawbacks of the diode-grounded system since location 2 is the only safe area for a ground fault to occur.

Location 1 of figure 9.63 includes the length of ungrounded conductor from its entrance point on the machine to the main contacts of the M_1 contactor. Since these machines normally utilize cable reels, slip-ring assemblies are required for connecting the incoming power conductors to the machine circuit. Hence, it is difficult to locate a circuit-interrupting device at the immediate cable entrance of the machine. If a fault occurs at location 1, the fault current flows through the machine frame, causing diode D_1 to become forward biased, and current then passes through the ground-fault relay. When the relay operates, the M_1 contactor resets, and the motor circuit is deenergized. However, this does not isolate the fault; isolation is solely dependent upon the opening of the M_1 interrupting device, and the fault is located on the line side of its contacts. In this case, fault current can be terminated only by the switching apparatus on the source side of the trailing cable (dual-element fuses for a trolley nip, or fuses or a circuit breaker for a power center).

Location 3 of figure 9.63 includes the entire length of the grounded conductor within the machine. A fault here renders the diode-grounded system unreliable, since it effectively provides a parallel path from the frame to the grounded conductor. The purpose of diode D_1 is to block the voltage drop across the grounded conductor from the machine frame. Therefore, if D_1 is bypassed, the frame potential is raised. A fault at location 3 can easily go undetected. As a result, a simultaneous fault at any location of the ungrounded circuit must again rely on isolation of the source side of the trailing cable.

Failure of the grounding diode is a common problem associated with the diode-grounded system. With D_1 in the shorted mode, the situation is the same as for a location 3 fault. However, when the grounding diode fails in an open mode, all ground-fault protection is lost. Faults occurring in location 1 or 2 can cause the frame to become raised to a value equal to the supply voltage.

The D_2 diode is referred to as the *polarizing diode*. Its purpose is to protect against installing the positive and negative conductors in switched positions, which can occur when adding a new trailing cable or while making a cable splice. If the polarity of the supply voltage is inadvertently reversed, the polarizing diode will block the current flow, and the motor circuit cannot be energized. If D_2 failed in the shorted mode when the dc circuit was under reversed conditions, the diode-grounded system would be rendered totally inoperative, but the machine would still operate.

Grounding Conductor

Other methods of grounding dc machines require the use of a separate grounding conductor and type G cable. The basic grounding-conductor system (fig. 9.64) and the relayed grounding-conductor system (fig. 9.65) are the simplest of these techniques and are commonly used when the source is a trolley system. With the basic system, a ground fault is detected only by the overload protection on the source side of the trailing cable, and the result is extremely poor sensitivity with respect to ground faults. In the relayed system, the ground-fault sensitivity is improved by adding a ground-fault relay in series with the grounding conductor, as shown in figure 9.65. However, the current flowing through the relay may not be the total ground-fault current, since a parallel path may be established through the earth.

Neutral-Shift System

The neutral-shift system is illustrated in figure 9.66. The resistors R_1 and R_2 create a dc neutral point for the system as well as limiting ground-fault current. A ground fault on either the positive or negative conductor will cause the neutral point to shift, which results in detection by the voltage-sensitive relays M_1 and M_2. These relays are usually given the device number 64, a potential relay, also termed a dc *unbalance relay*. The primary limitation of this system is that it cannot discriminate between ground faults for individual pieces of equipment fed by the same rectifier. If a ground fault occurs on one machine, all circuits are interrupted.

Differential Current

As shown in figures 9.67 and 9.68, differential-current dc ground-fault protection utilizes the neutral of the wye-connected secondary feeding the rectifier. With a delta-connected secondary, a neutral connection can be provided through a grounding transformer. A grounding resistor is inserted in the neutral circuit to limit dc ground-fault current, typically to 15 A. Ground faults are then detected using either a saturable-reactor or saturable-transformer system. Actually, this results in an alternative method for grounding dc off-track equipment to that presented in chapter 7.

With the saturable-reactor system (fig,. 9.67), a voltage-sensitive ac relay is placed in series with the winding of a reactor that encircles the positive and negative outgoing conductors. The relay and reactor are excited by a 60-Hz control voltage, and under normal conditions, the current through the positive conductor equals that through the negative conductor. The magnetic fields about both conductors tend to cancel each other, and thus the reactor winding exhibits maximum impedance and prohibits the ground-fault relay from operating. However, when a ground fault occurs, the currents in the positive and negative conductors become unequal in magnitude

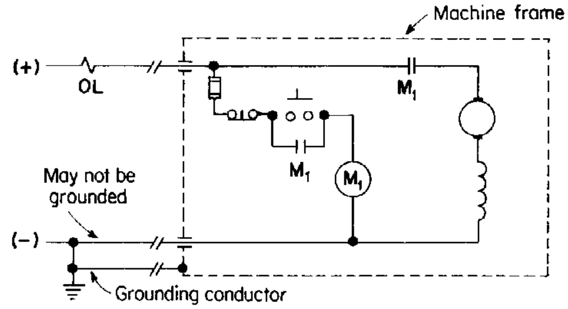

Figure 9.64.—Basic grounding-conductor system.

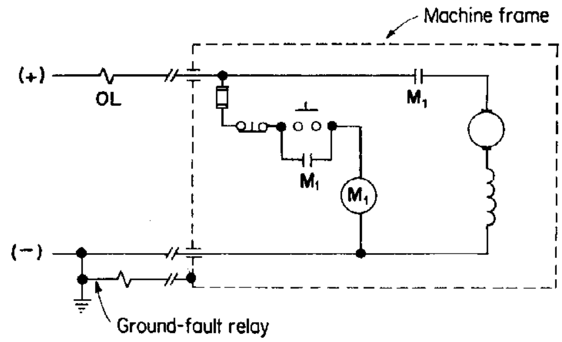

Figure 9.65.—Relayed grounding-conductor system.

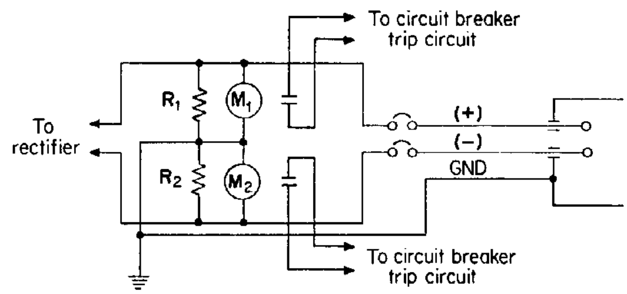

Figure 9.66.—Neutral-shift system.

because current is flowing in the grounding conductor. The magnetic fields about each power conductor are no longer equal, and the resultant magnetic field causes the reactor core to saturate. This in turn reduces the impedance of its winding, causing the ac control voltage to be impressed across the relay. Typical relay pickup is from 4 to 6 A of ground-fault current.

The saturable-transformer system (fig. 9.68) operates on the same principle. Here, the ac control voltage excites the primary (bias) of a two-winding toroidal transformer. The transformer secondary is connected to a rectifier bridge whose output feeds an adjustable resistor in series with a dc relay. The relay has *NO* contacts when deenergized. Under normal operation, ac is induced in the transformer secondary, and the rectified ac causes the relay to pick up its contacts. During a ground fault the core again saturates, but in this case the saturation stops transformer action. In turn, the dc is removed from the relay, which resets its contacts to cause circuit breaker

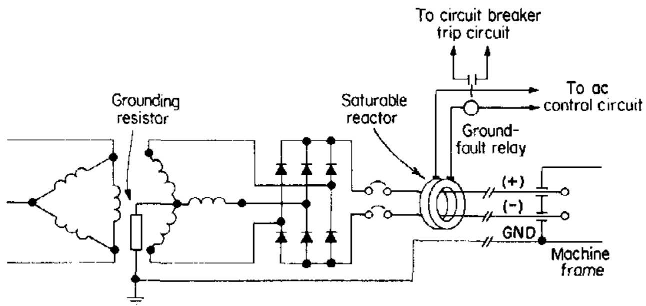

Figure 9.67.—Current-balance dc ground-fault relaying using saturable reactor.

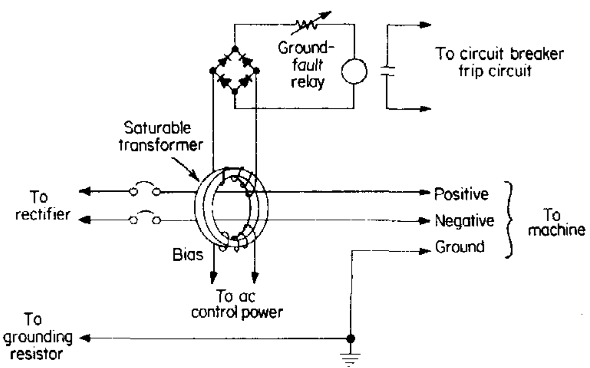

Figure 9.68.—Current-balance dc ground-fault relaying using saturable transformer.

tripping. The resistor allows adjustment of the pickup setting.

Another feature of differential current relaying in addition to sensitive detection of dc ground faults is that the technique senses dc unbalance only in the conductors that pass through the core. By using a detection device for each outgoing dc circuit, the individual relay system is sensitive only to ground faults existing downstream from its reactor or transformer. Thus, selective dc ground-fault relaying can be realized, an advantage not available with the *neutral-shift* system.

The material in this chapter has served to introduce the complexities of protective equipment and relaying, culminating with figures 9.60 and 9.62, where it was shown that the protective relaying in a mine power system can be divided into zones of protection for the various power equipment. The application of this circuitry is the subject of chapters 12 and 13, where additional variations of these basic techniques are discussed. But first, a careful study of the overall mine power system is needed to emphasize the correct coordination of the various protective devices. This is the topic of chapter 10.

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CHAPTER 10.—SIZING PROTECTIVE DEVICES[1]

In chapter 9, it was shown that circuit breakers, fuses, and switches are rated in terms of the nominal circuit voltage, the continuous currents they may carry, the sort-circuit currents they may interrupt, and the fault-through currents they must withstand. To ensure that these interrupting devices disconnect faulted equipment promptly and correctly, it is necessary to have a separate protective system that recognizes the presence of a fault, determines what is faulted, and supplies energy to the mechanisms that will terminate current flow. It then becomes necessary to calculate the maximum fault currents and, in many cases, the minimum sustained overcurrent values in the mine system in order to determine the sensitivity requirements for the current-responsive protection devices (*3*).[2] For multistage time-delay devices, the operating time of each device and the time relationships between devices must be found. These parameters are mandatory for the successful selection, installation, and coordination of protective equipment and relaying.

FAULT CURRENT

The fault current at any point in the mine complex is limited by the impedance of circuits and equipment from the source or sources to the fault point. The level is not directly related to the load on the system (*5*). When a mine is in development, system additions are often made to increase the capacity to handle the growing load. While these changes will usually not change the normal load on preexisting system portions, they may substantially increase short-circuit current during three-phase and line-to-line faults. Nevertheless, ground-fault currents, except for some special cases, remain relatively constant in high-resistance grounded applications. Whatever the situation, the available fault currents must be predetermined to ensure adequate protective-device operation.

Fault-Current Sources

To find the available fault current correctly, all sources of fault current should be known. The main sources in mining are electrical utility systems, synchronous motors, induction machines, and capacitors (and capacitance). Synchronous generators are also a significant source, but these are only a concern if the mine generates its own power. Of the others, some can be eliminated because of their negligible contribution to fault current, but with caution, depending upon the installation.

In most mine systems, the generators of the electric utility system are the principal source of short-circuit current. Because these generators are often remote from the mine, the current that results from a fault within the mining operation usually appears as merely a small increase in the generator load current, and the contribution remains fairly constant (*3–4*). In this case, the electric utility can be represented at the mine as a Thévenin's equivalent source—in other words, a single-value equivalent impedance driven by a constant voltage and referred to the point of connection. Even in proximate locations such as mine-mouth electric plants, this approximation usually gives adequate calculation results.

Synchronous motors can be a substantial contributor to fault current, but because their prime coal mining application is as drive motors for excavators, they are mainly of concern to surface mines. The current supplied to a fault can be described as follows (*3–4*). After the occurrence of a fault, system voltage tends to decrease: the motor receives less power to rotate its load, and simultaneously the inertia of the motor and load, as the "prime mover," causes the motor to act as a generator. The motor thus supplies fault current, which diminishes as the motor field excitation and kinetic energy decay.

Induction motors can also contribute fault current to the system when inertia drives the machine as a generator (*3–4*). Here, however, the presence of field flux in the rotor is produced by induction from the stator; this decays rapidly with motor terminal voltage and disappears completely after a few cycles. Accordingly, the effect of induction machines, except for large horsepower ratings, can sometimes be neglected but should always be considered.

Because of the widespread use of shielded cables, power-factor correction capacitors, and surge capacitors, capacitance is found in great abundance in mine power systems. Although the most destructive power-frequency fault currents come from rotating machinery (including the utility), capacitance can produce very high transitory overvoltages and fault currents (*4*). Fortunately, these are usually of short duration and of a natural frequency much higher than the power frequency, but they can lead to insulation damage. This subject is discussed in chapter 11.

The stored charge of the capacitance acts as the prime mover, but because the fault-current contribution is of such short duration (less than 1 cycle), it can often be neglected when calculating line current (*4*). The main problem with fault conditions lies in the area of ground overcurrent during line-to-neutral faults. For instance, most high-voltage resistance-grounded distribution systems have a 50-A ground-current limit or less, and some mines use a bank of surge capacitors across the terminals of all high-voltage loads. In 12.47-kV systems, the standard 0.25-μF surge capacitor adds about 5-A capacitive ground-charging current, directly translating to ground-fault current during a line-to-neutral mishap. Adding to this current is the charging current of high-voltage feeder cables, typically 0.2 A per 1,000 ft. With just 10 capacitor banks on the distribution system in a moderately sized mine, a 50-A limit is exceeded. If a line-to-neutral fault occurs, the capacitance could discharge and feed the fault with capacitive current in excess of the ground-current limit. As this could happen within the distribution circuitry, there is a possibility that the ground resistor would not see it, and because the duration could be less than 1 cycle, the protective circuitry might not react.

Source Equivalent Circuit

The representation of equivalent circuits is the principal difficulty in the calculation of fault currents. An

[1] The author wishes to thank Frederick C. Trutt, professor and chairman of electrical engineering, the University of Kentucky, for his assistance in preparing the fault-calculation and computer-analysis sections of this chapter.

[2] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

equivalent circuit for the most important typical source, the utility system connection, has already been presented and is usually straightforward. Rotating machinery is less straightforward because the fault current contributed by each machine is limited by the machine impedance, which is unfortunately a rather complex and time-dependent variable. To simplify calculations of fault currents, Thévenin's equivalent sources are again assumed.

Three specific values of reactance are used to establish the fault current delivered at three points in time (*3*):

1. The subtransient reactance, X''_d, for current during the first cycle after the fault occurrence;
2. The transient reactance, X'_d, for current after several cycles at 60 Hz; and
3. The synchronous reactance, X_d, which determines current flow for the steady-state region.

Substransient reactance is assumed to last about 0.1 s, after which the machine impedance increases to the transient reactance value. After 0.2 to 0.5 s, the value again increases to the synchronous reactance. With each increase, current contribution decreases.

Depending on the maintenance of field excitation, synchronous motors could use all three values, but X_d and sometimes X'_d are not needed in mining applications since the values approach infinity. Because of rapid current decay, X''_d is the only value used for induction motors, as these motors do not contribute significant fault current beyond the first cycle after the fault. Table 10.1 lists some typical motor reactances, given in per-unit based on the machine kilovoltampere rating (*3*). Following the previous statement and considering the type of rotating machinery, calculations on underground mining systems sometimes ignore the motor contribution, but computations for surface mines and surface facilities, especially where thermal dryers are involved, generally cannot.

Table 10.1.—Sample reactances for synchronous and induction motors

Motor type	Subtransient reactance (X''_d)	Transient reactance (X'_d)	Comments
SYNCHRONOUS			
Motors:			
6 poles	0.15	0.23	None.
8 to 14 poles	.20	.30	None.
16 poles or more	.28	.40	None.
Converters:			
600-V dc output	.20	(1)	None.
250-V dc output	.33	(1)	None.
INDUCTION			
Individual motors	.17	(1)	Motors generally above 600 V.
Groups of motors each less than 50 hp.	.25	(1)	Motor voltage usually 600 V and below. Subtransient reactance is increased slightly because of very rapid current decay. Lower value of X''_d would be in order for groups of larger motors.

[1]Transient reactance not usually needed in calculations.

NOTE.—The above values are in per-unit referenced to the motor kilovoltampere base. Approximate power bases can be determined from the motor horsepower rating: 0.8 pf motor, kVA base = hp rating; 1.0 pf motor, kVA base = 0.8 (hp rating).

FAULT CALCULATIONS FOR THREE-PHASE SYSTEMS

In specifying equipment or checking protective-circuitry performance, certain simplifying assumptions are normally made to calculate fault currents. An important assumption when finding line currents on three-phase systems is that the fault is three-phase. This type of fault generally causes the maximum short-circuit current to flow. Perhaps the only exceptions are on utility systems and similar industrial applications where a line-to-neutral fault can cause current up to 125% of a three-phase fault (*3*), but these fault currents are substantially limited in most mining systems. The significance of assuming a three-phase fault is that symmetrical-component methods are not needed for the solution of routine fault calculations, although detailed investigations may sometimes require looking at asymmetrical problems.

Because mine power systems are normally radial or operated radially, calculating a three-phase fault is a rather simple task. Basically, all that is required is Ohm's law and an equivalent circuit. Even during a line-to-line fault, positive-sequence and negative-sequence impedances are essentially equal. A good estimation can thus be made by applying a fixed fraction of the three-phase case,[3] which has been found to be about 0.87 (*3*).

A second assumption is that the fault is customarily assumed to be bolted; that is, it has zero impedance (*3–4*). This not only simplifies calculations but also applies a safety factor, since the results provide a value greater than maximum and equipment is rarely stressed beyond its full rating. For instance, analytical studies on low-voltage systems have shown that minimum arcing fault currents, expressed as a factor of bolted faults, are typically 0.89 at 480 V for three-phase arcing faults, and 0.74 at 480 V for line-to-line arcing faults (*3*). As the voltage level is increased, the arcing current level approaches but is always less than that of the bolted case.

IEEE standards 141–1976 and 242–1975 detail extensive fault-calculation recommendations, many of which are directly adaptable to mine power systems (*3–4*). A summary of these follows, after which an example will be discussed. In both, specifics about sizing high-voltage switchgear will be presented to illustrate the method, but the same line of reasoning can be applied to low voltage and medium voltage.

Short-Circuit Calculation Procedures

The calculation procedure for finding currents resulting from faults between power conductors, often termed *short-circuit currents*, can be divided into a series of steps. By assuming a bolted three-phase fault condition, the power-system parameters remain symmetrical regardless of the neutral conditions or even delta-wye transformer connections. The balance is especially close in high-resistance grounded systems, as line-to-neutral loads are not allowed. Therefore, the balanced fault currents can be calculated from a single-phase equivalent circuit, which has only per-phase impedance and line-to-neutral voltage. The procedure is to find the Thévenin's equivalent impedance of the entire system (Z), then solve for fault current

[3] Personal communication from E. K. Stanek, West Virginia University, Aug. 1977.

using Ohm's law. Calculations may use real, percent, or per-unit values. As shown in chapter 4, per-unit methods have the advantage of greatly simplifying the work when the system has different voltage levels.

The first calculation step is always the preparation of a good one-line diagram that shows all fault current sources and all significant elements. All major impedances, both resistance and reactance, should be included: the utility, transformers, conductors, cables, and rotating machines. The collection of this information is perhaps the principal difficulty in fault calculations. Impedances for cables and conductors can be assembled from chapter 8. If transformer impedances are not known, the values can be estimated from the standard values given in tables 10.2 and 10.3 (other impedances are given in chapters 12 and 13) (*3*).

Table 10.2.—Three-phase transformer per-unit impedances[1] for liquid-immersed transformers, 501 to 30,000 kVA

High-voltage rating, V	Low-side rating, 480 V	Low-side rating, 2,400 V and up
2,400 to 22,900	0.0575	0.055
26,400, 34,500	.0625	.060
43,800	.0675	.065
69,000	NAp	.070
115,000	NAp	.075
138,000	NAp	.080

NAp Not applicable.

[1] Actually a reactance, the value is normally expressed as a percentage. The per-unit values given use the transformer self-cooled kilovoltampere and voltage ratings as a base. Transformer resistance is usually well below 0.01 pu.

Table 10.3.—Three-phase transformer impedances for distribution transformers, including load centers

High-voltage rating, V	kVA rating	Per-unit impedance[1]
2,400 to 13,800	112.5- 225	≥0.02
	300 - 500	≥.045
	750 -2,500	.0575
22,900	All	.0575
34,500	All	.0625

[1] Actually a reactance, the value is normally expressed as a percentage. The per-unit values given use the transformer self-cooled kilovoltampere and voltage ratings as a base. Transformer resistance is usually well below 0.01 pu.

In terms of system impedance, fault current has been shown to be mainly dependent upon the reactance between the sources and the fault. This holds true except where there is substantial resistance such as with the extensive use of cables, overhead lines, and buses, which is the case in mining. However, if the reactance-to-resistance ratio, X/R, for the entire system from the source to the fault is greater than 5.0, negligible error is introduced by ignoring resistance. In fact, omitting resistance actually provides a small safety factor and has become common practice in other industrial applications.

For most cases, it is recommended that the per-unit system be used in calculation. Therefore after the one-line diagram is complete, all parameters must be converted to per-unit values on a set of consistent bases. Normally, the base power, kVA_b, is selected first. The current base, I_b, and impedance base, Z_b, are then derived, using the nominal voltage at each system level as the other base, V_b. Accordingly, the base voltages must be related to the turns ratios of the interconnecting transformers. The power base may be any convenient level, but the main-substation kilovoltamperes is often selected.

The next step in the calculation procedure is to reduce the system to its Thévenin equivalent by combining all impedances. The result of the reduction is a single driving voltage in series with a single impedance and the fault. Ohm's law can then be used to compute the per-unit fault current, or

$$I_{pu} = \frac{V_{pu}}{Z_{pu}}, \tag{10.1}$$

where V_{pu} = driving voltage of circuit, pu V,
Z_{pu} = equivalent impedance from sources to fault, including source impedances, pu Ω,
and I_{pu} = symmetrical rms fault current, pu A.

The prefault voltage, the level existing just prior to the fault occurrence, is assumed to be the nominal system voltage at the fault location. With this assumption, short-circuit current will approach maximum. Furthermore, when using the per-unit system, if the voltage bases are equal to the system nominal voltages, the driving voltage, V_{pu}, is simply equal to 1.0 pu.

The per-unit current can be converted to amperes using the base current multiplier. This calculated fault current is an alternating symmetrical quantity, because the sources are rms voltages, and can be used to compare equipment ratings that are expressed in symmetrical rms currents. However, the fault calculations must also recognize the asymmetry of typical fault currents and account for it. Fault current waveforms are discussed in chapters 4 and 9, but the typical asymmetrical type is again reproduced in figure 10.1 for convenience (*3*). The compensation for asymmetry considers current composed of two components:

1. The ac symmetrical component, taken as the calculated symmetrical value, and
2. The dc component, with its initial maximum magnitude taken as the peak of the ac symmetrical component.

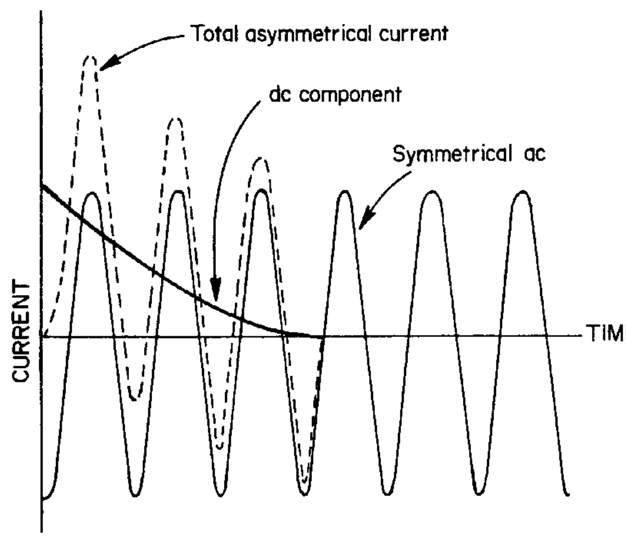


Figure 10.1.—Fault current waveform illustrating asymmetry.

The time period in which the dc component decays is related to the reactance-to-resistance ratio, (X/R), of the system. Hence, the first cycle maximum fault current is estimated as 1.6 times the symmetrical rms value. Depending on the X/R ratio, various other multiplying factors are used to approximate maximum asymmetrical levels throughout the dc decay. These result in estimates of asymmetrical rms current that can be used to compare with ratings based on total (asymmetrical) rms current.

Because fault current varies with time, several different results are commonly desired from the fault calculations. To obtain these might require carrying out simultaneous impedance reductions that account for X_d'', X_d', and X_d of the rotating machinery. However, when the system has just induction motors as loads, these usually contribute fault current only within the first cycle, and at most only two reductions are necessary: one with the motors as sources, the other with just the utility system. In marginal situations, the motor contribution could be the extra current that results in equipment destruction during line-conductor faulting.

Various applications of fault computations are shown in table 10.4. When aimed toward line-current applications in mining, the first-cycle maximum symmetrical current is always needed. The maximum value that occurs between 1.5 and 4.0 cycles after the fault is used for sizing high-voltage circuit breakers. Fault current levels, possibly existing beyond 6.0 cycles, are needed to estimate the performance of time-delay relays and fuses. The calculation result listed in table 10.4 refers to the ac symmetrical value of fault current. Figures 10.2 and 10.3, mentioned in the table, relate the system reactance-to-resistance ratio to the circuit-breaker contact parting time to obtain multiplying factors for momentary or close-and-latch rating comparisons (*3*). To assist in using these curves, typical breaker speeds are 3, 5, and 8 cycles, with respective contact parting times of 2, 3, and 4 cycles (a 2-cycle circuit breaker has 1.5-cycle contact parting).

As a summary of the foregoing procedures, the calculation steps could be listed as follows:

1. Express all impedances between the sources and the fault in per-unit on a set of consistent bases. (In rare cases where the sources and fault are at the same voltage level, such conversion is not necessary.)
2. Reduce the system to one equivalent impedance.
3. Calculate the three-phase bolted-fault current, I_{sc}, by

$$I_{sc} = \frac{V_{pu}}{Z_{pu}} I_b, \tag{10.2}$$

where V_{pu} = driving voltage of circuit, 1.0 pu, if nominal voltages are taken as the prefault level, pu V,

Z_{pu} = magnitude of equivalent impedance, pu Ω,

I_b = base current for system portion in which fault exists, A,

and I_{sc} = symmetrical rms fault current, A.

4. Apply appropriate multiplying factors to account for fault current asymmetry.
5. If line-to-line fault current is desired,

$$I_p = \frac{I_{sc}\sqrt{3}}{2} = 0.87\ I_{sc}, \tag{10.3}$$

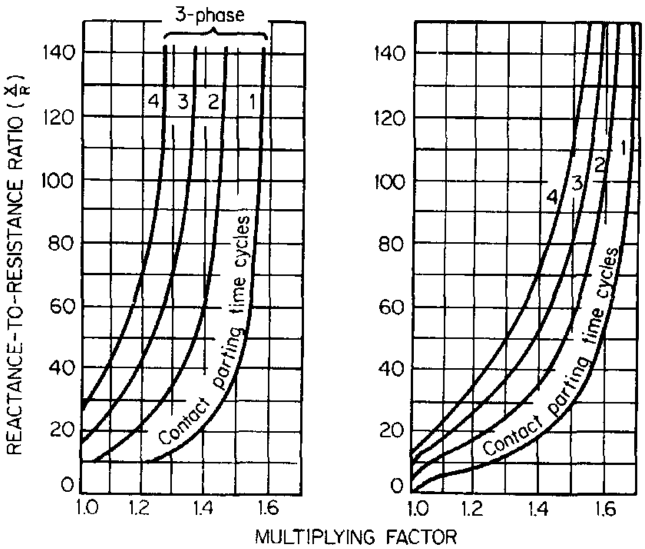
where I_p = approximate symmetrical rms fault current resulting from bolted line-to-line fault, A.

(Note that this is derived from the fact that positive-sequence and negative-sequence impedances are basically equal for phase-to-phase faults). For line-to-line-to-neutral faults, equation 10.3 also approximates the line-current level if the system is high-resistance grounded.

Because of the high mobility of mine power equipment, any specific unit, say a switchhouse, could be installed anywhere, and the location is controlled more by the operation personnel than engineering personnel. It is often desirable that portable apparatus be able to withstand or interrupt the worst fault conditions, and equipment should have a certain uniformity in design. To size this equipment correctly, repetitive fault calculations must be performed, assuming various fault locations throughout the mine system, to arrive at a worst case short-circuit current. More repetitions are usually needed at distribution levels than at utilization. Such extensive fault computations sometimes call for computer analysis, which will be discussed shortly.

Table 10.4.—Sample applications of fault calculations

Application	Operation	Fault current	Comments
1. Fuses and low-voltage circuit breakers.	Contact parting and momentary withstand.	First cycle maximum...	Machine subtransient reactance, X''_d, must be included. If ratings are symmetrical rms current, symmetrical fault current values are used directly.
2. High-voltage switching apparatus, excluding fuses.	Momentary rating or close-and-latch rating.	..do	Machine subtransient reactance, X''_d, must be included. Use values given in table 10.1, except use 1.2 X''_d for individual induction motors 50 to 250 hp, neglecting those below 50 hp. Multiply symmetrical fault current values by 1.6 to obtain total short-circuit current.
3. High-voltage circuit breakers....	Interrupting duty..............	From 1.5 to 4 cycles after fault.	Multiply all motors X'' from table 10.1 by 1.5, except use 3.0 X''_d for individual induction motors 50 to 250 hp, neglecting those below 50 hp. Determine X/R ratio of Thévenin's equivalent impedance. Select appropriate multiplying factor from figure 10.2 or 10.3.
4. Time-delay relays....................	Operation under short-circuit currents.	Beyond 6 cycles.........	All motor contributions omitted. Procedure same as application 1.



A Not more than 1 transformation (local generation)

B 2 or more transformations (remote generation)

Figure 10.2.—Multiplying factors applied to three-phase faults to obtain *momentary* ratings for switching apparatus.

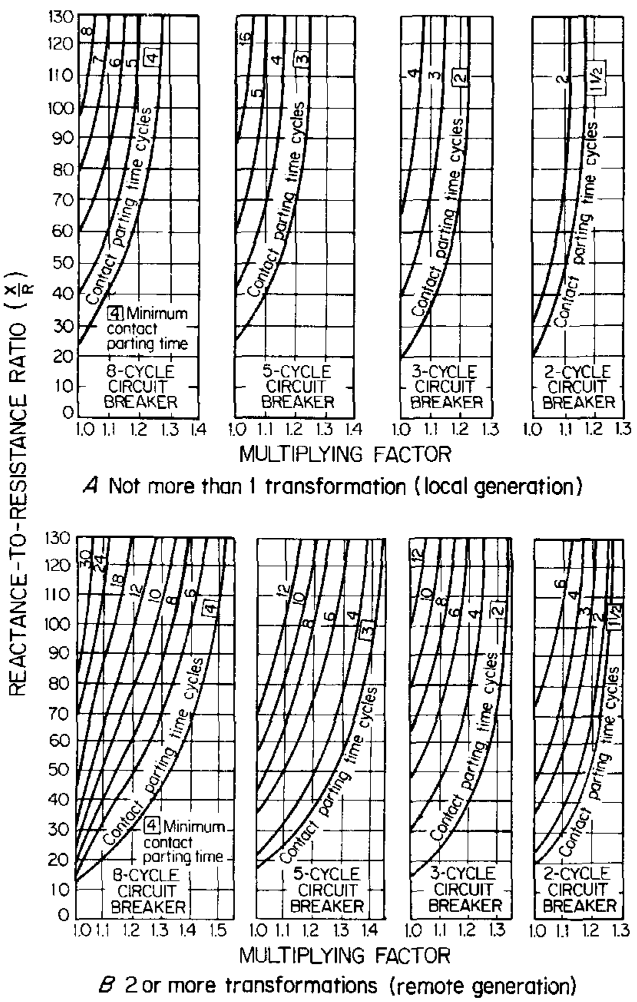


Figure 10.3.—Multiplying factors applied to three-phase faults to obtain *close-and-latch* ratings for switching apparatus.

Three-Phase Calculation Example

To illustrate an example of bolted three-phase fault computations, consider the one-line diagram in figure 10.4, which approximates an underground coal mine in its early stages of development. All cables are given by their conductor size; transformers by their voltages, capacity, and percent reactance; utility supply by its ability to deliver short-circuit current (a kilovoltampere capacity); and motors by their horsepower. Transformer resistance, power-equipment bus work impedance, and interrupting-device impedance are assumed to be negligible. Three possible fault locations are indicated, but for this example suppose only fault 1 has occurred. To ensure correct results, the contribution of rotating machinery is included.

The initial direction is to compute the first-cycle maximum fault current. Following the recommended procedure, the first concern is to convert all impedances to per-unit. For calculation convenience, a base power, kVA_b, of 5,000 kVA is chosen, a level not too large to make the per-unit values of any component insignificantly small. Prefault voltages are taken as nominal system voltages and define the second required base quantities, or for 7,200-, 600-, and 480-V line to line converted to line to neutral,

$$kV_{b69} = 39.838 \text{ kV},$$
$$kV_{b7.2} = 4.16 \text{ kV},$$
$$kV_{b.6} = 0.346 \text{ kV},$$
$$kV_{b.48} = 0.277 \text{ kV}.$$

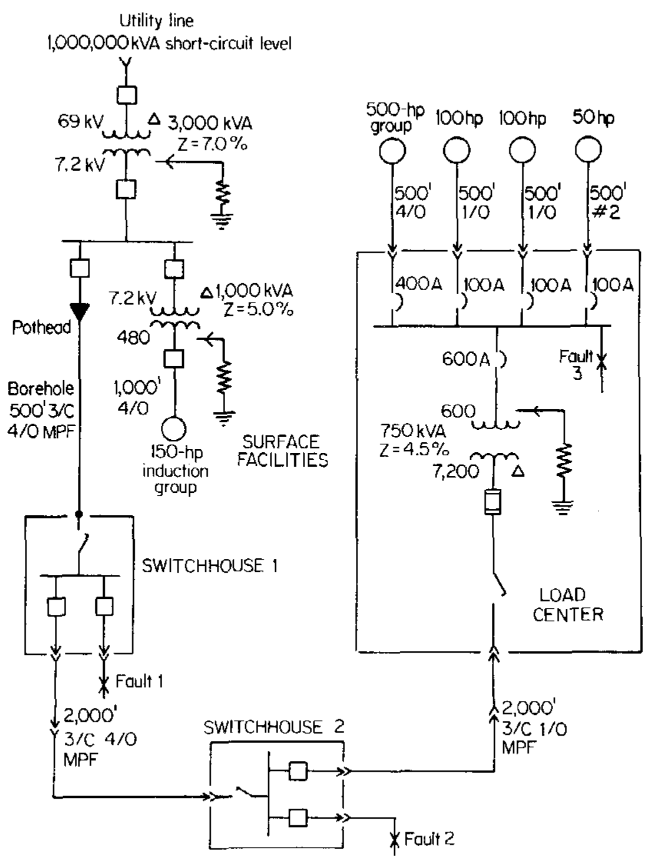


Figure 10.4.—One-line diagram for fault calculations.

Calculations then provide the base current and base impedance for each base voltage. Thus, the base quantities for the 7,200-V system are

$$kVA_b = 5{,}000\ kVA,$$
$$kV_{b7.2} = 4.16\ kV,$$
$$I_{b7.2} = 401\ A,$$
$$Z_{b7.2} = 10.37\ \Omega.$$

At 600 V,

$$kVA_b = 5{,}000\ kVA,$$
$$kV_{b.6} = 0.346\ kV,$$
$$I_{b.6} = 4{,}811\ A,$$
$$Z_{b.6} = 0.072\ \Omega.$$

At 480 V,

$$kVA_b = 5{,}000\ kVA,$$
$$kV_{b.48} = 0.277\ kV,$$
$$I_{b.48} = 6{,}014\ A,$$
$$Z_{b.48} = 0.046\ \Omega.$$

The base impedance and current at 69 kV will not be used.

The next process in the calculations is to reference the power-system parameters to these base quantities. The short-circuit level of 1 million kVA shown in figure 10.4 relates to 1.0-pu impedance on the utility system at 69 kV. It is common practice to assume that this impedance is pure reactance, X_{pu}, so the utility system base values can be taken as

$$kVA_e = 1{,}000{,}000\ kVA,$$
$$kV_e = 39{,}838\ kV,$$
$$x_{pue} = 1.0\ pu.$$

But the calculation base values at 69 kV are

$$kVA_b = 5{,}000\ kVA,$$
$$kV_{b69} = 39.838\ kV.$$

The utility per-unit reactance must therefore be converted using

$$X_{pu} = \frac{X_{pue}\ kVA_b}{kVA_b}\left(\frac{kV_e}{kV_b}\right)^2 \qquad (10.4)$$

or

$$X_{pu1} = \frac{(1.0)(5{,}000)}{1{,}000{,}000}\left(\frac{39.8}{39.8}\right)^2 = 0.005\ pu.$$

Likewise, the per-unit reactance of transformers in the power equipment must be referenced to the calculation base quantities. The main substation contains a 3,000-kVA transformer, which has a 7% reactance referred to the high side, and 69 kV, and is based on the transformer rated kilovoltamperes. The conversion is thus

$$X_{pu2} = \frac{X_{\%}\ kVA_b}{100\ kVA_e}\left(\frac{kV_e}{kV_b}\right)^2 = \frac{7(5{,}000)}{100(3{,}000)}(1) = 0.117\ pu.$$

The 1,000-kVA load-center transformer has a percent reactance of 5% based on the rated capacity of 1,000 kVA and referred to the high side, 7.2 kV. Therefore,

$$X_{pu3} = \frac{5}{100}\left(\frac{5{,}000}{1{,}000}\right)(1) = 0.250\ pu.$$

The 750-kVA load center has a 4.5% reactance similarly referenced, so

$$X_{pu3} = \frac{4.5}{100}\left(\frac{5{,}000}{750}\right) = 0.300\ pu.$$

The resistance and reactance for each cable can be found from tables in chapter 8, and each impedance is easily changed to per-unit with

$$Z_{pu} = \frac{Z_A}{Z_b}. \qquad (10.5)$$

Table 10.5 provides the results of all these cable computations.

The reactances in table 10.1 can be used to approximate the variable impedance of each rotating machine. As a time saver, it is often best to determine all reactances at this point and carry them through the subsequent calculations. However, because there are only induction motors in this example, subtransient reactance, X_d'', is the only value defined, with transient and synchronous reactances approaching infinity. The load in the 480-V system is a 150-hp group of induction motors with each motor less than 50 hp. Here the subtransient reactance is 0.25 pu, based on the motors combined with kilovoltamperes which is approximately the combined horsepower rating, or 150 kVA. This reactance referenced to the base power for the calculations is thus

$$X_d'' = \frac{(0.25)(5{,}000)}{150} = 8.33\ pu.$$

The 600-V system contains a 500-hp motor group, two 100-hp motors, and one 50-hp motor. Each motor in the 500-hp group is about 100 hp. Hence, from table 10.1, this

Table 10.5.—Impedance of cables in figure 10.4

System, V	Base impedance, Ω	Cable				Actual impedance, Ω		Per-unit impedance	
		Length, ft	Size, AWG	Type	Voltage, kV	Resistance (R)	Reactance (X)	Resistance (R)	Reactance (X)
7,200	10.37	500	4/0	MP+GC[1]	8	0.032	0.016	0.0031	0.0015
		2,000	4/0	MPF	8	.126	.064	.0122	.0062
		2,000	1/0	MPF	8	.254	.070	.0245	.0068
480	.058	1,000	4/0	G+GC	2	.068	.027	1.1724	.4655
600	.0752	500	4/0	G+GC	2	.035	.0135	.4514	.1875
		500	1/0	G+GC	2	.064	.0145	.889	.201
		500	2	G+GC	2	.109	.0145	1.514	.201

[1] Borehole.

subtransient reactance is about 0.17 pu on the combined kilovoltampere base, 500 kVA, and on the new base power

$$X''_d = \frac{(0.17)(5{,}000)}{500} = 1.7 \text{ pu.}$$

Continuing the process for the single-motor loads, the reactance for each 100-hp motor is

$$X''_d = 0.17 \text{ pu on a 100-kVA base,}$$
$$X''_d = 8.5 \text{ pu on the 5,000-kVA base,}$$

and for the 50-hp motor,

$$X''_d = 0.17 \text{ pu on a 50-kVA base,}$$
$$X''_d = 17 \text{ pu on the 5,000-kVA base.}$$

Motor resistance could also be assinged, but since the X/R ratio for induction machines is about 6, this is neglected.

The next step is to draw an impedance diagram as in figure 10.5 to show the calculated per-unit resistance and reactance values. The diagram should then be simplified as much as possible to show clearly the data required for calculations, as in figure 10.6. Even though this example concerns fault 1 only, all three fault locations are shown. All rotating machines, including motors and any generators, are represented by their per-unit reactances, X''_d, X'_d, and X_d, and are connected to an equivalent source bus (the dashed line in figure 10.6). Thus the utility supply equivalent reactance is in parallel with the rotating-machine reactances, and the combined potential driving the fault is 1.0 pu.

Further simplifications can now be made for each specific fault location. Figures 10.7 through 10.9 show the process graphically for fault 1. The operations involve no more than combining parallel and series impedances to obtain a single impedance between the source bus and the fault, which is the Thévenin's equivalent of the faulted system (fig. 10.9).

The single equivalent impedance can now be used to calculate the first-cycle symmetrical rms current of the bolted three-phase fault, or

$$I_{sc} = \frac{V_{pu}}{Z_{pu}} I_b = \frac{1.0}{0.1132}(401) = 3{,}542 \text{ A.}$$

With the fault located in high-voltage distribution, this value can be used to compare fuse ratings based on symmetrical rms currents. If the fault had been in a low-voltage portion, the current would also apply to circuit breakers. It is obvious that in this example, resistance could have been omitted with negligible error.

Figure 10.10*A* is a simplification of figure 10.5 to demonstrate the error that could be introduced by neglecting the motor fault current contribution. As shown, the only driving source is the utility, and the equivalent circuit in figure 10.10*B* can easily be found by series combination. The first-cycle symmetrical rms current in this situation is

$$I_{sc} = \frac{1.0}{0.1235}(401) = 3{,}247 \text{ A,}$$

which is 295 A less than the previous value. Thus by omitting the motors, the result is an 8.3% error, which emphasizes the need to be cautious about neglecting the motor contribution. Nevertheless, this calculated value should be very close to the symmetrical component of fault current and can be used for time-delay relaying comparisons (see application 4 of table 10.4).

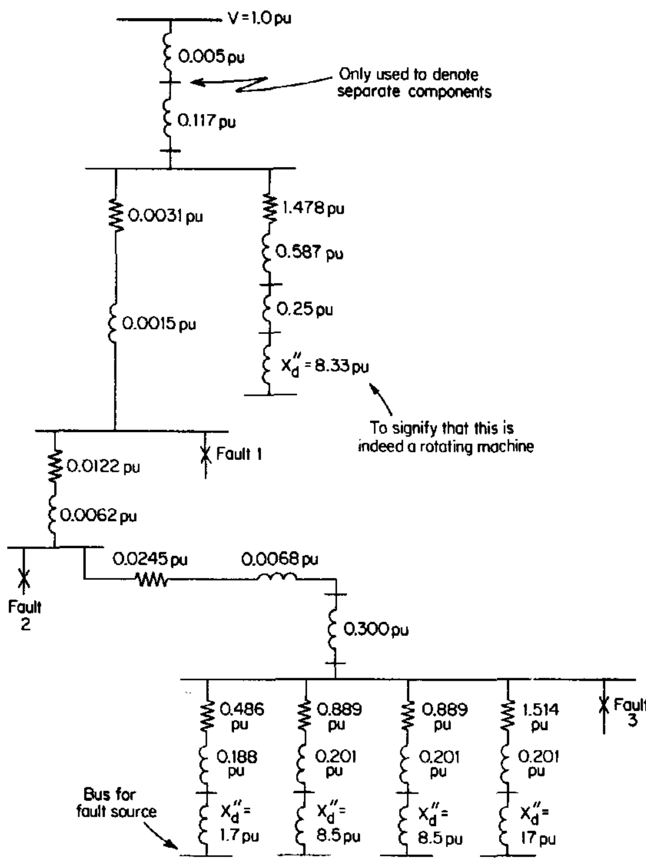


Figure 10.5.—Impedance diagram for one-line diagram of figure 10.4.

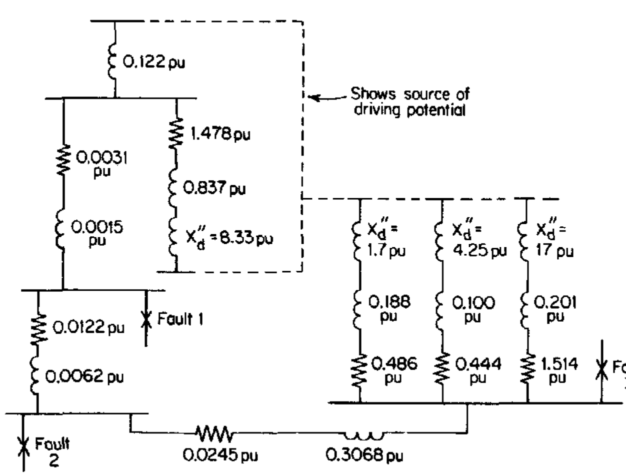


Figure 10.6.—Simplification of figure 10.5.

If fault calculations are to be applied to high-voltage switchgear, the momentary or close-and-latch current duties must be calculated. The next step in the calculations would therefore be to account for fault current asymmetry by employing multiplying factors that adjust for the dc component decay. Here the subtransient reactance of all induction motors greater than 50 hp is modified by multiplying by 1.2. The contribution of motors less than 50 hp is neglected. These adjustments can be compared to a decrement factor, allowing for fault current decay with time. Figure 10.11*A* reflects these changes and also omits resistance (compare with figure 10.6). Figure 10.11*B* shows the reduced single equivalent reactance, and using the formula,

$$I_{sc(mom)} = \frac{V_{pu}}{Z_{pu}} I_b \,(1.6) \qquad (10.6)$$

or

$$I_{sc(mom)} = \frac{1}{0.1154}(401)(1.6) = 5{,}559 \text{ A}.$$

This current represents the maximum asymmetrical rms value in the first cycle and can be compared with close-and-latch capabilities of high-voltage switching apparatus. These are normally circuit breakers, but any device that needs to be rated on total fault current also applies. Interestingly, if subtransient reactances were not modified

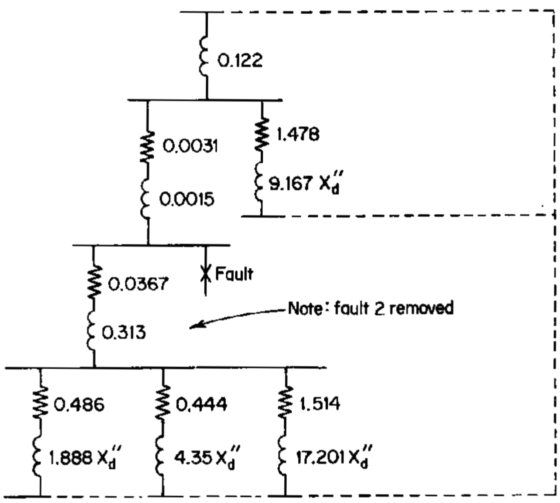


Figure 10.7.—Simplification of figure 10.6.

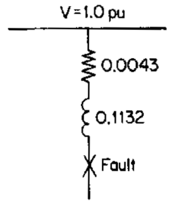


Figure 10.9.—Equivalent circuit of figure 10.6.

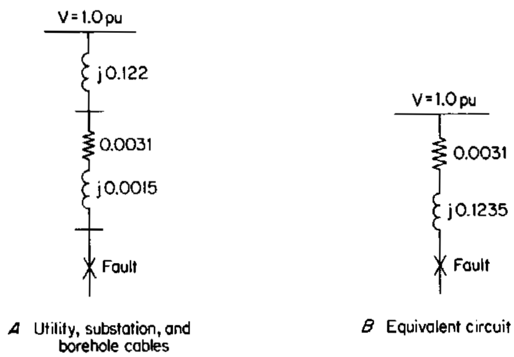


Figure 10.10.—Example problem with motor contribution neglected.

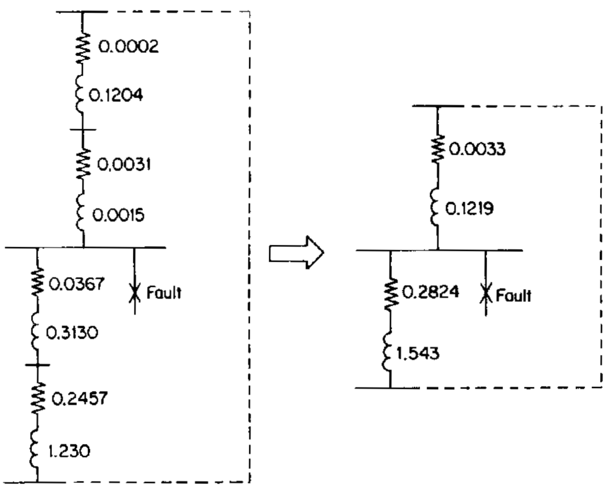


Figure 10.8.—Further reduction of example network.

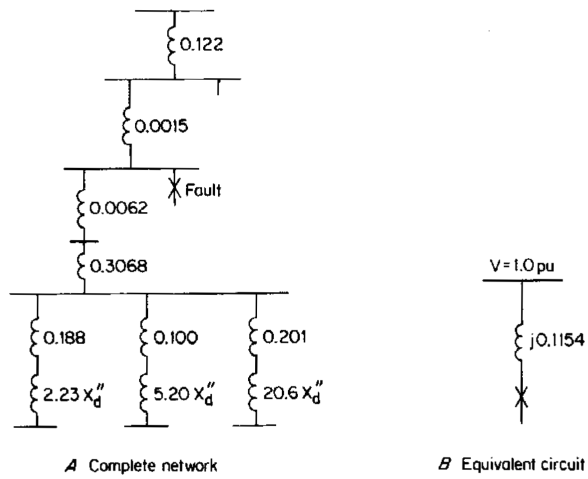


Figure 10.11.—Network to calculate momentary or close-and-latch current duties.

in this example, the calculations would produce an asymmetrical rms current of 622 A, thus creating a small safety factor for switching-apparatus comparison.

To illustrate a further application of fault calculations (table 10.4, item 2), the circuit of figure 10.6 can again be adjusted to determine the interrupting duties of high-voltage circuit breakers that have minimum contact parting times between 1.5 and 4.0 cycles. Again, to allow for decrement of the dc component, subtransient reactances are multiplied by (*3*)

1.5, for synchronous motors,
1.5, for 3,600-r/min induction motors over 250 hp and 1,800-r/min induction motors over 1,000 hp, and
3.0 for induction machines of 50 hp and larger (item 2 supersedes this item where there is any overlap in coverage).

Smaller motors are considered insignificant. To obtain the necessary multiplying factor from figures 10.2 or 10.3, the X/R ratio of the equivalent impedance or *fault-point impedance* is needed. This can be found by reducing resistance-only and reactance-only networks, which eliminates handling complex numbers. The formula for interrupting duty is then

$$I_{sc(int)} = \frac{V_{pu}}{Z_{pu}} I_b \text{ (multiplying factor)}, \tag{10.7}$$

where $I_{sc(int)}$ = total rms current interrupting duty, A,
and multiplying factor = value from figure 10.2 or 10.3.

For comparison with circuit breakers rated in megavoltamperes, the interrupting capacity can be converted to an interrupting-current capability by

$$I_{sc(int)} = \frac{\text{MVA}}{\sqrt{3}\,V_{oper}}, \tag{10.8}$$

where $I_{sc(int)}$ = interrupting current, kA,
MVA = system power, MVA,
and V_{oper} = line-to-line system voltage, kV.

The system voltage is used when it is within the minimum and maximum rated limits of the breaker, but the minimum rated breaker voltage is used for formula 10.7 if the operating voltage is less. For instance, if the minimum rated voltage is 12.47 kV and the circuit breaker is being used on a 7.2-kV system, then 7.2 kV is used for calculations.

The foregoing example is just a simple demonstration of bolted three-phase fault calculations in a small mine. As a result, the fault current was rather small but still large in comparison with normal load current. In practice, more fault locations would be taken, and the computations repeated until the worst case condition was verified. As more mining sections or loads are involved, the complexity of these calculations increases substantially, oftentimes to the point where hand computing becomes too cumbersome and a computer analysis must be adopted.

Computer Fault Analysis

Systematic digital computer methods, developed to aid the designer in specifying system protection requirements, are usually called upon for major fault studies (*3*). A complete discussion of the theoretical bases of such techniques is beyond the scope of this chapter, but is available in the literature (for instance, reference *15*). However, a few thoughts about the methods follow.

Balanced-fault calculations using a computer require either a generalized impedance formulation of the power-system network (referred to as *ZBUS fault analysis* where the network is similar to the preceding calculations) or a generalized admittance formulation, which is termed *YBUS load-flow analysis*. The impedance network has the advantage of also allowing the evaluation of unsymmetrical-fault behavior by applying symmetrical-component theory. With this approach, line-to-neutral, line-to-line, and line-to-line-to-neutral faults may be simulated by proper connection of the power-system network representation for the positive, negative, and zero sequences. On a large digital computer, computations are quite rapid, and the amount of output information is limited primarily by the user's requirements. Available information generally includes fault currents, as well as postfault currents and voltages, for any fault at any system location. Information on sequential faulting of any system bus or location is possible. Prefault load currents are normally neglected but may be included if a more exact analysis is required.

Input-output techniques vary with the program type but may be either batch or interactive mode, and in either case the system is generally input in terms of impedance or even cable sizes between bus pairs. Prefault loading information may also be input in terms of kilovoltamperes or load type. In batch-processed programs, the run is then completed and required output information displayed. Interactive techniques allow the additional capability of on-line system modifications to evaluate the effects of design changes (*7–8*).

Ground-Fault Current Calculations

Federal regulations require high-resistance grounding on portable or mobile coal mining equipment. For low-voltage and medium-voltage systems, the neutral grounding resistor must limit the maximum ground-fault current to no more than 25 A. For high-voltage systems, the voltage drop in the grounding circuit external from the resistor must be 100 V or less under grounding-fault conditions. To meet these conditions, as well as to exceed capacitive charging-current constraints, high-voltage mining systems typically have maximum ground current limits at about 25 A, but rarely above 50 A. As the grounding conductor cannot carry load current, there can be no neutral-connected loads in these systems; thus, ground-fault current is contributed only from generating stations and capacitance. In other words, motors have no effect. Power-factor correction capacitors are not connected to the neutral, so capacitance is contributed only from conductor and cable capacitance and surge capacitors. This contribution can be substantial, but it is always less than the maximum ground current limit and is of very short duration, much less than 1 cycle.

Under these conditions, line-to-line-to-neutral faults cause about the same current level as line-to-neutral faults, and the symmetrical rms ground-fault current can be calculated from the line-to-neutral case, or

$$I_{gf} = \frac{3V_{Ln}}{Z_1 + Z_2 + Z_0 = 3Z_G = 3R_G}, \tag{10.9}$$

where I_{gf} = symmetrical rms ground-fault current, A,
V_{Ln} = line-to-neutral voltage of system, V,
Z_1 = positive-sequence impedance, Ω,
Z_2 = negative-sequence impedance, Ω,
Z_0 = zero-sequence impedance, Ω,
Z_G = sum of impedances for fault and grounding circuit, Ω,
and R_G = resistance of neutral-grounding resistor, Ω.

The sequence impedances are usually so close that they can be assumed equal, so that

$$I_{gf} = \frac{3V_{Ln}}{3Z_T + 3Z_G + 3R_G} = \frac{V_{Ln}}{Z_T + Z_G + R_G}. \quad (10.10)$$

The sequence impedances (Z_T) are usually so small that they can be neglected, and

$$I_{gf} = \frac{V_{Ln}}{Z_G + R_G}. \quad (10.11)$$

Consequently, the calculation of ground-fault currents on high-resistance-grounded systems is a rather simple task. To obtain maximum values, many engineers assume a bolted fault and sometimes neglect conductor impedance. For minimum available ground-fault currents, conductor impedance can be the maximum possible at the mine, but a bolted fault is again usually assumed.

This simplification does not apply to grounding systems that are not high resistance grounded, but equation 10.9 can still be used to approximate ground currents from line-to-neutral faults. The method of symmetrical components should always be used in these asymmetrical fault cases.

DIRECT CURRENT SYSTEM FAULTS

The preceding material covered ac occurrences almost exclusively, yet knowledge about fault current levels on the dc portions of mine power systems is also imperative for the correct application of protective equipment, since faults can occur within motors, on trolley lines, trailing cables, cable reels, surface-excavator loop circuits, and so on. Trolley lines can pose extensive problems because of their normally heavy load currents as well as their exposed nature; for example, a roof fall can cause a serious arcing fault.

In underground mines, the principal source of dc fault current is from the dc distribution, through rectifiers and to a lesser extent batteries (see chapter 15) and motor-generator (m–g) sets. For surface mining, Ward-Leonard systems are also involved, but these fault locations are commonly confined to inside unit-designed equipment. As with ac, the voltage drop associated with a fault can cause motors to go into a generation mode and act as a source as long as the field and the applied inertia are maintained. Prime movers are perhaps a more serious concern with dc than with ac since they can exist for an extended time, for instance, the inertia of a large trolley locomotive being pushed by loaded cars. However, the motor contribution varies widely from application to application, so that assigning values for series impedances, like subtransient reactances, can be a very difficult task. Furthermore, procedures for simplifying calculations are not well established, and many authors recommend very detailed techniques (*1*, *3*).

Fortunately, dc systems in mining are almost exclusively at utilization, and in them the fault current seen by protective devices is usually dependent upon location and not motor contribution. In other words, a motor might pump substantial current to the fault but not usually through a protective device unless the fault is upstream from the device. Thus, in many cases, the fault network can be effectively constructed using only the principal sources and the impedance in series with the fault. When basing the reasoning on this apparent simplicity, it is all too easy to treat dc faulting as just an application of Ohm's law and resistance and fail to recognize that the system current is no longer steady state after the fault. Figure 10.12 shows an example in which the fault current is interrupted before the maximum is reached (*14*). It is apparent that the rate of rise as well as time to interruption must be considered, and circuit inductance becomes a very important factor (*1*). In assembling a fault network, trolley-system inductance is not an easy parameter to find, but values between 0.2 and 2.0 mH have been found to be typical (*14*).

Figure 10.13 gives plots for some rectifiers versus distance of the fault from the rectifier (*14*), to illustrate the available fault current level that can exist on rectifier-fed trolley systems. One of the more prevalent dc protective-relaying problems in mining is not maximum fault level

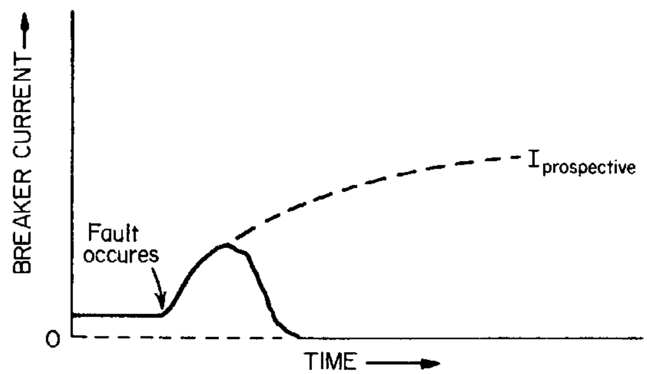


Figure 10.12.—Fault current in dc system.

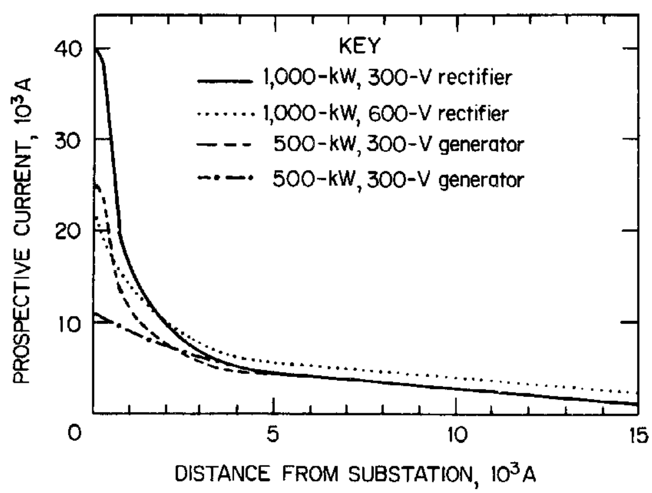


Figure 10.13.—Available fault current versus distance of fault from rectifier on typical trolley systems.

but rather the possibility of having small undetected faults occur on trolley systems. In Pennsylvania, during 1978 alone, two extensive mine fires were initiated in this way. A trolley-wire-to-rail fault, for example, could occur as the result of a roof fall but be of high enough resistance so the fault current would be less than normal load currents; hence, the fault would not be detected by conventional overcurrent relaying. Several methods have been proposed to detect such illegitimate loads, including discriminating relaying and rate-of-current-rise detection (*11*). At this writing, these are still in the demonstration stage.

DEVICE SETTINGS

Coordination of a mine power system entails complete organization of time settings and/or current settings for all protective devices from the loads to the sources. This necessitates a comprehensive coordination study of the entire system to determine the range of correct values for all instrument transformers, pickup and time settings, fuse ratings, and circuit breaker trip ratings, which will provide effective coordination and selectivity and ensure that the minimum of unfaulted load is disturbed when protective devices isolate a fault. The prime concern is overcurrent, since the circuitry must provide simultaneous overload, short-circuit and ground-fault protection without causing nuisance tripping. The application of this "art" is perhaps the most perplexing problem facing practicing engineers. A system fault analysis is vital input for any comprehensive coordination study and should address not only maximum values but also minimum values, together with the normal operation and maximum allowable currents.

The balance of this chapter is broken into the major aspects of the coordination study: relay pickup settings, CT matching, circuit breaker trip settings, fuse characteristics, and overall coordination. Emphasis is placed on radial ac systems that are high-resistance grounded.

RELAY PICKUP SETTINGS

Pickup has already been defined as the minimum value of the actuating quantity that will cause a relay to operate its contacts. Whether the application is at a trailing cable, feeder cable, or overhead conductor, the requirements for overload, short-circuit, or ground-fault protection usually translate into current pickup settings. The values used in this chapter are generally in line with those contained in 30 CFR 75 and 77, which are in effect at this writing (*16*). The quantities that define pickup may change in the future, but the techniques presented here for establishing relay pickup settings should not. Because of their widespread use, induction-disk relays are implied in most of the following pickup applications; however, pickup techniques for other relays are basically the same. To avoid confusion, molded-case circuit breaker trip settings will be covered later.

Establishing a pickup setting for an ac relay involves selecting a CT ratio and operating-coil current. For marginal short-circuit currents, the accuracy of the combination might require verification. The following material describes pickup settings for a single zone in a mine power system, but it must be remembered that the overall goal is to obtain coordination, and the settings at any location can be affected not only by requirements and regulations but also by other upstream and downstream relaying.

Short-Circuit Protection

Short-circuit protection can be obtained with an instantenous element (no intentional delay) or an inverse-time overcurrent relay using the minimum time dial setting (maximum time delay here is often restricted to no more than 0.6 s). The general requirements can be determined by selecting the lower value calculated from the following:

1. 115% of the maximum starting current or 115% of the peak load current, whichever is higher, for the equipment being protected; or
2. 60% of the smallest bolted three-phase symmetrical rms fault current for any point of the zone protected by the relay.

The first value is designed to have pickup above the normal operating current to prevent nuisance tripping. Usually, the bolted fault current value is higher than the first value.

If the maximum starting currents of the motors are not known, each can be approximated by

$$I_s = 1.25 \left(\frac{1}{X_d''}\right) I_{fl}, \tag{10.12}$$

where I_s = approximate maximum starting current, A,
X_d'' = per-unit subtransient reactance of motor or motor group, pu Ω,
and I_{fl} = motor full-load current, A.

The motor full-load current can be estimated from

$$I_{fl} = \frac{746(\text{hp})}{\sqrt{3}\, V \eta\, (\text{pf})}, \tag{10.13}$$

where hp = rated machine horsepower,
V = rated line-to-line voltage of motor, V,
η = motor efficiency,
and pf = full-load power factor, which can be assumed to be 0.85.

Inrush currents for transformers usually range from 8 to 12 times the full-load current rating for a duration of 0.1 s; typical values for inrush currents should be available from the transformer manufacturer.

To show how short-circuit pickup is selected, consider that a production shovel in a surface mine has 2,000-hp connected load rated at 4,160 V line to line. As shown in figure 10.14, the shovel is powered through 1,000 ft of 4/0 AWG cable. The shovel induction motor is 85% efficient at full load and operates at 0.8 pf. The minimum value of bolted three-phase fault current has been found to be 6,130 A. CT ratios are 400:5 A, and the instantaneous element has a pickup range from 10 to 50 A. The procedure is to find the minimum current according to the above criteria.

First, for the fault current, the pickup would be 60% of the bolted value divided by the CT turns ratio:

$$\text{pickup} = \frac{(0.6)(6{,}130)}{80} = 46 \text{ A}.$$

This value must now be compared with the pickup needed to slightly exceed maximum starting current, which may be estimated using equations 10.13 and 10.12. Here, the full-load current of the shovel motor is about

$$I_{fl} = \frac{746(2{,}000)}{\sqrt{3}\,(4{,}160)(0.85)(0.8)} = 305 \text{ A}.$$

From table 10.1, the per-unit subtransient reactance of individual induction motors is 0.17, and the estimated maximum starting current is then

$$I_s = 1.25\left(\frac{1}{0.17}\right)(305) = 2{,}243 \text{ A}.$$

Allowing 115% to prevent nuisance tripping, the pickup setting is then

$$\text{pickup} = \frac{(2{,}243)(1.15)}{80} = 32 \text{ A}.$$

As this is less than the value for the fault current, 32 A is the selected pickup setting for short-circuit protection.

Overload Protection

The prime purpose of overload pickup settings is to protect conductors and insulation from damage by excess temperature. Temperature here is a function of the ambient temperature and the I^2R power loss in the conductors; temperature rise also involves time; therefore, inverse-time overcurrent relays are used on high-voltage systems to provide this protection. (The thermal overloads and fuses commonly used for low-voltage and medium-voltage systems will be discussed later.) The general overload requirement is that relay pickup should occur whenever 125% of the ampacity of the smallest power conductor in the protected zone is exceeded.

When this requirement is directly applied to components such as transformers, the overload value could be 125% of the rated full-load current. However, the percentage recommended for transformer overload protection varies depending upon the protection scheme associated with the transformer. For transformers rated greater than 600 V, the National Electrical Code allows 300% of rated primary current with circuit breaker protection, and 150% for fuses when there is no protection at the transformer secondary (*10*). With transformers rated less than 600 V, overload protection for the primary winding is 125% when there is no overload protection at the secondary and 250% if overload protection for the secondary is set at 125% (*8*). On the other hand, IEEE standard recommendations for transformer overload protection state that time-relay pickup should be set at 150% to 200% of the primary full-load current (*3*).

Considering figure 10.14 as an example for overload pickup, the trailing cable is 4/0 AWG, three-conductor, 5-kV, with 90°C rated insulation, and the relay operating coil (for the 51 contacts) has a pickup range of 4.0 to 12 A. The maximum ambient temperature at this location is 30°C. From the data given in chapter 8, the cable ampacity at 40°C is 321 A, and when applying the temperature correction factor for 30°C,

$$\text{ampacity} = (1.1)(321) = 353 \text{ A}.$$

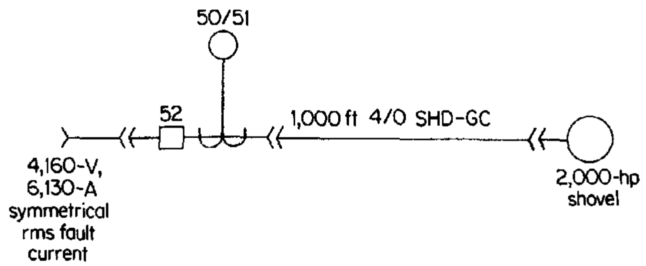


Figure 10.14.—One-line diagram for pickup setting example.

Allowing for 125%, the minimum conductor current that defines overload is

$$I = 1.25(353) = 442 \text{ A}.$$

The CT ampere-turns is 400:5 A, so the required pickup setting is

$$\text{pickup} = \frac{442}{80} = 5.5 \text{ A}.$$

A tap setting corresponding to this pickup would allow the time-current characteristics of the relay to determine an overload condition. If the next tap setting available was 6.0 A, this would relate to 480-A conductor current as an overload, which is too high. If an instantaneous element with a 32-A pickup is available in the relay, short-circuit protection is also provided with one CT per line conductor.

Ground-Fault Protection

Most experts suggest that ground-fault relays should pick up at no more than 30% of the maximum current limit of the grounding system to provide an ample margin of safety in high-resistance grounded systems (*9*, *13*, *19*). For a 25-A current limit, this represents a line-conductor unbalance producing a zero-sequence current of about 8 A. However, such a demand is lower than present protective-circuitry detection capabilities when electromechanical relays are used. For instance, the optimum arrangement with induction-disk relays is zero-sequence circuitry with a 25:5 ampere-turns CT in which the most reliable pickup performance is not less than 12 A. One reason for this limit is connected to the fact that the window-type CT needs a large opening in order to pass the three line conductors through. Here, zero-sequence currents less than 12 A do not generate enough capacity from the CT to drive the relay adequately.

Another problem is that induction-disk relay operation is not reliable when the magnitude of actuating current is only slightly above the tap setting. This is because the net actuating force is so low that any additional friction in the rotating-disk mechanism can prevent operation or increase the operating time. Even if the relay does close its contacts, the contact pressure may be so low that contamination of the contact surface can prevent electrical contact. To minimize this problem, it is common practice to apply induction relays in such a way that their actuating quantities are at least 1.5 times the tap setting (*6*). In fact, time-current curves are rarely shown for less than this amount. On the other hand, an induction relay is most effective if its pickup is selected so it will operate on the most inverse part of its time curve. Thus, the minimum value of actuating current should be only slightly higher than 1.5 times the tap setting. Often a pickup corresponding to 2.5 times is selected for very inverse relays, the type usually applied to ground-fault protection.

It might be thought that requiring relay operation below the ground current limit is too stringent and that relays should only be required to operate at the fault current available on the system. This is a logical deduction, which relates that on a properly installed grounding system no potential greater than that allowed can ever exist between any metallic object and earth under fault conditions (40 V on low-voltage and medium-voltage systems and 100 V on high voltage). Nevertheless, in terms of personnel safety, setting a relay pickup at less than the limited fault current provides viable backup protection. In other words, machine frames would remain well below the current allowed during any ground fault. Any trend toward hazardous conditions could also be detected, although such trends involving the system neutral are infrequent on high-voltage distribution systems.

Considering these thoughts and in-mine practice, ground current pickup should not be greater than 50% of the current rating of the grounding resistor. This level should provide reliable repetitive operation of zero-sequence protective circuitry. As technology improves, 30% pickup should be the goal, as is suggested in the literature (*9*, *13*, *19*).

To give an example of pickup settings, consider that ground-fault protection is provided by zero-sequence relaying A very inverse induction-disk relay is used and has a tap-setting range of 0.5 to 4.0 A for the time element coil. The CT has a 25:5 ampere-turns ratio, and the ground current limit is set at 25 A. Applying the 50% recommendation, zero-sequence current flow in the three line conductors cannot be greater than 12.5 A for relay pickup; thus, the maximum tap setting of the operating coil is 12.5/5 or 2.5 A. However, the selected tap setting should be lower to allow for effective time-delay operation. Using 2.5 times the minimum value of actuating current, the tap setting for the relay should be 2.5/2.5 or 1.0 A.

On solidly or low-resistance grounded systems, the available ground-fault current is substantial enough that there are usually no relay pickup problems, even if residual ground-fault relaying is used. Accordingly, the pickup level in this case could be defined by 30% of the minimum bolted line-to-neutral fault current. Similar logic could also apply to ungrounded systems, but here the maximum ground current requirement or even a ground-fault requirement is not easy to define. As a result, ground-fault pickup could, if necessary, be related to a decrease in the line-to-neutral potential of any power conductor and perhaps be 30% of the nominal system voltage. This line of reasoning could apply to dc as well as ac systems.

CURRENT TRANSFORMER MATCHING

From the foregoing, it appears that the selection of CT ratios and relay pickups to provide protection against excessive currents is quite a straightforward process. However, CT's are magnetic devices and can give inaccurate results when improperly applied. CT performance is an important factor in protective-relay design because relays are only as accurate as the CT's that energize them. The main CT problem is core saturation caused by excessive primary current, incorrect secondary burden, or a combination of both these factors (*4*). When a CT is in saturation, its accuracy is very poor, secondary current is actually less than it should be, and relays tend to operate more slowly than intended. One danger is the loss of relay coordination.

Current Transformer Accuracy

A model of a CT and its burden is shown in figure 10.15. The dependent current source delivers a current equal to the primary current (I_p) divided by the turns ratio (N) of the CT. The remaining impedances, voltages, and currents are defined as

Z_e = secondary excitation impedance, Ω,
Z_s = secondary winding impedance, Ω,
Z_b = burden impedance, Ω,
E_s = secondary excitation voltage, Ω,
V_t = secondary terminal voltage, V,
I_e = secondary excitation current, A, and
I_s = secondary current, A.

The burden refers to the impedance of the external load applied to the CT secondary, including the impedance of the relay, its associated wiring connections, and any meters. As shown in figure 10.15, a portion of the generated current (I_e) is consumed in exciting the CT core. The remainder of the generated current (I_s) is the true value of the secondary current.

The percent ratio correction error (I_e/I_s) is defined as that factor by which the nameplate ratio of a CT must be multiplied to obtain the true ratio (*6*). It is apparent that this error will remain small as long as the excitation current is small. The magnitude of the excitation current is a function of the excitation voltage (E_s) as illustrated in the typical secondary-excitation characteristics of figure 10.16 (*4*). It can be noted that the secondary-excitation curves are linear until the saturation point is reached. Beyond the saturation point, a small increase of E_s results in a large increase of I_e, which in turn causes the percent ratio error to increase dramatically. The magnitude of E_s is primarily a function of the secondary current (I_s) and the burden impedance (Z_b). Therefore, to minimize inaccuracies of the relay system, the burden impedance should be kept as low as possible. The pickup value for the relay must always result in the excitation current's lying in the linear portion of the secondary-excitation characteristic. It should also be noted that the impedance of electromechanical relays is not constant. Since they are magnetic devices, their impedance decreases as the secondary current increases because of saturation. Thus, the

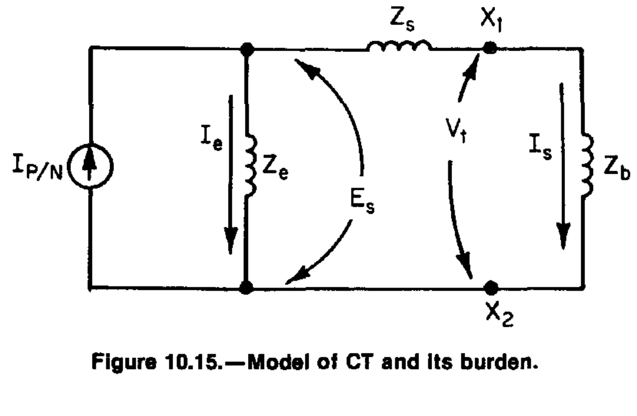


Figure 10.15.—Model of CT and its burden.

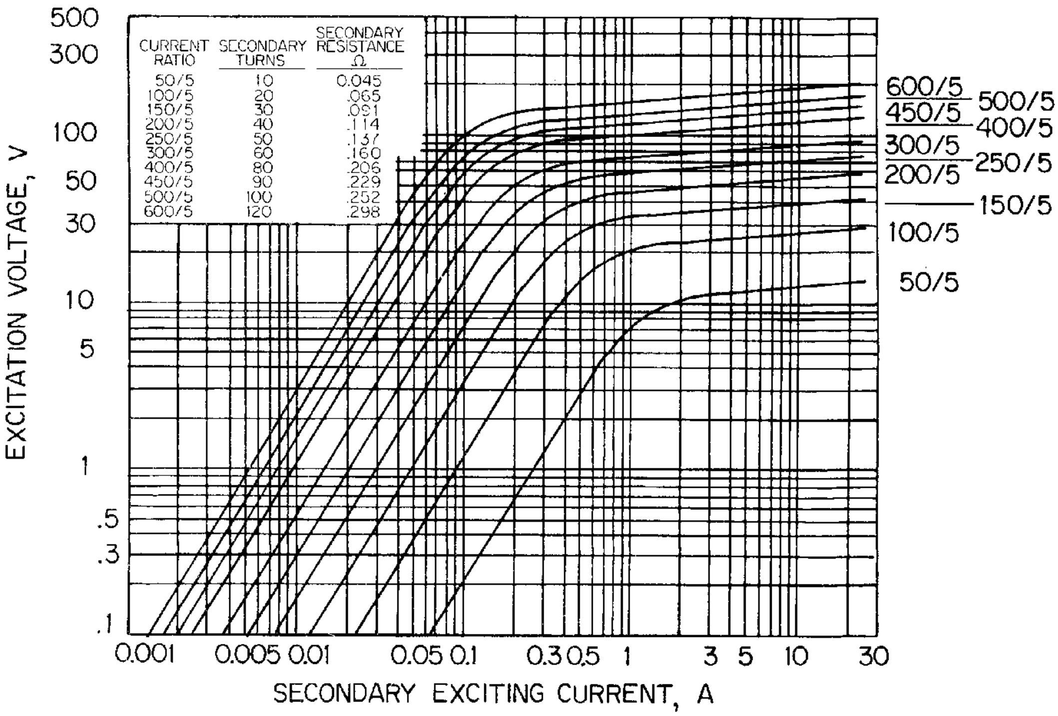


Figure 10.16.—Typical set of saturation curves for 600/5 multiratio bushing-type CT.

relay impedance should be considered over the entire operation range of currents when matching the relay to the CT.

Beyond keeping secondary burden as low as possible, there are other recommended guidelines for protective relaying that help to minimize the effects of saturation. CT turns ratios should be maintained as high as practical. Usually, saturation is only a problem on low-ratio current transformers, such as 50:5 versus maybe 300:5 ampere-turns. CT secondaries with a higher ratio for a specific application develop higher voltage and are less likely to be saturated under normal burden. Hence the lower the ratio, the greater is the chance that a fault will not be cleared within the intended zone. Operation of an upstream circuit breaker could then cause substantial outages.

Since underutilizing a CT also produces inaccuracies, the maximum anticipated load current should be as close to the CT current rating as possible without exceeding it. On CT's with 5-A secondaries, IEEE suggests that the CT be operated at 3 to 4 A during normal full-load currents (*4*). It has also been suggested that the CT and relay values be chosen such that the relay tap setting is at least one-half the current rating of the CT secondary (*5*). If this cannot be achieved, the performance of the CT should be checked.

In applications where saturation must be completely prevented, the CT should be sized to carry twice the peak flux associated with the symmetrical ac fault current (*5*). In most cases, it is not necessary to prevent saturation totally in order to provide adequate relaying. For instance, immediately after the initiation of a fault, the fault current might contain a substantial dc component. This component could cause the CT to become saturated, but in most mining applications, the dc component decays in less than ½ cycle. As high-voltage circuit breaker trip times are usually 3 to 4 cycles, the saturation during the first ½ cycle would be insignificant in terms of protection. If operation is in the linear curve portion, the CT would be functioning satisfactorily. Nevertheless, the maximum available fault current should be not than 20 times the CT current rating, and CT performance should be checked if the maximum is above this limit (*6*).

Accuracy Calculations

For a CT under a specific burden, the accuracy of its operation (transformer performance) can be calculated easily using manufacturer data, provided that the CT has a "C" accuracy class. As an example, suppose a 600:5 multiratio CT is connected through 50 ft of No. 12 AWG conductor to an induction-disk relay and an ammeter. Both overload and short-circuit protection are intended. Figure 10.16 gives the typical set of CT saturation characteristics, and burden data are contained in table 10.6 (*4*). The primary current (system line conductor) has 24,000-A symmetrical rms of available fault current. The objective of the calculation procedure is to find the percent ratio error.

Table 10.6.—Burdens of relay elements and ammeter connected to CT's

Element	*Burden*
Relay, timed element, 4–12 A pickup.	2.38 VA at 4 A at 0.375 pf; 146 VA at 40 A at 0.61 pf.
Relay, instantaneous element, 10–40 A pickup.	4.5 VA at 10 A; 40 VA at 40 A at 0.20 pf.
Ammeter	1.04 VA at 5 A at 0.95 pf.
Conductor (interconnecting)	0.08 Ω at 1.0 pf.
Transformer secondary resistance	0.298 Ω at 25°C.

Although the error should not exceed 10% under the most extreme circumstances (20 times the secondary current), satisfactory protective-relay operation often calls for smaller error, on the order of 2% or less. To compute the error, CT secondary burden and relay operating voltage must be known. The voltage shown in figure 10.16 is related to the CT secondary exciting current, the source of the secondary current error.

Burden should be calculated at maximum secondary current. In most cases this is the pickup current for the element providing short-circuit protection. Thus, the instantaneous element of the relay is assumed to be set at 40 A, which is 8 times rated secondary current or 4,800 A primary current. It was stated in chapter 5 that a direct summation of the burden voltampere ratings will produce adequate results, and while this is generally true, for precise work the impedance of each CT load should be found and then summed.

The time-element burden is 146 VA at 40 A and 0.61 pf, or an impedance,

$$Z_t = \frac{S}{I^2}\underline{|\cos^{-1}\text{pf}} = \frac{146}{(40)^2}\underline{|52^\circ}$$

or

$$Z = 0.091\ \underline{|53^\circ} = 0.0557 + j0.0723\ \Omega.$$

For the instantaneous element,

$$S = 40 \text{ VA at } 40 \text{ A } 0.20 \text{ pf}$$

or

$$Z_i = \frac{40}{(40)^2}\underline{|78^\circ} = 0.005 + j0.025\ \Omega.$$

For the ammeter, negligible saturation will occur at 8.0 times rated current (essentially an air-core magnetic circuit), and its impedance at 5.0 A is about the same as at 40 A,

$$Z_a = \frac{(1.04)}{(5)^2}\underline{|18^\circ} = 0.039 + j0.012\ \Omega.$$

The interconnecting wire resistance (R_w) and CT resistance (R_{ct}) are 0.08 and 0.298 Ω, respectively. Consequently, as all burdens are in series, the total burden impedance at 40 A is

$$Z = Z_t + Z_i + Z_a + R_w + R_{ct}$$

$$= 0.477 + j0.110 = 0.489\ \underline{|12.9^\circ}\ \Omega.$$

The magnitude of CT secondary voltage needed to produce 40 A is then

$$V = I|Z| = (40)(0.489) = 19.6 \text{ V}.$$

From figure 10.16, the secondary exciting current, I_e, at this voltage is

$$I_e = 0.03 \text{ A}.$$

Therefore, the percent ratio error correction can be found by

$$\%\ \text{ratio error} = \frac{I_e}{I_s}(100), \tag{10.9}$$

or at a secondary current, I_s, of 40 A,

$$\%\ \text{ratio error} = \frac{0.03}{40}(100) = 0.1\%.$$

Obviously, the accuracy for this application is extremely good.

Only one accuracy calculation example has been shown because as long as the data are available, this technique can be used for any CT application at any pickup current.

LOW-VOLTAGE CIRCUIT BREAKER TRIPS

In the preceding discussion, overload and short-circuit pickup settings were primarily related to external relays driven by CT's. The principles of pickup can also be applied to low-voltage power circuit breaker settings and the instantaneous magnetic settings of molded-case units. In this context, pickup would be the minimum current within a specified tolerance (usually ±10%) that would cause the trip element to activate the operating mechanisms (*4*). Although not entirely accurate, pickup can be loosely used to describe overload current as specified by the thermal trip elements of molded-case circuit breakers. However, as element tripping is a function of stored heat, tripping times rather than pickup are often employed. This section will focus on overload and short-circuit trip settings with molded-case breakers and then briefly outline trip levels for low-voltage power circuit breakers.

Overload Protection

Overload protection by thermal-magnetic molded-case breakers is mentioned in chapter 9. To review, most manufacturers calibrate their breakers to carry 100% of the continuous current at 40° C; thus, maximum continuous current is the rating of the thermal-trip unit. Derating is only necessary in noncompensating devices when the ambient temperature exceeds 40° C. With these thoughts, the overload procedure given for relay pickup protection can be adapted to this continuous current. The thermal element itself defines overload at 125%. For instance, the minimum trip time for a typical unit is around 10 s for 500% or more of the current rating, with time increasing inversely to about 1,800 s at 135% of the current rating (*20*).

Sections 75.900 and 77.900, 30 CFR (*16*), define breaker use according to Federal law for short-circuit and overcurrent (overload) protection of low-voltage and medium-voltage power circuits serving three-phase ac equipment. No further references are made to overcurrent protection except in section 75.900-2(c), which specifies that a circuit breaker protecting more than one branch circuit must be sized to afford overcurrent protection for the smallest conductor. Despite these references to overcurrent protection, Federal regulations are regularly interpreted and enforced to mean that overload protection is not required at each circuit breaker protecting a trailing cable (*17*).

Perhaps the lack of mandatory cable overload protection can be best explained in the following quotation from a design engineer working with ac power centers (*21*).

> Breaker thermal trip ratings should be applied to insure complete continuity of operation through all overload current peaks rather than to give complete cable protection. Many applications are being made with no breaker overload protection. Short circuit protection only, by means of the adjustable magnetic trip units, takes care of line to line cable faults. Motor control equipment on the machine takes care of motor overloads and may give some overload protection to the cable.

Some States have more stringent requirements than the Federal law for overload protection. For example, Pennsylvania requires overload protection on all breakers serving trailing cables (*12*). However, there are no specific requirements for sizing this overload protection, and so the engineer must select an appropriate size for the thermal-magnetic circuit breaker.

Short-Circuit Protection

The most common short-circuit protection is for trailing cables and is provided by the magnetic-trip elements of molded-case breakers. For surface mines, the pickup requirements listed for relay pickup protection are used; however, for underground coal mines, a set of maximum instantaneous settings based on conductor size are mandated in section 75.601-1, 30 CFR (*16*). These settings, listed in chapter 9, table 9.3, apply to both ac and dc systems and are based on (*2*):

1. An ideal 250-Vdc source feeding a bolted fault at the extremity of a 500-ft two-conductor trailing cable, and
2. A 50% safety factor to allow for circuit breaker tolerance, system impedance, and so forth.

These values are presently in effect for all underground coal mine trailing cables.

Fesak (*2*) has expressed concern that the dc basis of these values does not provide an adequate safety margin for resistance-grounded three-phase cables and has recommended a new set of maximum allowable instantaneous settings, which are shown in table 10.7. Here, a specific setting is selected not only by conductor size, but also by cable length and system voltage. The table stops with 4/0 cables because this is the maximum practical trailing-cable size for underground mines, but the recommendations extend to 1,000 MCM. The calculations on which these values are based assume an arcing line-to-line fault to find the minimum short-circuit current. The multipliers to obtain the minimum values from bolted three-phase fault currents are 0.85 at 480 V, 0.9 at 600 V, and 0.95 at 1,040 V.

Similarly, Vilcheck (*18*) has recommended a refined set of maximum instantaneous settings for dc trailing cables, based on minimum expected short-circuit current. These pickups are given in table 10.8 and correspond to conductor size, cable length, system voltage, and the method for grounding. Two types of grounding are included in the table: by means other than a grounding conductor, such as diode grounding, and via a grounding conductor. Where there is no grounding conductor, only line-to-line faults are of concern and the ℓ–ℓ values in table 10.8 are used. With grounding conductors, line-to-line ground faults can also occur, and if no ground-fault protection is used, a lower circuit breaker setting is needed for cables No. 6 AWG and larger because of the smaller grounding conductor (ℓ–g values in table 10.8).

Table 10.7.—Recommended instantaneous trip settings for 480-, 600-, 1,040-V three-phase trailing-cable protection

Conductor size, AWG	Cable length, ft	Maximum instantaneous circuit breaker setting, A		
		480 V	600 V	1,040 V
14	0– 500	75	100	NAp
12	0– 500	125	150	NAp
10	0– 500	200	250	NAp
8	0– 500	300	400	NAp
6	0– 550	400	550	850
4	0– 500	700	850	1,200
	501– 600	600	750	1,050
3	0– 500	850	1,050	1,350
	501– 650	700	850	1,150
2	0– 500	1,000	1,200	1,450
	501– 600	900	1,050	1,350
	601– 700	750	950	1,250
1	0– 500	1,200	1,350	1,600
	501– 600	1,050	1,200	1,450
	601– 750	850	1,050	1,350
1/0	0– 500	1,400	1,550	1,750
	501– 600	1,250	1,400	1,650
	601– 750	1,050	1,200	1,500
	751– 800	1,000	1,150	1,450
2/0	0– 500	1,600	1,700	1,800
	501– 600	1,400	1,550	1,750
	601– 750	1,200	1,350	1,650
	751– 850	1,100	1,250	1,550
3/0	0– 500	1,900	1,950	1,900
	501– 600	1,700	1,800	1,800
	601– 750	1,450	1,600	1,700
	751– 900	1,300	1,450	1,600
4/0	0– 500	2,050	2,100	1,950
	501– 600	1,850	1,950	1,900
	601– 750	1,650	1,750	1,800
	751–1,000	1,350	1,500	1,650

NAp Not applicable.

Low-Voltage Power Circuit Breakers

Trip settings for low-voltage power circuit breakers can involve three direct-acting elements (*4*). The requirement for long-time delay elements is the same as for the overload discussed in relay pickups, and the unit is usually set at 100% of the rating, regardless of tolerance. In general applications, the short-time-delay element is set at five times the overload current point, with the instantaneous element set at nine times the overload level. However for mining applications, except for trailing-cable protection, the short-circuit requirements covered in relay pickups also apply to the instantaneous device. Trailing cables demand the same maximum short-circuit settings discussed in the preceding section.

Table 10.8.—Recommended instantaneous trip settings for 300- and 600-Vdc trailing-cable protection (*19*)

Conductor size	Cable length, ft	300–V maximum instantaneous setting, A		600–V maximum instantaneous setting, A	
		ℓ–ℓ	ℓ–g	ℓ–ℓ	ℓ–g
AWG:					
14	0– 500	50	50	50	50
12	0– 500	75	75	100	100
10	0– 500	75	75	200	200
8	0– 500	100	100	300	300
6	0– 500	200	100	450	350
4	0– 500	400	250	650	500
	501– 600	300	150	550	450
3	0– 500	500	350	700	550
	501– 650	400	250	600	450
2	0– 500	600	400	800	650
	501– 600	500	350	750	600
	601– 700	450	300	700	550
1	0– 500	850	600	1,350	1,050
	501– 600	700	500	1,200	900
	601– 750	600	400	1,050	750
1/0	0– 500	1,050	750	1,550	1,200
	501– 600	900	600	1,400	1,050
	601– 750	750	500	1,200	900
	751– 800	700	450	1,150	850
2/0	0– 500	1,250	900	1,750	1,400
	501– 600	1,100	750	1,600	1,250
	601– 750	900	600	1,400	1,100
	751– 850	800	550	1,300	1,000
3/0	0– 500	1,500	1,100	1,950	1,600
	501– 600	1,300	950	1,800	1,450
	601– 750	1,100	750	1,600	1,250
	751– 900	950	650	1,450	1,100
4/0	0– 500	1,750	1,350	2,150	1,800
	501– 600	1,550	1,150	2,000	1,650
	601– 750	1,350	950	1,800	1,450
	751–1,000	1,050	750	1,550	1,200
MCM:					
250	0– 500	1,950	1,500	2,300	1,950
	501– 600	1,750	1,300	2,150	1,800
	601– 750	1,500	1,100	1,950	1,600
	751–1,000	1,200	850	1,700	1,350

ℓ-g Line to ground (with grounding conductor).
ℓ-ℓ Line to line (without grounding conductor).

FUSES

Overload and short-circuit protection can be provided by dual-element fuses on low-voltage systems or by power fuses on high-voltage systems. The sizing of fuses is basically the same as that for relays or tripping devices in conjunction with circuit breakers. In cable overload protection, the continuous-current rating of a fuse cannot exceed the lowest ampacity of any conductor in the protected system. Beyond this, as a general protection guide, fuses should always be sized to the very smallest continuous-current rating that will safely carry any load that should be maintained (*4*).

Transformer inrush currents, if not heeded, can be a problem with high-voltage fuses and for relay pickup as well. For instance, upon energizing, a power transformer can draw 8.0 to 12 times its full-load current rating for a duration up to 0.1 s (*4*). Expulsion and boric acid fuses can be obtained that have inverse-time characteristics to withstand this, thereby affording both overload and short-circuit protection for the transformer. High-voltage current-limiting fuses, because of their fast response, must often have a continuous-current rating to account for inrush. In these instances, current-limiting fuses can provide only short-circuit protection.

The use of fuses to provide ground-fault protection is more complex. In high-resistance-grounded systems, fuses cannot be used to protect against ground faults; in fact, their use in series with grounding conductors can create a serious personnel hazard if they open. During line-to-neutral faults, the current in these systems is limited to a level lower than typical load current in the line conductors; line-to-line-to-neutral faults cause about the same current levels. As there are no neutral-connected loads in these systems, fault-current contribution is only from generating stations and capacitance; that is, motors have no effect.

However in ungrounded or solidly grounded systems, ground-fault currents can be substantially greater; thus, line-interrupting devices including fuses can provide ground-fault protection. For instance, in an ungrounded system, fuses sized to the short-circuit requirements discussed in this chapter would probably cause interruption during two simultaneous line-to-neutral faults on different power conductors.

COORDINATION

The basic principles of coordination have already been discussed in chapter 9 and earlier in this chapter, but here the specific procedures used in coordination studies will be demonstrated. References *1–2* are highly recommended for more detailed coverage of this topic.

A coordination study is basically a comparison of the times-to-operate for individual protection devices under both normal and abnormal current flows (*3*). A preliminary study should always be made in the early stages of mine power-system design, since it could indicate that transformer sizes or impedances require modification or that cable sizes need to be changed. After final selection of devices and components, a second study must be made to confirm the tentative study. A new coordination investigation is required whenever the original system is changed, whenever new loads are added, or whenever existing equipment is replaced with higher rated components. An additional study should always be made if a fault causes a major shutdown of an existing system.

Coordination has two conflicting objectives: protection and selectivity (*3*). Each relay, tripping unit, or device is designed to protect a specific circuit segment and for maximum safety must isolate each faulted circuit as fast as possible. At the same time, continuity of service must be provided, and this requires selectivity. Again, selectivity is the concept of isolating only that portion of the system experiencing failure. This demands successively longer fault-clearing times or higher pickup settings for each protection zone from the loads to the source. It is the role of the coordination study to optimize these requirements without compromising personnel safety.

A coordination-curve plot is one of the best methods for achieving these objectives. For resistance-grounded systems, two plots can be made, one for line-conductor protection and the other for ground faults. The protection described here refers mainly to line-conductor protection plots but also applies to grounding.

Before a curve plot can be constructed, a good one-line diagram of the system must be prepared. In addition to the normal diagram features discussed in chapter 4, it should include the following information (*3–4*):

1. Maximum and minimum levels of short-circuit current that are expected to flow through each protective device (obtained from a system fault analysis);
2. Normal and maximum load currents for each system portion (from a load-flow analysis);
3. CT ratios; and
4. Relay, circuit breaker, and fuse ratings and adjustment ranges (time-current characteristics must also be known).

With these data available, as shown in figure 10.17, the coordination study can proceed to plotting.

Coordination curve plots are normally drawn on standard log-log graph paper, using the vertical scale for time and the horizontal axis for a common current source (*3*). For good results, this scale usually corresponds to currents at the lowest voltage level, while currents at higher voltage levels are plotted on the same scale as equivalents. For instance, if utilization is at 600 V with distribution at 7,200 V, the current scale is calibrated for 600 V. Currents in those system portions are plotted directly, but distribution currents are multipled by 7,200/600 or 12, then plotted. However, note that this is not a firm rule, as any convenient scale can be used. Important current levels such as transformer inrush, motor starting, and fault current can be placed on the graph as points.

A critical protection path is then chosen from the one-line drawing by selecting a critical load or safety hazard and following the protection path back to the source. Fortunately, because of radial distribution, this selection is an easy matter in most mine power systems. The time-current characteristics of all protective devices along the path or affecting the path are then plotted on the graph.

As shown in figure 10.18 (*4*), such a plot is invaluable as it allows protection-device action to be visualized throughout the system on the same current basis. This is an advantage similar to that of per-unit analysis. It also provides for the normalization of the characteristics for each protection device, which rarely have the same shape or even scale. The plot simplifies the selection of current and time settings that will provide the best possible protection and safety while also giving selectivity. One way to do this is by specifying the characteristics of the most downstream protection, then sequentially positioning each protection step toward the source (or vice versa).

As is evident in figure 10.18, operating time intervals must be maintained between protective devices in order to

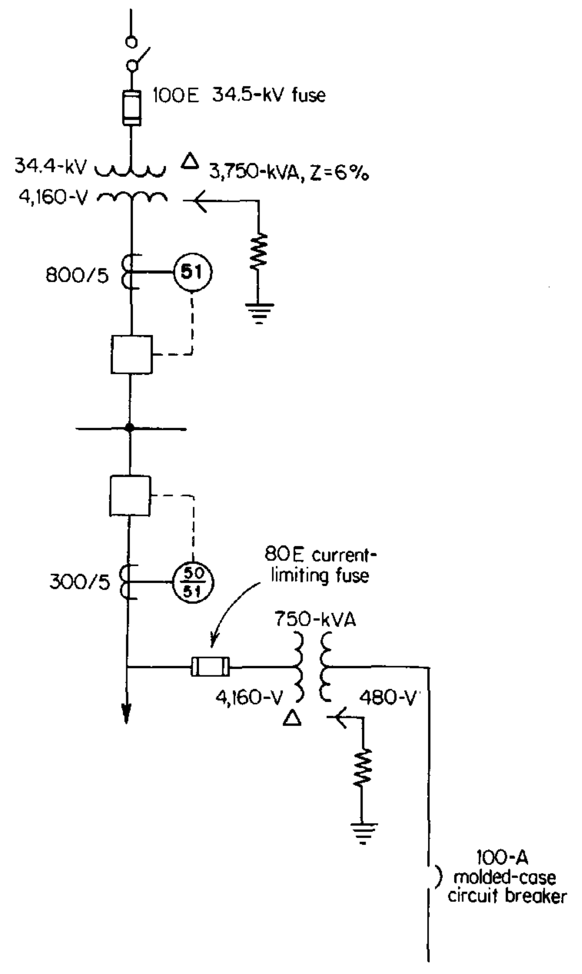


Figure 10.17.—Example of one-line diagram for preparing a coordination curve plot for one path.

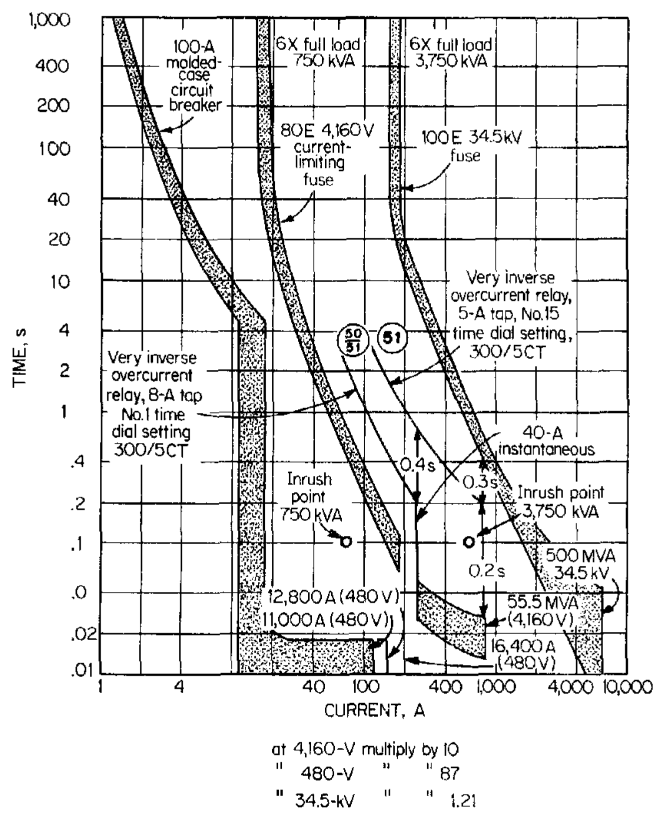


Figure 10.18.—Coordination curve plot for figure 10.17 showing various protective-device characteristics.

achieve correct sequential operation. When applied to relay-activated circuit breakers, this time margin must allow for circuit breaker interrupting time, overtravel time, and a safety factor (*4*). All these times are additive.

The time required by a circuit breaker to interrupt current once it receives the trip signal is equal to its speed in cycles divided by 60. A typical power circuit breaker has an operative speed of 5 cycles, corresponding to an 0.08-s operating time. Overtravel is important in electromechanical time-delay relays, especially inverse-time, induction-disk types. Owing to moving-part inertia, these relays will continue to close their contacts even after the fault is removed, and overtravel can thus create a relay operation delay. Typical overtravel time for inverse-time, induction-disk relays is 0.10 s. The safety factor mainly allows for variations in relay characteristics due to manufacturing differences, tolerance, aging, and dust; the commonly assigned range is 0.12 to 0.22 s. Summing these factors results in a necessary time margin range of 0.3 to 0.4 s (*4*). While the typical case is considered to be 0.4 s, the margin can be reduced to 0.3 s for carefully tested systems. Note that static relays eliminate overtravel and allow the use of the minimum safety factor, so that the typical time margin can be reduced to 0.2 s (see chapter 14).

The typical time interval of 0.4 s is used for relay-to-relay coordination in figure 10.18, and the time selections for line-conductor relays can be seen. Although this time interval considered only external relay tripping, the 0.4-s time margin is often utilized even when direct-acting low-voltage circuit breakers are coordinated together or with relayed interrupters. The same is true when a circuit breaker is coordinated with an upstream fuse if the total fuse clearing time is greater than 1.0 s. With less than 1.0 s, the fuse-to-circuit-breaker time margins can be reduced as low as 0.1 s (*4*).

For an additional example of coordination, consider ground-fault protection in a high-resistance-grounded distribution system. A separate plot can be made for the ground-fault conditions. If the induction-disk relay at the most downstream switchhouse is set to pick up with the minimum time dial setting, the time dial on the ground-fault relay in the next upstream switchhouse would be set 0.4 s higher. At each subsequent upstream switchhouse, the ground relay time would be set progressively higher until the neutral bushing of the source transformer is reached. These time allowances also involve the current equivalent of any potential relaying about the grounding resistor. Because of the need to maintain the time intervals, many engineers have found the coordination of more than five relays in one ground-fault current path nearly impossible, since the maximum practical time setting is about 2.0 s. Although coordination curve plotting has been discussed here, in practice the manufacturer characteristic curve could be used in this case. The reason is that available ground-fault current in a high-resistance-grounded mine distribution system remains fairly constant, so that the same very inverse relays could be used for all ground-fault protection, with the possible exception of resistor potential relays.

The material presented in this chapter and the last has perhaps given the impression that in order to create a safe mine power system all circuits should have overload, short-circuit, and ground-fault protection. While this is generally true, it is not quite the entire picture. Worker safety must always come first; hence, equipment protection is not applied in those cases where its use can cause hazards to personnel. These exceptions almost always involve unit-designed equipment, such as elevator motors or the swing, hoist, and propel motors of surface excavators. The only other exceptions to mandatory equipment protection are cases where there is no possibility of compromising personnel safety if it is omitted.

This chapter has introduced the important parameters used in the selection and adjustment of protective devices. This information together with the material in chapter 9 serves as one basis for chapters 12 and 13, where the components and techniques are applied to the equipment used in the mine system. But first another type of protection must be discussed, protection against the sometimes hidden dangers of transients and overvoltages.

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CHAPTER 11.—TRANSIENTS AND OVERVOLTAGES

The term *electrical transient* can mean different things to people of different interests. However, there are some common ideas. As defined by Greenwood (*19*),[1] "an electrical transient is the outward manifestation of a sudden change in circuit conditions, as when a switch opens or closes, or a fault occurs on a system." The circuit parameters of inductance and capacitance are found in any power system to some extent. When the system is changed, the quantities of current, voltage, magnetic flux, and so on, do not instantly assume new values. Rather, they go through a transition to reach the new steady-state condition (*33*). It is this transition period that gives rise to the transient voltages and currents.

Transients are of short duration, and the time in which they occur is an extremely small percentage of the total operating time. But it is during these short periods that some of the greatest electrical stresses can occur, mainly because of excessive currents or voltages. The excessive voltage is often critical in the design of mine power systems. In extreme cases, system parts are damaged and equipment failure follows. Safety can be further compromised because anomalous voltages may exist at machine frames.

Transients are a fact of life on every power system, yet careful design within the technical and economic constraints of a system can result in a reduction of transient-related component failures. However, mine power systems are particularly vulnerable to transient-induced failures because their operation and arrangements are extremely dynamic. As a mining activity advances, the electrical system is expanded, often on a weekly basis. Although this expansion is normally designed into the system, circumstances in the mine can call for additional modifications. In some cases, these on-the-spot modifications may not be in line with sound engineering principles.

A lack of knowledge about transients has resulted in power-system requirements that could be misunderstood by some mine power engineers. Furthermore, in the process of maintaining the system, faulty components are sometimes replaced with devices having different specifications or even total incompatibility. These two factors also increase the vulnerability of mine power systems to transient occurrences.

TRANSIENT SOURCES

There are several discrete sources of transient overvoltages. Although slight differences exist in classifying these events, IEEE standards catalog seven types (*23*):

1. Lightning,
2. Switching surges,
3. Static,
4. Contact with a higher voltage system,
5. Line-to-ground faults,
6. Restriking ground faults, and
7. Resonant conditions.

The following paragraphs introduce these sources briefly in light of mining applications. It should be noted that references *19*, *33*, and *37* are invaluable sources of transient information, and these should be consulted for detailed coverage of transient phenomena and problems.

LIGHTNING PHENOMENA

On a global scale, the earth and the lower part of the ionosphere may be considered as conductive bodies separated by a rather poor conductor, the atmosphere. The entire system is analogous to an enormous capacitor with a leaky dielectric. Any charge unbalances that accumulate are dissipated by sudden breakdowns in the dielectric (atmosphere), and the resulting current (lightning) is of short duration but high amplitude. At any given time, electrical storms are taking place somewhere on earth, and the average current flow between air and earth is more or less constant at a level of 1,500 A. During fair weather, the normal voltage gradient in the air near the earth's surface is about 3.0 V/cm, but this rises to around 500 V/cm beneath a thundercloud. The potential difference needed to initiate a lightning stroke is on the order of 50 million V (*12*). In the continental United States, a typical stroke lasts for only a fraction of a second, yet releases a tremendous amount of energy, approximately 200 million J. The total quantity of charge involved in the average stroke is about 20 C, and the peak current value is around 20 kA (*28*).

As shown in figure 11.1, the shape of the discharge starts with an extremely short-duration voltage pulse, whose crest may reach 5 MV or more (*29*). This is closely followed by the current waveform, which rises more slowly to its peak value and lasts longer. About 82% of the strokes occurring in the United States are negative in polarity; that is, they transfer electrons from the clouds to earth. Figure 11.2 illustrates the wide distribution of current magnitudes that may be expected in various lightning strokes (*27*).

Figure 11.3 shows the expected number of thunderstorm days per year for any of the 48 contiguous States (*27*). The probability that a particular object will be hit by lightning depends upon the cloud charge intensity, the geographical locale, and the height of the object in relation to its surroundings. Charge intensity is a parameter because an increased charge in the tip of the stroke generates a proportionally higher electric-field gradient. This determines the attractive range of the stroke, in other words, the horizontal distance from the tip of a downcoming leader to an object receiving the stroke. As

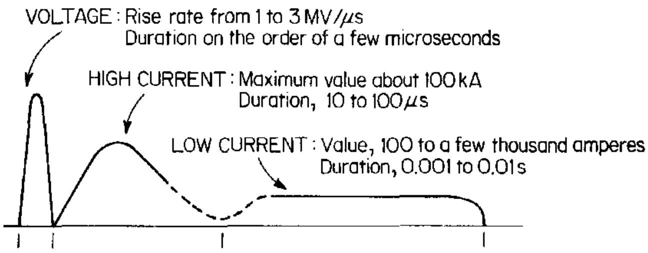

Figure 11.1.—Schematic representation of lightning stroke discharge.

[1] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

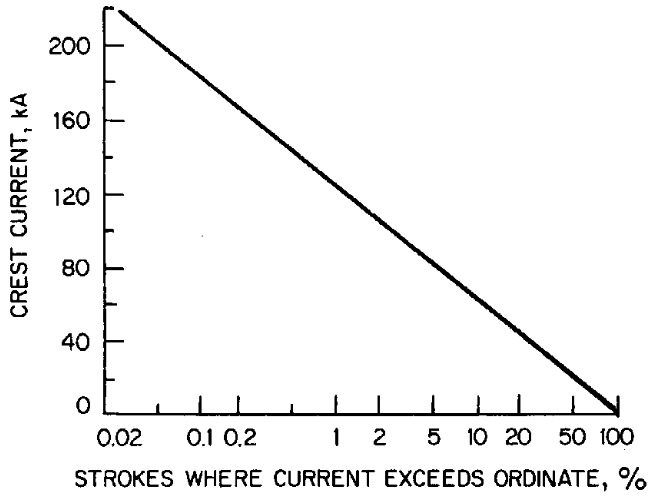

Figure 11.2.—Distribution of crest currents in lightning strokes.

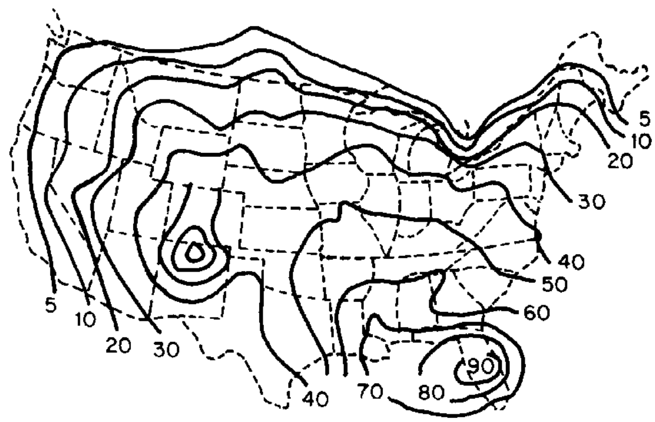

Figure 11.3.—Map showing average number of thunderstorm days per year in the United States.

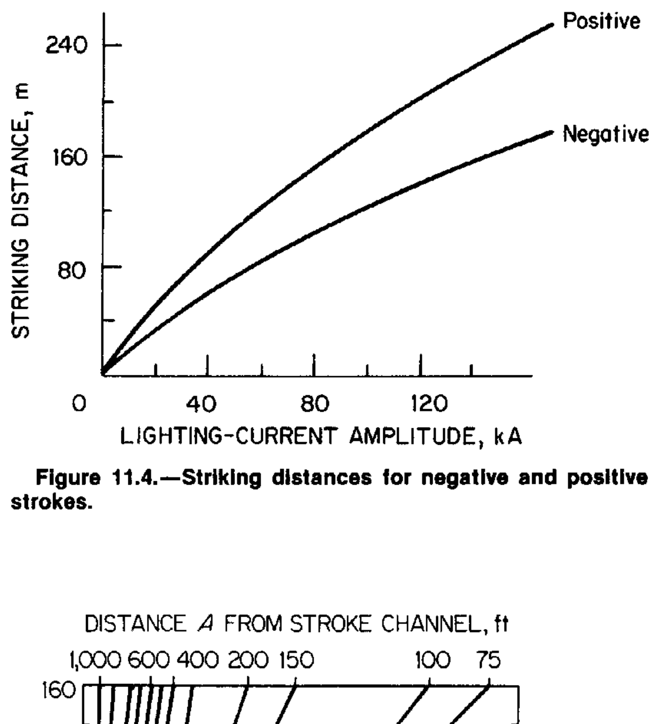

Figure 11.4.—Striking distances for negative and positive strokes.

shown in figure 11.4, the striking distance for a typical 20-kA stroke is about 30 m or 100 ft (*27*).

The exposed surface portions of a mine electrical distribution system are particularly vulnerable to lightning. For instance, overhead lines on wooden poles are often used to carry three-phase power from substations to surface facilities of underground mines. In surface mines, the overhead lines can extend close to the shovels and other electrically operated equipment in the pit. Since these poles can also carry the grounding conductor from the safety ground bed, it is doubly important that they be adequately protected from lightning.

Three separate failure modes are recognized whereby an electrical distribution system can be damaged by lightning. If the impedance to ground of a supporting structure is high and the structure is struck by lightning, its potential may become elevated above ground to the point that a flashover occurs from the tower to a power conductor. This is known as backflashover. If a high-intensity stroke to earth occurs nearby, the resulting electric field may be great enough to induce excessive potentials on conductors; a power conductor can then arc over to the tower. In a shielding failure, lightning hits a power conductor directly, raising its potential to the point where arcing takes place between the conductor and the support structure (*27*). Induced potentials on grounding conductors, including equipment frames, can elevate the potential of the grounding system above that of earth and cause damage to protective devices or create a personnel hazard. With nearby strokes to earth, the magnitude of the induced voltage depends upon stroke current, distance, and height of the conductor, as illustrated in figure 11.5 (*25*).

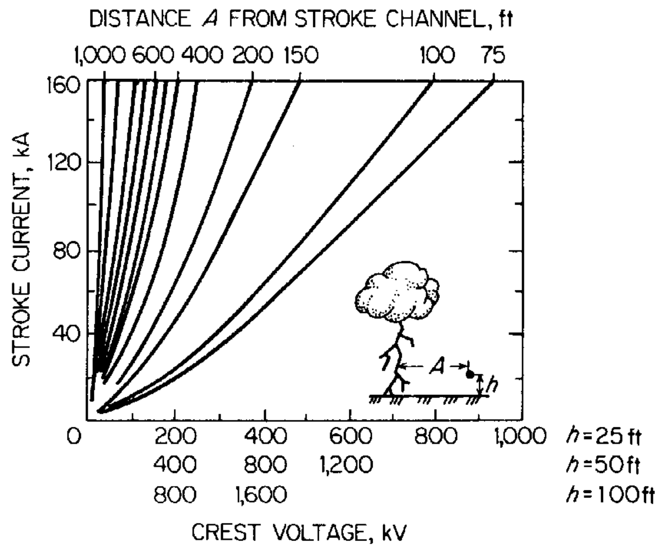

Figure 11.5.—Crest voltages induced on transmission lines by nearby strokes.

SWITCHING TRANSIENTS

Switching operations account for the majority of all transient phenomena on mine power systems, and except for direct strokes by lightning, the most destructive power-system transients are initiated by this source. Every time a switch is operated, a transient can occur, and these transients can be classified into two types: normal and abnormal (*33*). Normal voltage and current transients are considered a characteristic of proper operation, and their maximum outcome is a transient voltage or current no greater (but usually less) than two times the peak steady-state voltage or current of the system. Transients exceeding this level are termed *abnormal*, and these are the result of incorrect component operation or design. Abnormal switching transients can occur in several ways, but all involve the release or exchange of trapped energy in the power-system inductance or capacitance (*19*, *33*). Interestingly, a normal transient can be generated on a quiet power system, but if energy from this transient is stored and afterwards a second transient is created, the latter can be abnormal.

Transient phenomena caused by switching can be divided into those created on ac systems and those associated with dc. The ac switching events can be further subdivided into capacitive switching, current chopping, and prestrike.

Capacitance Switching

Because of the extensive use of cables, surge capacitors, and power-factor correction capacitors, capacitance plays a major role in most mine electrical systems. Abnormal transients can occur when ac current to capacitance is interrupted (*6*, *19*).[2] The problem is that current leads voltage by nearly 90°. Capacitive currents are usually rather small so there is a good chance that current flow is stopped at a very early current zero. Thus, the load-side capacitance would be charged to the peak voltage of the system, and this energy becomes trapped.

A simple circuit to demonstrate capacitance switching is shown in figure 11.6, where an ac source is feeding cable or line lumped capacitance through a circuit breaker. Source voltage and current waveforms are given in figure 11.7*A*; the point of current interruption is signified by the dashed vertical line. Figure 11.7*B* shows the voltage across the capacitance, and figure 11.7*C* relates the potential or recovery voltage across the circuit breaker contacts, which is the difference between the source and load sides. System inductance and stray capacitance are neglected in these waveforms to illustrate the effects of the trapped charge (*19*).

Because early interruption is possible, the recovery voltage may attain two times the system peak when the contact gap is quite small. Hence the arc may reignite or restrike. With restrike, an inductance-capacitance circuit is formed with a resonant frequency of

$$f_o = \frac{\omega_o}{2\pi} = \frac{1}{2\pi\sqrt{LC}}, \qquad (11.1)$$

where f_o = resonant or natural frequency of system, Hz,
L = system inductance, H,
and C = system capacitance, F.

[2] Personal communication from E. K. Stanek, West Virginia University, Aug. 1977.

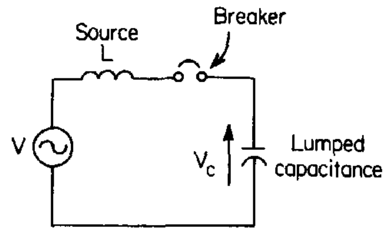

Figure 11.6.—Simple circuit to illustrate capacitance-switching voltage transients.

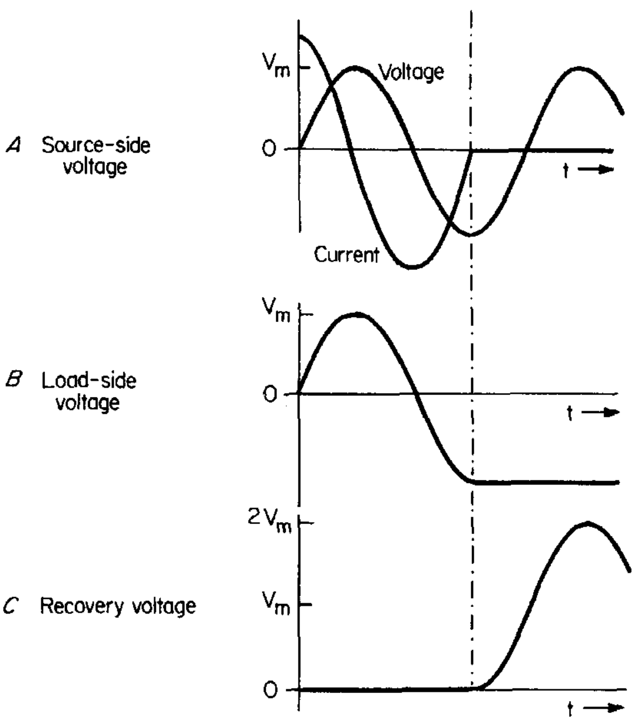

Figure 11.7.—Voltage and current waveforms before and after current interruption.

The basic equation for the per-phase voltage after restrike is

$$v = L\frac{di}{dt} = v_c, \qquad (11.2a)$$

where $v = V_m\cos(\omega t)$ = source voltage, V,
V_m = peak source voltage, V,
$v_c = V_{c(o)} + \frac{1}{C}\int i\,dt$ = capacitance voltage, V,
$V_{c(o)}$ = voltage across capacitance at restrike, V,
and i = current through circuit breaker, A,

or $$L\frac{di}{dt} + \frac{i}{C}\int i\,dt = V_m\cos(\omega t) - V_o. \qquad (11.2b)$$

To investigate the current transient in the short time interval after restrike,

$$\cos(\omega t) \approx 1,$$

and thus,

$$L\frac{di}{dt} + \frac{1}{C}\int idt = V_m - V_o. \qquad (11.2c)$$

Solving this equation for the time-domain current, it is found that

$$i(t) = (V_m - V_o)\left(\frac{C}{L}\right)^{0.5} \sin(\omega_o t). \qquad (11.3)$$

Accordingly, the voltage across the capacitance would be

$$v_c = V_o + \frac{1}{C}\int_o^t (V_m - V_o)\left(\frac{C}{L}\right)^{0.5} \sin(\omega_o t)\,dt$$

or

$$v_c = V_o + \frac{V_m - V_o}{(LC)^{0.5}}\int_o^t \sin(\omega_o t)\,dt$$

or

$$v_c = V_o + \frac{V_m - V_o}{(LC)^{0.5}}[1 - \cos(\omega_o t)]. \qquad (11.4)$$

For the worst case,

$$V_o = -V_m$$

or

$$V_m - V_o = 2V_m,$$

and the transient current is

$$i(t) = 2V_m\left(\frac{C}{L}\right)^{0.5}\sin(\omega_o t), \qquad (11.5a)$$

with transient voltage per phase for the load-side capacitance being

$$v_c = V_m - 2V_m\cos(\omega_o t). \qquad (11.5b)$$

Both these sinusoidal waveforms are illustrated in figure 11.8; the oscillations are at the system natural frequency (*19*). Equation 11.5*b* is of special interest as it indicates that the transient voltage can reach three times the peak system voltage. The decay shown in the waveforms is due to damping by system resistance. Obviously, system resistance is neglected in the foregoing derivation, but its effect is considered small compared with capacitance and inductance in this type of switching.

To show the significance of the current transient, consider figure 11.9, which is a per-phase diagram of a three-phase motor circuit. Here, three-phase 4,160-V power is being fed through a circuit breaker (used as a motor starter) through 25 ft of SHD cable to a 1,500-hp wound-rotor motor. A 300-kvar capacitor bank is used for power-factor correction and is placed at the load side of the breaker. Assume that the motor is drawing very little current, the breaker is opened, and interruption occurs at the first current zero. If the motor current is very small compared with the current drawn by the capacitor, the stage is set for restrike. Peak restrike current would be approximately

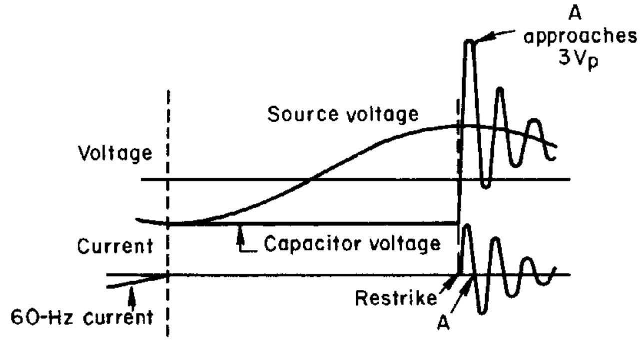

Figure 11.8.—Voltage and current transient waveforms occurring with capacitance switching and restrike.

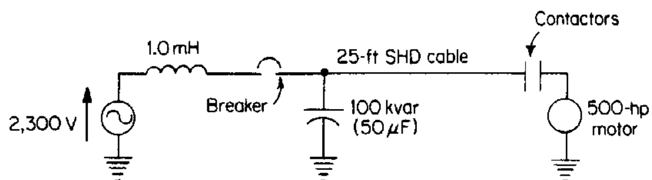

Figure 11.9.—Per-phase diagram of 4,160-V pump-motor circuit.

$$I_p = 2V_p\left(\frac{C}{L}\right)^{0.5}$$

or

$$I_p = 2(3{,}396)\left(\frac{5 \times 10^{-5}}{10^{-3}}\right)^{0.5} = 1{,}520 \text{ A}.$$

Since the steady-state current through the capacitor is around 45 A, this transient-current peak is about 34 times the normal capacitance current flow, approaching a level expected for motor-starting inrush.

Even though this is significant, the serious problem is not the current transient but the possible overvoltage produced. It can be seen from equation 11.5*b* that, after one restrike, the per-phase voltage can approach 3 V_m, 3(3,396) V, or about 10 kV. This is substantial but still within the insulation capabilities of most 4,160-V systems. However, if the capacitor voltage is still around 3 V_m when current becomes zero again, a current interruption will trap 3 V_m across the capacitor. Now, the circuit breaker recovery voltage may approach 4 V_m, causing a second restrike, which can then cause a capacitor voltage of −5 V_m. This situation can continue, developing even higher voltages. Such a multiple-restrike process is shown graphically in figure 11.10 (*33*). The voltages created can cause insulators to flashover.

The problem here is large capacitance as seen by the switching apparatus, with a very small impedance between that capacitance and the load-side contact. Examples of this situation could include the charging current to an unloaded cable or distribution line. Yet perhaps the most drastic instance would be interrupting current to a static capacitor bank, involving one line of a grounded bank or two lines of an ungrounded bank.

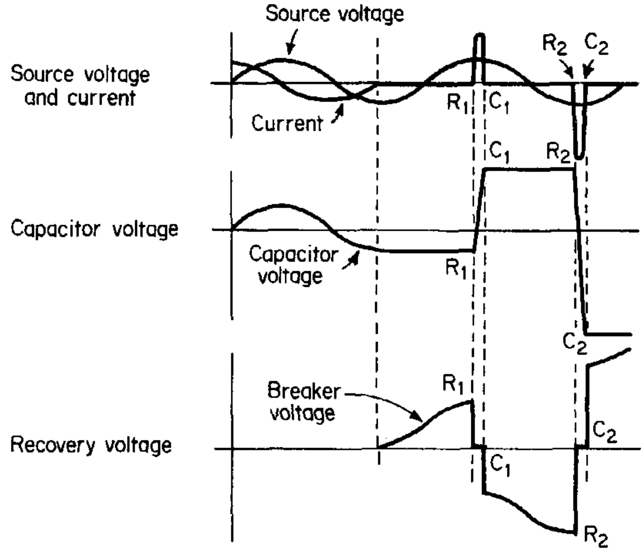

Figure 11.10.—Voltage and current waveforms resulting from multiple restrikes after capacitance switching.

Current Chopping

Current chopping is the phenomenon of forcing current to zero before a natural current zero. This can occur when small currents, such as transformer magnetizing current, are interrupted by switching apparatus, as illustrated graphically in figure 11.11 (*19, 26*). Current chopping can trap magnetic energy in the power-system segment being interrupted, and the result can be severe transient voltages.

In recent years, current-chopping transients have received more attention than any other type. The concern has been connected with the increased use of vacuum circuit breakers (VCB's), perhaps to the point where these transients have become associated with the use of VCB's. Through their high efficiency, these interruptors can easily chop small current flow. However, it should be noted that all circuit breakers can cause current chopping, as well as certain fuses, especially the current-limiting types.

To appreciate the magnitude of the created overvoltages, consider the simplified equivalent circuit of a power-system segment shown in figure 11.12. The components can be considered per phase for the three-phase system. If the circuit breaker chops the current at magnitude I, magnetic energy is stored at that instant in the system inductance (mainly the transformer) at a level (*33*)

$$W_L = \frac{LI^2}{2}, \tag{11.6}$$

where W_L = energy stored, J,
L = transformer magnetizing and cable inductance, H,
and I = magnitude of chopped current, A.

This stored energy is then transferred to the capacitance and charges the capacitance to

$$W_c = \frac{CV^2}{2}, \tag{11.7}$$

where V = voltage produced across capacitance, V,
and C = lumped capacitance of cable and transformer, F.

Neglecting losses, both energies are equal, producing a voltage

$$V_p = I\sqrt{\frac{L}{C}} = IZ_o, \tag{11.8}$$

where V_p = peak transient voltage, V,
and Z_o = system surge (or characteristic) impedance, Ω.

Following the capacitance charging, the energy is transferred back to the system inductance. The effect of transferral back and forth causes oscillations with a frequency equal to

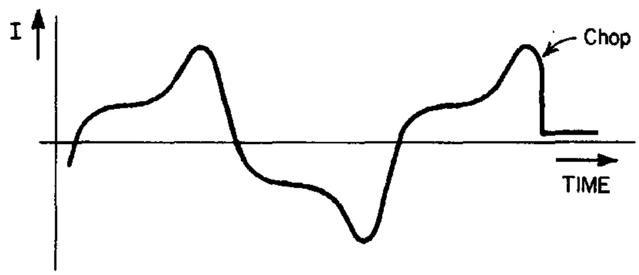

Figure 11.11.—Graphic example of current chopping by breaker interruption.

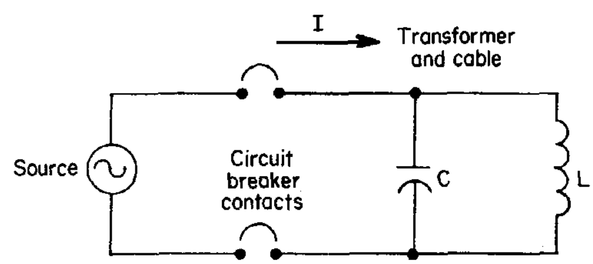

Figure 11.12.—Equivalent circuit of power-system segment with lumped components per phase, neglecting resistance.

$$f_o = \frac{1}{2\pi}\left[\frac{1}{LC} - \left(\frac{R}{2L}\right)^2\right]^{0.5}, \tag{11.9}$$

where f_o = oscillation frequency, Hz,
and R = system resistance, Ω.

Equation 11.9 shows the damping effect of system resistance, but for very small resistances (as are usually the case in power systems) the equation reduces to approximately

$$f_o = \frac{1}{2\pi\sqrt{LC}}. \qquad (11.10)$$

This is again the natural frequency of the power system.

The preceding has considered the theoretically pure world and, of course, actual systems exhibit losses. Present-day dry-transformer hysteresis losses limit energy storage to about 40% (*20*). Therefore, the peak transient voltage is restricted to about 63% of equation 11.8 or

$$V_p = 0.63I\sqrt{\frac{L}{C}} = 0.613IZ_o. \qquad (11.11)$$

An illustration of a voltage transient resulting from chopping (by trapping energy) is given in figure 11.13.

Consider the power system shown in figure 11.14, where a switchhouse is connected to a load center by a short cable. A typical switching procedure in mining is to interrupt or switch out an unloaded load center, for example, during a maintenance shift. Even though there are no connected loads on the secondary, the transformer still draws a small amount of magnetizing current, about 0.03 to 0.05 pu of rated current for most mining applications. Assume that the cable is so short that it has negligible capacitance and inductance as compared with the transformer. Thus, the equivalent circuit would be as shown in figure 11.12.

A typical load center in underground coal mining is 750 kVA, and common unloaded transformer values for a 7,200-Vac system are C = 3,000 pF and L = 15 H. If 1.0 A is chopped by the circuit breaker, the per-phase crest voltage that can be produced on the system is

$$V_p = (0.63)(1.0)\left(\frac{15}{3 \times 10^{-9}}\right)^{0.5} = 44.5 \text{ kV}.$$

This level is indeed high, being directly proportional to the product of the system surge impedance and the chopped current. Hence, if 2.0 A is chopped (for example, the peak due to distortion), V_p would equal 89.1 kV. Significantly, the level is independent of rated system voltage but is inversely proportional to the square root of the load-side capacitance. Common mine power-distribution systems employ 4,160, 7,200, or 12,470 V, and transformers of the same capacity (kVA) have similar cores, surge impedances, and magnetizing currents; therefore, similar energy levels can be stored by chopping, and similar peak transient voltages produced, regardless of system voltage.

The magnitude of chopping obviously depends upon the instant of switching. For example, if switching occurred at the natural current zero, no chopping could result, and the transient would be normal. Yet the probability of this happening is extremely small. Consequently, if the surge impedance is high, there is a likelihood that serious transients can be created. Configurations similar to that described above can also result in large chopping transients, for example, a cable-connected motor. However, surge impedance and magnetizing current are smaller in a motor than in a transformer, which reduces the danger of large chopping transients.

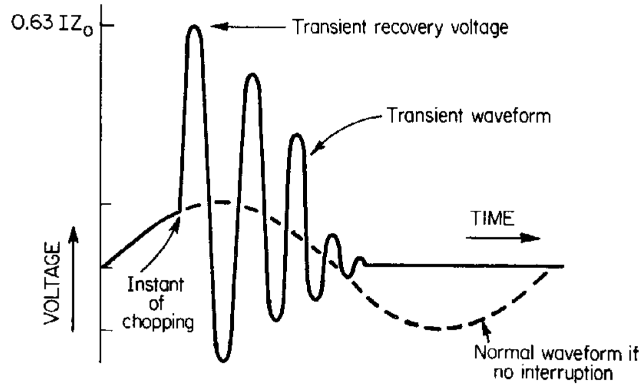

Figure 11.13.—Graphic example of chopping voltage transients.

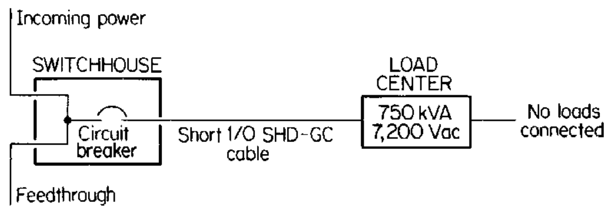

Figure 11.14.—Segment of mine power system.

In addition to system losses, there are other phenomena that can reduce the transient voltage. After chopping, with only a small VCB contact separation, the dielectric in the circuit breaker is often unable to support the transient voltage, and an arc restrikes (*19*). Sometimes the circuit breaker will make successive attempts to clear the circuit in this manner, reaching progressively higher voltages as the contact gap increases, until isolation is finally established. But the maximum voltage attained may not be as great as when switching is clean; the stored energy is not allowed to accumulate, and the effect can reduce V_p by as much as one-half (*20*). Nevertheless, the sequential restriking and clearing of a circuit breaker can create dangerous overvoltages, and the voltage escalation can be much more rapid (*19*).

If the cable length between the switchhouse and the load center is increased, the cable capacitance will be proportionally increased. This will reduce the surge impedance and therefore the possible severity of the chopping transient. Yet too large a capacitance can result in other transient phenomena, not only from restrikes as previously covered, but also from prestrikes.

Prestrike

During the initial energization of system capacitance, an arc ignition or prestriking may occur across the contacts of a circuit breaker, prior to (final) mechanical closure (*32*). This phenomenon appears to be enhanced when the load-side capacitance is considerably in excess of the source-side capacitance. These events seem dependent upon the capacitive inrush current, and the magnitude of inrush current is mainly controlled by the amount of load-side capacitance.

Prestrike transients have usually been associated with vacuum interrupters, and overvoltages have been found to reach substantial levels if the initial inrush current caused by the prestrike is momentarily stopped, then followed by a subsequent prestrike or contact closure. Situations can involve energizing one line of grounded capacitance or two lines of ungrounded capacitance.

Hypotheses that have been advanced to account for prestriking include the following (*10*):

- *Contact "whiskers."* When exposed to a high electrostatic stress in a vacuum, extremely fine filaments can grow from the surface of metals. Such filaments could cause an arc ignition.
- *High electric-field strength.* Stress created by a high electric-field strength could cause breakdown of the dielectric.
- *Contact bounce.* Contact bounce is the deflection of the moving contact after impact with the fixed contact of the interrupter. This is caused primarily by a weak operating mechanism, and the resulting separation could allow arc extinction.

Whatever the source, if the load-side capacitance is uncharged and a prestrike occurs, the voltage on the immediate load side of the breaker will collapse to zero. This can cause voltage and current traveling waves to radiate on cable and lines. As will be discussed later, the waves can reflect and refract at discontinuities in the system characteristic impedance, and the typical frequencies of the resultant waveforms can be 60, 600, 6,000, 60,000 and 600,000 Hz, all superimposed. The combination of the harmonics obviously has many current zeros, and when these zeros occur between the prestrike and the mechanical contact closing, the circuit breaker can easily interrupt the inrush-current flow. In the same manner as in capacitance switching, energy can be trapped on the capacitance being energized. Subsequent arc reignitions followed by interruptions can create an unusually rapid escalation of high overvoltage. Pflantz (*32*) found that prestrike transients can have crest voltages up to 7.0 pu of the system peak, with an oscillating frequency band that spans from 60 Hz into the megahertz region.

The combination of capacitance switching, chopping, and prestrike creates many problems in the design of high-voltage distribution. If capacitance on the downstream side of an interrupter is small, destructive voltage transients can occur from current chopping. When this load-side capacitance is large, overvoltages may result from capacitance switching or a prestrike event. Consequently, the placement of capacitance within a mine power system is critical, and this will be discussed later in this chapter.

It is interesting to note from the foregoing discussion of ac switching transients, that the resulting overvoltage in each case can crest at about 7.0 pu of the system peak voltage or more. The oscillation frequency provides the key to distinguishing among the three types: the system natural frequency indicates capacitance switching or chopping (generally less than 10 kHz), and much higher frequencies indicate prestrike (often these have frequency components greater than 100 kHz).

Direct Current Interruption

The dc systems are also subject to abnormal voltage transients from circuit openings. Unlike ac, where interruption generally occurs at a current zero, dc must be forced to zero. Large amounts of magnetic energy can be stored in the system inductance, and any sudden current decrease can result in an overvoltage. The energy might be transferred to capacitance as in ac circuits, but the energy can be dissipated in an arc.[3] The arc voltage tends to drive current in the direction opposite to the source, forcing the circuit current to zero.

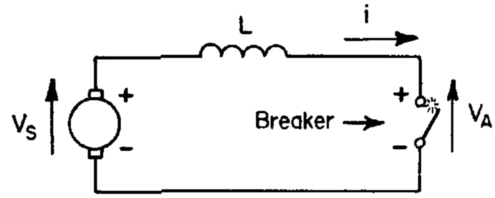

Figure 11.15.—Circuit to demonstrate voltage transients in dc systems.

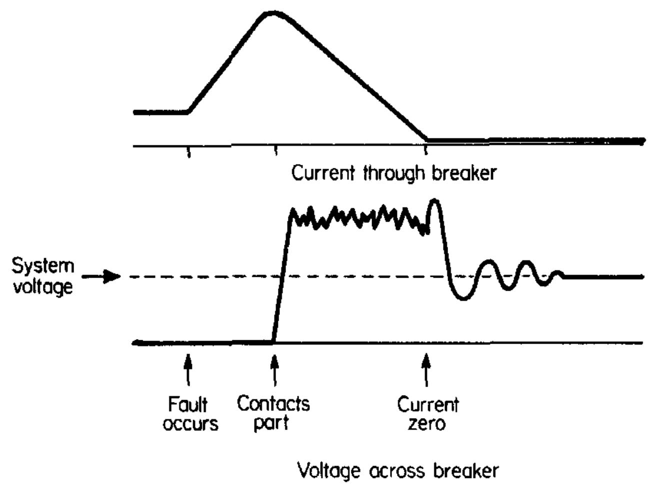

Figure 11.16.—Transient overvoltage resulting from current interruption on dc system.

Figure 11.15 shows a simple dc circuit to illustrate the overvoltage phenomenon associated with arcing. The overvoltage could be caused by a switching operation to clear a short circuit. The relationship for voltage and current is

$$L\frac{di}{dt} = V_s - v_A, \tag{11.12}$$

where L = system inductance, H,
i = system current, A,
di/dt = rate of current change, A/s,
V_s = source voltage, V,
and v_A = arc voltage, V.

To stop current flow, the arc voltage must be greater than the source potential (fig. 11.16). As the source is constant, the peak transient voltage is proportional to the rate of current decrease, that is, the faster the decrease, the higher the voltage produced.

[3] Personal communication from E. K. Stanek, West Virginia University, Aug. 1977.

The same reasoning can also be applied to interrupting dc motor operation as well as other current flows. For instance, if a dc motor current is terminated, there exists a high rate of current change, which can develop a voltage across the system inductance between the source and the motor contactors, equal to Ldi/dt. A dramatic example of this would be dropping the contactors on a large trolley locomotive while drawing full-load current.

General Switching Transients

Switching transient problems are not restricted to main power components. In either ac or dc systems, small tripping and relay coils, when coupled with a very small capacitance, can exhibit high voltages even though they operate in low-voltage circuits (*33*). Silicon diodes and thyristors can create considerable overvoltage by current chopping, sufficient to destroy themselves (*19*). In fact, solid-state conversion equipment is constantly in a transient state.

OTHER TRANSIENT PHENOMENA

Because of the existing protection methods in mine power systems, transients resulting from line-to-ground faults or accidental contact between two lines of differing voltage are minor, although destruction from localized heating may be severe. However, if the protective circuitry malfunctions, as might occur in the harsh mining environment, the problem can become critical. Local heating at the fault site can cause the conductors to melt. As this happens, the potential gradient across the conductors can be sufficient to strike an arc. The arc will extinguish and reignite, all things remaining equal, at each zero current crossing. The random fluctuation of arc impedance, as well as phenomena related to arc reignition, results in large transient overvoltages (*33*).

Resonant conditions can result in transients exceeding 10 times the nominal line voltage (*33*). Since the power system contains inductance and capacitance that are normally much greater than system resistance, resonance can occur at the natural frequency of the system. Resonance may result from the presence of line-frequency harmonics (harmonic overvoltage) or from the frequency components of other transients (dynamic overvoltage). Although dynamic overvoltage is probably not a frequent form of transient voltage, the number of failures resulting from this mode are believed by some engineers to be significant. When transformers are involved, transient resonant conditions creating dynamic overvoltages have traditionally been given the term *ferroresonance*. Considering circuit component values, the occurrence of these transients in mine power systems should be rare, but a possible problem area could be a pole-mounted transformer powered through a cable feeder. Here, the series-resonant condition could easily be corrected by simply changing the cable length.

Restriking ground faults, which are found primarily on ungrounded systems, can cause transients several times greater than the nominal line voltages. Some systems are operated ungrounded because when a line-to-neutral fault occurs, little or no fault current flows and the system remains operational. Yet some current will exist because of system capacitance. To illustrate the effect of this capacitive current during faulting, figure 11.17*A* shows a normal ungrounded system and figure 11.17*B*

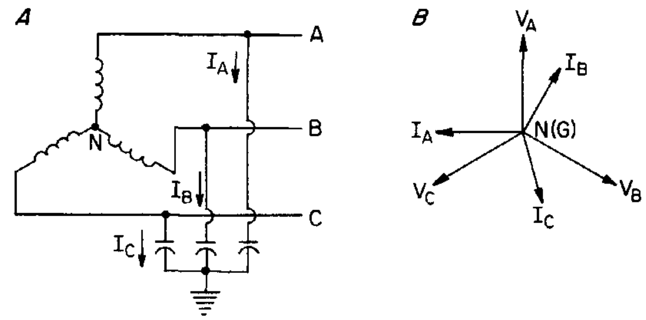

Figure 11.17.—An ungrounded system, showing capacitive-current flow.

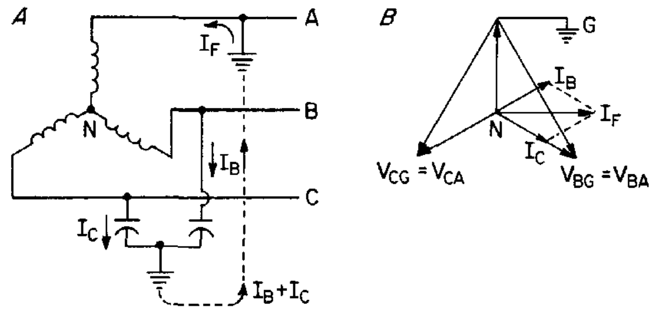

Figure 11.18.—An ungrounded system, with fault on phase A.

provides the voltage and current phasors. Consider that a line-A-to-ground fault occurs as shown in figure 11.18*A*, where I_F is a small fault current but sufficient to support an arc. The arc may extinguish itself at a current zero, but when this happens line-to-line voltages are trapped on lines B and C. The system voltage will thus be offset on these lines (fig. 11.18*B*). In the oscillatory return to steady state, it is possible to get restrikes of the fault current and further self interruption: in other words, a large voltage can build up. This is a major reason why systems should not be operated ungrounded, unless stern overvoltage precautions are heeded.

Since coal mine power systems are required to use resistance grounding for portable and mobile equipment, restriking ground-fault transients are also rare, except when the protective circuitry malfunctions. A specific case would be an open grounding resistor, where uncleared faults could cause transients.

TRAVELING WAVES

The discussion of transients has considered only circuits that have lumped resistance, capacitance, and inductance, except for circuits in prestrike conditions. In many circuits, transient behavior can be accurately predicted even though these parameters are distributed (*19*). But there are other power-system portions where circuit-element concentration will result in too large an approximation. An outstanding example is a transmission or distribution line, be it overhead or cable (*16*). Fortunately, these exhibit a certain resistance, inductance, and capacitance per unit length of line, respectively, R, L, and C. For analysis, the line can be divided into small but finite elements, as shown in figure 11.19 (resistance and leakage are neglected) (*16*).

The voltages created by transients can have a rise time in microseconds; in other words, they rise from zero to peak in that time. When very fast voltage changes occur, it is often best to analyze the system in terms of traveling waves, rather than by conventional methods (*16*). To show why this is so, consider figure 11.19, given that the switch has just closed. Conditions existing at the source end are not immediately observed at the load end because it takes time for the voltage-current conditions to pass through each LC segment. The movement of these conditions with time is known as traveling waves, and a characteristic feature of a circuit with distributed impedance is its ability to support these waves of voltage and current (*19*).

From analysis of this circuit, where Δx is made very short, it can be shown that the voltage-current relationship for each incremental section is (*19*)

$$dv = \sqrt{\frac{L}{C}}\,di$$

or

$$v = \sqrt{\frac{L}{C}}\,i = Z_o i, \qquad (11.13)$$

where v = voltage existing in incremental element, V,
i = current existing in that element, A,
L = total inductance of line, H,
C = total capacitance of line, F,
and Z_o = surge impedance or characteristic impedance of line, Ω.

The propagation velocity, U, of voltage and current is (*19*)

$$U = \frac{dx}{dt} = \frac{1}{\sqrt{LC}}. \qquad (11.14)$$

For open lines, the propagation velocity is approximately the speed of light, 1,000 ft/μs; with solid-insulation cable, the speed is about 500 ft/μs (*22*).

To illustrate the time behavior, suppose an overhead line with 400-Ω surge impedance, as depicted in figure 11.20*A*, is hit with an 100-kV step function at time t = 0 (*16*). At this instant, voltage and current exist only at the source end and nowhere else on the line (fig. 11.20*B*). At t = 1.0 μs, the voltage-current conditions have propagated down the line for 1,000 ft, and between zero and 1,000 ft, the voltage is 100 kV with the current

$$i = \frac{v}{Z_o} = \frac{100{,}000}{400} = 250\ \text{A}.$$

Beyond 1,000 ft, voltage and current are zero (fig. 11.20*C*). At t = 4.0 μs, the surge has moved 4,000 ft, voltage and current being 100 kV and 250 A to the left of that point and zero to the right (fig. 11.20*D*).

If the line shown in figure 11.20*A* was of infinite length, the bundle of energy would theoretically travel forever. Because conductors have finite length, problems occur at the line end or at a discontinuity, which is a point where the surge impedance changes (*16*, *19*). At either location, the strict proportionality between the voltage wave and the associated current wave must be satisfied by natural adjustment. Reflected and refracted waves are the result. The reflected wave propagates back down the line and is superimposed on the original or incident wave, whereas the refracted wave travels beyond the discontinuity. The reflected and refracted amplitudes are such that the voltage-to-current proportionalities are preserved, as dictated by equation 11.13 and the surge impedances of the lines on which they travel (*19*). Expressed mathematically,

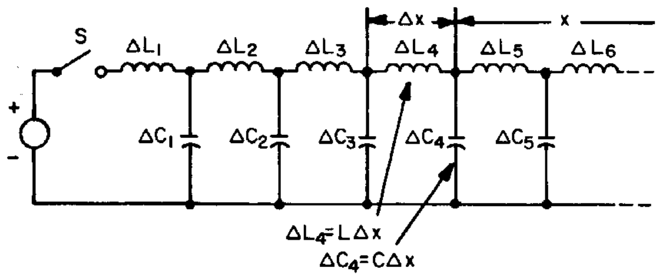

Figure 11.19.—The distributed inductance and capacitance of two-wire line shown as incremental sections.

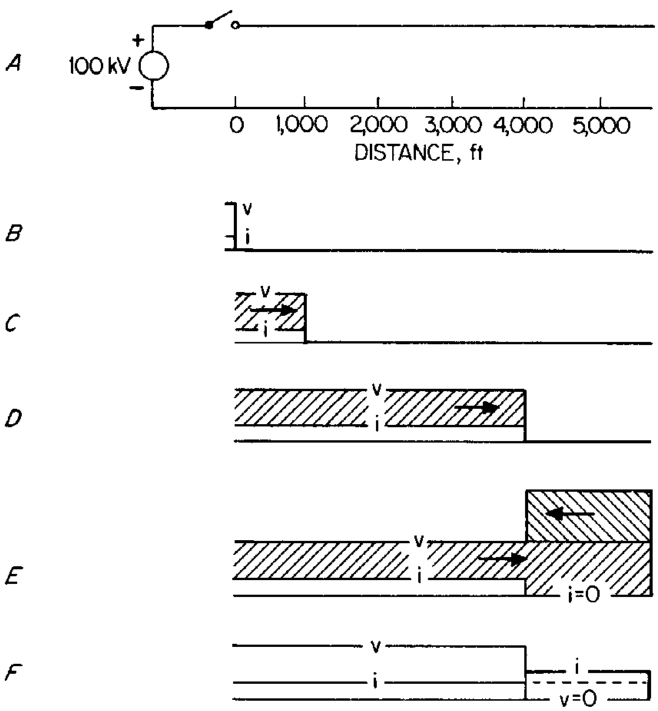

Figure 11.20.—Demonstration of traveling wave on overhead line.

$$v_1 + v_2 = v_3, \qquad (11.15a)$$

$$i_1 + i_2 = i_3, \qquad (11.15b)$$

where

$$v_1 = Z_1 i_1, \qquad (11.15c)$$

$$v_2 = -Z_1 i_2, \qquad (11.15d)$$

$$v_3 = Z_3 i_3, \qquad (11.15e)$$

and v_1, i_1, Z_1 = conditions for incident wave, V, A, Ω,
v_2, i_2, Z_2 = conditions for reflected wave, V, A, Ω,
v_3, i_3, Z_3 = conditions for refracted wave, V, A, Ω.

The assumption is that energy is conserved. The reason for the minus sign indicated in equation 11.15*d* is that i_2 is traveling in a minus-x direction, and thus has a sign opposite to v_2. By combining the above equations, expressions for the reflected and refracted voltage-wave magnitudes in terms of the incident wave can be obtained:

$$v_2 = \left(\frac{Z_3 - Z_1}{Z_3 + Z_1}\right)v_1, \tag{11.16a}$$

$$v_3 = \left(\frac{2Z_3}{Z_3 + Z_1}\right)v_1. \tag{11.16b}$$

Traveling-wave behavior under reflection and refraction can be demonstrated using the preceding example and terminating the overhead line by either an open circuit, short circuit, or a line with a different surge impedance.

Figure 11.20*E* illustrates the conditions if the line end is open-circuited. At the instant the incident waves reach the end, the current at that point must be zero, as equation 11.15*b* relates,

$$i_1 + i_2 = 0$$

or

$$i_2 = -i_1,$$

but, by equation 11.15*d*,

$$v_2 = -Z_1 i_2 = Z_1 i_1 = v_1.$$

In other words, the reflected current wave will have a magnitude of $-i_1$, and the reflected voltage wave will have v_1. Moving to the left, both will superimpose on the incident waves. Therefore at t = 6.0 μs, as shown in the figure, the voltage from 4,000 to 5,000 ft is 200 kV and the current is zero.

Instead of an open circuit, consider that the line end is short-circuited. When the incident waves reach the short circuit, the voltage at that point must be zero, or from equation 11.15*a*,

$$v_2 = -v_1,$$

yet, by equation 11.15*d*,

$$i_2 = -\frac{v_2}{Z_1} = \frac{v_1}{Z_1} = i_1.$$

In this case, the reflected waves superimposed on the incident waves will result in zero voltage, but at two times the incident current magnitude. Figure 11.20*F* shows the conditions at t = 60 μs, with current to the right of 4,000 ft as 500 A.

Now suppose that the overhead line is terminated by a cable with a 50-Ω surge impedance, a typical value for feeder cable. The situations before and after the incident waves reach the junction are shown in figure 11.21 *(19)*.

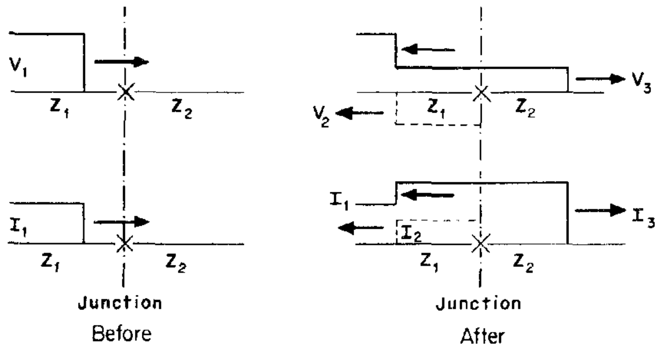

Figure 11.21.—Incident waves being reflected and refracted at discontinuity.

For the reflected wave (equation 11.16*a*),

$$v_2 = \left(\frac{50 - 400}{450}\right) 100 \text{ kV} = -78 \text{ kV}$$

and (equation 11.15*d*),

$$i_2 = \frac{(-78{,}000)}{400} = 195 \text{ A}.$$

For the refracted wave (equation 11.16*b*),

$$v_3 = \left(\frac{2\,(50)}{450}\right) 100 \text{ kV} = 22 \text{ kV}$$

and (equation 11.15*e*),

$$i_3 = \frac{22{,}000}{50} = 445 \text{ A}.$$

Therefore, the surge that penetrates the cable has a marked reduction of voltage magnitude. This benefit is employed in power systems to protect equipment from surges coming down connected lines *(19)*; instances include machine trailing cables in surface mines and feeder cables in underground mines. It should be noted that for this case, the refracted waves propagate at 500 ft/μs, whereas the reflected waves are traveling at 1,000 ft/μs.

The foregoing discussion can be expanded to any line termination, including those with more than two connections. Within a specific line, reflections can occur at both ends, causing the traveling waves to move back and forth. In this theoretical model, resistance and leakage have been neglected so the energy of the traveling wave is maintained. In practical circuits, the presence of resistance and leakage means losses are incurred when current flows *(19)*. These losses serve to attenuate the magnitude of the waves as they propagate. Even though the preceding demonstrations were simplified through use of a step function and by ignoring losses, the concepts can be applied to any waveform with a very fast voltage rise time.

Two important points can be extracted from this brief outline of traveling waves *(16)*. First, if a line is open ended, any terminal equipment on the line may experience a potential up to two times higher than that of the traveling wave that produced it. Next, it is a common practice to protect personnel working on exposed powerlines by placing protective grounds on each power conductor. Consider what would happen if a traveling wave caused by a lightning stroke was on the line and a person was touching the line between the surge source and the protective ground. The person would be exposed to the incident voltage for the time it took the wave to travel from that location to the protective ground and back. It is then obvious that protective grounds should always be installed right at the worksite, or at least between probable surge locations and the workers.

ELECTROMAGNETIC PHENOMENA

It should now be apparent that electrical transients are in essence electromagnetic phenomena. According to Faraday's law, transients existing on a specific circuit can produce electrostatic and electromagnetic induced voltages on nearby or associated circuits.

It was shown in chapter 3 that whenever two charged conductors are separated, an electric field and a potential difference exist between them. The charge is related to the potential difference by the proportionality of constant capacitance. The physical arrangement between the conductors is shown in figure 11.22*A* (*19*). As shown by figure 11.22*B*, when another conductor is placed in the same space, it distorts the original electric field and there will be a charge separation on the third conductor surface. The conductor will assume a potential somewhere between the original two conductors. The relationship will also establish capacitances among the three conductors (*19*).

Suppose that conductors 1, 2, and 3 are, respectively, high-voltage power, grounding, and ground-check conductors. If the potential difference between the power and grounding conductors suddenly changes, an electromotive force is induced in the ground-check conductor, which can cause current flow. This redistribution of charge could operate a relay, thereby nuisance-tripping a circuit breaker. If the third conductor is for some other control, monitoring, communication, or grounding circuit, the result, beyond false relay operation, could be incorrect machine operation, erroneous meter readings, communications noise, or injury to personnel.

TRANSIENT-INDUCED FAILURES

Deteriorating electrical insulation affects the entire mine power system and may jeopardize its safe operation. Whether the dielectric is in a transformer winding, motor winding, portable cable, or rectifier, it is a critical factor in the safe, economical, and reliable operation of any mine power system. Disturbances that threaten to compromise the integrity of the power system must be eliminated at the source. This is why attention must be paid to the causes of electrical transients as well as to their elimination. Consequently, it may be helpful to examine the effect of abnormal voltages on dielectrics.

Each type of insulation or dielectric is designed for a safe maximum applied voltage and a transient overvoltage. The transient overvoltage rating is given in terms of BIL, the basic impulse insulation level. The most common BIL measurement is the 1.2 × 50 wave test (fig. 11.23), where the voltage impressed across the dielectric reaches its peak in 1.2 μs (*22*). Thus a dielectric with a 95-kV BIL rating can safely withstand a 1.2 × 50 pulse of 95 kV

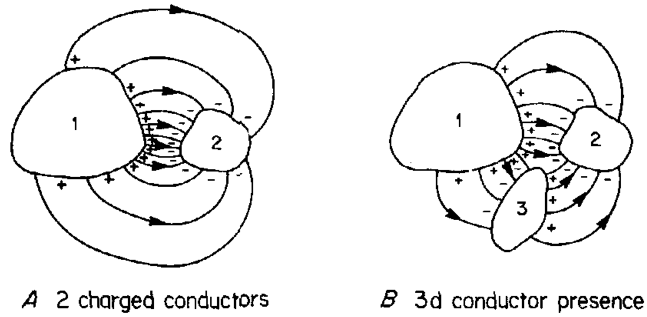

Figure 11.22.—Electric field between conductors.

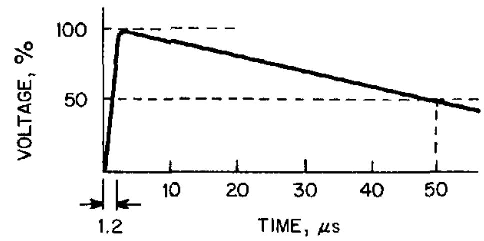

Figure 11.23.—A 1.2 × 50 wave test used for BIL measurement.

peak. The peak voltage in the 1.2 × 50 test is considered more severe than the transients actually found in mine power systems.

Dielectric deterioration is created in large part by the rise time of the transient, as well as the crest magnitude. Furthermore, the greater the overvoltage pulse width (duration), the greater the probability for failure, since more energy is involved. These voltage anomalies break molecular bonds in the dielectric, which reduces its effectiveness. If the overvoltage contains sufficient energy, the dielectric can fail immediately; however, this is usually not the case. Instead, the insulation is progressively weakened until it finally fails, resulting in a line-to-ground or line-to-line fault. If the weakened insulation is in a portable cable or splice, a considerable personnel safety hazard arises since the insulation appears to be functional when in reality a lethal potential may exist on the cable surface. Although the deteriorating dielectric may not present a direct safety hazard, a complete failure can, because it may cause an explosion or arcing. In either case, the equipment faces serious problems in terms of repair, replacement costs, and downtime.

Winding Response

The physical construction of equipment may increase its susceptibility to transient failure; for instance, motor and transformer windings often fail at the end of a coil because of increased electrical gradient. Figure 11.24 can be used to explain why this can happen (*19*). It shows the distributed nature of the winding inductance and capacitance, where capacitance is assumed to be uniformly divided among the windings and to consist of capacitance to ground and to adjacent turns; resistance is neglected. The winding neutral may be grounded depending upon the position of the switch.

From an analysis of the circuit, the following general equation can be obtained:

$$C\frac{\partial^2 V}{\partial t^2} = K\frac{\partial^4 V}{\partial X^2 \partial t^2} + \frac{1}{L}\frac{\partial^2 V}{\partial X^2}, \qquad (11.17)$$

where C = winding capacitance to ground, F/m,
K = winding series capacitance, F/m,
L = winding inductance, H/m,
and V = voltage applied across the winding, V.

By simplification, this can be used to solve for the response of the winding to a surge. Consider that surge waveform is a step function with amplitude V. If the neutral is grounded, at the instant the step function hits the winding, the initial voltage distribution across the winding is

$$V_x = V\frac{\sinh(\alpha x)}{\sinh(a\ell)}, \qquad (11.18)$$

and for the ungrounded neutral,

$$V_x = V\frac{\cosh(\alpha x)}{\cosh(a\ell)}, \qquad (11.19)$$

where $\alpha = \sqrt{\frac{C}{K}}$,
ℓ = winding length, m,
x = distance between neutral and a point on the winding, m,
and V_x = voltage at that distance x, m.

For a specific α, the initial voltage distribution across the winding in response to the surge can be plotted as shown in figure 11.25 (*19*). This figure relates that as α increases, the distribution becomes nonuniform. When α = 10, 60% of the voltage is initially impressed across the first 10% of the winding, with 75% across the first 20%. Therefore, under surge conditions, very high stresses can occur on the first few turns, and if precautions are not taken, transformers and motors can fail by breakdown of the turn-to-turn insulation in this region.

Coupling Through Transformers

When a fast rise-time voltage surge hits a transformer, as just shown, the parameter of concern is the winding capacitance. Series capacitance and capacitance to ground exist not only in the primary, but in the secondary winding as well. Figure 11.26*A* approximates this situation for a two-winding transformer, and the equivalent circuit shown in figure 11.26*B* forms a crude capacitive voltage divider (*19*). When a change of voltage is applied to the primary, the voltage divides inversely with capacitance (*15, 35*):

$$V_s = \left(\frac{C_1}{C_1 + C_2}\right) V_p, \qquad (11.20)$$

where V_p = magnitude of transient voltage impressed on primary, V,
V_s = magnitude of transient voltage transmitted to secondary, V,
C_1 = primary-to-secondary winding capacitance, F,
and C_2 = secondary-to-ground winding capacitance, F.

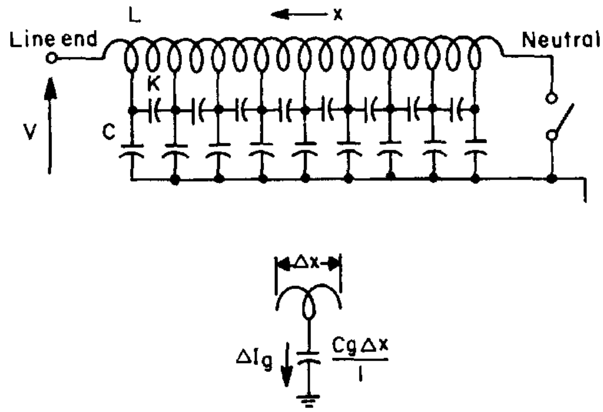

Figure 11.24.—Equivalent circuit of multiturn winding showing distribution inductance and capacitance.

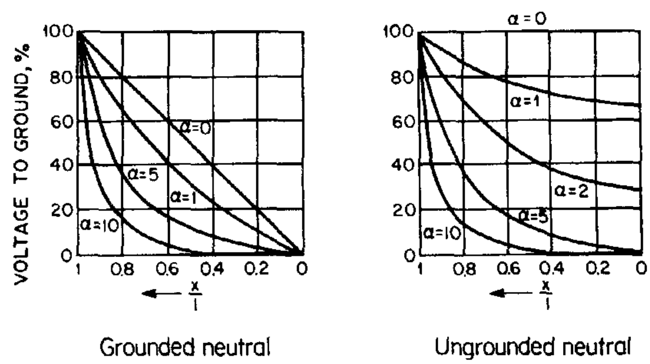

Figure 11.25.—Initial voltage distribution across uniform winding from step function.

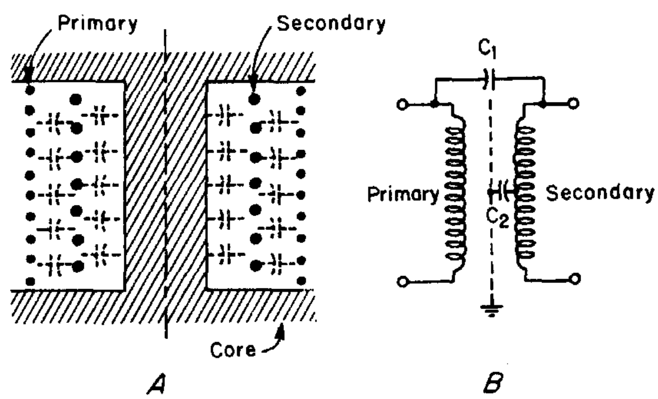

Figure 11.26.—Capacitive coupling of transient voltage through two-winding transformer.

Obviously, the voltage transmitted from the primary to the secondary is not tied to the transformer turns ratio. Typical values are 35% to 40% of the impressed primary voltage (*15*). If the transformer is a step down, for example, from distribution to utilization, the result can be low-side per-unit voltage levels much greater than those that existed on the high side.

TRANSIENT PROTECTION

Ideally, the elimination of transient-voltage problems begins with an excellent power-system design applying time-tested principles. The basic goal is to avoid the situations covered in this chapter. However, even this ideal situation can address only normal conditions; unpredictable abnormal conditions can still arise, producing destructive transient overvoltages (*19*). To address this problem, additional overvoltage control must be placed in the mine power system through use of such protective devices as surge arresters, surge capacitors, shielding, and circuit arrangements.

The role of these protective schemes is to ensure that equipment dielectric strengths are not exceeded, so that if a transient attempts to raise the voltage above an insulation withstand level, the protective device will exert a clamp or restraint to maintain the voltage within acceptable limits (*19*). Effective transient voltage suppression is basically the dissipation of transient energy.

Surge Arresters

The simplest form of overvoltage device is the spark gap, which is essentially two conductors separated by air, as in the tips of two rods where one side is connected to a powerline, the other to the neutral or earth. The spacing between the conductors establishes a specific dielectric strength so that voltage above that level will cause the gap to spark over. The spark gap has no effect on normal system operation, but its main disadvantage is that once an arc occurs, a fault is created on the system, which remains until the gap is deionized. Often, the attendant current flow or follow current can only be interrupted by a circuit breaker or a fuse. Surge arresters, formerly called lightning arresters, use this spark-gap principle to clip the peak of a voltage and divert the excess current to ground. They also contain a device to interrupt the follow current (*31, 34*). There are two common surge-arrester types, expulsion and valve; they differ in the scheme they use for interruption.

The expulsion surge arrester extinguishes the arc in a manner similar to an expulsion fuse (see chapter 9); that is, an overvoltage establishes an arc across the spark gap and also across a gas-evolving material (usually organic). Ignition of the material causes the expulsion of gas, which blows out the arc. The operation has three disadvantages. First, some gas-producing material is destroyed during each operation, and only a limited number of interruptions are available. Second, because of the gaseous discharge, care must be taken in placement and installations are restricted to outdoors. Lastly, the arrester has an assigned current-interrupting rating and cannot be used on circuits that have a greater available fault current than this rating.

In valve surge arresters, the spark gap is in series with a nonlinear resistor or valve block, as shown in figure 11.27. A property of the block, which is commonly made of silicon carbide, is that the resistance diminishes sharply as the voltage across it increases. To increase gap efficiency, a number of short gaps are used because these spark over more consistently and in less time than one long gap. On an overvoltage, the gaps spark over and the valve block operates in its highest conductance to pass the surge current safely to ground. After the surge voltage diminishes, the block changes to a low-conductance mode to limit the follow current, such that the gaps can provide an interruption. The valve surge arrester has none of the disadvantages of expulsion types, and it is used extensively for equipment protection, especially on distribution systems. The balance of this section will thus cover only the valve units.

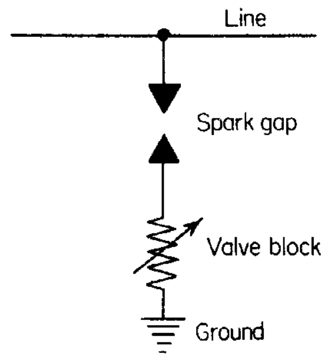

Figure 11.27.—Basic valve surge arrester.

Four important parameters are connected with the proper application of surge arresters (*31, 34*):

1. *Voltage Rating.* The power frequency sparkover voltage is the lowest rms 60-Hz ac voltage across the arrester at which it will perform the operating cycle. This level is 1.5 times the arrester voltage rating for arresters rated at 60 kV and below.
2. *Sparkover Voltage.* The highest crest voltage at which arcs will form across the spark-gap electrodes, initiating the operating cycle.
3. *Discharge Current.* The current through the arrester created by the overvoltage immediately after sparkover.
4. *IR Discharge Voltage.* The voltage formed across the arrester during the discharge of surge current.

Ideally, gap sparkover should occur on any dangerous system overvoltage and ignore all minor and harmless transients. Proper sparkover requires high-speed response to fast rise-time wave fronts (as in lightning surges) and consistent response to slower rates of voltage rise (as in system-generated surges). Both requirements are satisfied by electrical grading of the spark-gap structure, which consists of shunting each gap with a high resistance. Figure 11.28 shows the technique in simplified form (*31*). After sparkover, the IR discharge voltage occurs, being equal to the product of the discharge current (I) and the arrester discharge-path resistance (R). As discharge current may be very large, the discharge voltage may equal or exceed the sparkover voltage. Thus, protected equipment is exposed to both the sparkover and IR discharge voltages, and the system insulation withstand ability must be safely above both.

To establish an IR discharge voltage, it is important to recognize the magnitude of possible discharge currents. A surge arrester is likely to be exposed to a wide discharge range, but experience from field measurements has shown that discharge currents typically range from 1,000 to 2,000 A, that 5.0% exceed 9,000 A, and that only 1.0% exceed 20,000 A (*31*). Even though 20,000 A is rare, it is often used as a worst case to estimate the discharge voltage.

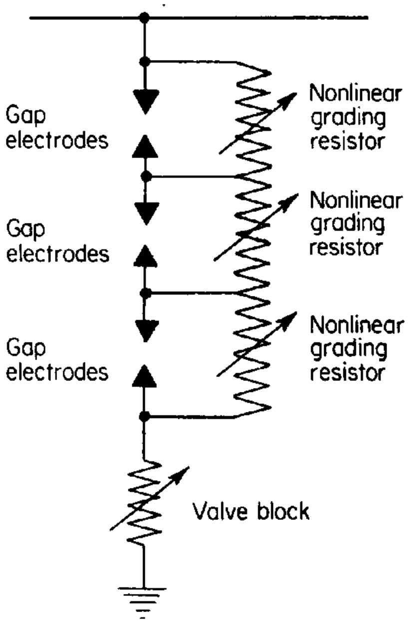

Figure 11.28.—Surge arrester with nonlinear resistance grading to equalize each gap structure.

There are three classes of valve-type arresters available (*34*):

1. *Distribution-class* arresters have the lowest cost and are satisfactory for general equipment protection purposes. Typical voltage ratings range from 1,000 V to 18 kV, and these arresters can withstand a 65-kA current surge. (Surge arresters are generally used on high-voltage systems in mining, but these devices are available with ratings as low as 50 V.) When protecting rotating machinery and dry-type transformers, the arresters must be the low-sparkover rotating-machinery (RM) type because of low BIL's.
2. *Intermediate-class* arresters have lower sparkover and IR discharge voltages than do distribution-class arresters. Available voltage ratings are from 3,000 V to 73 kV, and the arresters can typically withstand a 65-kA surge current. The cost is about five times that for a distribution arrester.
3. *Station-class* arresters can handle discharge currents up to 100,000 A and are available for almost any distribution or transmission voltage application. They are considered to provide the best possible protection, but cost about twice as much as intermediate-class arresters. Both station-class and intermediate-class arresters have pressure-relief systems; if stressed by a surge beyond their capability, the internal pressure is vented to prevent housing rupture.

Surge Arrester Applications

The factors to consider when selecting an arrester class are the degree of transient exposure and the importance of the equipment being protected. This is basically an economic question, but in general, intermediate or station arresters are justified for surface substations, with distribution-class arresters being suitable for distribution and utilization equipment protection.

An arrester voltage rating of a certain class has associated sparkover and IR discharge voltages. System voltage, as well as the method of system grounding, affects the voltage that the arrester is exposed to and therefore the selection of the arrester voltage rating. Consequently, once an arrester voltage rating is set, the system insulation withstand ability must be coordinated with it. This is most often related as a BIL for the equipment being protected.

The voltage-rating selection is affected by the system grounding categories: *effectively grounded* or *noneffectively grounded*. The coefficient of grounding can be employed to find which category is being used. The coefficient of grounding is defined as the percent ratio of the highest rms line-to-ground voltage existing during a line-to-ground fault to the nominal line-to-line voltage (*31*). If the ratio does not exceed 80%, the system is termed *effectively grounded*. Solidly grounded and typical low-resistance-grounded systems are in this class (the neutral potential remains rather constant during line-to-neutral faults). However for high-resistance and ungrounded systems, the occurrence of a line-to-neutral fault can shift the neutral to near the faulted line, with the potential to ground of the other two lines approaching line-to-line system voltage. The coefficient of grounding here can be from 80% to 100%. These systems are termed *noneffectively grounded*. Resistance-grounded mine power systems for portable and mobile equipment are included in this category.

In either grounding case, the arrester voltage rating should be above the possible exposed crest voltage; if not, a disruptive discharge might occur.

Therefore, on effectively grounded systems, the arrester is sized to maximum expected line-to-ground voltage, whereas maximum line-to-line voltage is used for noneffectively grounded systems (*2*). To allow for the expected increase due to voltage-regulation compensation, the arrester voltage rating should be 5% to 10% above these values. Tables 11.1 and 11.2 list the recommended sizing for resistance-grounded mine power systems (*3, 34*). The first table refers to station and intermediate arresters, the second to the low-sparkover distribution class. Note that the transformer BIL's specified are according to ANSI C57.12.00–1973 (*1*) and may be too low for some mining applications (see chapters 12 and 13).

The equipment protection from voltage surges can be verified for any arrester selected. Full coordination requires checking the arrester performance over a full time range for an expected surge (*1*). However the following quick guidelines will ensure safe protection (*34*).

- The insulation BIL ratings of equipment should be at least 20% greater than the arrester sparkover voltage.
- The BIL rating should be above the IR discharge voltage of the arrester.

As mentioned earlier, a discharge current of 20,000 A to establish IR discharge voltage may be used as an approximation of worst case conditions. Tables 11.1 and 11.2 also provide typical sparkover and IR discharge voltages for arresters used in mining service (*34*). However, manufacturer catalog values should be consulted for actual applications, as differences exist among products. Additional details of surge-arrester equipment protection are provided in chapters 12 and 13.

Another important point about maximum exposed surge voltage concerns the arrester connections to line and

Table 11.1.—Recommended station and intermediate surge arresters for resistance-grounded mine power systems to protect oil-immersed transformers

Insulation class, kV	System voltage,[1] V		Transformer BIL,[2] kV		Arrester rating, kV	Front-of-wave sparkover,[3] kV		IR discharge voltage[3] (20,000 A) kV		Maximum 3-phase line-to-line voltage, V
	Nominal	Maximum	Power	Distribution		SC	IC	SC	IC	
2.4	2,400	2,540	60	45	3	13	12	7.8	12	3,000
5.0	4,160	4,400	75	60	6	19	21	15.5	20	6,000
	4,800	5,080	75	60	6	19	21	15.5	20	6,000
8.7	7,200	7,620	95	75	9	30	31	23	27	9,000
15	12,470	13,200	110	95	15	51	51	39	45	15,000
	13,200	13,970	110	95	15	51	51	39	45	15,000
	13,800	14,520	110	95	15	51	51	39	45	15,000
	14,400	15,240	110	95	18	NA	61	NA	54	18,000
	14,400	15,240	110	95	21	70	NA	54	NA	21,000

NA Not available.
[1] Maximum system voltages are from ANSI C84.1-1970 (*4*).
[2] BIL ratings are from ANSI C57.12.00-1973 for oil-immersed transformers (*1*).
[3] SC, station class; IC, intermediate class.

Table 11.2.—Recommended distribution-class, RM-type, surge arresters for resistance-grounded mine power systems to protect rotating machinery and dry-type transformers

Insulation class, kV	System voltage,[1] V		Transformer BIL,[2] kV	Arrester rating, kV	Front-of-wave sparkover, kV	IR discharge voltage (5,000 A), kV	Maximum 3-phase line-to-line voltage, V
	Nominal	Maximum					
2.4	2,400	2,540	20	3	13	10	3,000
5.0	4,160	4,400	25	4.5	17	15	4,500
	4,800	5,080	25	6	22	20	6,000
8.7	7,200	7,620	35	[3]7.5	24	25	7,500
15	12,470	13,200	50	15	44	50	15,000
	13,200	13,970	50	15	44	50	15,000
	13,800	14,520	50	15	44	50	15,000
	14,400	15,240	50	[3]15	44	50	15,000

[1] Maximum system voltages are from ANSI C84.1-1970 (*4*).
[2] BIL ratings are from ANSI C57.12.00-1973 for oil-immersed transformers (*1*)
[3] Arresters may occasionally be exposed to voltage above their rating.

to ground. Conductors extend from each ungrounded power conductor to an arrester and from the arrester to ground so that in resistance-grounded three-phase systems, a minimum of three arresters is needed. For stationary equipment on the surface, an arrester ground bed, such as a substation system ground bed, serves as the grounding medium (it must be low resistance); on portable equipment, the frame would be the ground. As the connections exhibit inductance, they should be of No. 6 AWG solid-copper conductor or larger and as short as possible because the inductance of too long a conductor can render an arrester ineffective (*19*). Sharp bends should also be avoided, since a bend substantially increases the effective inductance. The inductance of arrester connections that adhere to these requirements is estimated at 0.4 μH/ft, with the voltage drop produced during a surge estimated at 1.6 kV/ft, using a current wave front of 4,000 A/μs (*31*). This voltage must be added to sparkover and IR discharge voltages to assess protective margins.

As a general rule, all arresters should be located as close as possible to the equipment they are to protect. Ideally, they should be across the protected equipment terminals, the connections for three-phase systems being a wye configuration with the common arrester connection grounded. First of all, this location minimizes the possibility of a destructive surge entering the circuit between the protecting and protected devices (*19*). Second, close proximity also reduces the change of surge-voltage amplification through refraction and reflection of a traveling wave. For instance, consider figure 11.29, where a transient voltage surge is traveling along a power conductor toward surge-arrester-protected equipment (*22*). The arrester is located a distance, d, from the line end. As the wave passes the arrester location, the arrester sparks over at its protective level, but lets a traveling wave with a crest equal to its sparkover value through. The voltage wave reaching the equipment terminals is amplified by the addition of the incident and reflected wave, with the resultant magnitude depending upon the line-end surge impedance. Consequently, surge-arrester locations other than directly across the equipment terminals can lead to higher surge voltages at the protected apparatus than the arrester sparkover voltage. The terminal voltage rise will be aggravated by a greater separation distance, d. Note

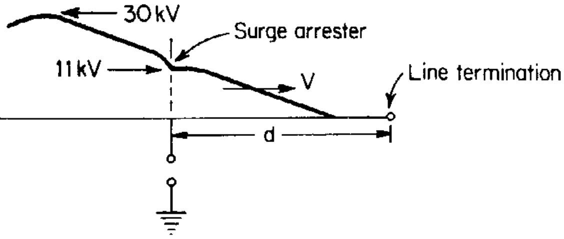

Figure 11.29.—Surge approaching surge-arrester-protected equipment.

that with a wavefront rise time no greater than 0.5 μs, a maximum distance of 25 ft is perhaps allowable, but shorter distances always afford greater protection (*28*).

Capacitors and System Capacitance

Surge capacitors, also termed *RM capacitors*, are special units with low internal inductance that are used extensively to protect rotating machinery and dry-insulated transformers (*28*). This equipment is very susceptible to line-end turn-to-turn failures caused by fast rise-time wavefronts, and the faster the rise time, the greater the probability for damage. Connected across equipment terminals in grounded wye, as shown in figure 11.30, the capacitors serve to limit the rate of rise of the transient voltage. Simply, the capacitor has to be charged before the overvoltage can be impressed on the system dielectric. Transient rise time is then largely determined by the charging rate. Coupled with the system inductance, the limiting criterion is that at least 10 μs is needed before the crest value of the protected-equipment nameplate voltage is reached (*22*). Low internal inductance of the capacitor is important because the presence of series inductance in the capacitor circuit deteriorates the wave-sloping action. Accordingly, the capacitors shown in table 11.3 have been standardized for this kind of protection (*22*). In combination with the recommended low-sparkover distribution-class surge arresters (table 11.2), the crest voltage of transients is considered restricted to harmless values for the utilization equipment (*31*).

Table 11.3.—Commonly used surge capacitors for limiting voltage rate of rise on rotating machinery and dry-insulated transformers

Rated equipment voltage, V	*Capacitance, µF*
650 or less	1.0
2,400 to 6,900	.5
11,500 or higher	.25

Surge-Impedance Reduction

Surge capacitors can also be used to control transient overvoltages by reducing the system's characteristic impedance. An increase of system capacitance, as exhibited by equation 11.8, can lower the surge impedance and therefore the possible peak transient voltage resulting from current chopping. A fine line exists here, because too much capacitance on the load side of switching apparatus can cause capacitive-switching or prestrike transients. For instance, prestrike events are dependent upon the capacitive inrush current during contact closure, and the magnitude of inrush current is controlled by the amount of load-side capacitance (*30*). With capacitive switching, the transient overvoltage is created by the ability of the load-side capacitance to hold a charge, thus causing a recovery voltage across the breaker so restriking occurs. System capacitance is, therefore, a critical factor in transient protection.

Hence, the problem of surge-impedance reduction mainly concerns how much capacitance is necessary to limit safely the transient overvoltages caused by current chopping. Actually, any magnitude below the minimum insulation withstand level or, if used, the surge arrester sparkover can be considered safe. Yet perhaps the most conservative approach would be to limit any chopping

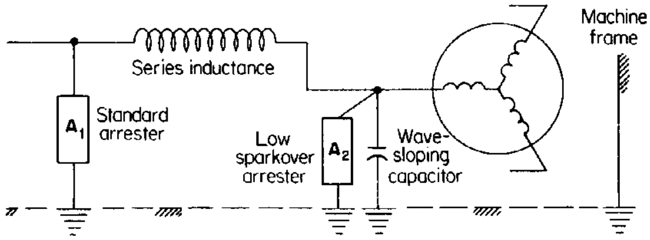

Figure 11.30.—Typical surge protection of rotating machinery and dry-insulated transformers.

event to two times the peak system voltage. The most critical portion of the mine power system would be where capacitance is minimum, such as the case of a switchhouse connected to a power center, as illustrated in figure 11.14, or rotating machinery. It can be inferred that current chopping is the result of VCB operation, but other switching-apparatus types, including current-limiting fuses, could be chopping sources.

Equation 11.8 can be used to select a value of capacitance that will include the capacitance inherent in the distribution system, that is, the equivalent line-to-neutral capacitance of all devices on the load side of the switching apparatus. Considering a three-phase transformer as the load, the procedure can be as follows:[4]

1. Determine the allowable peak voltage, V_p;
2. Find the exciting current and assume interruption is at peak level, I_m;
3. Determine the transformer exciting inductance, L_m;
4. Assume all transient energy is absorbed by the capacitance, C; then
5. The necessary per-phase capacitance, in farads, referred to the transformer primary circuit is

$$C = \frac{I_m^2 L_m}{V_p^2}. \tag{11.21}$$

However, a more useful form of this equation can be found if (*24*)

1. The allowable peak voltage is limited to two times the peak system voltage;
2. The inductance of the transformer and the power system is no greater than 20%; and
3. The peak exciting current is expressed in terms of the rated capacity and rms voltage of the transformer.

With these parameters,

$$C = \frac{10\ S\ (60)}{V_s^2\ f}, \tag{11.22}$$

where C = necessary capacitance, μF,
S = per-phase transformer capacity, VA
V_s = line-to-line voltage rating of transformer primary, V (if it is desired to connect the capacitance across the secondary, secondary line-to-line voltage is used),
and f = power-system frequency, Hz.

[4] Personal communication from E. K. Stanek, West Virginia University, Aug. 1977.

The value resulting from equation 11.21 or 11.22 is total system capacitance per phase. If the level is above that supplied by the system, additional capacitance might need to be added.

For three-phase systems, the common surge capacitor connection is shown here in a wye configuration with the center connected to ground. The typical location is directly across the protected equipment terminals, as in wave-sloping applications (fig. 11.30). Another popular location has been at the switching-apparatus load terminals. The philosophy here is that the interrupter sees the increased capacitance directly, and thus, chopping transients are limited more effectively; in other words, transients are best eliminated at their source. There is a general feeling that in this way the entire system downstream from the capacitor location would receive protection. However, surge capacitors have an extremely low internal series impedance; therefore, at the interrupter load terminals, they can be a significant source of capacitive inrush current as well as having the ability to exchange transient energy effectively to and from the system. Consequently, the best location for applying surge capacitors is at the protected-equipment terminals.

Two advantages are gained through the ground connection: Transient energy is shunted to ground, and compared with delta connections, a lesser voltage rating is necessary. The capacitor working-voltage rating (WVDC) should be at least three times the exposed rms system voltage, usually line-to-neutral for wye grounded and line-to-line for delta connections.[5] However, for the same reasons given for surge arresters, it is perhaps best to rate surge capacitors in resistance-grounded systems to three times the line-to-line voltage.

Buss (*11*) and Morley (*30*) have performed extensive tests on actual underground coal mining equipment to determine the severity of transients existing on these distribution systems. For the most part, these tests involved recording staged transients on unloaded and loaded systems by chopping (tripping the interrupter) and prestrikes (engaging the interrupter). In every instance, the system segment was similar to figure 11.31 and consisted of a VCB-equipped switchhouse with various lengths of cable supplying a power center. Various switchhouses and power centers were used, all typical of actual mine installations. Beyond the principal goal of uncovering the nature of any transients, the overall objective was to see if surge capacitors were necessary to limit chopping voltage transients and what effect they have on prestrike events. Buss (*11*) backed up the actual equipment testing with computer simulations but primarily addressed chopping events. Morley (*30*) extended the research to cover prestrike activity.

In terms of transient protection by surge capacitors, the results of both research programs were practically identical and apply directly to underground mine power systems or similar surface arrangements. Specifically, it was found that both grounded wye or ungrounded wye surge capacitors do significantly reduce chopping transients when applied as given in table 11.3. With capacitors removed and practical cable lengths held to a minimum, the chopping transient voltages reached substantial levels (about 4.5 pu system nominal voltage for a common event) but were still below the sparkover level of correctly applied low-sparkover distribution-class surge arresters. Thus they remained within the BIL withstand of system equipment. However, with surge capacitors installed and a 1,000-ft cable length, some prestrike transient activity was present, with peak voltages similar to the maximum observed for chopping but again not exceeding the surge arrester sparkover. The worst case prestrike activity was associated with surge capacitors located at switchhouses.

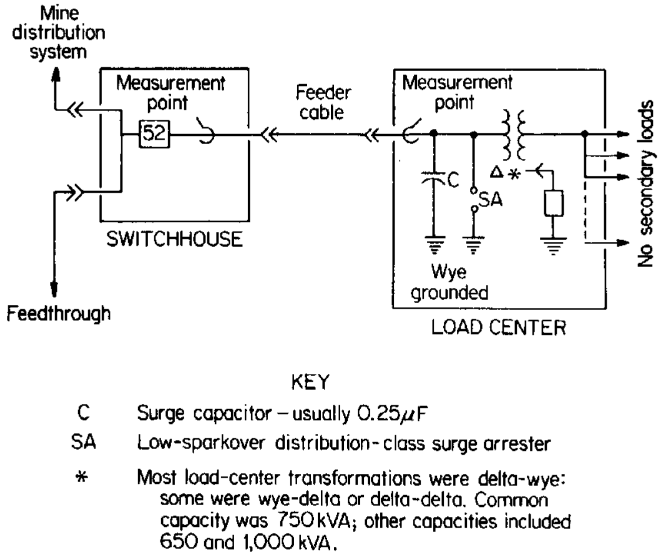

Figure 11.31.—Simplified sketch of mine power-system segment.

The conclusion was that surge arresters alone were sufficient transient protection for underground coal mine distribution systems. Surge capacitors were unnecessary redundant protection, and their general use could result in more severe transients than those they are installed to correct. Furthermore, as long as surge arresters protect each distribution load, VCB's can be employed on any mine distribution system without a need to add surge capacitors (*11*).

However, when surge capacitors are not used, a significant length of cable should be installed between switchhouses and loads to limit chopping events to safe levels. For general mining applications, 500 ft of SHD cable was recommended, although 100 ft would probably be satisfactory (*11, 30*). When the capacitance is calculated from equation 11.21 and compared with cable values (in tables 11.4 and 11.5), comparable results are supplied. Beyond overall surge-impedance reduction (if lumped circuit elements are considered), the normally lower characteristic impedance of the cable compared with that of the switchhouse circuits reduces the crest of voltage traveling waves through refraction.

The above findings were for common system arrangements in mining. Some installations could still benefit from the wave-sloping action of surge capacitors, as for example a surface mine where there is a high incidence of lightning and the distribution loads are rotating machines or dry-insulated transformers.

Capacitive Charging Current

Wye-grounded surge capacitor combinations have been recommended for several years to prevent severe chopping transients. Therefore, mines often employ them

[5] Personal communication from E. K. Stanek, West Virginia University, Aug. 1977.

Table 11.4.—Typical capacitances, in microfarads, per phase of power-system components, for shielded power cable SHD, SHD-GC, and SHD + GC

Insulation class................kV..	5	8	15	23
AWG:				
8	0.0607	—	—	—
6	.0709	—	—	—
4	.0836	—	—	—
2	.0993	0.0850	0.0401	—
1	.1096	.0938	.0456	0.0345
1/0	.1202	.1029	.0524	.0388
2/0	.1321	.1131	.0507	.0441
3/0	.1452	.1243	.0661	.0504
4/0	.1600	.1369	.0715	.0546
MCM:				
250	.1598	.1368	.0776	0588
350	.1846	.1580	.0844	.0634
500	.2150	.1840	.0920	.0685
750	.2410	.2062	.0981	.0743
1,000	.2740	.2345	.1118	.0789

NOTE.—Dashes indicate cable is not made.

Table 11.5.—Typical capacitances per phase of power-system components

Power-system component	*Capacitance, μF*
Nonshielded cable	0.02 to 0.05 per 1,000 ft; typically 0.03 per 1,000 ft.
Nonshielded cable in conduit	0.02 to 0.06 per 1,000 ft; typically 0.04 per 1,000 ft.
Overhead open-wire lines	Negligible for line lengths used in typical mine distribution.
Surge capacitors, by insulation class:	
600 V	1.0.
2,000 V	1.0.
5 kV	0.5.
8 kV	0.25.
15 kV	0.25.
23 kV	0.25.
Synchronous and induction motors, by insulation class:	
600 V	0.032.
2,000 V to 23 kV	See figures 11.32 and 11.33.

at the high side of every distribution transformer. Such extensive use can pose problems in the high-resistance grounding system of a mine.

As stated in chapter 7, the ground-fault current that is limited by the grounding resistor should not be less than the system capacitive charging current. This requirement restricts the amount of capacitance that can be placed on a system of a specific size. In other words, the size of the mine distribution system is limited by the total connected capacitance (*11*). Some relief is gained through power-transformer configurations, because the zero-sequence system is isolated in each voltage level. Nevertheless if there is excessive capacitance in distribution, for example, the capacitance can discharge during a line-to-neutral fault and feed the fault with capacitive ground current in excess of current limit.

Tables 11.4 and 11.5 and figures 11.32 and 11.33 provide typical system capacitances to assist in estimating a per-phase system capacitance (*38*). The following can be used to compute system charging current per phase:

$$I_{co} = \frac{V_{ln}}{X_{co}}, \tag{11.23}$$

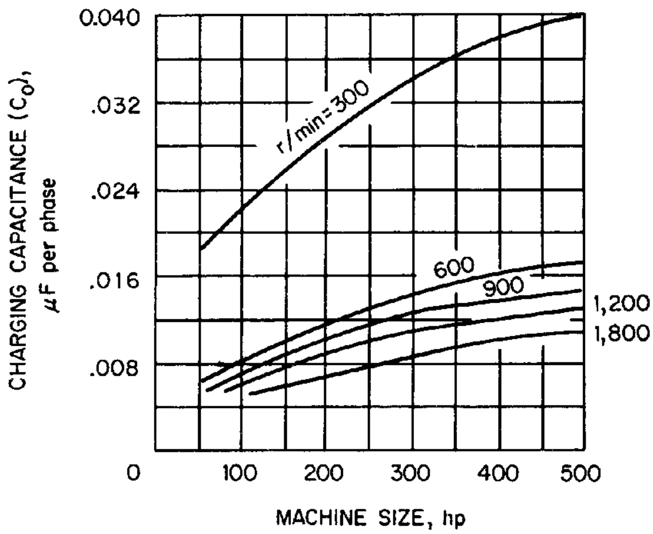

Figure 11.32.—Capacitance for 2,300-V induction motors; for motors up to 7,200 V, value will not vary more than ± 15% of above.

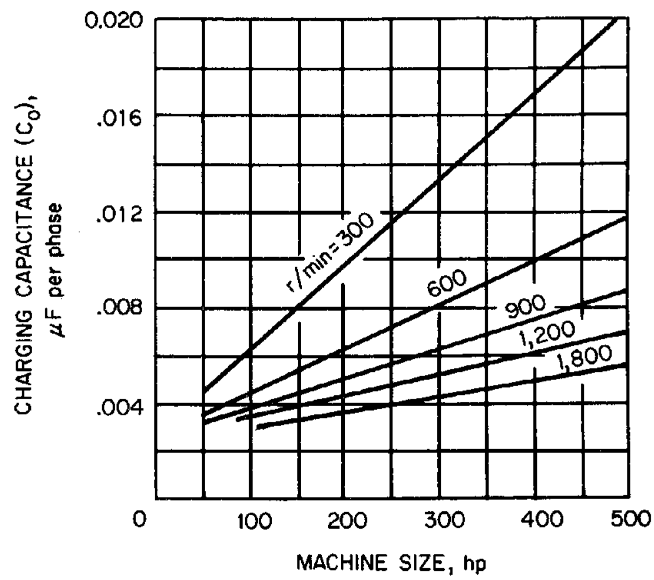

Figure 11.33.—Capacitance for 2,300-V synchronous motors; for motors up to 7,200 V, value will not vary more than ± 15% of above.

where I_{co} = per-phase system charging current, A,
V_{ln} = line-to-neutral system voltage, V,
X_{co} = per-phase capacitive reactance, Ω, = $1/2\pi fC_0$,
and C_o = lumped charging capacitance per phase, F.

This current can be compared with ground-current limits to assess the effect of adding surge capacitors. The values in the table can also be compared with the results from equations 11.21 or 11.22 to obtain greater understanding of chopping events.

Other Suppression Devices

The discussion thus far has been mainly concerned with surge suppression on high-voltage distribution. Transient-related failures can also be severe on mine systems below 1,000 V, an outstanding example being the destruction of solid-state elements in such equipment as ground-check monitors, communications apparatus, and power supplies. These transistors, integrated circuits, and thyristors are the devices most sensitive to transient overvoltages, and problems can occur even through induction from transients occurring on power conductors to a neutral or communications circuit. Two suppressor types already presented also offer effective protection for low-voltage systems: valve-type surge arresters and surge capacitors (also termed *snubbers*). In this section, several other common protection devices designed principally for but not restricted to low-voltage applications will be discussed. These can generally be divided into two classes: transient suppressors and circuit-shorting devices.

Transient Suppressors

Transient suppressors, also called constant-voltage devices, are basically nonlinear resistances placed across the circuit to be protected (*13*, *18*). They act directly to clamp or limit voltage rise much like the valve block of a surge arrester, but no series spark gap is used. Typical transient suppressors are power zeners and metal oxide varisters.

Powers zeners are primarily for dc protection, working on the zener-regulator principle covered in chapter 5. They can be used in ac circuits when two devices are placed back to back or anode to anode to give bidirectional operations. Zeners have the capability to clamp transient voltage rise to a level largely independent of the impedance or voltage-current characteristic of the transient. The response is extremely fast, and the clamping action is very firm. Although they are effective for low-energy transients, many events common in industrial power systems can readily destroy all but the most expensive high-energy zeners, which are called avalanche diodes (*18*).

Metal oxide varisters (MOV's) are ceramic suppressors that use zinc-oxide-based materials such as a zinc oxide and bismuth oxide ceramic body (*18*). Their construction provides a voltage-dependent, very nonlinear resistance with symmetrical conducting properties. The bidirectional breakdown allows their use on either ac or dc circuits. The response is similar to that of back-to-back zeners, but the clamping action is softer than with zeners, yet faster than with valve-type surge arresters (*13*). As with the silicon carbide valve block in surge arresters, transient energy is dissipated throughout the entire volume of material, making the MOV a very rugged suppressor.

MOV's can be used for a wide range of applications from power-supply voltage regulation to power-system transient suppression, including high-voltage systems. Rated voltages extend from 22 V to the thousands (*18*). A very popular application in mining is to suppress transients across thyristors in solid-state motor starters, where an MOV is mounted across each thyristor. A projected employment at this time is for direct replacement of valve surge arresters in ac distribution and dc trolley-line applications.

Circuit-Shorting Devices

A circuit-shorting device or "crowbar" can be described as a device such as a spark gap, gas-discharge tube or thyristor that senses a high voltage and throws a short circuit or low resistance across the line (*18*). The low resistance is not removed until the current through the crowbar is brought to a low level; hence, these devices need to have the power removed momentarily before resetting can occur (*13*). To facilitate this requirement, crowbars are often used in conjunction with circuit breakers, where the device could be connected in series with a shunt-tripping element, and the combination located across the line on the load side of the breaker. Sensing an overvoltage, the current through the crowbar trips the circuit breaker; normal circuit operation is not resumed automatically. These quick-acting devices are available for ac or dc systems, typically 250 V and below.

Faraday Shields

Faraday shields are used to protect the low side of a transformer against surge voltages. The shield is a turn of nonmagnetic metal sheeting placed between the primary and secondary windings; it is insulated from all windings and connected solidly to ground (*15*). This location effectively destroys interwinding capacitance and substantially reduces the transfer of surge conditions. The shield has the further advantage of practically eliminating interwinding faults. Other advantages of these shields are covered in chapters 12 and 14.

Circuit Arrangements

As induced voltages on low-voltage circuits from transients on high-voltage or other power-system conductors can be a serious problem, there are four recommendations to reduce the possibility of induction.[6]

1. Separate low-voltage circuits from high-voltage systems by a large distance.
2. Use shielded conductors on the low-voltage circuit or maintain shielding between circuits serving different purposes.
3. Twist low-voltage conductors (this effectively cancels many induced voltages).
4. Place conductors on the high-voltage circuit close together.

Protection of Overhead Lines

Exposed overhead lines are very susceptible to direct contact from a lightning stroke, which obviously produces severe transient overvoltages. An excellent means of protecting distribution lines from such occurrences involves the use of overhead ground wires or static wires (*25*, *37*). One or two ground wires are strategically situated above and between the power conductors to provide a shielding effect, as illustrated in figure 11.34 (*27*). Here, line a bisects a line drawn from the ground wire to the outer

[6] Personal communication from E. K. Stanek, West Virginia University, Aug. 1977.

power conductor; lightning strokes in area 1 are intercepted by the static wire. Line b is equidistant between the outer power conductor and the earth's surface, so that lightning streamers in area 2 will discharge to the earth. The position of line b in figure 11.34 varies with the relative height of the supporting structure, and line c is described by an arc whose radius depends upon the size and construction of the supporting structure. A stroke in area 4 will be borne by the supporting structure if it is metallic. Area 3 is the danger region (and may include area 4) where lightning flashes will strike the power conductor (*27*).

Extensive tests and observations have been conducted to determine the optimum angle Θ, which is measured from the vertical up to the line joining the power conductor and the static wire. The angle is dependent upon the height of the supporting structure, as shown in table 11.6 (*27*), but field results have shown that a good average for this angle is 30° (*37*). (Actually, 45° provides satisfactory performance if the line is situated on a level surface, but if on a hillside, the angle should be decreased from 45° by the slope angle of the hill.) Importantly, the design of static protection is practically independent of the system voltage. Figure 11.35 illustrates two approaches for shielding overhead lines by static wires when the support structure is wooden (*37*).

Table 11.6.—Protective angle versus structure height

Tower height, ft	*Angle (Θ), deg*
50	45
100	25–30
150	10–12

In addition to intercepting direct strokes, the static wires perform another function: when static wires are struck by lightning, the resulting surge current is immediately halved since the impulse travels in both directions from the point of contact. The magnitude of the induced voltage on the static wire is thereby halved also, leading to a similar reduction in the induced potentials seen by the power conductors.

Adequate grounding must be used in conjunction with the overhead static wires. The system generally consists of downleads extending from the static wire to grounding electrodes so that induced voltages are minimized by conducting stroke currents to earth as quickly as possible. This also helps to reduce the surge impedance of the overhead static-wire system. A grounding electrode may be provided by driving several ground rods at the base of each supporting structure and connecting them to the static wires via the downlead (*23*). An alternative method is based upon the use of a counterpoise, consisting of one or more buried horizontal conductors located at the base of each tower (*5*). The counterpoise may be a single continuous conductor buried directly beneath the power conductors and running from tower to tower, or it may consist of several short conductors radiating outward from each tower base (*9*). The most common technique in mines is to wrap heavy wire around the base of each wooden pole before it is placed in the hole, the so-called butt wrap. Whichever technique is selected, the grounding electrode must always have as low a surge impedance as practical, and the downleads must adhere to the guidelines stated earlier for surge arrester connections. As a general rule-of-thumb, downleads and grounding electrodes should not be spaced apart more than 500 ft. Considering normal spans in mining, this translates to grounding the static wire no more than every other supporting structure.

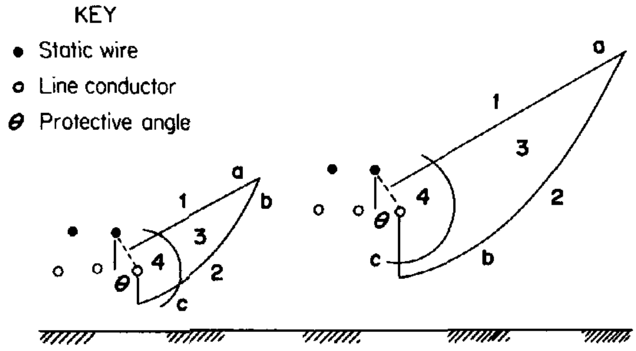

Figure 11.34.—Overhead ground-wire shielding for low and high distribution towers.

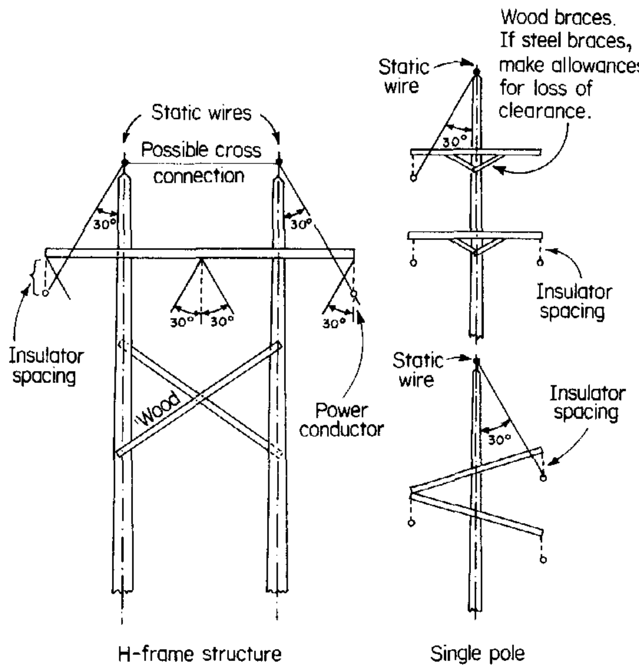

Figure 11.35.—Static-wire-protection designs of wooden support structures using 30° protective angle.

A few additional comments are needed on the application of static wires in mining. At the voltage levels used in conjunction with overhead lines to distribute mine power, the normal separation distance used for insulation between power conductors and static wires is perhaps insufficient to prevent a lightning stroke that hits a static wire from jumping directly to the power conductors (*37*). Often, the static wire is connected to earth only at the system ground bed at the substation. This can lead to a high-impulse impedance for the static-wire system, thereby reducing its effectiveness. An alternative method could be the use of protector tubes or surge arresters on the power conductors in conjunction with pole grounds (*37*). Additional points about static-wire applications in mining are presented in chapter 13.

Impulse Performance of Ground Beds

The impedance of a ground bed is directly connected with its ability to help dissipate a transient. The general consensus about the performance of ground beds when subjected to impulse currents is that the surge resistance of an electrode (R_s) is almost invariably less than the normal resistance value (R_n) as measured by an earth tester (*9, 14, 21*). Examination of oscillograph records has shown that under surge conditions, the ground system exhibits a resistance equal to or slightly higher than its normal value for the first 1 or 2 μs. The surge resistance then quickly decreases to a level usually from 20% to 80% of the normal value (*36*). Figure 11.36 illustrates that as the value of peak surge current increases, the ratio of impulse to 60-Hz resistance (R_s/R_n) decreases (*7*). Furthermore, ground beds that exhibit a rather high normal resistance show a proportionally larger resistance decrease when subjected to an impulse (*36*). Thus, two ground beds with dissimilar values of R_n may exhibit R_s values that are very close to each other (*8*).

Experiments have shown that in many respects soil behaves as a dielectric material (*17*). When a certain critical potential gradient is reached, the soil breaks down or arcs internally. Thus, the ground-bed resistance decrease under surge conditions is due to electric discharges within the soil, which spark across areas of high electric-field intensity. As shown in figure 11.37, the presence of moisture in the soil acts to increase its dielectric strength (*17*). Accordingly, very dry soil breaks down much more readily than soil containing a minor amount of water. Grounding systems can be designed in such a way that their performance under surge conditions is optimized. Figure 11.38 shows how increasing the pointedness of electrodes results in lower values of surge resistance (*21*). As a result, driven ground rods may be superior to meshes in lightning prone areas since the pointed electrodes are conducive to high stress concentration buildup and will therefore cause the soil to break down more readily under surge conditions.

This chapter has described the electrical transient and its ramifications on mine power-system safety and integrity. An electrical transient has been defined as the outward manifestation of a sudden change in circuit conditions. The magnitude and fast rise time of the overvoltage can cause damage to electrical components, particularly insulating materials. In addition to sound design practices, techniques and devices are available to reduce the damage done by surge voltages. The application of these in the design of mine power equipment is covered in the next two chapters.

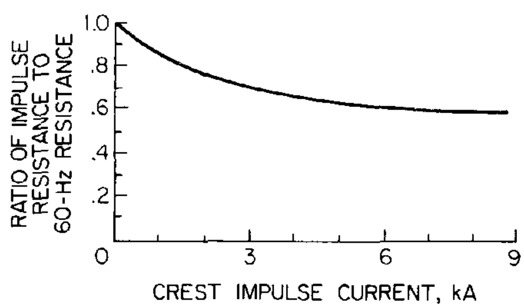

Figure 11.36.—Ratio of impulse to 60-Hz resistance as a function of peak impulse current, for driven rods.

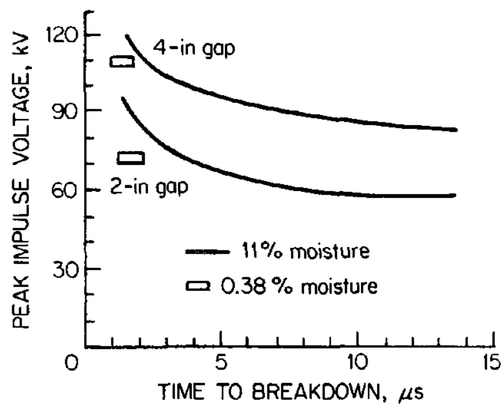

Figure 11.37.—Impulse breakdown of sand for two moisture conditions using spherical electrodes.

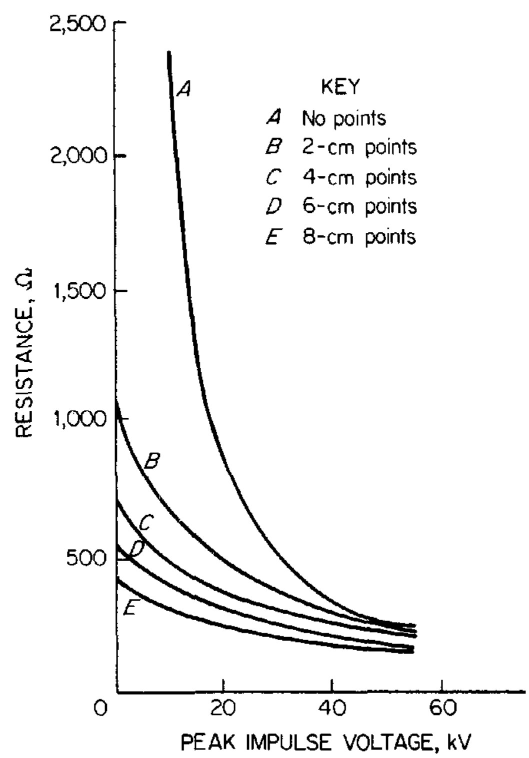

Figure 11.38.—Impulse characteristics of spherical electrode, with seven attached pointed protrusions of various lengths.

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CHAPTER 12.—MINE POWER CENTERS[1]

The major power equipment in mining power centers, switchhouses and substations, was introduced in chapter 1, and an elementary overview of the protective circuitry involved was given in chapter 9. Other chapters have added numerous basic concepts and techniques whose thrust has actually provided the background for this chapter and the next, where the major power equipment used in mines is discussed in detail. In this chapter, the internal components and construction of typical mine power centers are covered. It should be noted that no formal standards have yet been developed for this equipment; hence, extensive use is made of information provided by major manufacturers serving the industry and by several mine operators.

Many circuit diagrams in this chapter apply directly to underground coal mining. With a few notable exceptions, underground power equipment is the most complex electrical equipment found in mining. An understanding of this equipment should, therefore, lead to comprehension of that used in any other mining system. In the paragraphs that follow, some material contained in preceding chapters is reviewed, but usually the content is either changed or expanded. The main reason is to enhance many section presentations without requiring reference to other information.

EQUIPMENT SPECIFICATIONS

An important objective of this chapter and the next is to provide sufficient information for an individual to draw up the detailed specifications required when procuring a piece of power equipment to fit a mining need. It is often stated that a manufacturer supplies what the customer requests. Since there are no official standards nor recommended practices at present, and given the variability of most equipment, every piece of mine power apparatus must be specified individually. If these specifications are not drawn up in complete detail, down to each nut and bolt, the purchaser might not receive what he or she intended.

Making detailed equipment specifications does not imply criticism of equipment manufacturers. It is a fact that many companies employ very capable applications-oriented engineers who have the total respect of the mining industry. Some can deliver precisely matched power systems just by knowing the mining equipment being used and the seam in which it is operating. Even in these cases, however, a complete specification from the buyer is still wise in terms of the protection it affords both the manufacturer and the customer. In situations where manufacturers receive specifications they believe are faulty or feel money can be saved by another approach, most manufacturers will contact the buyer for clarification rather than following the specifications rigidly and without question.

A listing of the minimum number of items recommended for a specification is provided below.

1.0. *Time Schedule*
2.0. *Work by Purchaser*
3.0. *General*
- 3.1. Intended use of equipment
- 3.2. Enclosure specifications
- 3.3. Special needs

4.0. *Internal Components*
- 4.1. One-line diagram
- 4.2. Description of each component

5.0. *Drawing and Manual Requirements*
- 5.1. With manufacturer's bid
- 5.2. Before construction starts
- 5.3. With equipment delivery

6.0. *Inspections*
- 6.1. During construction and prior to delivery, by purchaser
- 6.2. Manufacturer compliance with local, State, and Federal regulations applicable to company
- 6.3. Equipment (being built) compliance with applicable local, State, and Federal regulations

7.0. *Guarantee*
- 7.1. Minimum guarantee or warrantees demanded by buyer
- 7.2. Request for proper storage requirements before service

8.0. *Delivery Dates*

A document prepared to this format could be used as part of a purchase order directly to one manufacturer or as a solicitation for bids from several companies. Usually, such information is preceded by a general title, a phrase expressing what the specification covers, and the number of units desired. The next paragraphs will present suggestions for the content within each item of the document. The individual making the specification is referred to as the *buyer* or *purchaser*.

The time schedule alerts the manufacturer to when the buyer expects work on the equipment to begin. Furthermore, it serves notice that the manufacturer must have enough personnel, equipment, and material to complete work within the delivery time stated. (Obviously, if the number of sources for the equipment is restricted—and sometimes there could be only one, the buyer must accept the seller's delivery dates.) Item 2.0 should contain a statement that the purchaser will accept only equipment that meets the specifications. In other words, the units may be rejected if they do not comply, and the purchaser has no obligation to pay.

A detailed account of the intended use for the equipment should be given under item 3.1 so the manufacturer knows exactly what the specification covers and what the buyer expects to do with the unit after receiving it. Item

[1] The author wishes to thank Thomas Novak, associate professor of mineral engineering, the University of Alabama, who prepared the original manuscript for this chapter while he was an instructor in mining engineering at The Pennsylvania State University.

3.2 primarily applies (but is not restricted) to unit-designed equipment. Content examples include:

- Minimum material requirements.
- Physical protection of internal parts.
- Desired mounting (tire, skid, or rail).
- Measures to prevent water from entering enclosure.
- Maximum length, width, and height.

Minimum materials should always be stated when the specification is going out for bid and when ruggedness for the mining environment is imperative. To prevent any inadequate bids, it is wise to provide minimum acceptable steel thicknesses for the base plate, frame, end plates, and covers. Specification of maximum dimensions is essential for underground operations. It may seem absurd, but there have been many instances where equipment has been purchased that is too large to fit down a shaft or slope. Special needs, item 3.3., refers to such things as engraved nameplates for all major components and labeling of internal wiring and terminal blocks.

Item 4.0, internal components, is commonly the largest part of a specification. It should provide a complete listing of all components with one-line diagrams to show how they are to be connected. It is also helpful to state how each component is to be used and why each is included. Special needs such as transformer ratios, trip settings, and minimum insulation levels should be given where applicable.

Item 5.0 is concerned with requirements for drawings and manuals. If the specification is being sent out for bids, it is good to request both one-line diagrams and schematics of physical layout from prospective manufacturers. With unit-designed equipment, details for physical layout would include the basic frame, the mounting arrangement, and major component layouts. The manufacturer should be asked to submit all applicable outlines, arrangements, and schematics for approval prior to starting construction. The specification should also state the number of parts manuals and instruction manuals to be supplied on delivery of the equipment. Drawings, including some of reproducible quality, should also be requested where applicable.

Item 6.0 should be included to protect the customer. The first entry must relate that the buyer has the right to inspect the equipment during manufacture and prior to delivery and that any discrepancies from the specification must be corrected. At times, orders can be construed as contracts; therefore, the buyer might be held (in part) legally responsible if the manufacturer violates any local, State or Federal ordinances, codes, acts, regulations, or laws. Thus, the specification must demand compliance with all applicable statutes; this should be extended to any subcontractors or suppliers that the manufacturer might use. The last entry for item 6.0 is a statement that the specified equipment shall comply with all Federal and State regulations and requirements that apply to the intended use in the State in which it will operate.

Guarantees or warranties (item 7.0) cannot be obtained on some mine power equipment. Regardless, a request for a minimum guarantee is wise, for example, a 1-yr guarantee that begins after the equipment is placed in operation. The manufacturer should also be asked to supply any special instructions or precautions necessary for proper equipment storage prior to its being placed in service.

Finally, the specification must relate the dates that the equipment is to be delivered to the mining operation.

Establishing an adequate specification is not always an easy process. As will be seen, certain types of mine power equipment could require a document of numerous pages. However, a good specification might be the difference between having a unit that performs superbly throughout the life of the mine or one that is a headache from the day it is placed in service.

MINE POWER CENTERS

The power or load center is one of the most essential power-system units for underground mines and, in a simpler form, for surface mines. Its primary function is to convert the distribution voltage to utilization voltage for operating equipment throughout the mine. It must incorporate protective circuitry to ensure safe, efficient, and reliable operation. In effect, the power center could be considered a portable substation, but because of the ways in which it is used, its main component (and perhaps the enitre unit) might also be classified as a distribution transformer.

The electrical components of the power center are usually metal clad, that is, housed in a heavy-duty steel enclosure that may be tire mounted, skid mounted, or track mounted. Illustrations of typical underground coal mining units are given in figure 12.1 (*9, 12*).[2] Towing lugs or pin-and-link couplers are commonly provided on each end of the enclosures to permit towing as mining advances or retreats. Bumpers or check plates are often installed to protect externally mounted components, such as couplers, from damage by mobile equipment. Similar enclosures can be used in surface mines, but here the simplest power center consists of outdoor components assembled on a flatbed trailer with a fence or gates to discourage unauthorized entry.

There is no such thing as a "standard" power center because of the variety of mining practices and the numerous types of mining equipment used. The power center may supply only 1 motor or as many as 20 pieces of machinery; it may be totally ac, dc, or a combination of ac and dc. The distribution voltage received by the power center can be 4.16, 7.2, 12.47, 13.2, or 13.8 kV. The outgoing ac utilization voltage may be 480, 600, 995, 1,040 V, or a combination of 995 or 1,040 V with one of the lower voltages. Dc can be at 300 or 600 V, but is almost always 300 V for face applications. As a result of this variety, manufacturers custom-build the units to meet the individual needs and specifications of the customer. However, there is a general design philosophy central to all power-center types and this forms the basis of the following discussion of a typical ac power center and a combination ac–dc power center.

[2] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

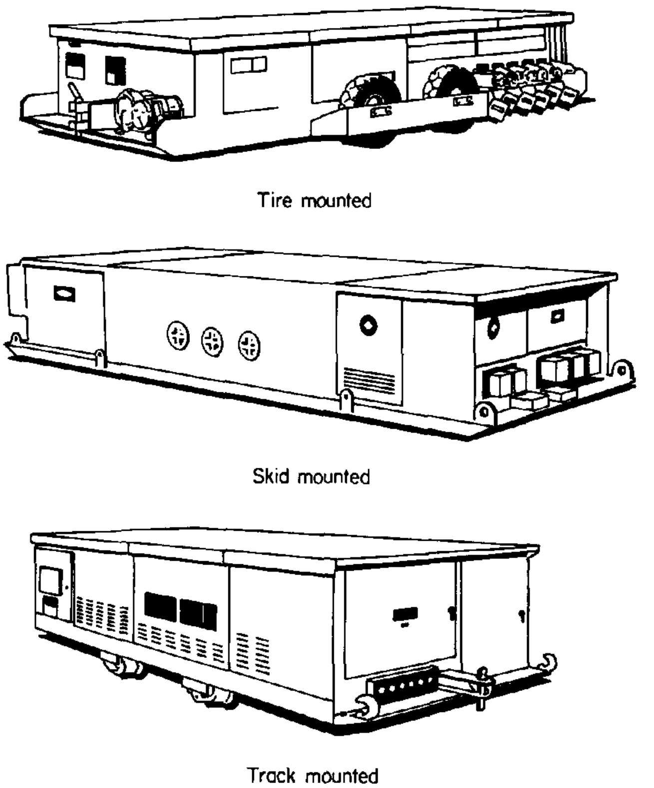

Figure 12.1.—Typical power centers used in underground coal mines.

The major electrical components of a typical mine power center are shown in the schematic of figure 12.2, and a possible component placement is provided in figure 12.3. The circled numbers in figure 12.2 will be used to indicate component location with respect to the overall system throughout the following descriptions of individual components.

HIGH-VOLTAGE CABLE COUPLER

Incoming power is usually supplied to the center from the distribution cable through a high-voltage cable coupler (fig. 12.2, No. 1). The receptacle (typically with female contacts) is cable mounted, while the plug (male contacts) is gear mounted. Although the conductor pins are recessed in the coupler housing, dust caps should be provided for each side of the coupler pairs to protect the contacts when disengaged.

When being disconnected, the coupler is designed so that the pilot contact is broken first, the line conductors second, and the grounding conductor last. As was discussed in chapter 9, the pilot contact is interlocked with the upstream high-voltage circuit breaker, which protects the incoming line. If the incoming power is energized, the power will be tripped by the associated ground-check monitor prior to breaking the power contacts of the coupler. Having the grounding contact break last ensures that the frame is tied to earth ground whenever the power center is energized.

A feedthrough receptacle may be provided as shown in figure 12.4; the contacts are typically female. This permits

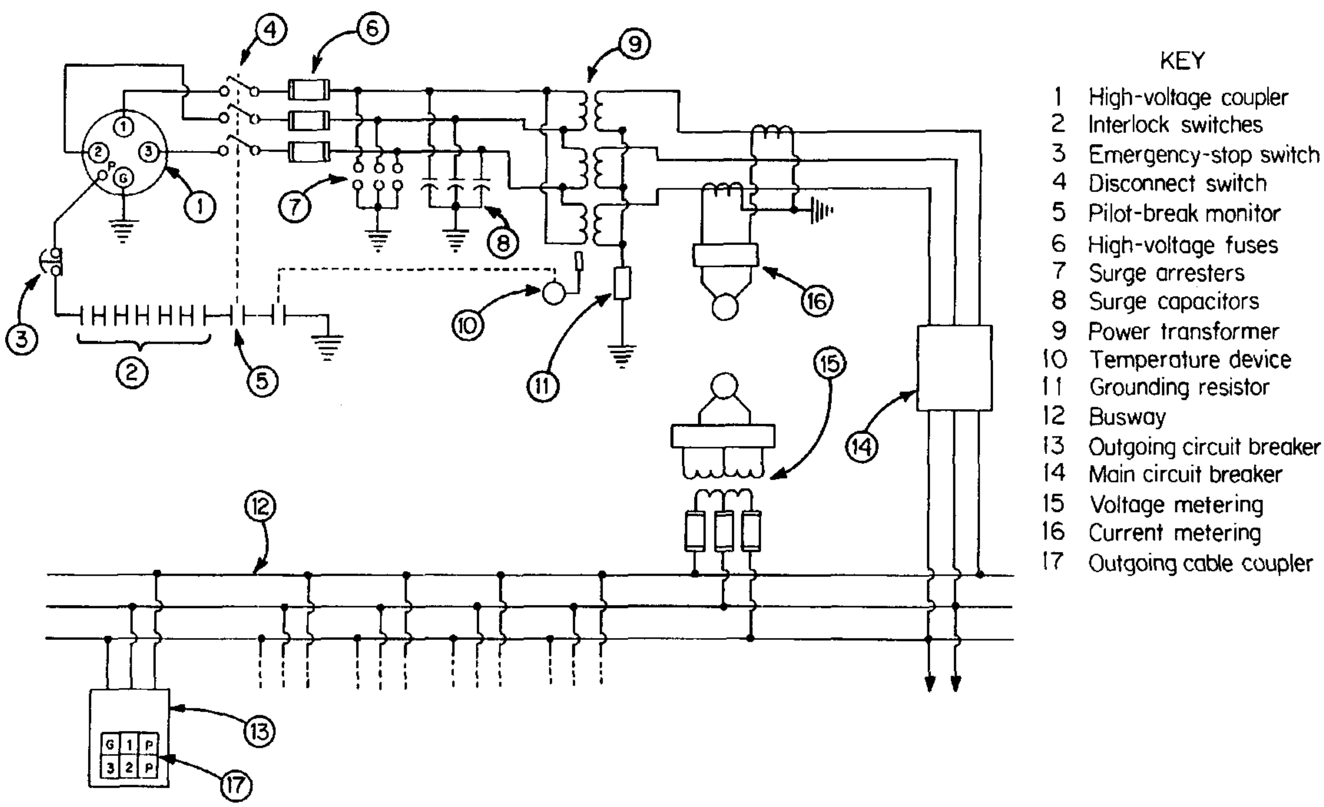

Figure 12.2.—Schematic illustrating major components in power center.

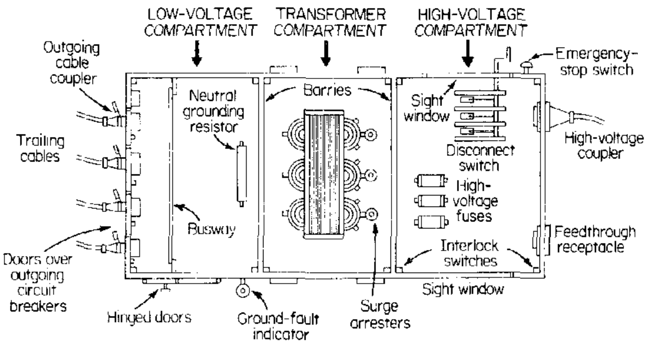

Figure 12.3.—Top view of mine power center showing placement of many internal components.

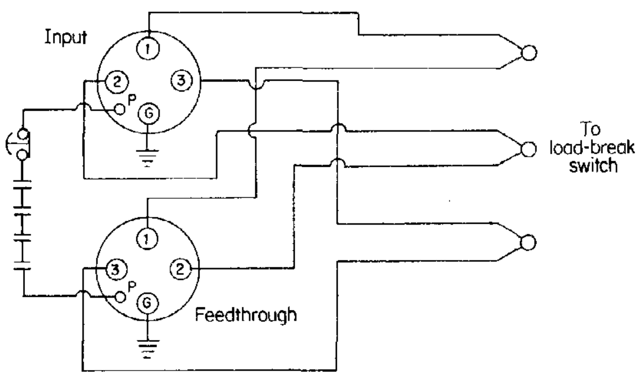

Figure 12.4.—Interconnections between input and feedthrough receptacles.

distribution power to be supplied (or continued) through the power center at other high-voltage loads. A dust cap is again provided for use when the feedthrough receptacle is not in service. This cap also shorts the pilot contact to the grounding contact to provide a closed path for the ground-check monitor interlocking with the upstream circuit breaker. The conductors between the input and feedthrough receptacles should be sized to the maximum capacity of the distribution system.

INTERLOCK SWITCHES

Electrical interlock switches (fig. 12.2, No. 2) are presently used in the high-voltage and transformer compartments of mine power equipment. Strategically located around exterior top and side protective covers, they act to deenergize the internal power circuitry if the panels are removed, and thus help to prevent accidents caused by a worker's contacting energized components. The normally open (*NO*) contacts of the switches are connected in the pilot circuit of the incoming distribution cable as shown in figure 12.2. With covers in place, the interlock switches are depressed and their associated contacts are closed. This provides a closed path for the pilot circuit. When a cover is removed, the switch contacts are opened, thus causing the upstream circuit breaker to trip because of the loss of continuity in the pilot circuit.

For added safety, some manufacturers sectionalize their power centers into three separate compartments: high voltage, transformer, and low or medium voltage (fig. 12.3). If the equipment has no barriers, interlock switches should be placed on the low-voltage or medium-voltage exterior covers, with their contacts in series with the incoming pilot circuit. These additional switches are recommended even with the compartment segregation, but then the interlocks are connected to trip circuit breakers associated with the transformer secondary, thus avoiding tripping the upstream switchhouse.

An emergency-stop button (fig. 12.2, No. 3) should also be provided, and its function is similar to the interlock. It consists of a normally closed (*NC*) set of contacts in series with the interlock switches. If the stop button is depressed, its contacts are opened, opening the incoming pilot circuit, and thus tripping the upstream circuit breaker. The switch should not automatically reset to the normally closed position after being depressed; manual resetting of the switch should be required.

DISCONNECT SWITCH

The disconnect or load-break switch (fig. 12.2, No. 4) is a mechanically operated air-type switch whose primary function is to allow a quick means of disconnecting the primary of the power-center transformer. A spring-loaded or torsion-bar mechanism provides the quick-make and quick-break operation, which is independent of the speed of the manually activated handle. Observation windows are provided in the power-center enclosure, and the switch can serve as a visual disconnect.

Disconnect switches have no interrupting capability, but load-break switches do. Load-break switches are able to interrupt currents that are not in excess of their continuous-current rating. They are also rated for a maximum interrupting capacity, but at this level they are designed for one-time interruption only. Load-break switches are more expensive than disconnects, yet many mine power engineers prefer them because of their extra ruggedness. Some States require their use. Continuous-current ratings of 400 and 600 A and a 15-kV voltage rating are common for mining applications. Table 12.1 illustrates typical ratings of a 400-A load-break switch.

Table 12.1.—Typical current ratings of 400-A load-break switch

	Rating, A
Full-load current	400
Rated interrupting capacity	400
Maximum interrupting capacity	1,200
Short-time current (1 s)	20,000
Impulse current	45,000
Close-and-latch current	37,000

The only means of activating a load-break switch should be by the manually operated handle accessible from outside the load-center enclosure. Its operating mechanism should not be tied into the power-center protective circuitry. More will be said about this recommendation soon.

If a disconnect switch is allowed and employed, a pilot-break monitor (fig. 12.2, No. 5) is required to interlock the switch handle with the pilot circuit of the incoming distribution system. This allows the upstream circuit breaker to interrupt the circuit prior to opening the switch contacts. It is also desirable to use pilot interlocking on load-break switches to extend the life of the contacts. Again, some States require both load-break switches and pilot-break monitoring in order to minimize any danger of operating the switch under load. The balance of this chapter will use or imply the term *load-break* for all disconnect-switch applications.

HIGH-VOLTAGE FUSES

A fuse has been defined as an overcurrent protective device with a circuit-opening fusible element that is heated and severed by the passage of current through it. Current-limiting fuses (fig. 12.2, No. 6) are the type used in mine power centers. Their main purpose is to protect the high-voltage section, particularly the transformer, during short circuits. High-voltage circuit breakers are not recommended here, because of their potential for creating transients.

For resistance-grounded mine power systems, the fuse voltage rating should be based on the line-to-line voltage. The following two rules-of-thumb are recommended for determining the fuse current rating that will ensure that fuse action will not be triggered by the transformer inrush current.

1. The fuse should be able to withstand 12 times the rated current of the transformer for 0.1 s without element damage.
2. The element should be able to withstand 25 times the rated current of the transformer for 1/2 cycle.

Based upon the above criteria, the continuous-current rating of the current-limiting fuses usually falls in the range of 1.5 to 2.5 times the rated transformer current.

Transient overvoltages can be generated during the operation of the fuses. The magnitude of the overvoltage depends upon the point of the waveform at which initiation of the fault occurs and the size and design of the fuse element. For ribbon-element fuses, the generated peak arc voltage is primarily a function of the system voltage, as shown in figure 12.5 (*10*). The peak arc voltage should be compared with the minimum 60-Hz sparkover level of the surge arresters. The sparkover level of the arresters, discussed in the next section, should be multiplied by $\sqrt{2}$ to obtain a peak voltage. If this value is greater than the peak arc voltage, arrester sparkover will not result from fuse interruption.

The fuses are sometimes used in conjunction with an automatic (fused) load-break switch, which has the objective of preventing the power center from operating under a single-phase condition if only one fuse blows. The fuses normally have actuators located at one end of their casings. In the event that a fuse element is blown, the actuator extends outward from the fuse end and activates the trip mechanism of the switch. This causes the switch to open and eliminates the single-phase condition. However, for a three-phase fault, the possibility exists that one fuse may blow before the others, which could result in the load-break switch attempting to clear a fault current higher than its designated rating. As a result, this type of automatic load-break switch is not recommended in power-center applications.

Single-phasing protection is better accomplished at the secondary bus, using relays to monitor for a single-phase or phase-reversal condition. The relay contacts can be employed to activate the undervoltage release or shunt trip of the main breaker on the secondary. If a main breaker is not used, the contacts can be connected directly at the output of the control transformer secondary winding, so all outgoing circuit breakers will trip through their undervoltage releases if the relay is actuated.

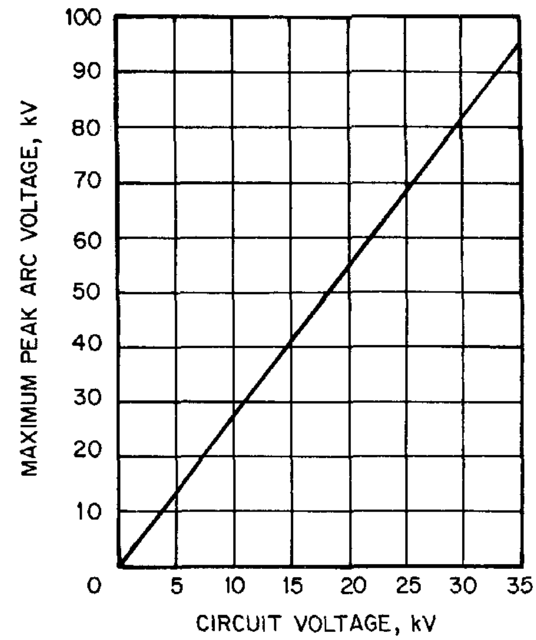

Figure 12.5.—Graph illustrating transient crest voltage caused by ribbon-element current-limiting fuse operation.

SURGE ARRESTERS

The surge arrester (fig. 12.2, No. 7) is a device designed to limit dangerous transient overvoltages to safe levels. Lightning, switching surges, and some faults result in transient overvoltages that can exceed the insulation levels of power-system equipment. Since it is not always economically possible to rate the equipment insulation above the surges, overvoltages must be clamped or suppressed to tolerable levels. The function of an arrester is

- To discharge the energy associated with a transient overvoltage;
- To limit and interrupt the 60-Hz current that follows the transient current through the arrester; and
- To return to an insulating state without interrupting the supply of power to the load.

Valve-surge arresters of the low-sparkover distribution class are used almost exclusively in mine power centers. A complete discussion of these devices as well as application tables was provided in chapter 11.

The voltage rating of a surge arrester is the highest power-frequency voltage at which the arrester is designed to operate. As mine power systems utilize resistance grounding and fall into the category of noneffectively grounded systems, the surge-arrester voltage rating must be selected on a line-to-line basis. However, the rating should be 5% to 10% above the nominal distribution voltage to allow for voltage-regulation compensation.

The component protected by the surge arrester is the main power transformer. The key to protection is coordination between the transformer insulation withstand, or BIL (basic impulse insulation level), and the characteristics of the transient voltage the arrester lets through. Dry transformers are used almost exclusively in present-day power centers, and their insulation strength does not increase significantly above their BIL as the duration of the applied pulse decreases. Therefore, the voltage-time withstand characteristic can be plotted as a flat line with a value equal to the BIL, as shown in figure 12.6 (*11*). The arrester characteristic, including impulse sparkover and discharge voltage, is also illustrated. The margin of protection is the difference between the transformer BIL and the arrester discharge characteristic at any given instant of time. This value should not be less than 20% if a 5,000-A surge discharge current is assumed. Note that the worst case discharge of 20,000 A need not be used in typical power-center locations.

The physical location of surge arresters within the power center is important for effective operation. The two critical distances are the conductor lengths to the line conductor and to the power-center frame ground, and the distance between the arrester and the transformer. To maximize arrester performance, the arrester leads must be as short and straight as possible and made of No. 6 AWG solid copper or larger. Also, the surge arrester and arrester connections to the line conductors should be as close to the transformer primary terminals as possible. Preferably, the surge arrester leads should be connected right at these terminals.

Surge capacitors (fig. 12.2, No. 8) can be found in numerous mine power centers. Their intended purpose is to lessen the severity of transient overvoltages caused by current chopping in vacuum circuit breakers. Their effectiveness is based upon the ability to limit the rate-of-rise of transient overvoltages and to reduce the system characteristic impedance. However, as was discussed in chapter 11, surge capacitor use in load centers is generally not required, provided that the surge arresters are correctly applied, and a minimum practical cable length (100 ft) exists between the power center and the upstream circuit breaker.

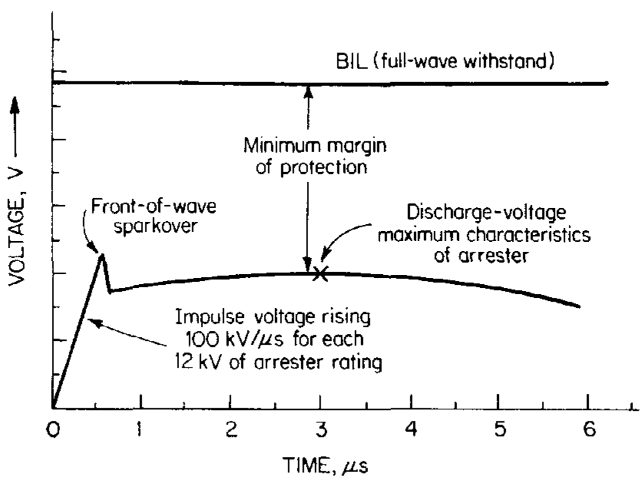

Figure 12.6.—Comparison of transformer withstand characteristic and surge arrester withstand characteristic.

TRANSFORMERS

The main transformer (fig. 12.2, No. 9) can be considered the heart of the power center, since its primary function is to convert the distribution voltage to utilization. Proper selection is therefore imperative from the standpoint of safety, efficiency, and reliability. Fortunately, the IEEE (*2, 4*) has established classifications and specifications for determining transformer characteristics as an aid in design and application, and these regulations are in general use in the mining industry.

The IEEE uses three items for the general classification of transformers: distribution or power, insulation, and substation or unit substation. The capacity or kilovoltampere rating determines whether the transformer is classed as a distribution or power unit. Distribution transformers fall in the range from 3 to 500 kVA, with power transformers having capacities greater than 500 kVA. Power-center transformers can be of either classification, since they may range from 150 kVA for a rectifier or belt drive, up to 2,250 kVA for a longwall section. Capacities are rarely greater than 1,250 kVA.

A transformer may be further classified by the insulation system as liquid or dry. Liquid insulation includes mineral oil or synthetic fluids, and dry transformers are ventilated or sealed gas-filled types. Ventilated dry units are used almost exclusively in mine power centers. With convection cooling and air insulation, dry transformers have the following advantages:

- Toxic gases cannot be released.
- The transformer cannot explode or catch fire.
- There is no oil or other liquid to spill, leak, or dispose of (a critical problem with PCB, polychlorinated biphenyls).
- The transformer is virtually maintenance-free because there are no valves, pumps, or gauges.

The substation-transformer classification refers to direct or overhead-line termination facilities, while a unit-substation transformer has an integral connection to primary or secondary switchgear. *Primary* refers to a voltage of 1,000 V or higher, while *secondary* refers to a rating less than 1,000 V. For practical purposes, 1,040 V is also considered 1,000 V. Therefore, a load-center transformer can be considered a unit-substation class.

Specifications

The IEEE transformer specifications (*4*) that apply to mine power centers include

- Capacity or kilovoltampere rating,
- Phases,
- Frequency,
- Voltage rating,
- Voltage taps,
- Winding connections,
- Impedance,
- BIL,
- Temperature rise and insulation,
- Cooling.

Each will be discussed in general terms in the next paragraphs. There are a few exceptions to the IEEE standard.

There is no set formula for determining the kilovoltampere rating for a power-center transformer. For constant loads, it is a common rule-of-thumb to allow 1 kVA for each horsepower of connected load. However, the mining process does not produce a constant load (that is, all connected motors are not operating at the same time on a continuous basis); hence, using the rule-of-thumb will normally result in oversizing the transformer. Past experience and demand factors established by manufacturers and operators, along with the horsepower of the connected load, are essential for determining the transformer capacity. For typical underground mining sections, the kilovoltampere rating may lie within the range of 50% to 80% of the connected horsepower. (Chapters 4 and 8 contain additional information on demand factors.)

As an example of sizing, consider the following continuous-mining section:

1. 370-hp continuous miner (gathering head, pump, tram, and cutter motors),
2. Two 125-hp shuttle cars (traction, conveyors, and pump motors),
3. 80-hp roof bolter (traction and pump motors), and
4. 50 hp of auxiliary equipment (section fan, sump pump, hand tools, etc.).

The total connected horsepower sums to 750 hp. If the demand factor has been determined to be 60%, the effective load would be 450 kVA, and a 500-kVA transformer would be selected. However, flexibility must always be considered in mining applications, and as a result, it may be necessary to select the next higher kilovoltampere rating to accommodate anticipated additional loads.

Over the years, power-center manufacturers have arrived at certain standard or typical transformer capacities, created mainly by repeat demand of the industry. These are—

Ac output only (section may have outboard rectifier for dc load): 150, 225, 300, 500, 600, 750, 1,000, and 1,250 kVA.

Dc only (often termed *rectifiers*; rating of unit commonly in kilowatts): 100, 150, 200, 300, 500, and 750 kVA.

Combination ac–dc (typical dc capacity; on larger units, 200 and 300 kW for dc also available): 300/150, 500/150, 600/150, 750/150, 1,000/150, and 1,250/150 kVA. Many manufacturers will supply capacities at any increment of 50 kVA up to 2,250 kVA, the maximum allowed by Federal regulations for face applications in underground coal mines.

Three-winding transformers are necessary in many power centers, for example, when ac face equipment has two rated voltages such as a 950-V continuous miner working with 550-V machines, or in the common application of mixed ac and dc machinery. In these cases, the capacity of each transformer winding (primary, secondary, and tertiary) must be individually rated.

All mine power-center transformers are three phase, being either three single-phase units where each transformer is rated at one-third of the total required capacity, or integral three-phase types with construction allowing field replacement of failed windings. The integral three-phase transformers are preferred because they have equal reliability, lower cost, and higher efficiency, take less space, and have fewer exposed interconnections than three single-phase units.

Transformers should be rated at the standard commercial frequency in the United States, which is 60 Hz. If a transformer rated at 60 Hz is applied to a 50-Hz system, the voltage and the kilovoltampere rating must be reduced by 20%. Transformers rated at 50 Hz can be operated at 60 Hz, but the efficiency and regulation are reduced, since the reactance is directly proportional to the frequency.

The primary voltage rating of the transformer is dictated by the distribution voltage (that is, 4.16, 7.2, or 12.47 kV, and so on). The voltage rating of the secondary winding must match the voltage of the utilization equipment. In practice, the transformer secondary should be rated at a higher voltage (10% is common) than shown on the nameplate of the utilization equipment, to allow for a voltage drop in the trailing cables. The secondary winding of power-center transformers is most commonly rated at 480 or 600 V, relating to 440- or 550-V equipment, respectively. Secondaries supplying 950-V machines are usually rated at 1,040 V, except for machines that cannot be remotely controlled or in States where face voltages over 1,000 V are not allowed. In these instances, 995 V is often specified. The transformer voltage for dc applications is dictated by the rectifiers and will be presented in a separate section.

Power-center transformers should be provided with voltage taps on the primary windings to account for voltage fluctuations and line losses in the distribution system. Five fully-rated taps at 2.5% increments are usually available, with the middle tap being rated at the nominal distribution voltage. Figure 12.7 illustrates this arrangement for a 7.2-kV primary on a per-phase basis.

Delta primary and wye secondary are the preferred connections for standard two-winding power transformers and are the connections commonly used in mine power centers. An external busing from the neutral of the wye secondary provides an easy means for resistance grounding, as required by Federal regulations. The delta-connected primary provides isolation of the distribution circuit from the utilization circuit with respect to zero-sequence currents resulting from exciting currents or secondary ground faults. The delta-wye connection also stabilizes the secondary neutral point and minimizes the production of third-harmonic voltages. Additional information concerning the reasoning behind using delta-wye transformers can be found in chapters 4 and 9, including the reasons wye-wye transformers are not recommended.

In certain situations, a delta-connected secondary may be specified or required, and the primary may be delta or wye in this case. If neutral grounding is desired or required, a grounding transformer is needed to derive a neutral. The zig-zag grounding transformer, illustrated in figure 12.8, is sized according to the continuous-current

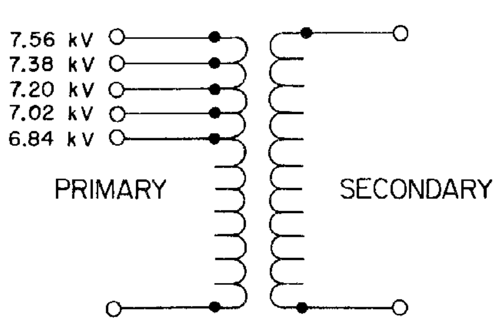

Figure 12.7.—Typical primary winding taps on power cable transformer.

rating of the neutral-grounding resistor. Standard single-phase *or integral three-phase transformers, arranged in a wye-delta configuration*, can be used as grounding transformers in lieu of the special zig-zag type, as shown in figure 12.9.

The transformer impedance is sometimes referred to an *impedance voltage* because of the common means of measurement. Impedance voltage is the voltage required to circulate rated current through one of two windings of a transformer when the other winding is short-circuited, with the windings connected as for rated voltage operation (*4*).

An example will help to illustrate this procedure. Figure 12.10 shows a three-phase delta-wye distribution transformer with the indicated ratings. The secondary of the transformer is short-circuited while a three-phase variable voltage source is connected to the primary. The voltage source is wye connected so the impedance voltage, V_i, will be a line-to-neutral voltage. The source voltage is increased until the ammeter reads the rated or full-load current of the transformer, I_{FL}, which can be determined by

$$I_{FL} = \frac{S}{\sqrt{3}\,V}, \qquad (12.1)$$

where I_{FL} = full-load current of transformer, A,
S = rated transformer capacity, VA,
and V = rated voltage at transformer winding terminals, V,

or for the primary,

$$I_{FL} = \frac{500{,}000}{\sqrt{3}\,(7200)} = 40.1 \text{ A}.$$

Consider the case where V_i = 250 V, when the ammeter reads 40.1 A. The impedance voltage is normally expressed in terms of a per-unit value; therefore,

$$V_i = \frac{250 \text{ V}}{4160 \text{ V}} = 0.06 \text{ pu}.$$

The line-to-neutral rating of the transformer (4,160 V) is used since the impedance is being measured as a line-to-neutral potential. For this example, the line-to-neutral impedance of the transformer, with all quantities reflected to the primary, would be

$$Z_T = \frac{V_i}{I_{FL}} = \frac{250 \text{ V}}{40.1\text{A}} = 6.23\ \Omega.$$

The transformer impedance can be measured in terms of a per-unit quantity by using the ratings of the transformer as base values, as follows:

$$\text{kVA}_b = 500 \text{ kVA},$$

$$\text{kV}_b = \frac{7{,}200}{\sqrt{3}} = 4.16 \text{ kV (line to neutral)},$$

$$I_b = \frac{\text{kVA}_b}{\sqrt{3}\,\text{kV}_b} = \frac{500}{\sqrt{3}\,(7.2)} = 40.1 \text{ A},$$

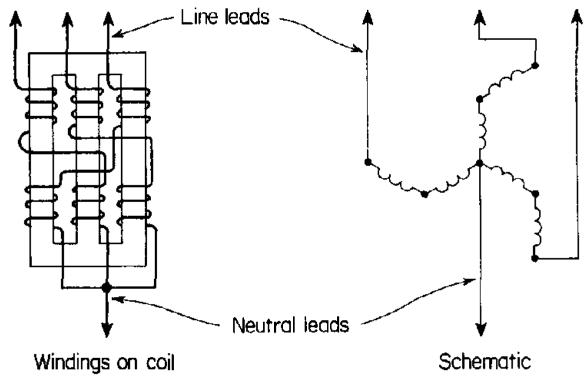

Figure 12.8.—Zig-zig grounding transformer.

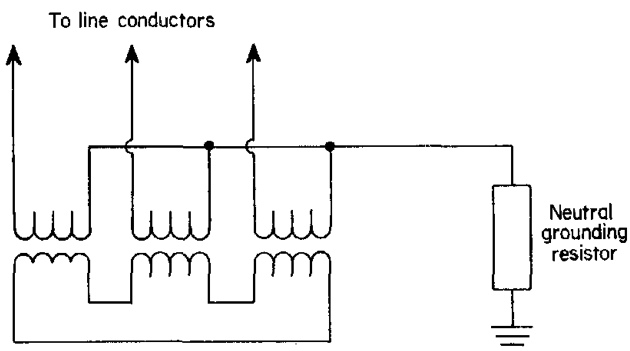

Figure 12.9.—Delta-wye connection for deriving a neutral.

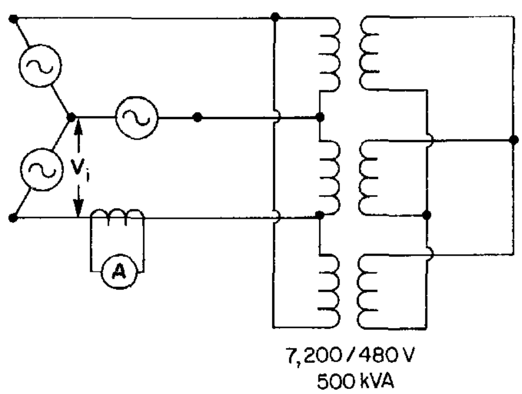

Figure 12.10.—Technique for measuring transformer impedance.

$$Z_b = \frac{V_b}{I_b} = \frac{4{,}160}{40.1} = 103.7\ \Omega.$$

Thus, the per-unit value of the transformer impedance is

$$Z_{T_{pu}} = \frac{6.23\ \Omega}{103.7\ \Omega} = 0.06 \text{ pu}.$$

This yields the same results; therefore in terms of per-unit quantities, the terms impedance and impedance voltage are interchangeable.

The vector sum of the leakage reactance and the resistance of the windings comprise the impedance of a transformer. Figure 12.11 shows that the X/R ratio of a typical transformer increases as the size (megavoltampere rating) increases (*13*). The transformer impedance is extremely important since it plays a major role in voltage regulation as well as dictating the amount of current that can be delivered to a fault on the secondary. Here, the maximum symmetrical secondary-fault current (three-phase fault) can quickly be estimated by

$$I_{f(max)} = \frac{I_{FLs}}{Z_{Tpu}}, \tag{12.2}$$

where $I_{f(max)}$ = maximum symmetrical fault current, A,
I_{FLs} = full-load current of transformer secondary, A,
and Z_{Tpu} = transformer per-unit impedance, pu Ω.

For the previous example,

$$I_{FLs} = \frac{kVA}{\sqrt{3}\ kV} = \frac{500}{\sqrt{3}\,(0.48)} = 601$$

and $$I_{f(max)} = \frac{601}{0.06} = 10{,}023\ A.$$

This technique produces a more pessimistic estimate than would actually occur, since it assumes that the transformer primary is connected to an infinite bus. A more realistic value for fault current should take into account the system impedance upstream from the transformer as well as an equivalent impedance for the utility. However, this system impedance is constantly changing because of the dynamic nature of the mining process. By using the above method, it is possible to obtain a conservative determination of the required interrupting capacity for protective circuitry serving the utilization equipment.

For transformers used in mine power centers, the per-unit values of impedance normally lie in the following range:

- 0.04 to 0.05 pu for 150 to 450 kVA, and
- 0.05 to 0.06 pu for 500 to 1,250 kVA.

The 0.04-pu lower limit is usually needed to limit short-circuit currents to reasonably safe values. For large power transformers, the leakage reactance may be intentionally increased in the design of the transformer to further limit the available fault current. External air-core reactors in series with the secondary windings are sometimes used to restrict fault current from transformers that feed rectifiers. This will be discussed in the section on ac–dc combination power centers.

For transformers used in mine power centers, the following minimum insulation ratings should be used (the BIL's in parentheses are optional):

4.16-kV primary, 35-kV (60-kV) BIL;
7.2-kV, 60-kV (75-kV) BIL;
12.47-kV (13.2 and 13.8 kV), 95-kV BIL; and
23-kV, 150-kV BIL.

Coordination with surge arrester characteristics must be maintained, as discussed earlier. (See chapter 11 for further information.)

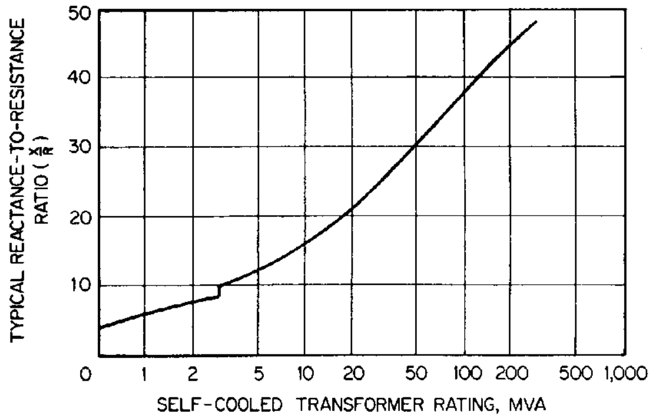

Figure 12.11.—Typical X/R ratio versus transformer capacity.

The kilovoltampere rating of a transformer is based upon continuous operation at rated voltage and frequency without exceeding the specified temperature rise or limiting temperature. The temperature rise depends upon the core and conductor losses. The core losses remain constant with load, while the conductor losses are determined by the I^2R of the windings.

Dry transformers used in mine power centers should have class 220°C insulation. This classification terminology has replaced the old class H 150°C rise notation that was based on an average ambient temperature of 30°C, allowing a maximum 150°C temperature rise in the windings (*2*). This restricted the average insulation temperature to 180°C, but a 220°C hot-spot temperature was also allowed. The new classification simply places an absolute allowable maximum temperature on the transformer when it is operating continuously under full capacity at rated voltage and current.

Because of the use of trailing cables and mobile equipment, short-circuit conditions are frequently encountered in mining, and transformer windings can undergo considerable thermal and magnetic stress during these occurrences. Consequently, the transformer should be designed to withstand 25 times rated current in any winding for a period of 2 s. One method of curtailing expansion is to add an insulated brace across the windings, and some engineers request adjustable braces, which allow periodic readjustments to be made when expansion causes loosening.

A temperature-sensing device (fig. 12.2, No. 10) can be placed in the transformer windings to prevent damage from overheating. The device controls a set of contacts located in the pilot circuit of the incoming distribution line. If the transformer temperature exceeds the specified limit, the contacts in the pilot circuit are opened, which results in tripping the upstream circuit breaker. As an alternative, the contacts can be used to activate the tripping element of a main secondary breaker or all outgoing breakers.

Mine power centers are usually cooled by natural convection, and the side panels of the transformer compartment normally have louvers to allow air circulation. The transformer windings are designed so that the heat generated by the I^2R losses is exposed to an adequate amount of cooling to handle the expected loads. The effective cooling areas are the inside of the winding, the outside of the winding, and the cooling ducts within the winding. The movement of air by convection carries away the heat generated by the winding (*13*).

Transformer Construction

The two main transformer components are the core and the windings. Figures 12.12 and 12.13 show a typical mine power-center transformer in the process of being constructed and in completed form. The transformer core provides a path of low reluctance to the flux produced by the primary windings. In effect, it comprises the magnetic circuit of the transformer. To reduce eddy currents, the core is constructed of laminated sheet steel. The eddy-current losses vary as the square of the thickness of laminations, and core laminations are usually from 0.01 to 0.02 in thick (*13*). The laminated material is cut from silicon-iron sheets. The silicon reduces the reluctance to hysteresis and prevents increased loss with age. The laminated material is specially annealed to obtain a high permeability and is also treated with a chemical coating to insulate the laminations from each other. The laminations are stacked one upon another to construct a closed magnetic path. Alternate layers are staggered in an interlaminar core-and-gap construction so all joints do not meet at the same place. This is done to reduce the effect of the air gap between the joints and make the entire structure function more like a solid piece of iron (*13*).

The primary and secondary windings comprise the current circuit of the transformer. The windings are designed to get the required number of turns into the minimum of space through the core opening, while also providing enough room for insulation and cooling ducts. Transformer windings are made of copper or aluminum. Because of its lower conductivity, aluminum requires a larger cross section of conductor and thus a larger opening in the core. The conductors used in the windings may be round, square, or rectangular in cross section. Aluminum secondaries are often wound using sheet metal. All windings may be insulated with enamel, mica paper, NOMEX paper, silicon glass tape, or a combination of these materials (*8*).

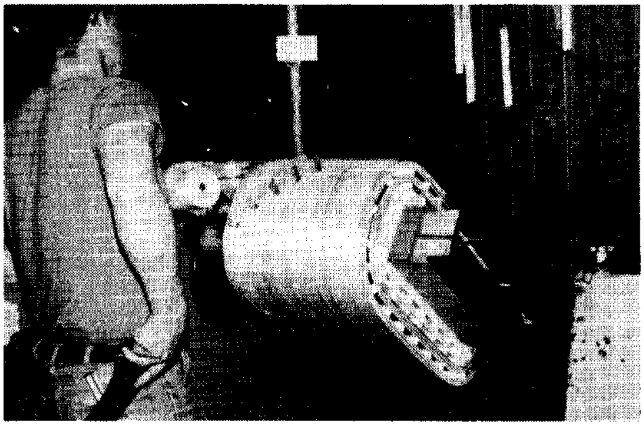

Figure 12.12.—Typical mine power-center transformer under construction. (Courtesy PEMCO Corp.)

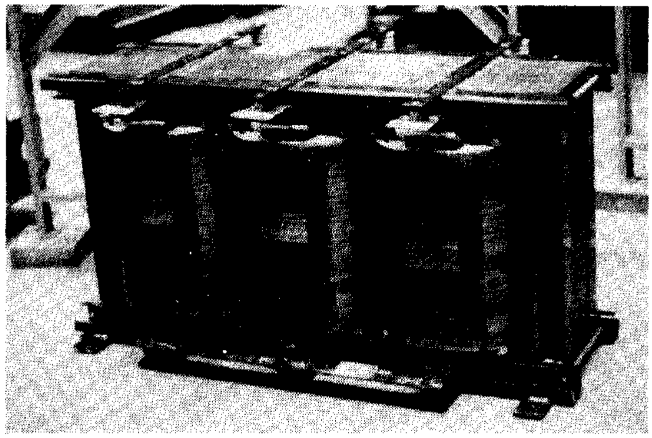

Figure 12.13.—Completed transformer prior to installation. (Courtesy PEMCO Corp.)

Faraday Shields

Transformer windings can be provided with a grounded Faraday (or electrostatic) shield to destroy interwinding capacitance between the primary and secondary, thus reducing the danger of transferring distribution transients to utilization. Another important use of the shield is to prevent interwinding faults between layered windings. This is especially critical when the secondaries are isolated above ground, as the primary voltage can be impressed on the secondary without detection. Using the shield, a ground fault will occur in this situation, and the resulting ground current will be detected by upstream ground-fault relaying.

The shield consists of a layer of nonmagnetic metal placed between the primary and secondary windings, insulated from all windings, and connected solidly to ground (*1*). Aluminum or copper can be used, but the shield should be made of the same material as the main windings. Physically, it may be a single turn of sheet metal or a closely wound single layer of wire. To obtain interwinding fault protection, the shield must have the same ampacity as the grounding conductors leading to the power center, in order to carry maximum available ground-fault current (at least, equivalent to one-half the cross-sectional area of the primary-winding wire).

GROUNDING RESISTOR

Federal regulations require the maximum frame potential on medium-voltage and low-voltage circuits to be limited to 40 V. In an attempt to ensure this, the regulations further require the maximum ground-fault current of these circuits to be limited to 25 A. However, many modern power centers are designed to limit the ground fault current to 15 A for an additional factor of safety. Limiting the ground-fault current results in the following additional benefits (*7*):

- Reduction in burning and melting of faulted electrical equipment,
- Reduction in mechanical stress of faulted electrical equipment,
- Ability to selectively clear the faulted circuit, and
- Reduction in overvoltages that might cause insulation failure.

The grounding resistor (fig. 12.2, No. 11) is inserted between the neutral of the transformer and the power-center frame as a means of limiting the ground-fault current. The ohmic value of the grounding resistor is based on the maximum ground-fault current condition; that is, a ground fault at the secondary terminal of the transformer. For this situation and neglecting the transformer impedance, the resistance is determined by

$$R_G = \frac{V_{ln}}{I_f}, \tag{12.3}$$

where R_G = ohmic value of grounding resistor, Ω,
V_{ln} = line-to-neutral potential of transformer secondary, V,
and I_f = maximum ground-fault current, A.

As an example, consider sizing a grounding resistor to limit the ground-fault current to 15 A for a 480-V system. The ohmic value of the resistor would be

$$R_G = \frac{480/\sqrt{3}}{15} = 18.5\ \Omega.$$

When sizing a grounding resistor, the time rating must also be considered. Under normal conditions, an insignificant amount of current flows through the resistor, but during the worst case situation, 15 A may flow through it. Thus the power (P) dissipated by the neutral-grounding resistor would be

$$P = I_f^2 R_G \tag{12.4}$$

or

$$P = (15)^2(18.5) = 4{,}162\ \mathrm{W}.$$

Normally the resistor would only be required to dissipate this power for the time it takes for a circuit breaker to trip, but the possibility of the circuit breaker's failing to trip must also be considered. Federal regulations require the resistor rating to be based on an extended time rating. This has been defined as 90 days of operation per year. The end of the resistor that is connected to the neutral of the transformer is also required to be insulated for line-to-line voltage.

BUSWAY

The power-center output requires numerous taps for feeding the utilization circuits. A busway (or busbars) provides a convenient and economical means of providing these taps (fig. 12.2, No. 12). The busway consists of flat, bare conductors supported within the power-center enclosure by means of insulators made of glass, polyester, or porcelain, as can be seen in figure 12.14. Busways are available with either copper or aluminum conductors. Aluminum has a lower electrical conductivity and lower mechanical strength and quickly forms an insulating film on its surface when exposed to the atmosphere. If aluminum conductors are used, they should have electroplated contact surfaces (tin or silver), and bolting practices that accommodate aluminum mechanical properties should be used at electrical joints (*5*).

The continuous-current rating of the busway is based on the cross-sectional area of the conductor and a maximum temperature rise of 55° C from ambient. The short-circuit rating should exceed the maximum available fault current. The sizing of the busway is basically the same as that discussed for other conductors in chapter 8.

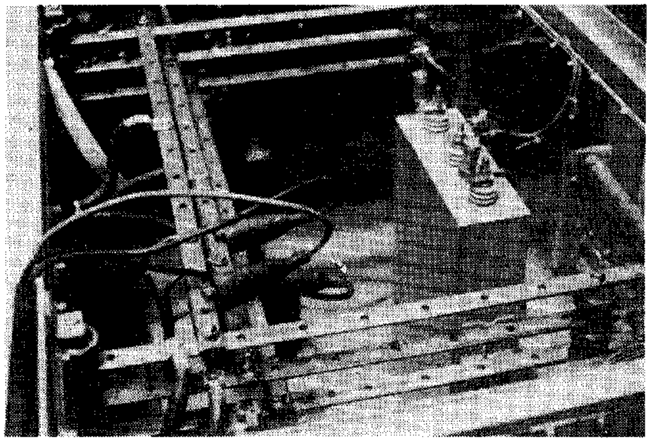

Figure 12.14.—Typical bus work in power center under construction. (Courtesy PEMCO Corp.)

OUTGOING CIRCUIT BREAKER

Molded-case circuit breakers (fig. 12.2, No. 13) are usually employed to protect ac utilization equipment and associated cables. Some manufacturers have special mine-duty breakers available, which have greater ruggedness and reliability to meet the demands of mining practices. Dual-element fuses are rarely used because of single-phasing problems and the lack of auxiliary protective-relaying capabilities. A complete discussion of molded-case breaker and fuse applications, sizing, and problems is available in chapters 9 and 10. A presentation of molded-case devices with solid-state tripping elements is contained in chapter 14.

The power-center circuit breaker compartment must be designed for easy access, but protection is necessary against accidental exposure to energized terminals and conductors. Dead-front panels are the preferred method of construction. Here the breaker is mounted on a recessed panel so that only the operating handle and adjustments are accessible from the outside. As an alternative, the breakers can be surface mounted, but here too a barrier should cover the power terminals. Doors should be used on the exterior frame to minimize exposure of the compartment to dust and other contaminants.

The two kinds of protection that can be provided directly by molded-case circuit breakers are short circuit and overload. Short-circuit protection is required on all outgoing power (ungrounded) conductors, and maximum allowable instantaneous-trip settings are established by Federal regulations. Overload protection is mandated in some States. Magnetic elements give instantaneous-trip protection, and thermal elements afford overload protection. Pickup settings for both are based on the smallest size of cable being protected.

Undervoltage protection of each outgoing circuit is almost always required, the only exception being if the protected equipment has its own. An undervoltage release (UVR) is invariably used and is an auxiliary solenoid that trips the operating mechanism of the breaker whenever its coil voltage drops below 40% to 60% of rated. Additional external protective relaying can be given to the outgoing circuit through the UVR. Note that even though undervoltage protection may be needed, each outgoing breaker in a mine power center should contain an UVR. Outgoing breakers are rarely provided with shunt-trip solenoids.

Terminal connectors are available for molded-case breakers for either single- or multiple-conductor entry. Copper conductors require the use of copper terminals, and aluminum conductors require aluminum-compatible terminals. If multiple-conductor terminals are employed, the terminal must be constructed so that each conductor can be tightened without removing another conductor. Figure 12.15 is a representative view of conductor connections to a molded-case breaker.

The selection of molded-case circuit breakers is based on the voltage, frequency, interrupting capacity, continuous-current rating, and trip settings. Even though this topic is presented in chapters 9 and 10, additional coverage of the practical considerations for interrupting-capacity and continuous-current selection is warranted here.

Molded-case circuit breakers often begin interrupting short-circuit currents during the first cycle after the fault, and so they must be selected on the basis of maximum first-cycle asymmetrical fault current. The breakers are usually rated on a symmetrical current basis, which eliminates applying dc offset multipliers when selecting the breaker. It is generally considered adequate to use calculated symmetrical short-circuit currents for load-center applications.

As an example, consider a 750-kVA power center with a secondary utilization voltage of 600 V. If the transformer impedance is 6%, the maximum symmetrical fault current, $I_{f\,(max)}$, can be calculated as

$$I_{FLs} = \frac{750}{\sqrt{3}\,(0.6)} = 722 \text{ A},$$

$$I_{f(max)} = \frac{722}{0.06} = 12{,}033 \text{ A}.$$

This of course assumes a worst case condition, since transformer impedance is the only limiting factor for all fault current. In this case, even a 100-A standard mine-duty breaker could interrupt the fault; the typical interrupting capacity is 14,000 A symmetrical at 600 V. However, if the power center contains a 1,000-kVA transformer with 6% impedance, fault current becomes

$$I_{FLs} = \frac{1{,}000}{\sqrt{3}\,(0.6)} = 962 \text{ A},$$

$$I_{f(max)} = \frac{962}{0.06} = 16{,}038 \text{ A}.$$

Under this situation, the standard 100-A molded-case breaker would have inadequate interrupting capacity. (A premium-duty 100-A unit would be safe; typical interrupting capacity is 18,000 A symmetrical at 600 V. See chapter 9 for other typical ratings.) The simple example given here shows just one reason for the minimum transformer impedance stated earlier in this chapter.

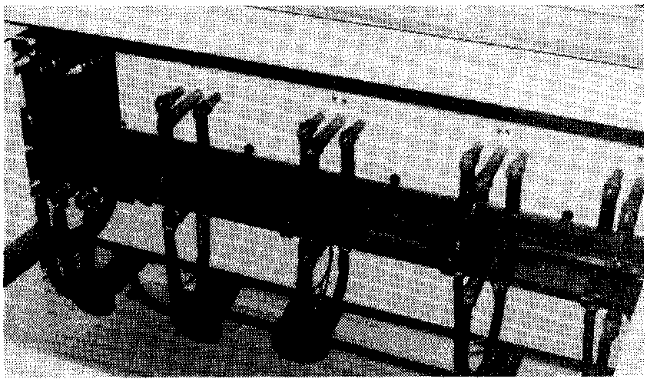

Figure 12.15.—Typical conductor connection to molded-case circuit breaker. (Courtesy PEMCO Corp.)

The continuous-current rating for a molded-case circuit breaker is actually the rating of the thermal-trip element, which can be less than the breaker frame size (a thermal-magnetic breaker). Magnetic-only devices have a continuous-current rating equal to the frame size. The rating is based upon 100% continuous current at 40°C, but sizing the breaker is normally related to 80% of the rating. This means that sizing is based upon 1.25 times the cable ampacity or the full-load circuit current. In other words, the breaker operates at 80% of the continuous-current rating at full-load current. (Note that some molded-case breakers are applied at 100% full-load current, particularly those with solid-state tripping elements.)

For instance, consider sizing a breaker to protect a 4/0 cable for overload. A typical ampacity for 4/0 three-conductor unshielded trailing cable is 287 A. Thus, the breaker continuous-current rating would be (1.25) (287) or 359 A. The next highest standard size, or a 400-A rating, would then be selected. However, when sizing outgoing breakers for a mine power center, the method of selection is not so simple. The process can approach something of an art based on prior experience, especially when underground coal mining equipment is involved. An actual situation is used here to illustrate some difficulties that can be encountered.

From a practical standpoint, 4/0 is the largest trailing-cable size that can be used in underground mining and is the most common conductor size for continuous miner applications. Assume that the continuous miner has the following motors;

- Cutting motors, two at 90 hp;
- Pump motor, 50 hp;
- Gathering-head motor, 70 hp; and
- Traction motors, two at 35 hp.

Assuming a 480-V system, an average efficiency of 90%, and an average power factor of 90%, the maximum theoretical full-load current for the machine (370 connected horsepower) would be from equation 8.3:

$$I_{FL} = \frac{(370)\,(0.746)}{\sqrt{3}\,(0.48)\,(0.9)\,(0.9)} = 410 \text{ A}.$$

This current exceeds the cable ampacity; thus, breaker sizing based on cable ampacity is not applicable. It should be realized that it would be a rare situation to have all motors draw rated current. Each motor or motor pair has its own noncontinuous duty cycle, which is a function of many variables. It is for this reason that an undersized trailing cable can be used. Yet the circuit breaker must be selected to handle the load demands of the continuous miner. Therefore, the sizing of the continuous-current rating for a breaker does not follow a set procedure. Most power-center manufacturers are well aware of the machinery available and can usually provide the optimum circuit-breaker size for specifications. (Again, see chapter 9 for more discussion.)

A main circuit breaker (fig. 12.2, No. 14) is often recommended if the number of outgoing circuits exceeds three. The breaker has the bus work as its primary protection zone, but also serves as a backup to the outgoing breakers. The rating can be based upon the full-load current of the transformer secondary or the ampacity of the bus work, whichever is lower. In general, the bus work is sized on the transformer full-load current. For example, the full-load current for the 600-V secondary of a 750-kVA power center is

$$I_{rated} = \frac{750}{\sqrt{3}\,(0.6)} = 722 \text{ A.}$$

Thus, the breaker continuous-current rating should be based on

$$I_{FL} = (1.25)\,(722) = 902 \text{ A.}$$

A 1,200-A frame with a 1,000-A continuous-current rating would be selected. The thermal elements would provide overload protection to the transformer and bus work; magnetic elements could be set to give short-circuit protection to the bus work. Some main breakers are provided only with magnetic elements. Obviously, the main breaker must be coordinated with the outgoing breakers and the high-voltage fuses on the transformer primary.

GROUND-FAULT PROTECTION

As mentioned earlier, a grounding resistor is placed between the transformer neutral and the power-center frame to limit ground-fault current to not more than 25 A. Ground-fault relaying must be used on each outgoing ac circuit to initiate circuit interruption during malfunctions. The common relay schemes for ac applications in load centers are zero-sequence, neutral (direct), and potential. Zero-sequence relaying is the most common ground-fault protection and is utilized on practically all outgoing ac circuits. As shown in figure 12.16, the three line conductors are passed through a window-type current transformer (CT), and burden for the CT secondary is a ground-trip relay. Relay operation was described in chapter 9.

The ground-trip relay must be set to pick up at 40% or less of the maximum ground-fault current. For this situation, a low-ratio CT might be expected to be used, such as 25:5 or 50:5 (ampere-turns ratio). However, better sensitivity has been realized by using a high-ratio CT, 350:5 or higher, with a voltage-sensitive relay (about 1.5-V pickup), instead of a current-sensitive relay for the tripping device. Some manufacturers employ a slight variation to this conventional scheme by rectifying the CT output and using a dc voltage-sensitive relay. Either way, these schemes work with a low-cost, low-burden CT. Since the CT is drastically underutilized, its secondary current cannot be predicted by knowing the turns-ratio of the CT. The pickup values must be determined by testing.

The normally open (*NO*) contacts of the relay usually parallel the UVR of the associated circuit breaker. When the relay pickup value is exceeded, its associated contacts close and short out the UVR coil. This technique has been adopted to eliminate nuisance tripping due to bounce and vibration, which hamper circuits that have contacts in series. One problem with paralleled contacts as shown in figure 12.16 is that ground-fault protection is lost if the relay is removed from its socket. To prevent this, UVR power can be supplied through a jumper in the ground-fault relay case (fig. 12.17). Without the relay, the circuit breaker cannot be closed, except in some small molded-case units, such as 50-A units.

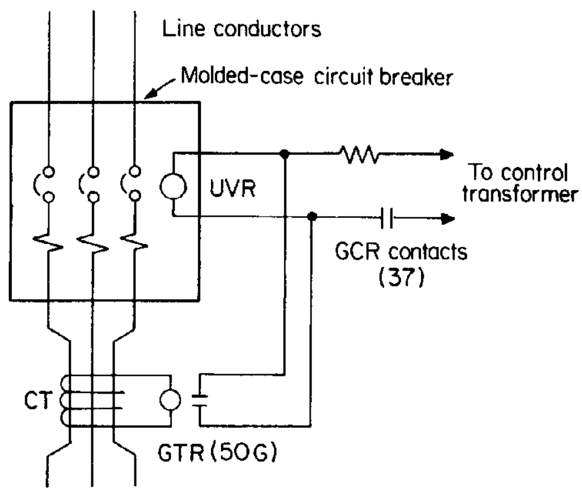

Figure 12.16.—Zero-sequence relaying on outgoing circuit with control connections to breaker.

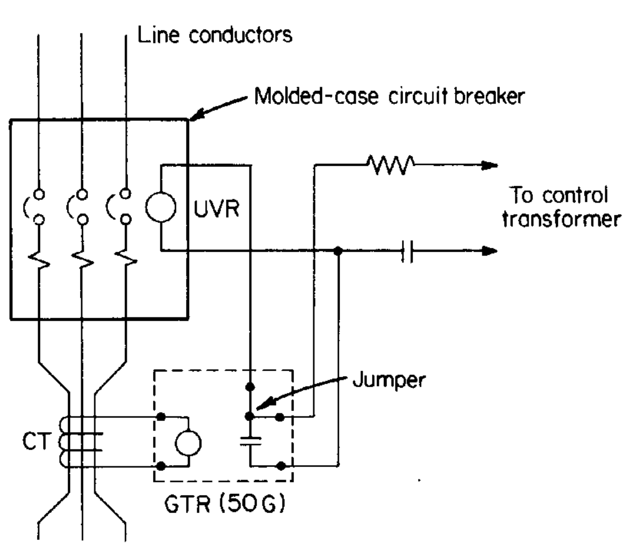

Figure 12.17.—Zero-sequence relaying with jumper in relay case.

Neutral relaying is sometimes used in mine power centers when a main breaker is used. The neutral conductor from the transformer is encircled by a CT, as shown in figure 12.18. The CT and the ground-trip relay are the same as discussed for zero-sequence relaying. However, a time delay should be introduced by the ground-trip relay so selective interruption of the faulted circuit occurs. If the circuit breaker of the faulted circuit fails to trip, the main circuit breaker will provide backup protection after the prescribed time delay. The time delay is on the order of 0.5 s and can be achieved by pneumatic or electronic means. The breaker trip device may be either a shunt trip or UVR, but the UVR is preferred.

Potential relaying can also be used as backup protection in conjunction with a main circuit breaker. Unlike zero-sequence and neutral relaying, potential relaying has the advantage of being able to detect a ground fault with the neutral grounding resistor open. Figure 12.19*A* shows a potential-relaying scheme using a PT for obtaining a voltage within the operating range of the relay whereas figure 12.19*B* shows a voltage divider to provide the same function.

For both schemes, the maximum pickup voltage of the relay should correspond to the voltage developed across the relay at 40% of the maximum fault current. When using a voltage divider circuit, care must be taken in sizing the resistors so the total equivalent resistance of the parallel combination is not reduced significantly. The combined value (R_1 + R_2) must be significantly higher than the value of the neutral grounding resistor. The impedance of the relay should also be significantly higher than the resistance of R_2 for the same reason. The power (I^2R) rating of the resistors should be capable of withstanding the currents expected through them under a maximum ground-fault condition.

Figure 12.19*C* illustrates a popular alternative to backup relaying. Here the relay coil is replaced by a red warning light located in a rugged or explosion-proof housing mounted on the outside of the power center. The light bulb voltage is matched to the PT to give maximum brilliance under maximum ground-fault conditions but noticeable light output at 40% of current limit. Many engineers prefer this technique as it provides an obvious indication of ground fault and is especially useful when primary ground-fault relaying is inoperative either through tampering or malfunction.

Test circuits for ensuring pickup at 40% of the maximum ground-fault current can be incorporated into the power center: figure 12.20 is an example. A control transformer with a secondary voltage of 12 V is used as a

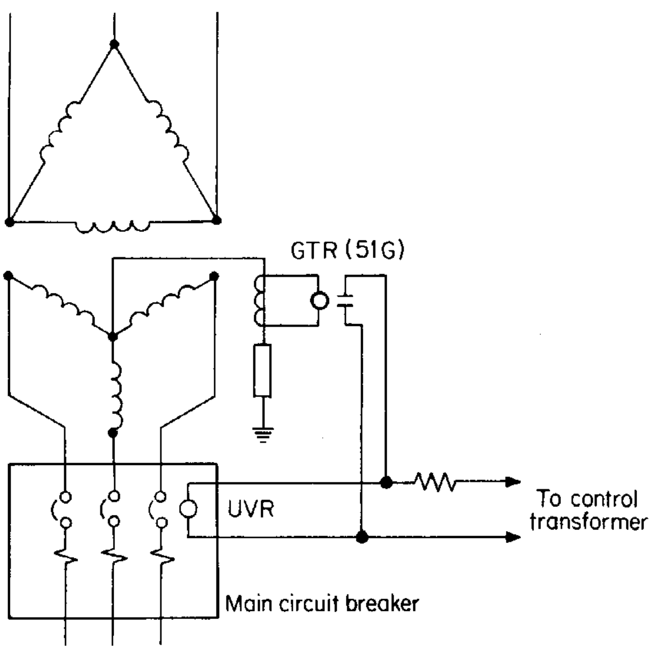

Figure 12.18.—Neutral relaying applied to grounding-resistor current as backup protection.

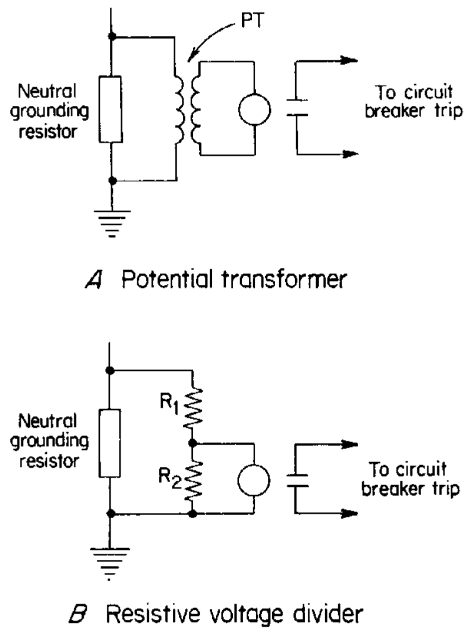

Figure 12.19.—Backup protection devices associated with mine power centers.

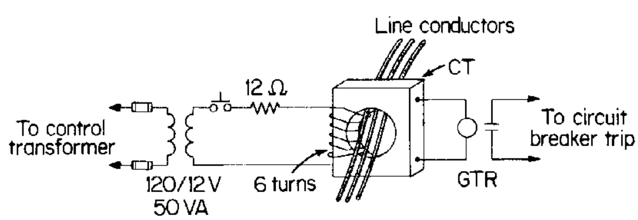

Figure 12.20.—Typical test circuit for zero-sequence relaying.

current source. The 12-Ω resistor is inserted in the circuit to limit the secondary current to 1.0 A. The six turns through the CT at 1.0 A produce the same effect as 6.0 A of zero-sequence current for the power conductors. The current corresponds to 40% of a 15-A system so that depressing the test button simulates a ground fault at 40% of the maximum current. The associated circuit breaker should then trip.

SINGLE-PHASE TRANSFORMERS

Single-phase transformers are used in power centers to supply 120 V to the control circuit and 240/120 V to convenience outlets. The control circuit consists of ground-monitoring systems, undervoltage releases, and ground-fault circuitry for each associated machine circuit, as well as relay connections for other protection devices. Convenience outlets can be used for portable power tools, area lighting, or external test instrumentation. Control transformer capacity normally ranges from 50 to 500 VA, whereas transformer capacity for 120/240-V outlets often falls in the range of 5 to 10 kVA. Larger capacities may be found, particularly in mine power centers with extensive control circuitry. A simple control circuit and a convenience-outlet circuit are shown in figures 12.21 and 12.22, respectively; in both circuits, fuses are used to protect the transformer primaries. Circuit breakers can be employed to protect convenience-transformer primaries, but fuses are recommended for control-power circuits because breakers can be tripped by an unwary miner, thus deactivating the load center.

Control-power fuses can be mounted in insulated dead-front holders or in a typical spring-clip arrangement that is only accessible to authorized personnel through a bolted cover. These two mounting types are illustrated in figure 12.23. The testing and changing of control-circuit fuses are among the more frequent electrical maintenance procedures for all types of equipment. Typical spring-clip fuse holders have uninsulated exposed metal clips, and rushed repair persons frequently remove the fuses from these holders for testing without deenergizing the circuits, thus placing their hands within close proximity of the energized clips. A slight inadvertent movement of the hand can result in electric shock. With dead-front fuse mounting, however, all energized components are enclosed in an insulated housing so that the fuses can be removed and replaced without exposing the electrician to the metallic clips and fuse ends.

A variety of schemes exist for handling the control- and utility-circuit voltages. Each machine circuit can have its own individual control transformer, but it is more common for a single control transformer to supply all the control circuits. When all control circuits are supplied by a single source, the control-circuit transformer can be eliminated, with the control voltage supplied by the secondary of the convenience-outlet transformer. Many mine power engineers dislike this combination because a failure or misuse of a convenience-outlet circuit could result in blowing the transformer fusing. This in turn would cause the loss of control power and all power to the mining machinery would stop. For this reason, some engineers prefer not to incorporate any 120/240-V outlets in their load centers, even with separate transformers.

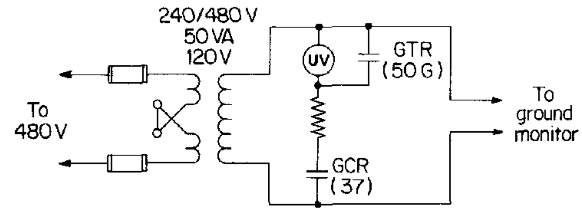

Figure 12.21.—Simple control circuit incorporating one ground-fault relay (GTR) and one ground-check relay (GCR).

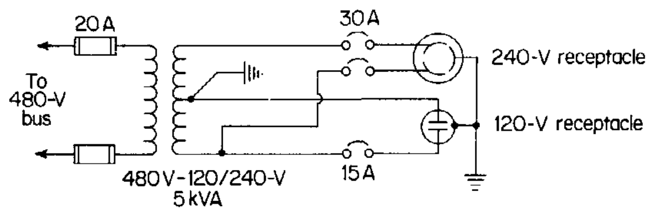

Figure 12.22.—Simple convenience-outlet circuit for 120- or 240-V single phase.

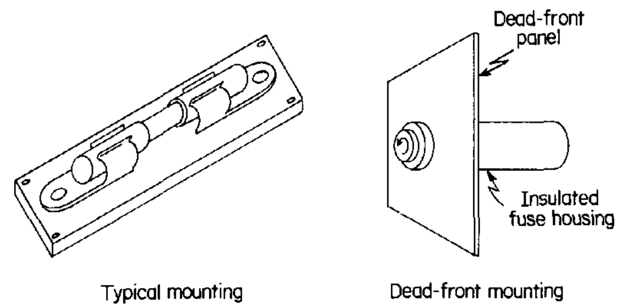

Figure 12.23.—Fuse mountings.

METERING CIRCUITS

The mine power center should include metering circuits to monitor line voltages and currents of all three phases (fig. 12.2, Nos. 15–16). The built-in instrumentation is an invaluable aid to maintenance, as it allows a firsthand look at the electrical operation of the load center and gives a composite view of the loads it serves. The metering is usually for distribution or utilization voltages and current, but rarely for both.

Figure 12.24 illustrates a common approach for metering voltage, where potential transformers (PT's) are used to reduce the line voltage to a value within the rating of the voltmeter potential coil. Two single-phase transformers are connected in open delta, and a four-position switch allows the three line-to-line voltages to be monitored in addition to the *off* position.

As shown in figure 12.25, two CT's are sufficient for metering the current in all three lines and isolating the meter circuit from the line. The CT ratio should be as low

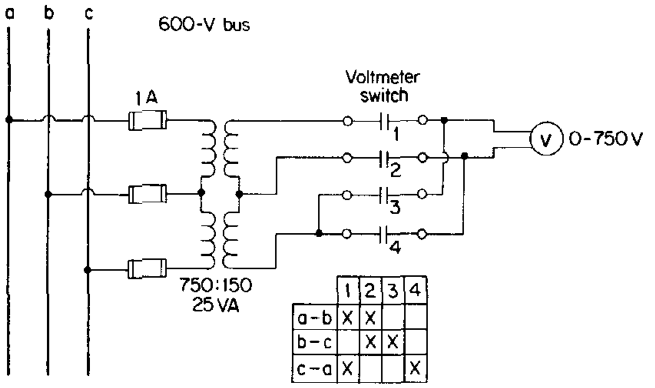

Figure 12.24.—Typical metering circuit for line-to-line voltages.

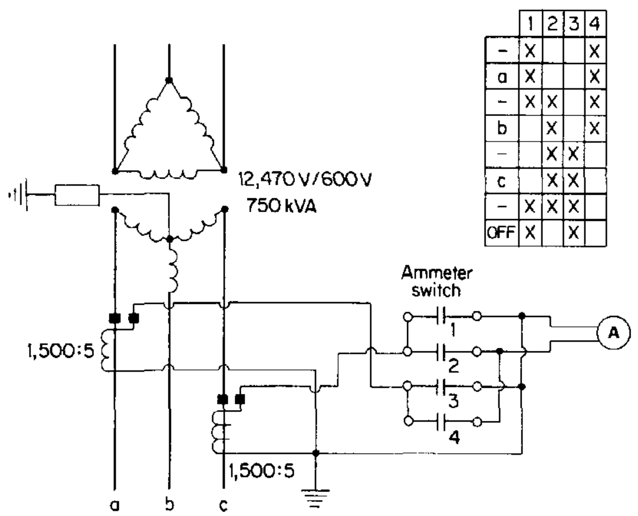

Figure 12.25.—Typical metering circuit for line currents.

as possible, without exceeding rated current in the secondary winding. A common recommendation is that the ratio be such that normal CT secondary current is 0.5 to 0.75 of the full-scale rating of the meter (5). The CT turns ratio can cause dangerously high voltage on the secondary if it is opened, so the ammeter switch must be designed so the secondary is not open circuited during the transition from one switch position to another. These are known as shorting switch contacts.

OUTGOING CABLE COUPLERS

The trailing cables for utilization equipment are almost always connected to the power center by means of low-voltage or medium-voltage cable couplers (fig. 12.2, No. 17). The couplers are rated at 600 V for low-voltage operation and at 1,000 V for medium voltages. Standard current ratings are 100, 225, 400, and 800 A, but larger sizes are readily available. Additional information can be found in the coupler section of chapter 8.

The outgoing cable couplers on the power center are female-contact receptacles, which connect with the male-contact plugs of the trailing cables. When disconnecting, the opening of contacts must follow a sequential order. The couplers serve as a visual disconnect for the low-voltage and medium-voltage circuits served by the mine power center, and for safety the couplers should always be locked out when not in use. Lockout is the process of deenergizing a circuit so that it cannot be energized without authority. Improper lockout or failure to lock out has been a leading cause of electrical accidents. For example, a repair person deenergizes a faulty circuit at the power center prior to performing repair work. While the repair person works on the faulty circuit, another worker who is unaware of the situation mistakes one cable for another, energizes the faulty circuit, and subjects the repair person to electric shock or electrocution.

Effective lockout can be provided by routine use of locking dust covers or keyed couplers. With keyed couplers, the receptacle of each outgoing circuit is matched to fit only one cable-mounted plug so that mistakes cannot be made. On the power center, the dust covers are hinged to the receptacle or connected by a chain. The hinged covers are preferred from a safety standpoint since they are less likely to become disengaged and lost. Locking dust covers on cable-mounted plugs offer even better protection, as once the plug is locked it is impossible to connect it to any receptacle. The covers also prevent damage and dirt accumulation when the plugs are not in use. Although chain-connected locking dust covers are the only type presently available for plugs, hinged lids would again offer improved safety.

Another lockout technique is to drill a hole in the spare pilot-circuit pin of the cable-mounted plug. A padlock can then be inserted in the hole by the electrician. This technique is extremely simple and inexpensive, but it provides excellent lockout protection by prohibiting plug-receptacle coupling.

The installation of two receptacles for each outgoing machine circuit can also provide lockout protection. One receptacle is used for powering the outgoing circuit while the other is used strictly for locking out the circuit when necessary. Actually, the lockout receptacle is not a true receptacle since its housing need not contain internal contacts. The lockout receptacle can also protect the plug from damage while the associated circuit is not in use.

GROUND-CHECK MONITORS

Low-voltage and medium-voltage resistance-grounded systems are required to have a fail-safe ground-check circuit to continuously monitor the continuity of the grounding conductor. The monitor must cause its associated circuit breaker to trip if the grounding conductor or a pilot wire is broken. An indicator lamp on the monitor should indicate a tripped condition. The monitors are usually enclosed in a dead-front package and mounted near the associated circuit breaker. As was seen in chapter 9, monitors in common use in mining can be divided into two general classifications: impedance types and continuity types.

Impedance monitors require the trailing cable to have a pilot conductor. The monitor is calibrated to the impedance of the loop formed by the pilot and grounding conductors, and the device then monitors the change of impedance from the initial calibration. If the impedance of the loop increases beyond a preset value, the monitor must trip its associated circuit breaker by opening a set of contacts in series with the undervoltage release.

The maximum allowable increase in impedance is dependent upon the maximum ground-fault current permitted by the system. For a 15-A neutral-grounding resistor, the monitor should cause tripping if the impedance increases by 2.7 Ω. This value is based upon the maximum allowable frame-to-ground potential of 40 V, as follows:

$$Z = \frac{V_{max}}{I_f} = \frac{40}{15} = 2.7\ \Omega.$$

From the above, it is apparent that a ground-check monitor does not ensure that the frame potential of a piece of equipment will not rise above 40 V, since the device monitors only the change in impedance of the pilot and grounding-conductor loop and not the actual impedance of the grounding conductor.

A schematic of a common impedance monitor is shown in figure 12.26. This particular monitor is powered from a 24- to 32-V source, but others use 120-V power. Out-of-phase induced currents during motor starting can result in cancelling out the monitoring current, which in turn can cause nuisance tripping. As a result, a polarity-reversal switch should be provided to change the phase relationship of the pilot current with respect to the induced current. Some manufacturers use impedance-matching transformers to amplify the change in impedance for easy detection. The monitor should also provide a test button. With the button depressed, the appropriately sized resistor is inserted into the pilot circuit, which should result in tripping the circuit breaker. A disadvantage of the impedance monitor is that it cannot detect a pilot-to-ground fault. This type of monitor is also susceptible to problems with parallel paths.

Continuity monitors do not monitor impedance change, but only the grounding-conductor continuity. However, they must be adequately immune to parallel paths. Continuity motors are audio units and do not require a pilot conductor for operation. Figure 12.27 contains a block diagram of a common unit. Most makes can also be wired for pilot operation (fig. 12.28), but only the operation of the pilotless configuration will be discussed.

The monitor generates an audio frequency, which is coupled to the grounding conductor by means of the transmitting coil. The pilot wire is eliminated by using the line conductors as a return path. Filters at the monitoring location and within the monitored machine are necessary for coupling and uncoupling the audio signal from the line conductors. If the grounding conductor is intact, the receiver coil picks up the audio signal. If the grounding conductor is open, the receiver coil will pick up no signal, and the monitor will cause a set of contacts to open the undervoltage circuit of its associated circuit breaker. Chapter 9 contains a discussion of a continuity monitor that operates in a somewhat different fashion; the signal is

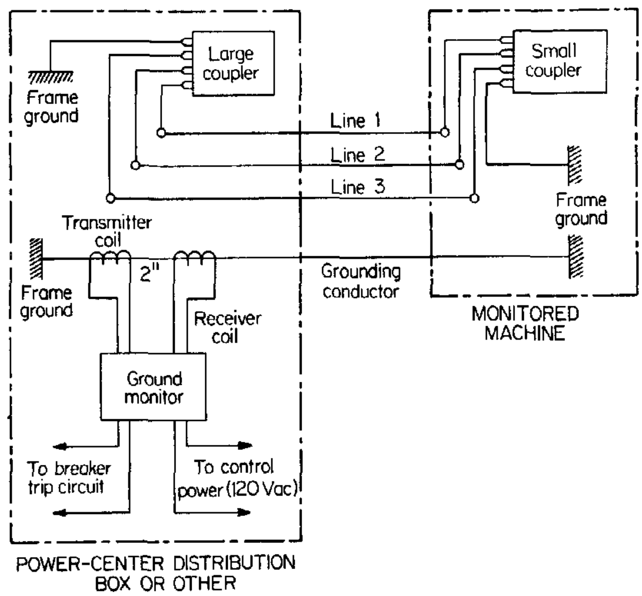

Figure 12.27.—Block diagram of continuity monitor connected in pilotless mode.

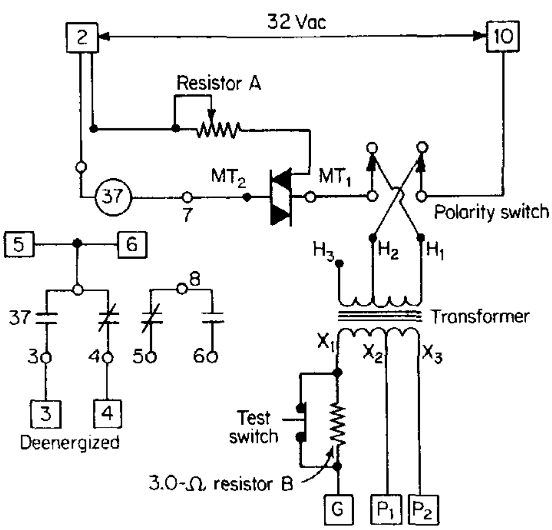

Figure 12.26.—Typical impedance monitor circuit.

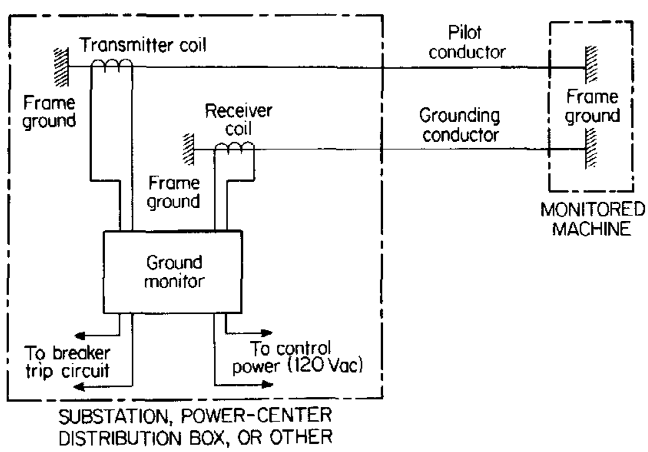

Figure 12.28.—Block diagram of continuity monitor wired for pilot operation.

impressed on and removed from the line conductors, with the grounding conductor used for the return path.

The major advantage of the continuity monitor is that it is immune to parallel paths and stray currents. It is also immune to pilot-to-ground faults if a pilot conductor is not used. However, the continuity monitor is expensive and complex when compared with an impedance monitor. A pilotless monitor must still be wired into the pilot and grounding contacts of the coupler in order to trip the circuit breakers when disconnecting the coupler. Another problem is that the grounding conductor must be isolated from the coupler shell, or the intended monitoring is bypassed. For metallic shells, a separate grounding conductor must be supplied through a spare coupler contact.

POWER-FACTOR CORRECTION

Some mining machines have notoriously poor power factors resulting from underutilization of induction motors. Perhaps the most outstanding example is the continuous miner, which can have a power factor that averages 0.6 lagging during the operational cycle. Whether it is this machine or others that create excessive reactive power, the result is poor power-system efficiency and utilization. If the power factor is poor at the purchase points (under 0.80, for example), the utility company will attach a penalty to the power bill.

There are many opinions on the best location at which to improve power factors. The most obvious place is at the utilization points. This strategy is common in surface mines, particularly with excavating machines. However, surface mining equipment has adequate space to incorporate the correction equipment, an advantage not common in underground mining machines. If the improvement is made at a convenient location immediately upstream from offending motors, the entire system up to and including the substation can benefit through lower apparent power, reduced line currents, and better voltage regulation. This location is commonly on the utilization buswork in mine power centers. On the other hand, if correction is not attempted here, power factors will be increased by the inherent distribution-system capacitance (for example, that associated with shielded cables) or somewhat cancelled by the diversity of operation that always exists between mining machinery and equipment sections. As a result, some engineers would rather correct the overall power factor at the distribution side of the substation.

Consequently, three general locations can be considered if power factors need correction: the machine, in the power centers, or at the substation. Each individual mine power system must be analyzed to find the solution, and the final decision must consider both electrical operation and economics. One consideration should be the cost of load-center correction versus correction at the substation versus the penalty added to the power bill. If a decision is made to add correction, the common approach is to use a bank of capacitors, also known as a static bank. (Synchronous rotating machinery is not logistically practical for most cases.)

Precautions must be taken when applying correction-capacitor banks, regardless of the location. They are a system load, and thus the bank must not be grounded on resistance-grounded systems. In the past, the common capacitor insulation was PCB (polychlorinated biphenyl), but because of the environmental hazards associated with this material, capacitors containing PCB must no longer be used. Owing to the dynamic operation of mining equipment, the correction should never be designed to obtain unity power factor on an average basis. If this were done, the resulting power factor could be leading (capacitive) for a significant portion of operation time, and the overall system operation could be worse than if no correction had been attempted. The last precaution concerns resonance. There is a chance that undesirable resonance could be created if a resistance-capacitance-inductance (RCL) combination is formed by adding capacitance from the static bank to the system portion. This problem has been observed particularly with power centers (belt transformers) supplying solid-state belt starters. It manifests itself as voltage and current frequencies other than 60 Hz.

Figure 12.29 shows the application of power-factor correction in a mine power center. The schematic is basically a reproduction of figure 12.2 with the addition of items 18 and 19, a static capacitor bank and its associated protection, respectively. Three capacitors connected in ungrounded wye are shown, but an integral three-phase capacitor with the same configuration can also be used. A molded-case circuit breaker affords short-circuit protection but also acts as a switch for removing the bank from the line. Sizing this breaker follows the same procedure related earlier for other applications and can be based on normal capacitor current. The current can easily be calculated from the per-phase capacity (kilovoltamperes reactive) and the system voltage. Some load centers are designed with more than one capacitor bank so that the magnitude of correction can be changed.

The sizing of the capacitor is also straightforward and follows the basic procedures presented in chapters 3 and 4. For instance, consider a mining section that has been found to be consuming 172 kW at an average power factor of 0.6 lagging. The average apparent and reactive powers are then

$$S = \frac{P}{pf} = \frac{172}{0.6} = 287 \text{ kVA},$$

$$Q_L = S[\sin(\cos^{-1} pf)] = 287\ (0.8) = 230 \text{ kvar}.$$

With a 90-kvar capacitor bank, the resultant reactive power and new power factor would be

$$Q_T = Q_L - Q_C = 230 - 90 = 140 \text{ kvar inductive},$$

$$\text{Tan } \theta_T = \frac{Q_T}{P} = \frac{140}{172}, \theta_T = 39°,$$

$$pf = \cos^{-1}39° = 0.78.$$

Thus, the average power factor has been improved from 0.6 to 0.78. As a general recommendation, power-factor correction in mine power centers should not try to correct an average power factor above 0.85.

Figure 12.29 also serves as a summary of the material presented on ac mine power centers. The internal components of an ac mine power center can be grouped into those associated with the high-voltage side, the transformer, and the utilization side.

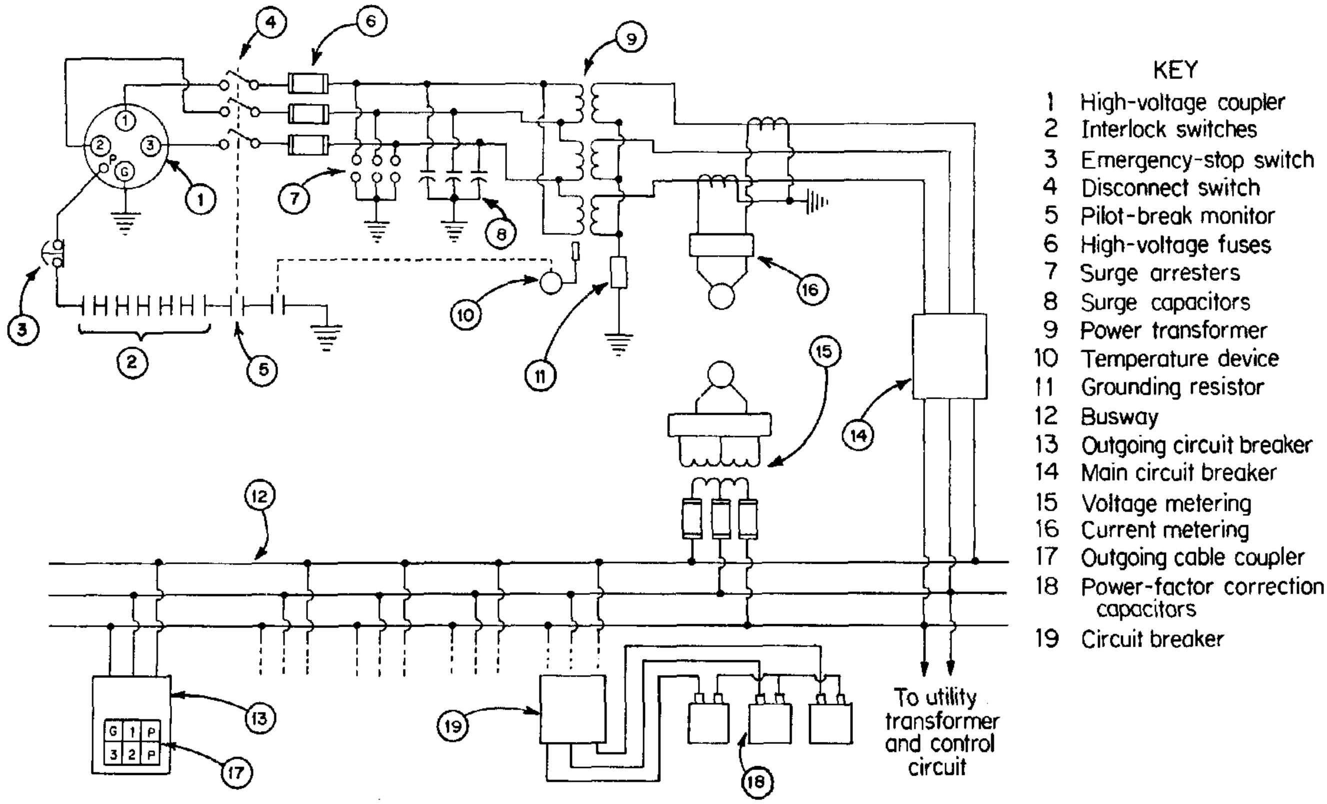

Figure 12.29.—Application of power-factor correction in mine power center.

DIRECT CURRENT UTILIZATION

Although ac utilization now dominates the mining industry, some segments of the mining process still remain better suited to the use of dc motors. The classic example is the use of dc series-wound motors for traction. The speed-torque characteristics of these motors are particularly well suited to this application, especially for locomotives and shuttle cars. This is the reason why dc utilization still plays a significant role in the mining process.

There are various ways of obtaining a dc voltage for powering the face equipment. If a mine uses rail haulage at a working section, the dc equipment can be powered directly from the trolley feeder line. However, if a working section does not have direct access to a trolley feeder line, a rectifier must be used to convert the three-phase ac voltage to dc voltage. The rectifier can be a separate piece of equipment housed in its own enclosure and powered by means of a feeder cable from the ac power center, or it can be incorporated into a single enclosure with the ac power center, with the total unit being referred to as an ac–dc combination power center. Some mining machinery manufacturers offer face equipment with on-board rectifiers so the benefits of dc motors can be obtained without the need for a section rectifier (see chapter 14).

Even if the rectifier is supplying a trolley, the basic internal components of dc power equipment remain essentially the same, and so only the dc section of a combination power center will be discussed. A general arrangement of the dc components is illustrated in figure 12.30. This diagram will be used as a reference to indicate the placement of individual components with respect to the

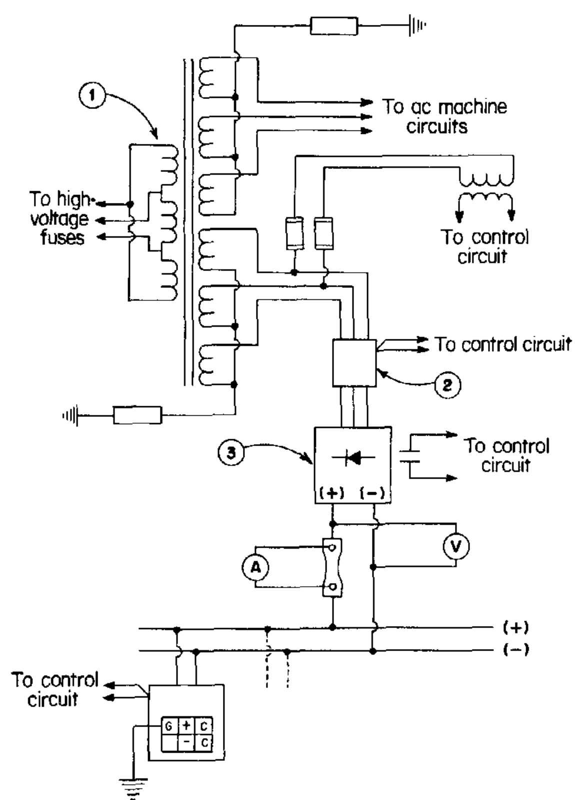

Figure 12.30.—General arrangement of dc components for combination power center.

overall system. The dc circuits that are frequently associated with trolley feeder lines will also be mentioned. Components and circuits that have been discussed in the preceding sections will be presented only when they are required for clarity.

RECTIFIER TRANSFORMER

A three-winding transformer (fig. 12.30, No. 1) is commonly used in a combination power center and consists of primary, secondary, and tertiary windings. Normally, the primary winding is delta connected with the secondary and tertiary windings wye connected, but the transformer may be designed such that the primary winding can be connected in either a delta or a wye configuration so it can be applied at two different distribution voltages: for example, 4,160-V delta, 7,200-V wye. For the latter situation, either the secondary or the tertiary winding must be delta connected. Thus, if a resistance-grounded system is to be used on the delta-connected winding, a zig-zag or grounding transformer must derive a neutral.

The full-wave bridge (fig. 12.31) is the most popular rectifier configuration used in combination power centers or section rectifiers. Here the relationship of the ac rms input voltage to the dc output voltage is

$$V_{ac} = 0.74\ V_{dc}.$$

The nominal 300-V system voltage is the one most commonly used in mining applications. Although some 600-V systems are still in operation, they are usually used only for trolley supplies. For a nominal system voltage of 300 V, the line-to-line ac voltage feeding the input to the rectifier would be

$$V_{ac} = 0.74\ (300) = 222\ V.$$

Hence, the voltage rating of the transformer winding feeding the rectifier would be 222 V. Some of the common ratings used in combination power center are listed in table 12.2.

Table 12.2.—Typical ratings for combination power centers.

dc, kW	ac, kVA	ac voltage, V	dc voltage, V	Input voltage, V
75	300	480Y/277	300	7,200
150	375	480Y/277	300	4,160
	400	480Y/277	300	7,200
200	600	600/220	300	7,620/13,200
300	500	480Y/277	300	7,200
	750	600/220	300	7,620/13,200
	1,000	600/220	300	7,200/4,160

A dc is much more difficult to interrupt than an ac, and the amount of available short-circuit current must be limited to less than that of ac equipment with similar capacity. As a result, the impedance of transformer windings feeding rectifier circuits is normally in the range of 0.06 to 0.08 pu. In order to obtain this high impedance, the leakage reactance of the associated winding is often deliberately increased during transformer fabrication. If this method does not provide the adequate impedance, a small additional impedance can be obtained by using air-core reactors at the output of the transformer, as shown in figure 12.32. However, in some circumstances, it is necessary to use a separate transformer (fig. 12.33) to limit the

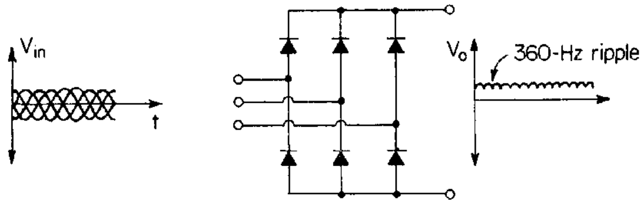

Figure 12.31.—Full-wave bridge rectifier.

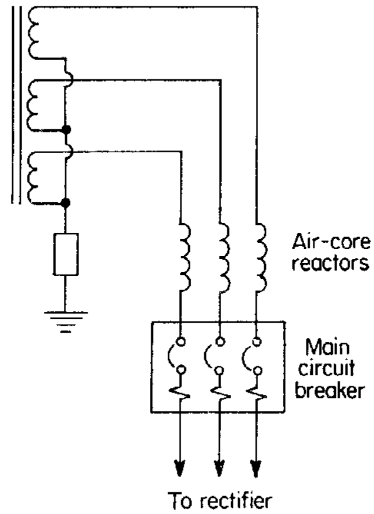

Figure 12.32.—Series reactance to reduce available short-circuit current.

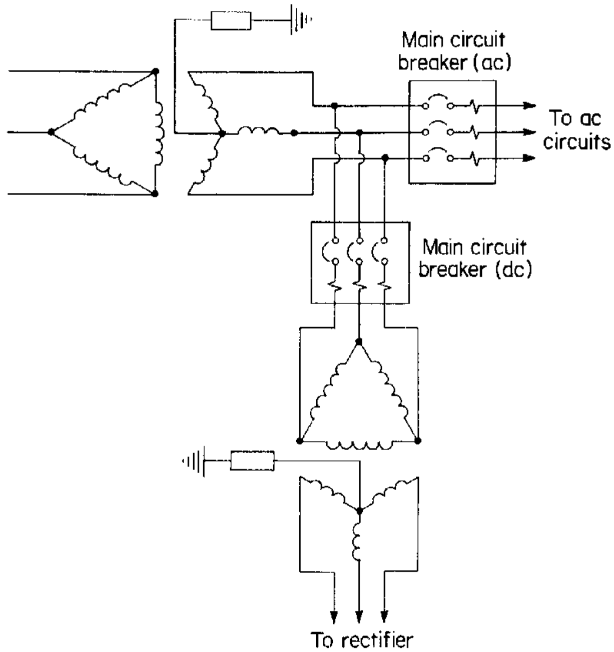

Figure 12.33.—Separate transformer to increase impedance of dc circuit.

maximum fault current to a level that can be safely interrupted and a magnitude that will not damage the rectifier bridge.

A main circuit breaker (fig. 12.30, No. 2) can be provided to protect the transformer winding that supplies the dc circuit. The thermal-trip setting of this breaker is usually based upon 125% of the rated current of the winding. As an example, a winding rated at 150 kVA would have a full-load current of

$$I_{FL} = \frac{150}{\sqrt{3}\,(0.222)} = 390\text{ A.}$$

The thermal-trip setting would be the next rated value above 1.25 (390), or 488 A. If the transformer impedance is 0.06 pu, the breaker should be capable of interrupting a fault current of

$$I_f = \frac{390}{0.06} = 6{,}500\text{ A.}$$

RECTIFIER

As seen in figure 12.30, the output of the transformer tertiary winding feeds the rectifier (No. 3) via the main circuit breaker. Again, the rectifier configuration is almost always a three-phase full-wave bridge. The power or kilowatt requirement of rectifiers used for mining is normally too high to permit the use of a single diode in each leg of the bridge. The current rating of the rectifier bridge, and thus the diodes, is selected on the expected bolted-fault current (faulted dc output) with the power-center input connected to an infinite bus. This current should also incorporate a multiplier to account for a worst case offset. The rectifier should be capable of handling this current for the time required for the circuit breakers to interrupt current flow. To achieve the adequate current-carrying capacity, diodes are paralleled in each rectifier leg to share the current.

Figure 12.34 shows a full-wave bridge rectifier with two diodes paralleled per leg (No. 1). The number of parallel diodes is determined by the individual diode current rating (all must be rated equally) along with the maximum available fault current. Each diode must also be rated in terms of a peak inverse voltage (PIV), which is the maximum voltage that can be applied across the diode in a reverse-biased mode without causing breakdown. For mine rectifier applications, the diode PIV should not be less than 2.5 times the nominal dc system voltage if proper transient suppression is used. Thus for a 300-V system, a rating of 800 V is commonly employed.

Matched diode characteristics should allow sufficient means for sharing current when used in a parallel configuration. Replacement through failure of any diode, however, would easily upset the balance. Therefore, although rectifiers are supplied with matched diodes, most manufacturers use current-balancing reactors (fig. 12.34, No. 2) to force uniform conduction of all diodes in parallel. (See chapter 4 for an explanation of operation.)

Each diode is accompanied by a circuit-protection fuse (fig. 12.34, No. 3). The purpose of the fuse is not to protect the diode but to prevent a catastrophic failure to a bridge leg (these diodes fail in a shorted mode). The fuse current rating must be such that it will not initiate interruption for a bolted fault at the dc bus until the clearing time for the downstream circuit-interrupting devices has elapsed (that is, the time-current characteristics of the two must be coordinated). Each fuse has a matching light (fig. 12.34, No. 4) to indicate when the fuse has blown and the diode needs to be replaced.

Suppression devices should always be used to protect the rectifier from transient overvoltages occurring from either the ac or dc side. Owing to their successful history in mining applications, selenium voltage suppressors (fig. 12.34, No. 5) are presently the most popular devices used in rectifiers. They are available with rms ratings ranging from 30 V to 480 V in 30-V increments. The clamping voltage at rated discharge is approximately 2.5 times the rms voltage rating. For instance, the input ac voltage to a 300-V rectifier would be 222-V rms; therefore, a selenium suppressor rating of 240-V rms should be selected. The resulting clamping voltage would be 600 V, which is significantly below the 800-V PVI rating determined earlier. Although not shown in figure 12.34, the suppressor often has each connection (five in all) protected by a fuse, again with a paralleled light to indicate failure. For rectifiers serving face equipment (150 or 300 kW), one selenium suppressor is used, but two or more may be paralleled for trolley rectifiers (450 kW or more).

An RC-snubber circuit, as shown in figure 12.35, is sometimes provided to reduce commutation transients.

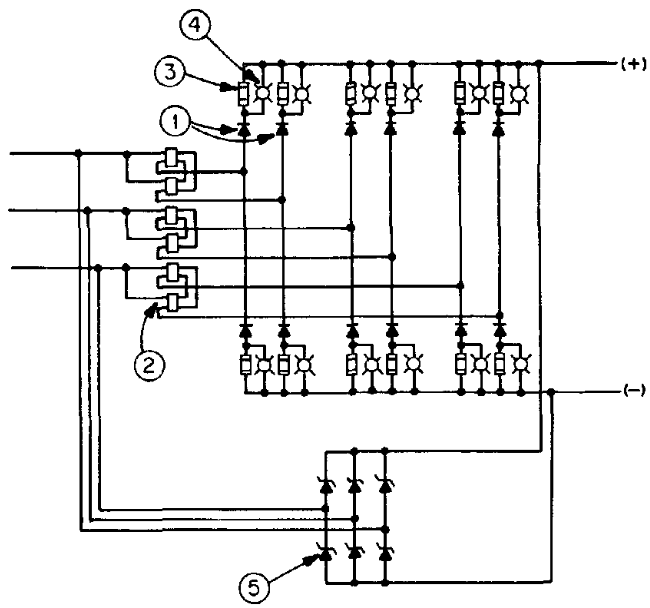

Figure 12.34.—Typical full-wave bridge rectifier with two diodes in parallel per leg.

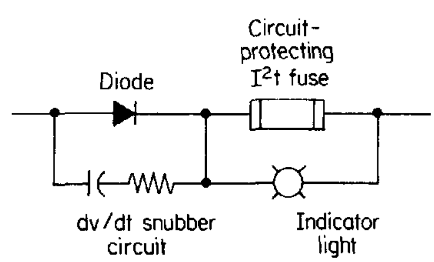

Figure 12.35—Diode with RC snubber protection.

The circuit serves to reduce the voltage rate-of-rise (dv/dt) that might be developed across the diode. Although snubber circuits are common in mine rectifiers, some manufacturers have achieved reliable diode operation without their use. The application of this protection technique is discussed in chapter 14 as dv/dt protection is required on thyristors.

DIRECT CURRENT GROUND-FAULT PROTECTION SCHEMES

Five basic systems of dc ground-fault protection are presently being used in the United States:

1. Diode grounding,
2. Basic grounding conductors,
3. Relayed grounding conductors,
4. Neutral shift, and
5. Differential current.

These are illustrated respectively in figures 12.36 through 12.40. The technique used dictates the complexity of the circuitry that will be associated with each outlet from the rectifier section. Because a complete discussion is presented in chapter 8, the systems will only be summaried here. The first three systems are more commonly used when a trolley feeder is the dc source. The neutral-shift and differential-current systems can only be used in conjunction with rectifiers serving face equipment. It should be noted that rectifiers for trolley systems must have both positive and negative conductors isolated above frame ground; thus, the subject of grounding is not applicable in that instance.

In the diode-grounded system (fig. 12.36) the machine frame is tied to the grounded conductor through a grounding diode. The basic grounding-conductor system (fig. 12.37) uses a separate conductor to provide grounding to the machine frame. In essence, adequate grounding-fault protection is not available from either system; for example, ground faults in the trailing cable rely on an interrupting device for sensing and clearing. No additional circuitry, other than the grounded power conductor, is used in the power center.

The relayed grounding-conductor system (fig. 12.38) can be sensitive to ground faults. A ground-fault relay coil is placed in series with the grounding conductor, and when a current threshold is reached, contact pickup can be used to trip a breaker UVR. However, parallel paths existing through the earth can cause fault current to partially bypass the relay coil, making for unreliable protection.

The neutral-shift system is shown in figure 12.39, where the resistors R_1 and R_2 create a dc neutral point. Ground faults on either the positive or negative conductor will cause the neutral to shift. The voltage-sensing relays, M_1 and M_2, detect the change. The result is sensitive ground-fault relaying, but the system is not selective with more than one outgoing circuit.

For one type of differential-current system (fig. 12.40), a grounding resistor is placed between the transformer

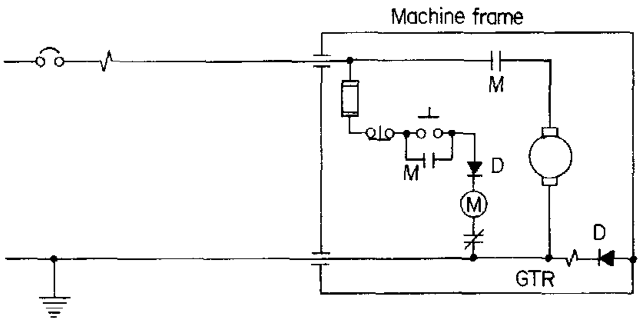

Figure 12.36.—Diode-grounded system.

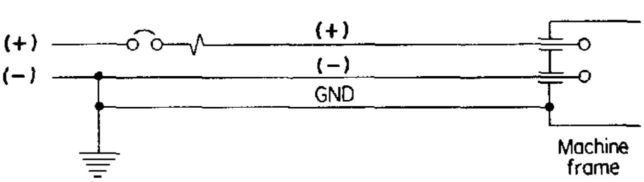

Figure 12.37.—Basic grounding-conductor system.

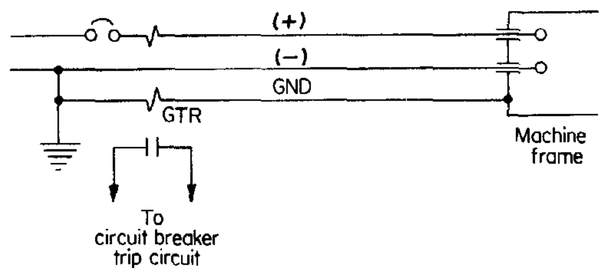

Figure 12.38.—Relayed grounding-conductor system.

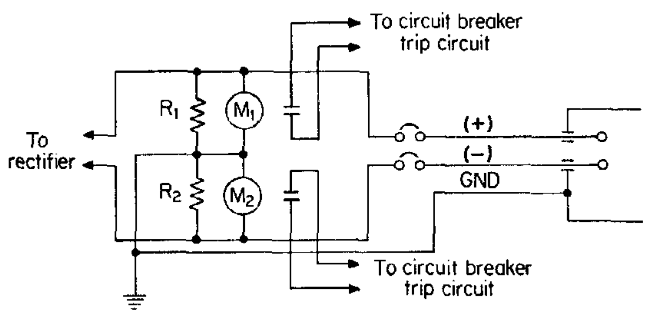

Figure 12.39.—Neutral-shift system.

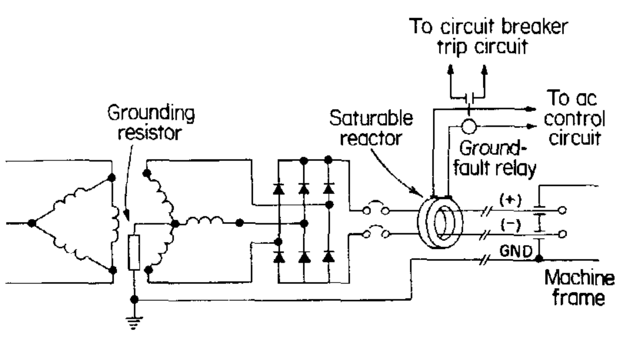

Figure 12.40.—Differential current scheme.

neutral and load-center frame and both positive and negative conductors pass through a saturable reactor. The differential current created by a ground fault causes magnetic saturation of the reactor core, which in turn allows the ground-fault relay to pick up. The system is sensitive and selective and by using the technique for each outgoing dc circuit, practical ground-fault protection is available. As a result, this system is considered in more detail in the following discussion.

DIRECT CURRENT CONTROL CIRCUITRY

A control circuit using a differential-current technique for ground-fault relaying is illustrated in figure 12.41. A main control transformer supplies 120 V to the entire circuit. Heat-sensing devices are mounted to the heat sink for each leg of the bridge rectifier. The overtemperature relay (OTR) coil is connected in series with the heat-sensing devices and is energized by a filtered single-phase bridge rectifier. The relay contacts are in series with the UVR of the main breaker so when a rectifier overtemperature condition occurs, a sensing device opens, the relay resets, and the UVR trips the breaker.

This version of differential-current relaying is supplied from the control power through a stepdown transformer with a 6-V secondary. With a ground-fault current from 4 to 6 A, voltage is impressed cross the ground-trip relay (GTR) in sufficient magnitude to cause contact pickup. This deenergizes the UVR on the machine circuit breaker, which in turn interrupts the offending circuit.

DIRECT CURRENT INTERRUPTING DEVICES

The problem mentioned earlier concerning interruption of dc requires the interrupting device to force the current to zero with an arc voltage greater than the line voltage. The device must be capable of withstanding the energy dissipated during the time that the arc exists across its contacts. This energy is a function of the circuit inductance as well as the current magnitude (*3, 14*). There are two basic interrupters used to protect dc machine circuits: the dc contactor and the molded-case circuit breaker. For trolley systems, power circuit breakers are employed.

A cross-sectional view of a dc contactor is shown in figure 12.42. This device consists primarily of stationary and movable contacts, which make and break upon energizing and deenergizing the operating coil. When the operating coil is energized, the armature assembly is attracted to the pole piece, which completes the magnetic circuit and causes the contacts to close. The contactor is designed so that the contacts close with a self-cleaning wiping action to ensure a good electrical contact (*6*).

Arc termination is similar to that of an air-magnetic circuit breaker (see chapter 9). A blowout coil is located between the front power terminal, Z, and the stationary contact. The purpose of the coil is to aid in extinguishing the arc and to minimize burning of the contact tips. To accomplish this, the coil produces a magnetic field perpendicular to the arc, which causes the arc to be lengthened and ruptured by deflection. Since the blowout coil is connected in series with the circuit to be interrupted, the strength of the magnetic field is proportional to the current. A flexible braided shunt completes the connection

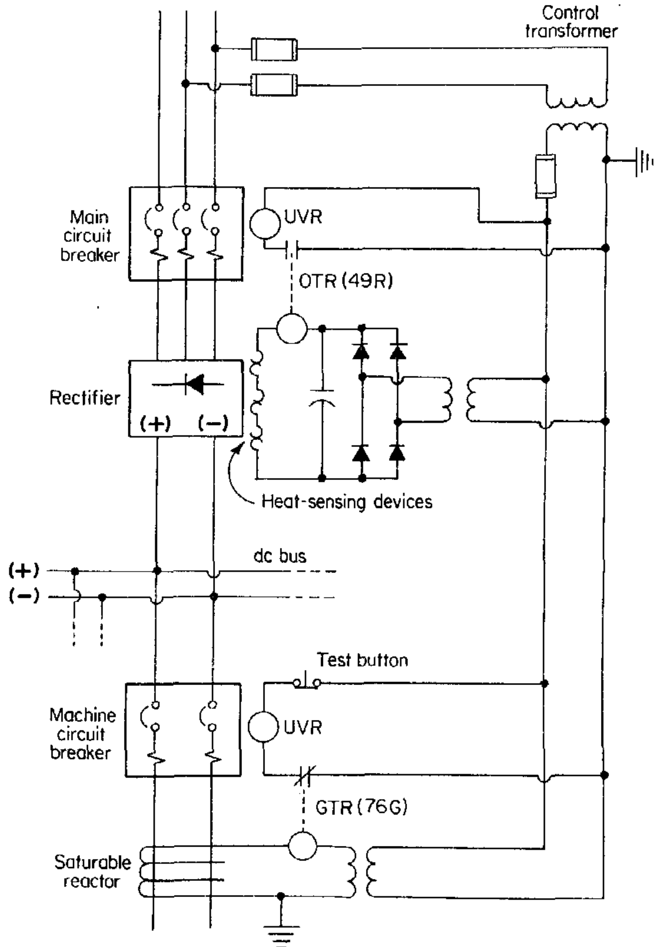

Figure 12.41.—Representative control circuit for rectifier.

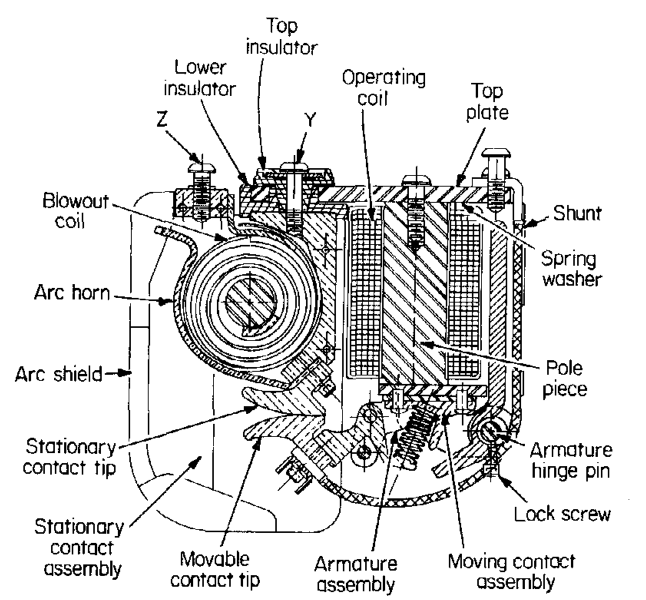

Figure 12.42.—Cross section of dc contactor. (Courtesy Joy Manufacturing Co.)

from the movable contact to the rear power terminal, Y. The arc horn protects the insulation of the blowout coil from being burnt by the arc, and an arc shield encloses the contacts to confine the arc and protect the adjacent parts (*6*).

The contactors most commonly used as interrupting devices for dc machine circuits have a continuous-current rating of 250 or 500 A. Their interrupting capability is approximately *10 times their continuous-current rating,* which is significantly less than that given for molded-case circuit breakers. One advantage of the dc contactor is its simple design and ease of maintenance. In some applications, molded-case circuit breakers are used as backup protection for dc contactors. It should be noted that molded-case circuit breakers are not rated for 600-V applications, but dc contactors can be applied at this voltage.

Molded-case circuit breakers that are rated at *600* Vac are also normally rated at 300 Vdc. The dc interrupting-duty rating for mine-duty breakers is either 10,000 or 20,000 A, depending on the product. However, as with the dc contactor, the actual interrupting capability varies with the inductance of the circuit being interrupted. Laboratory tests have indicated that for dc operation, the instantaneous dc tripping current is about 1.3 times the magnetic-element trip setting for ac (*14*).

Even though molded-case breakers have higher interrupting capacity, many engineers in mining operations have expressed concern over using them for dc applications. The engineers relate that in practice molded-case devices cannot adequately interrupt dc faults, whereas low-voltage power circuit breakers appear to work satisfactorily. On the other hand, manufacturers report that they have had few problems with field applications of molded-case breakers on dc.

In summary, the principal understanding that should be acquired from this power center presentation is that the equipment is assembled to match the mining operation and application. This is true whether the unit is a belt transformer, a trolley rectifier, or a combination ac–dc power center. Consequently, there cannot be a standard mine power center, but nevertheless, there are many practices recommended for mine power center construction that must be considered in order to achieve an efficient operating system.

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CHAPTER 13.—SWITCHHOUSES AND SUBSTATIONS[1]

Switchhouses, substations, and power centers comprise the major power equipment in mine power systems. Chapter 12 covered the design and construction of mine power centers, and this chapter will follow a similar format to present switchhouse and substation arrangements.

SWITCHHOUSES

Switchhouses are contained in metal-clad enclosures, similar in construction to those of power centers. This power equipment consists of visible disconnects and sectionalizing units, although the term *switchhouse* is normally equated with just the sectionalizing equipment. The principal function of the disconnect switch is to remove power manually from downstream mine power equipment that is connected to distribution. As in power centers, it must be possible to determine the position of the disconnect (load break) switch visually through a window in the enclosure wall. In underground coal mines, the power removal function must be located within 500 ft of the point where power enters the mine, and additional disconnects are incorporated into all power equipment that receives power from high-voltage distribution. This disconnect-switch equipment can be described simultaneously with switchhouses. The high-voltage side of the power center that was shown in figure 12.2 contained the same components.

The prime role of the switchhouse is to provide protective relaying in the distribution system and to allow branching of the radial system. The principal component is an automatic circuit breaker. The equipment name is modified depending upon the number of circuit breakers and associated protected outgoing circuits: for example, a *double switchhouse* contains two breakers and circuits. Because of size limitations, the units in underground mine power systems are rarely larger than double switchhouses, but surface mines often incorporate four-breaker switchhouses, or switching skids as they are commonly called. The schematic diagram for surface or underground applications is practically the same: the only basic difference is the repetition of internal components to correspond to the number of circuit breakers used.

SWITCHHOUSE INTERNAL COMPONENTS

A general arrangement for a single-breaker switchhouse is illustrated in figure 13.1. Incoming high-voltage power enters the switchhouse through the input receptacle to the load-break switch and the feedthrough receptacle. Some circuit-interrupting devices have disconnect switches incorporated in their construction, as shown in this diagram; others have a load-break switch located on the line side of the breaker. In either case, the switch must have an external operating handle with a mechanism for locking the switch in the open position. The output (load side) of the interrupting device feeds the branch receptacle. The current-sensing devices for the protective-relaying circuits are usually situated between the breaker and the branch receptacle, while the control transformer is located on the incoming side. The associated control circuitry for figure 13.1 is given in figure 13.2. The operation of this circuit will be discussed later.

As in mine power centers, interlock switches are positioned around the side covers and top covers of the switchhouse and are wired into the incoming pilot circuit to trip the upstream breaker in the event that a cover is removed. An emergency stop button is also provided.

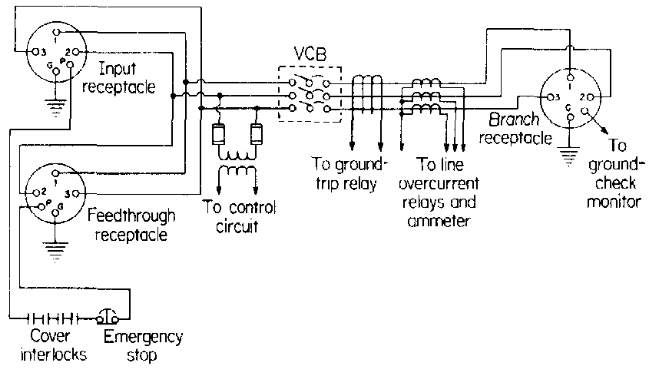

Figure 13.1.—Diagram for typical single switchhouse.

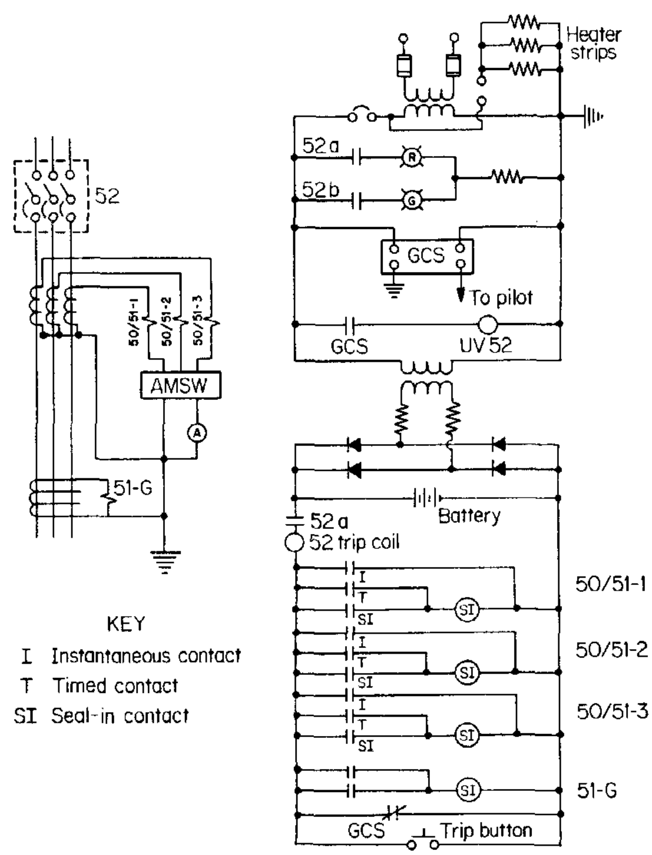

Figure 13.2.—Control circuitry for single switchhouse using battery tripping.

[1] The author wishes to thank Thomas Novak who prepared original material for many sections of this chapter.

A typical arrangement of a double switchhouse is presented in figure 13.3, and its control circuitry is shown in figure 13.4. The figures again show breakers with incorporated switches, and the repetition of circuitry mentioned earlier is obvious. In addition, the circuits in branch A work independently of branch B, and vice versa. Where the breakers do not have built-in switches, one load-break switch is usually located upstream from both breakers, and serves as a disconnect for all branch circuits, but not the feedthrough.

The incoming circuitry to the circuit breakers, shown in figures 13.1 and 13.2, is nearly identical to the incoming circuitry to the power center transformer discussed in chapter 12 and shown in figure 12.2. Surge arresters and switch prebreak monitors are not incorporated in the switchhhouse schematics, but they are often found in practice and are recommended. A typical schematic for the disconnect switch (the equipment, not the component) would be similar to figure 13.1 if the circuit breaker and

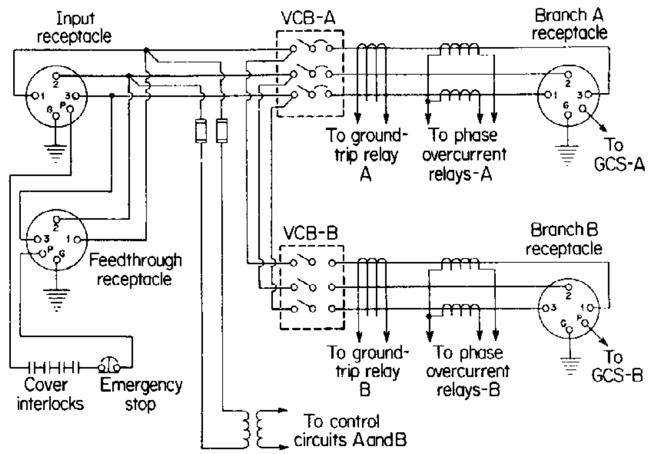

Figure 13.3.—Diagram for typical double switchhouse.

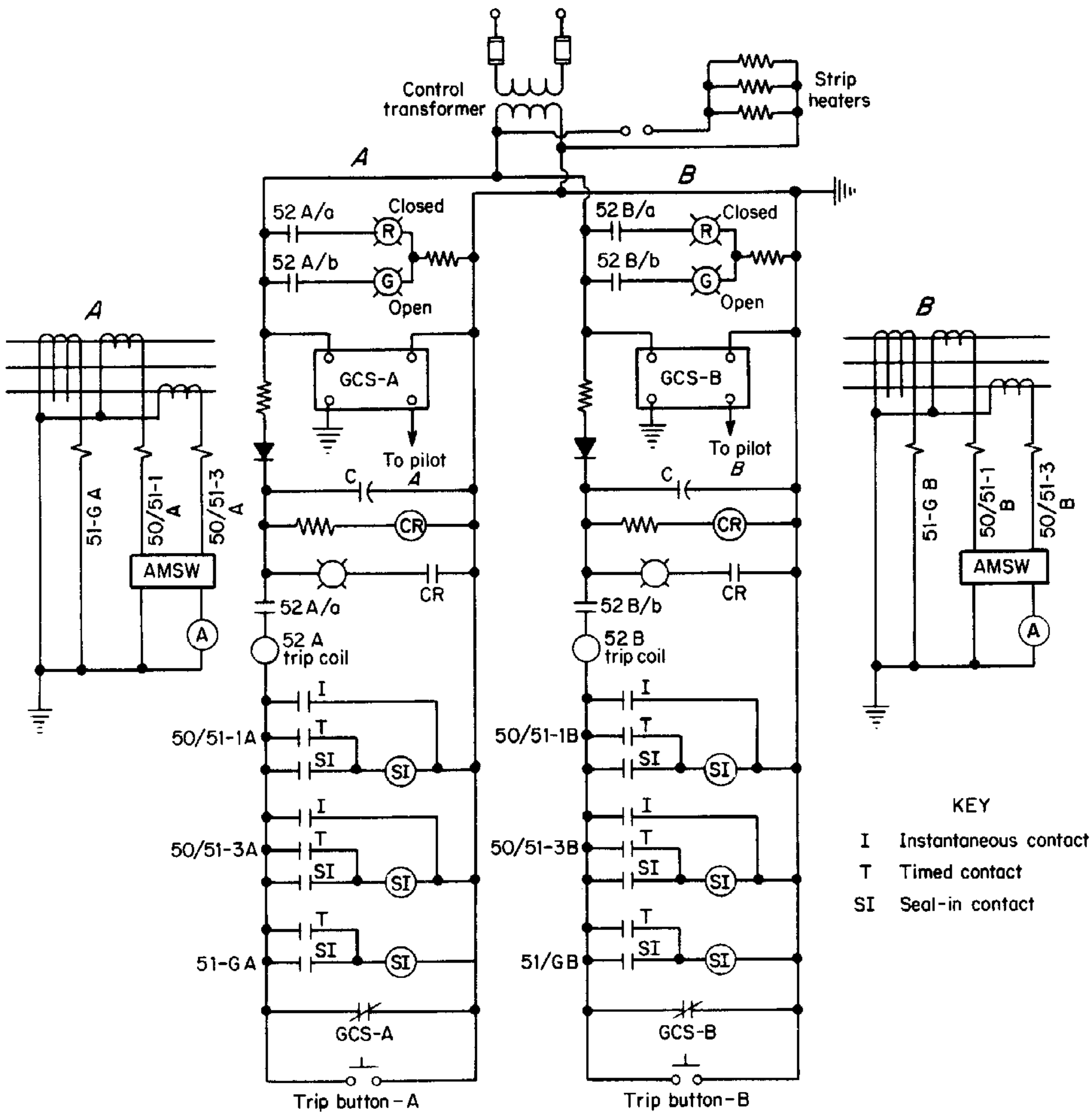

Figure 13.4.—Control circuitry for double switchhouse using capacitor tripping.

current transformers (CT's) were removed. This similarity is important in mine power-equipment design because when circuits serving identical functions have the same arrangement and construction, the maintenance crews can work on all equipment containing these circuits without surprise. This both simplifies mine training and increases the safety factor for maintenance personnel.

SWITCHHOUSE PROTECTIVE RELAYING

Induction-disk overcurrent relays are used for most protective-relaying applications in switchhouses. General relay operation and characteristics have already been discussed in chapter 9. For mining applications, inverse-time characteristics are usually employed for line-overcurrent relaying, while very-inverse-time relays are commonly used for ground-fault protection.

These relays have a range of adjustments so they can be applied to a variety of situations: the operating time can be controlled by the time dial setting; pickup current is adjusted by the main coil taps. Since the resulting time-current characteristics are the same for each tap, curves are plotted in terms of multiples of the pickup value. A typical family of curves for an inverse-time relay is given in figure 13.5 (*11*).[2] An instantaneous relay can be incorporated with the induction-disk (timed) relay, which will respond to the same actuating quantity. The actuating currents that are delivered to overcurrent relays are obtained by CT's. The CT's provide insulation from the high-voltage circuit and supply the relays with currents that are proportional to the currents in the line conductors, but sufficiently reduced. When applying a CT, critical consideration should be given to transformer burden and performance. Chapter 10 can be consulted for information on necessary calculations.

The pickup of an overcurrent relay should be selected so it will operate for all short circuits within its primary zone of protection and provide backup protection in adjoining zones. Figure 13.6 will be referred to as an example. Breaker 1 should provide primary protection for line 1. At the same time, its relays must be coordinated with the relays of breakers 2 and 3 to afford backup protection for lines 2 and 3. To ensure selectivity under all circumstances, the pickup value of any given relay should be somewhat higher than the pickup of downstream relays. The time delay of overcurrent relays must also be adjusted to provide selectivity with the relays of the immediately adjoining zones.

To provide proper selectivity, a bolted three-phase fault can be assumed as a basis for adjusting line-overcurrent relays, and a bolted line-to-ground fault would form the basis for the ground-relay setting. Selectivity under these conditions should ensure selectivity at lower currents.

For the fault shown in figure 13.6, the relay at breaker 2 must close its contacts, and breaker 2 must trip and interrupt the flow of short-circuit current before the relay at breaker 1 closes its contacts. The calculation must also account for the overtravel of the relay at breaker 1. The time delay of breaker 1 that is needed to provide selectivity with breaker 2 can be determined by

$$T_1 = T_2 + B_2 + O_1 + F, \tag{13.1}$$

[2] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

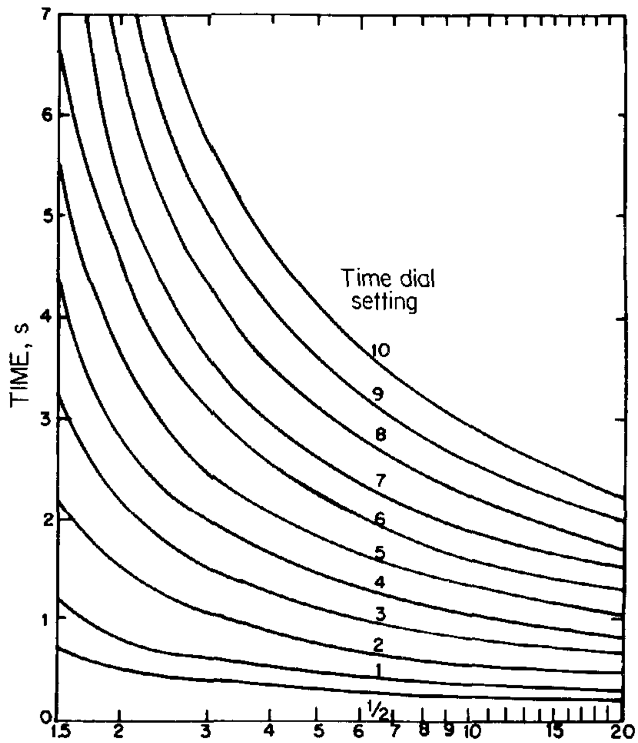

Figure 13.5.—Typical family of curves for inverse-time relay.

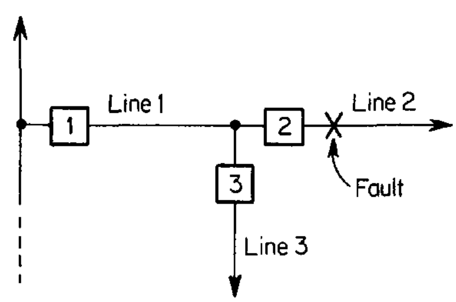

Figure 13.6.—Illustration of fault location for adjusting selectivity.

where T_1 = operating time of relay at breaker 1, s,
T_2 = operating time of relay at breaker 2, s,
B_2 = short-circuit interrupting time of breaker 2, s,
O_1 = overtravel time of relay at breaker 1, s,
and F = factor of safety, s.

As was shown in chapter 10, the sum of B_2, O_1, and F can normally be assumed to be 0.4 s. When coordinating the time delays of inverse-time relays, the process should start with the most downstream relays and work back toward the substation.

The typical connections for the line-overcurrent relays and CT's used in switchhouses are illustrated in figures 13.2 and 13.4. The inverse-time units are labeled 51, while the instantaneous units are labeled 50. Figure 13.2 shows three wye-connected CT's driving three wye-connected overcurrent relays, whereas figure 13.4 shows two open-delta-connected CT's driving two open-delta-connected overcurrent relays. Although protection for all three lines can be obtained with the open-delta connection, this approach is neither as accurate nor reliable as the wye connection.

The best ground-fault protection is obtained through zero-sequence relaying, which is shown in both figure 13.2 and figure 13.4. The CT ratio is typically 25:5 or 50:5. Unlike the power-center applications in chapter 12, the secondary current from the CT, which has an induction-disk relay as burden, is proportional to the ground-fault current as dictated by the turns ratio of the CT. However, because of the necessarily large CT windows, burden matching and actual testing are essential to obtain reliable sensitive relay pickup. Here the aim should be to obtain 50% of the current limit.

Residual ground-fault relaying can be found in some switchhouses, but adequate sensitivity is usually not available with this system because of errors caused by CT saturation and unmatched characteristics. Some success in eliminating this problem has been achieved with the use of solid-state relays, as discussed in chapter 14.

POWER CIRCUIT BREAKERS

Three types of power circuit breakers can be found in mine switchhouses: live-tank oil (or minimum oil), dead-tank oil (OCB), and vacuum (VCB). Live-tank oil circuit breakers have been plagued with maintenance problems, the most outstanding being inadequate oil levels after five or fewer interruptions. This requires extremely frequent inspections, which are impractical for most mining operations. Three-phase dead-tank OCB's in 250- to 500-MVA interrupting capacities are used exclusively by many mining companies with tremendous success. Despite this, their use has been substantially curtailed in recent years because of cost, availability problems, polychlorinated biphenyl (PCB) contamination, and their large size for thin coal seams. In fact, some States prohibit the use of OCB's in underground applications above 10 kV. VCB's have been criticized in terms of the transient overvoltages that their high efficiency can create. However, as demonstrated in chapter 11, these problems can usually be related to poor power-system design practices. When reasonable precautions are met, the VCB is perhaps the most effective high-voltage power circuit breaker compared with other types that have equal voltage and interrupting-capacity ratings. It is by far the most popular circuit breaker used in switchhouses.

The voltage ratings of power circuit breakers correspond to the insulation class used in the mine distribution system, but manufacturers offer VCB's rated at 15 kV, which are applicable for 4.16-kV through 14.4-kV nominal voltage systems. Continuous-current ratings of 400 and 600 A are the most common in switchhouse applications. Since high-voltage circuit breakers begin interruption a few cycles after the first-cycle fault current peak, the interrupting rating of these breakers is based on symmetrical fault current. (Many power breakers are also rated on an asymmetrical interrupting-current basis.) A 400-A VCB has a typical interrupting rating of 4,000-A symmetrical, with the 600-A model rated at 12,000 A. The breaker must also be able to withstand the physical stresses resulting from first-cycle fault currents. Thus, the breaker must have a close-and-latch rating, based on asymmetrical current, that is greater than 1.6 times the maximum symmetrical fault current.

Sometimes an automatic load-break switch interlocked with fuses (as discussed in chapter 12) replaces the power circuit breaker in the most downstream switchhouse in the distribution system. A small delay is built into the switch tripping circuit to remove the problem of the switch's trying to clear fault currents. The time delay is coordinated with the maximum period that a second fuse takes to clear a line-to-line or three-phase fault. Manual interlocking with the fuse elements is therefore not applicable. The only automatic load-break switch suitable for this approach is the type where the fuses have elements that activate a contact set upon separation and can be used to power the tripping mechanisms in conjunction with the control circuitry.

For additional information, note that chapter 9 contains extensive information about different power circuit breakers and their advantages and disadvantages, and chapter 10 covers the selection of continuous-current, interrupting, and close-and-latch ratings for these devices.

SWITCHHOUSE CONTROL CIRCUITS

Most control-circuit schemes found in switchhouses have a similar operation. Thus, for convenience of discussion, the typical control shown in figure 13.4 is reproduced in figure 13.7 and the control shown previously as figure 13.2 is repeated as figure 13.8. Figure 13.7 illustrates the control circuit of a double-breaker switchhouse. Each circuit breaker has its own independent control circuit. Control circuit B is an exact duplicate of control circuit A; therefore, only circuit A will be discussed.

The actuating currents for the line overcurrent relays (50/51-1A and 50/51-3A) are obtained by the open-delta-connected current transformers. The relays are also connected in an open-delta configuration through the ammeter and ammeter switch. The ground-fault relay 51-GA is used in a zero-sequence ground-trip circuit.

A single-phase control transformer steps down the distribution voltage to 120 V to supply both control circuits and the strip heaters. The strip heaters are resistive devices placed in the switchhouse enclosure to provide heat that minimizes moisture accumulation. The red lamp is connected in series with the normally open (*NO*) auxiliary contacts (52 A/a) of the circuit breaker, while the green lamp is in series with the normally closed (*NC*) auxiliary contacts (52 A/b). The auxiliary contacts are mechanically controlled by the opening and closing mechanism of the breaker. When the breaker is closed, 52 A/a also closes and causes the red lamp to give a visual indication that the breaker is closed. At the same time, 52 A/b opens, which causes the green (open) lamp to be extinguished. The reverse procedure follows when the circuit breaker is opened.

The ground-check system of branch A (GCS–A) also obtains its power from the control circuit as shown. The output of the GCS is connected to the pilot and grounding conductors of the outgoing cable of branch A to ensure continuity of the downstream grounding conductor.

A capacitor-trip device is used to supply power to the circuit breaker trip circuit in the event of a fault or overcurrent condition. A diode is used as a half-wave rectifier to charge the capacitor (C). The capacitor stores energy to provide reliable tripping of the breaker and minimize nuisance trips. The voltage-sensing relay (CR) operates when the capacitor is fully charged and causes the lamp to provide a visual indication of a full charge.

Another *NO* set of auxiliary contacts (52 A/a) is placed in series with the trip coil of the circuit breaker to disconnect the tripping circuit from the control voltage when the breaker is open. The contacts of the line-overcurrent relays and the ground-fault relays are connected in parallel as shown. If either the instantaneous or the inverse-time unit of any of these relays is actuated, its associated contacts close, which completes the path and allows the capacitor to discharge through the trip coil. When the trip coil becomes energized, it causes the trip mechanism of the breaker to release and open the breaker.

A push button and an *NC* set of ground-check contacts (GCS–A) are also parallel with the contacts of the overcurrent relays. The GCS–A contacts should remain open, provided that the control circuit is energized and the integrity of the associated pilot and grounding conductor is maintained. If the continuity of the loop circuit is lost, the GCS–A contacts will close, which results in tripping the breaker. Depressing the push button provides a convenient means of tripping the breaker, as well as a means of testing the operation of the capacitor trip device.

The control circuit of figure 13.8 is similar to the above control circuit, with some minor exceptions. Three CT's and three line-overcurrent relays are connected in a wye configuration to provide a more precise and reliable operation. A battery, which is charged by a full-wave bridge rectifier, is used as the energy storage device, rather than a capacitor. An undervoltage release (UVR) (UV 52) is also included, with an *NO* set of ground-check contacts in series with it.

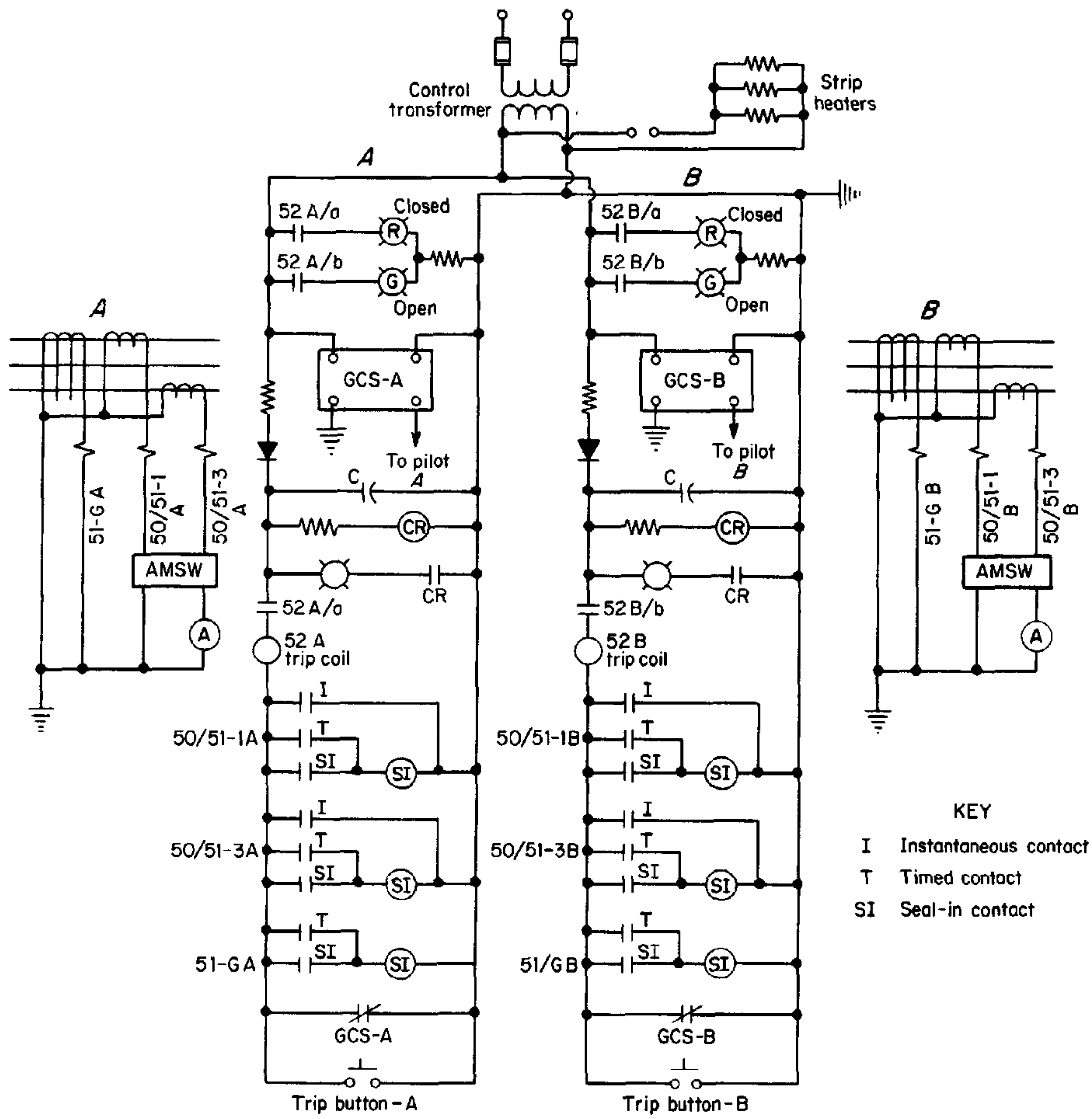

Figure 13.7.—Typical control circuit for double switchhouse using capacitor tripping.

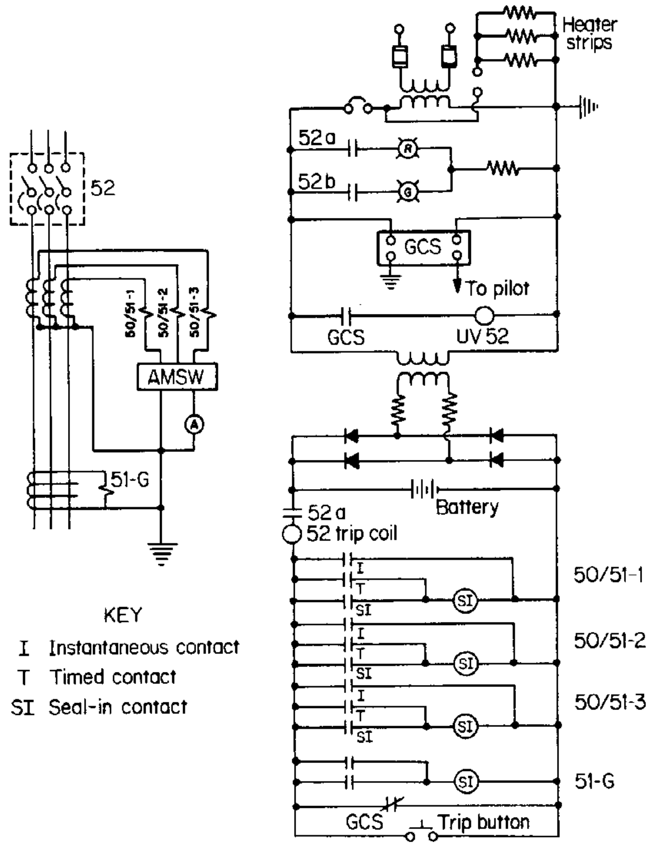

Figure 13.8.—Typical control circuit for single switchhouse using battery tripping.

SWITCHHOUSE DESIGN

At first glance the design of switchhouses is not nearly as complicated as that of mine power centers, but the simpler circuitry can be misleading. Incorrect sizing or adjustment of the internal components can have a disastrous effect on the mining operation. Perhaps the most serious concern is coordination, because it is in switchhouses that most adjustments must be made to obtain sensitive and selective protective relaying. For this reason alone, load-flow and fault analysis of the entire mine power system must be available. Comparison of these data allows the judicious selection of the required transformer ratios, relay time-current characteristics, pickup ranges, and device ratings. Some additional comments about the process are in order.

Because of the need for a uniform design, it is best for all switchhouses to have an identical assembly. In other words, all single switchhouses should have the same components, and one section of a multiple-breaker switchhouse would be basically equivalent to a single switchhouse. This may not be economically possible in some mines and not practical in others, yet there would be a specific advantage: switchhouses could be placed anywhere in the mine and then adjusted to suit that location. This would benefit any major movement of equipment within the complex.

To provide such uniformity, all power circuit breakers should be the same, with

1. The continuous-current rating sized to the maximum continuous current through any switchhouse in the mine (demand factors should be applied),
2. The interrupting-current rating selected to stop the maximum short-circuit current at any location in the distribution system (opposed to the preceding statement, the required rating here might be greater than symmetrical rms short-circuit current, depending upon the time to interruption), and
3. Close-and-latch rating sized to the maximum asymmetrical first-cycle rms fault current (convention is to use 1.6 times the interrupting-current rating).

Load-break switches need this same close-and-latch rating, rather than a multiple of their interrupting duty. The ampacity of all power conductors within the equipment would be sized and braced accordingly, but the size should never be less than 4/0.

For overload and short-circuit protection, multiratio CT's might be necessary, but experience has shown that no more than one or two types are required for the majority of applications: for example, 150/300/600:5 A, 300/600:5 A, 75/150/300:5 A, and 150/300:5 A are common ampere-turns ratios. With precise burden matching and accuracy calculations, common induction-disk relays can work reliably and selectively. Obviously the pickup range for the timed and instantaneous elements must correspond to the entire range needed for coordination. One range for each element might be inadequate for the mine's needs, but the relays are easily interchangeable, and one or two spare sets of relays can be maintained at the mine to meet all demands of line-overcurrent protection. Correct burden and accuracy must be verified for any relay-CT combination.

Adequate ground-fault protection, however, can be a difficult goal to attain. As stated earlier, burden and accuracy matching become critical. Yet even with precise matching, reliable pickup at a desirable ground-fault current threshold sometimes cannot be achieved. The large window required on the zero-sequence CT introduces induction problems that can cause inadequate secondary current at rated burden. The main result is that both the timed and instantaneous elements of induction-disk relays cannot be used simultaneously for ground-fault protection with a 50:5 or even a 25:5 CT ratio.

A solution to this ground-fault problem is to use only timed elements and apply the minimum time dial setting to the most downstream switchhouse. The radial distribution arrangement can then be established such that no more than five relays exist between the substation transformer secondary and the extreme end of distribution (see chapter 10 for discussion). This enables practical ground-fault coordination to be achieved from one tap-setting range and only one relay type. However, such problems as tolerances in off-shelf components can still remain. The best recourse is to apply testing that simulates a ground fault through the CT, then observe the response of the induction-disk relay. The optimum location for this testing is at a switchhouse in place within the mine power system. During initial equipment installation, the tests are not difficult, but subsequent maintenance checks may be impractical. As an alternative, it is possible to include a test circuit, such as shown in figure 12.20, in the switchhouse circuitry for each ground-fault relay. The resistor and wire turns about the CT would then be adjusted for the required relay pickup; for instance, 2 A and 6 turns would simulate a 12-A ground-fault current or 48% pickup on 25-A ground current limit.

A final point must be made about switchhouse construction. Reference has been made to surge capacitors in chapters 11 and 12: in the past, some manufacturers and operators incorporated these devices to increase the system's characteristic impedance in an attempt to limit transient overvoltages created by VCB chopping. However, the use of discrete capacitors of any kind on the load side of a vacuum interrupter, including power-factor correction types, can set the stage for prestrike and capacitance-switching transients. Hence, such devices should never be specified within a switchhouse. (A surge capacitor used directly across transformer and motor windings to limit the rate of voltage rise is a feasible means of protection if needed.) For similar reasons, switchhouse and power-center components must never be combined into a single enclosure, for instance, by using a VCB in place of the high-voltage fuses that were shown in figure 12.2. Conversely, surge arresters are desirable transient protection, and distribution-class types should be installed in all switchhouses.

SUBSTATIONS

It is common practice in the mining industry to purchase electrical power from a utility company whenever possible. A *substation* is used to receive the utility power before distribution to the mine and other facilities. This complex of equipment contains switching and protective relaying, establishes the grounding system, and may or may not involve transformation to a distribution voltage. The substation design should be based on personnel safety, reliability of operation, and ease of maintenance, and limit damage from fire, lightning, and equipment malfunction (*2*). The utility supply voltage for mines may range from 230 to 13.2 kV and less, depending upon utility company availability and the economics and location of the mine.

The substation components are often mounted on concrete pads. Power-conductor terminal structures, commonly made of galvanized steel but also of wood, support the conductors and cables that provide connections for transformers and circuit breakers. Insulators, usually made of glazed porcelain or Pyrex heat-resistant glass, insulate the overhead conductors from the supports. An overall view of a typical mine substation is given in figure 13.9. The main substation is usually a permanent installation, but portable or unit substations are becoming very popular for small load requirements.

BASIC SUBSTATION ARRANGEMENTS

The most popular distribution system used in mining is expanded radial A simplified sketch of this system is illustrated in figure 13.10, with the substation noted. Radial techniques have the lowest initial cost since a single transformer supplies all mine circuits and there is no duplication

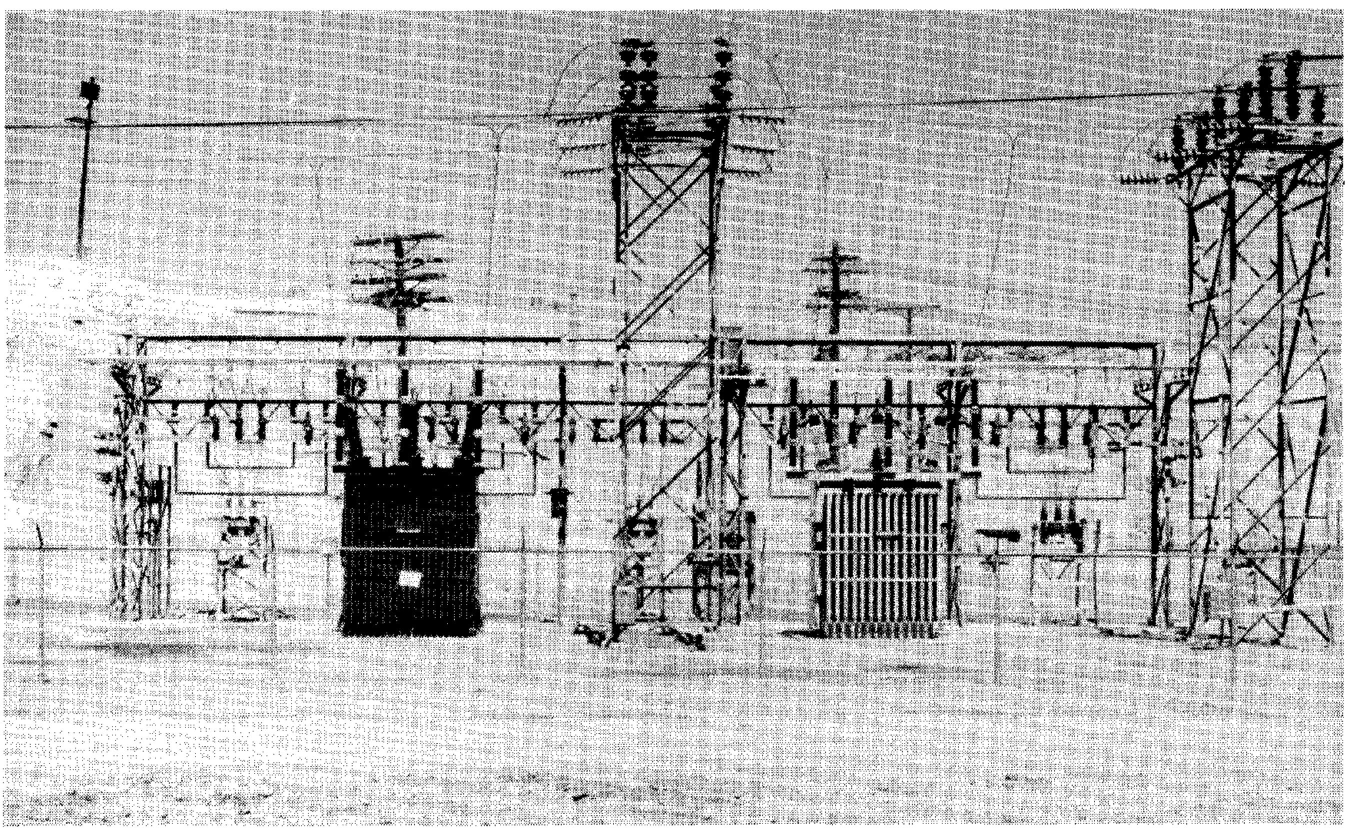

Figure 13.9.—Overall view of main substation serving mine.

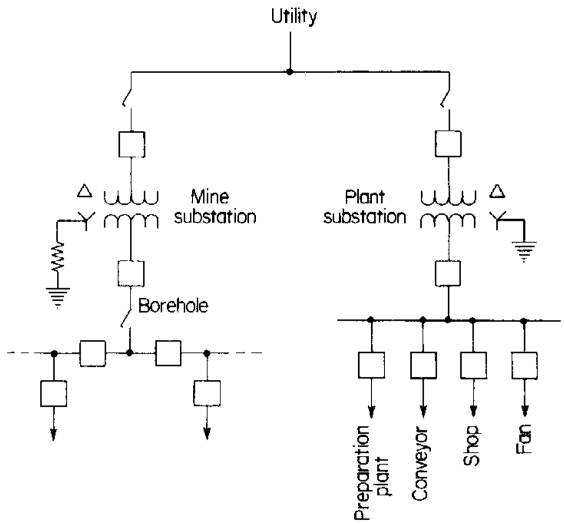

Figure 13.10.—Radial distribution applied to underground mine and its surface facilities.

of equipment. Operation and expansion is simple, but this simplicity leads to its major disadvantage: if a major component fails, the entire system can be effected. Nevertheless the logistics of mining, especially underground mining, often dictate the use of radial systems.

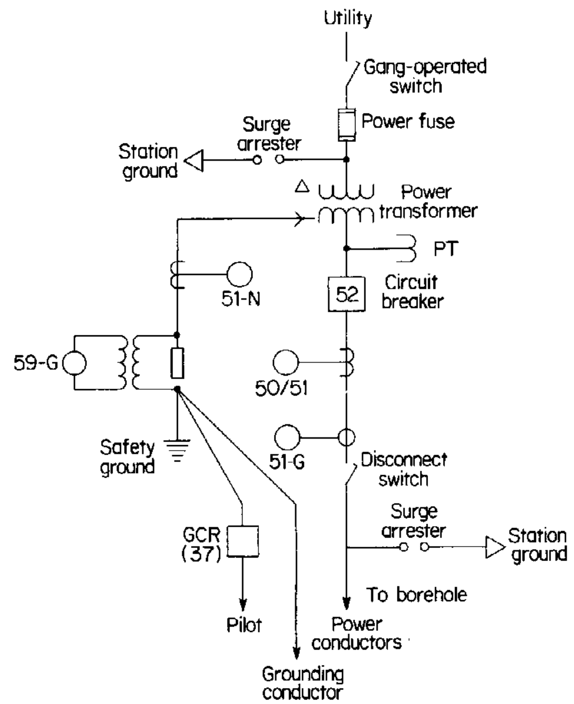

Figure 13.11.—One-line diagram for single-ended substation with fuse-protected transformer.

Single-Ended Substations

The substation design employed for radial systems is often termed *single-ended.* Figures 13.11 and 13.12 are one-line diagram examples, and the only difference between the two is found on the primary side of the transformer. The substation components, much like those contained in power centers, can be grouped into three general sections. The primary section provides connection with one or more incoming utility lines, switching and interrupting devices, and transformer protection. The transformer portion may include one or more transformers, a neutral point to establish grounding, and automatic tap changing, if used. The secondary section or distribution side provides for the connection of one or more secondary feeders, each having switching and interrupting devices.

Transformers (primary side) may be protected by high-voltage power fuses (fig. 13.11) or by a circuit breaker and associated overload and short-circuit relaying (fig. 13.12) or, less frequently, by both. Fuses or a circuit breaker is usually all that is necessary. Ground-fault protection could also be provided on the primary (not shown), but this relaying is restricted to circuit breaker applications and its inclusion depends to some extent on utility company requirements. The sizing of components is similar to that already discussed for power centers and switchhouses.

The outgoing-circuit protective relaying covers ground faults, overloads, and short circuits, and establishes a primary protection zone as far as the first downstream switchhouse. There are two recommended backup ground-fault relays: neutral-current sensing between the transformer bushing and the top of the grounding resistor, and potential relaying about the grounding resistor. The first ensures backup if the resistor shorts, while potential relaying provides protection against open-resistor hazards.

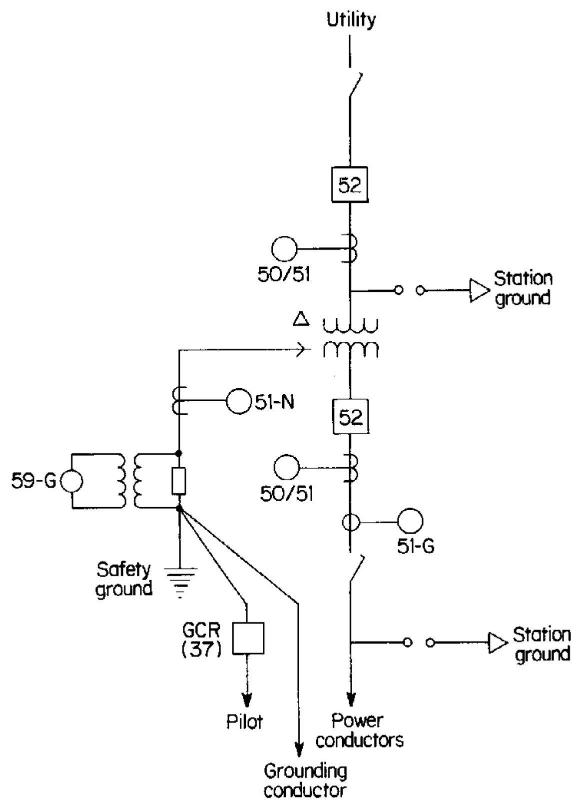

Figure 13.12.—One-line diagram for single-ended substation with circuit-breaker-protected transformer.

Both problems can occur in this surface installation because of exposure. The ground-fault protection must be coordinated with that in the downstream switchhouses. Line-overcurrent relays must coordinate with downstream protection as well as with the transformer primary protection and the utility. Circuit breaker, relay, and transformer sizing and selection generally follow the procedures given earlier, this time closely related to switchhouses.

The substation has two separate ground beds: the station (or system) and the safety (or mine). The grounding resistor is tied to the safety bed, and a grounding conductor extends into the mine. Ground-check monitoring of this conductor is required. The surge arresters, substation fencing, and other metallic parts are grounded to the station ground bed.

All substation components should be located within a fence posted with danger signs; the internal area, including the fence, is called the substation area. Specific concerns about substation components will be given in the following sections.

Double-Ended Substations

To enhance the reliability of the distribution circuit, some mines utilize a secondary-selective system. Two substation secondaries, as shown in figure 13.13, are connected with an *NO* tie circuit breaker. The combination is commonly but not always in the same substation area. Both substation halves are identical, and each is basically a single-ended unit but often with only one station and one safety ground bed. Under normal operation, each substation independently feeds 50% of the load, and the distribution system on either side is actually expanded radial. If a primary feeder or a transformer fails, the main secondary breaker on the inoperative circuit is opened, and the tie breaker is closed either manually or automatically. With automatic operation, relaying between the secondary breakers and the tie breaker must be interlocked so that the tie breaker will not close on a faulted distribution system. For instance, if the main secondary breaker is tripped by its associated relaying, the same relays should prevent tie breaker closure.

To permit continued operation with one transformer, consideration must be given to the following (*8*):

- Oversizing both transformers (instead of 50%, 80% of load) so one can temporarily carry the total load (at 125% of its capacity),
- Providing forced-air cooling to the transformer that remains in service during the emergency period, or
- Oversizing both transformers so that one can carry the total load.

These substations, also called *double-ended*, can be economical when the total substation capacity is above 5 MVA. Two separate incoming utility lines maximize the advantage of secondary-selective over radial systems. However, because of high transformer reliability, the economics of using a double-ended substation with one incoming line is considered questionable by some. Yet a definite advantage of this technique over single-ended types is that maintenance chores can be performed without power outage, such as insulator cleaning, component adjustment and replacement, and no-load tap changing.

SUBSTATION TRANSFORMERS

Transformers for permanent surface substations are almost always liquid (oil) immersed and built to IEEE standards (*5*). A typical substation transformer is shown in figure 13.14. Capacities commonly range from 5,000 to 30,000 kVA (sometimes less, but rarely more). In addition to the mine, the loads may include a preparation plant and general surface loads, such as pumps, ventilation fans, maintenance shops, and office and bathhouse facilities. When an underground mine is involved, it is recommended that a separate transformer be used for the underground power system.

The selection of transformer capacity must be based on estimation of electrical load. Up to 2,000 or 3,000 kVA, a general rule of 1 kVA/hp of connected load is perhaps satisfactory. This provides more than adequate capacity

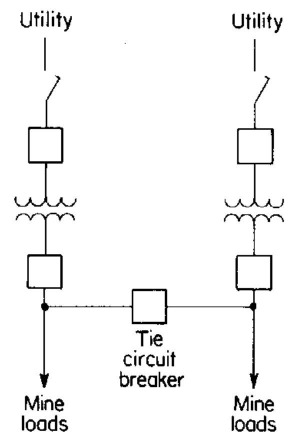

Figure 13.13.—Simplified one-line diagram for double-ended substation.

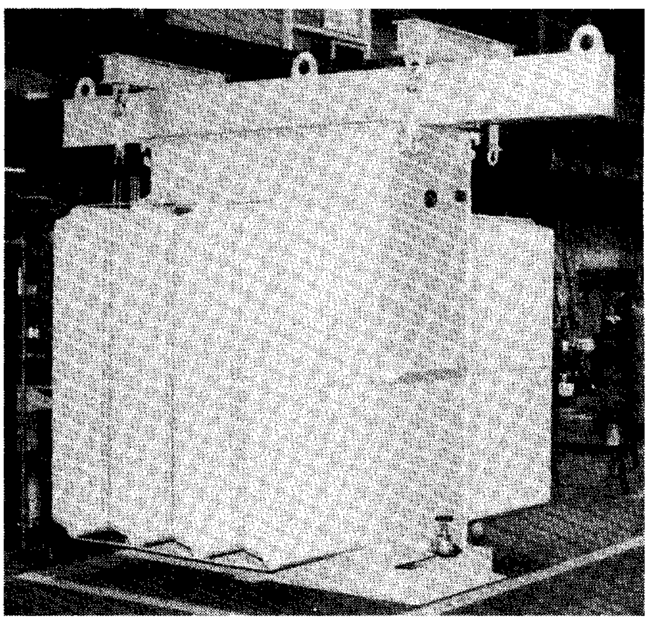

Figure 13.14.—Typical liquid-immersed transformer in substation.

for the load and will allow minimal load growth. Note that a worst case demand factor of 0.75 to 0.85 is built into the selection. Such oversizing will also allow for an adverse power factor.

When the requirements are greater than this, a more precise load estimate is required if optimum equipment costs are to be achieved. Considering the common capacity needs for the majority of mining operations, a load-flow analysis must be performed. The estimation may be made by using the demand factors discussed in chapters 4 and 8. In some cases, capacity might be determined on a per-connected-kilovoltampere basis (about 0.75 per connected horsepower) to allow for the growth in demand factor that always exists.

Table 13.1 shows recommended impedance values on a percentage basis for a liquid-immersed transformer (*5*). These values cover a capacity range from 500 to 30,000 kVA and are based on primary voltage, whereas mine power-center values are based on capacity. To provide a reasonable limit to short-circuit currents, the reactance should never be less than 5.0% (0.05 pu).

Past practice in the industry was to use three single-phase transformers for easy replacement of failed units, and a spare was often included to facilitate the changeover. However, improved manufacturing techniques have now resulted in transformers' being among the most reliable components; thus, the common practice today is to install three-phase integral units (*13*). The same advantages given for power centers hold for substations.

The standard ratings for substation liquid-immersed transformers are based on an allowable average winding-temperature rise of 65°C. A rating related to a 55°C rise is also available but is no longer listed as a standard (*5*). A transformer rated with the 65°C rise can be loaded to 112% of the 55°C-rise specification. The transformer capacity always involves a self-cooled [over-air (OA)] rating but may also have a forced-air-cooled rating (FA). Standard capacities for OA-rated and FA-rated substation transformers are listed below (*8*):

55°C OA:		
500	2,500	12,000
750	3,750	15,000
1,000	5,000	20,000
1,500	7,500	25,000
2,000	10,000	30,000
65°C OA:		
560	2,800	13,440
840	4,200	16,800
1,120	5,600	22,400
1,680	8,400	28,000
2,240	11,200	33,600
55°C FA:		
575	3,125	16,000
862	4,687	20,000
1,150	6,250	26,667
1,725	9,375	33,333
2,300	12,500	40,000
65°C FA:		
644	3,500	17,920
966	5,250	22,400
1,288	7,000	29,867
1,932	10,500	37,333
2,576	14,000	44,800

If the transformer is so equipped, the unit could also have a forced-oil-and-air (FOA) rating.

Table 13.1.—Standard impedance for liquid-immersed three-phase transformers, percent

Primary voltage, V	Secondary <1,000 V	Secondary 2,400 V and above
2,400 to 22,000	5.75	5.5
26,400 to 34,500	6.25	6.0
43,800	6.75	6.5
67,000	NAp	7.0
115,000	NAp	7.5
138,000	NAp	8.0

NAp Not applicable.

SUBSTATION SWITCHING APPARATUS

Dead-tank OCB's, such as those illustrated in figure 13.15, are the most common interrupters used in substations. In recent years, air-magnetic, vacuum, and sulfur hexafluoride (SF_6) circuit breakers have gained in popularity because of their reduced and simple maintenance, but for applications at utility voltages of 69 kV and higher, OCB's or SF_6 breakers are employed almost exclusively (*13*). SF_6 circuit breakers use a fluorocarbon gas for the arc-interruption medium, and their high cost usually restricts their use at 69 kV and above. On the other hand, the use of OCB's at lower substation voltages often results in lower costs and easier installation than the alternatives.

Because of the transformer overload capacity and the lack of overload capacity in circuit breakers, main breakers of the transformer secondary should have a continuous-current rating 25% greater than the anticipated top continuous-current rating of the transformer. Obviously, the voltage, interrupting-current, and close-and-latch ratings must be sufficient for the system to which the breaker is applied. These demands may be greater than for interrupters in switchhouses.

Reclosers

Oil circuit reclosers have proved to be extremely reliable as interrupters on the transformer secondary at distribution voltages to 15 kV (*13*). A circuit recloser is a circuit breaker

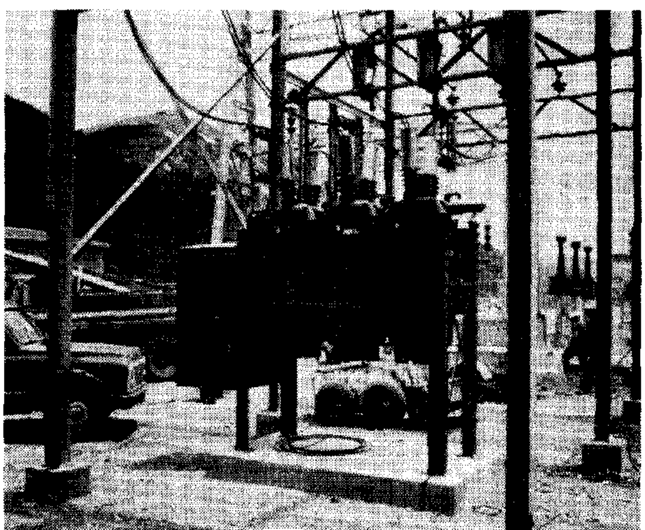

Figure 13.15.—Dead-tank OCB in substation.

with the necessary self-contained ability to detect line overcurrents, to time and interrupt the overcurrents, to reclose automatically, and to reenergize the line. If the line overcurrent is permanent, the recloser will lock open after a preset number of operations (usually three or four) and isolate the failure. Thus, the recloser can eliminate prolonged outages of the distribution system due to temporary faults or transient overvoltage conditions (*12*).

Reclosers can be hydraulically or electronically controlled (*12*). With the hydraulic control, an overcurrent is sensed by a trip coil in series with the line. When the minimum trip current of the recloser is exceeded, the trip coil actuates a plunger and causes the recloser contacts to open. The timing and sequencing are accomplished by the pumping of oil through separate hydraulic chambers. Electronically controlled reclosers provide a more easily adjusted, flexible, and accurate control then the hydraulically controlled recloser. The electronic control gives a convenient means for changing time-current characteristics, trip-current level, and the sequence of the recloser operation without deenergizing the recloser. Auxiliary tripping devices are available to allow additional protection, such as ground-fault and ground-check monitoring. Activation of the auxiliary tripping causes the recloser to lock open.

The selection and application of a recloser is basically the same as for an interrupter. The necessary items to consider are system voltage, maximum available fault current at the recloser location, maximum load current, maximum fault current in the zone protected by the recloser, coordination with other protective devices upstream and downstream, and ground-fault tripping or sensing. The common ratings are voltage, basic impulse insulation level (BIL), continuous current, minimum trip current, and interrupting current.

Disconnect Switches and Fuses

To provide a visual disconnect for maintenance purposes, knife-blade load-break switches or fuse cutouts are located on the primary and secondary of the substation transformer. The switches are usually housed in metal-clad enclosures, while the cutouts are pole mounted. Gang-operated switches are recommended over hook-stick devices, since pulling one line at a time with the circuit under load can obviously single-phase the system.

Fuse cutouts have an interrupting capacity instead of a short-time current rating. They should be used to isolate parts of a deenergized circuit, even if the cutouts used are designed with mountings that operate as a load-break switch. Cutouts alone cannot be relied upon to protect the transformer in all cases, especially when the available short-circuit current is above their interrupting rating. Power fuses should therefore be used as backup protection.

High-voltage fuses may be employed as the main means of transformer protection. As with any protective device, their time-current characteristics must be coordinated with upstream and downstream devices, here the utility and the protective relaying on the secondary. Fuses alone cannot provide ground-fault protection of the transformer primary. When fuses are used and ground-fault protection is needed, relaying could be used to trip the mechanism of an automatic load-break switch. A general protection rule, though, is to use a primary circuit breaker with protective relaying when the three-phase transformer capacity is 5,000 kVA and above (*13*). For any capacity, the primary breaker prevents the single phasing that fuses allow and permits easier coordination with other relaying.

PROTECTIVE RELAYING IN SUBSTATIONS

The sizing of overload, short-circuit, and ground-fault relays is basically no different from that described already in this and previous chapters for other power equipment. Transformer protection is a protective-relaying problem in substations. With larger substation capacities, relay pickup for overloads and short circuits alone is normally too high to provide adequate protection. Sudden-pressure relays that sense the rate of pressure rise in the gas cushion of the transformer tank are sensitive to small arcs under oil, and their use is warranted (*13*). Further, percentage-differential relaying is also strongly suggested for internal fault protection of transformers rated at 5,000 kVA or higher, although it is not as yet a widespread practice.

A standard percentage-differential relaying system is illustrated in figure 13.16. Figure 13.17 is a typical one-line diagram of a substation with this differential relaying added. The CT's on a wye-connected winding of a transformer should be connected in delta, while CT's on a delta winding are connected in wye. There are two basic requirements that percentage-differential relaying must satisfy (*11*).

1. The relays must not operate for load currents or external faults.
2. The relays must operate when internal faults are severe enough.

Relay pickup is used to trip the breaker on the transformer primary.

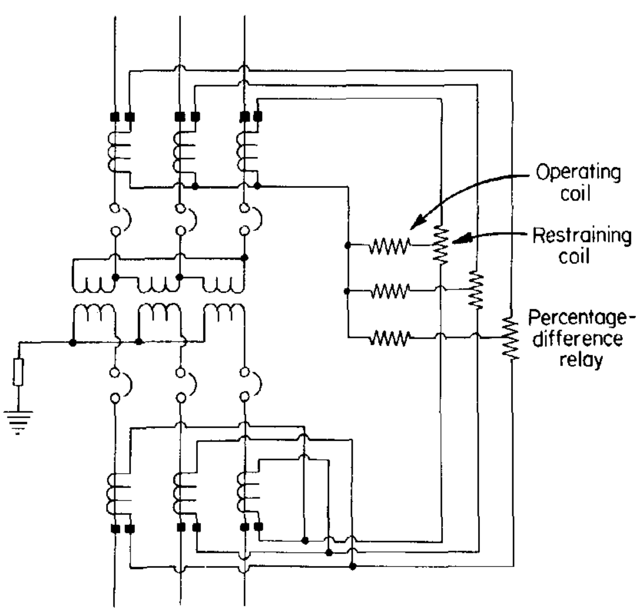

Figure 13.16.—Standard percentage-differential relaying system for transformer protection.

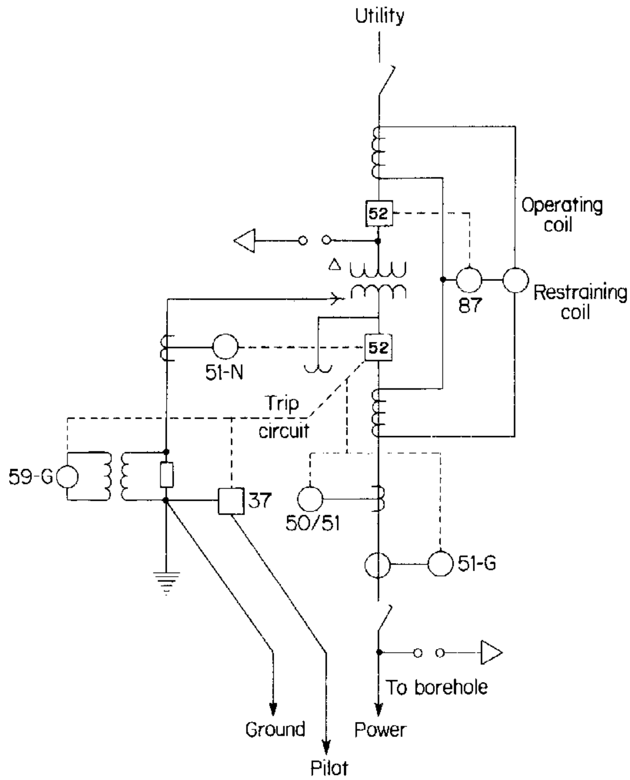

Figure 13.17.—One-line diagram of substation with percentage-differential relaying.

Differential relays usually have coil taps to compensate when the CT's are not perfectly matched. When selecting a CT for differential relaying, the common practice is to choose the highest CT ratio that will provide a secondary current as close as possible to the lowest-rated relay tap. This minimizes the effect of impedance in the wiring connections between the CT's and the relays. To assure that the relay will operate a maximum sensitivity, the current supplied to the relay under maximum load conditions should be as close as possible to the continuous-current rating of the tap (*11*).

Percentage-differential relays usually have an adjustment to vary the percent slopes. The adjustment provides a means for preventing unreliable relay operation due to unbalances between CT's during external faults. Unbalances can occur from the following:

- Tap changing of the power transformer,
- Mismatch between CT secondary currents, and
- The difference between CT errors on either side of the power transformer.

If a power transformer is rated at 10,000 kVA or greater, a harmonic-restraint circuit is recommended in addition to the percentage-differential relays (*13*). This circuit causes a differential relay to be self-desensitizing during magnetizing inrush periods, but the relay is not desensitized if a short circuit occurs in the transformer during a magnetizing inrush period. Only the fundamental component of the differential current is delivered to the operating coil. The harmonics are separated, rectified, and delivered to a restraining coil (*11*).

The pressure relays mentioned earlier can play a valuable role as supplemental protection to differential relaying. In fact, with sensitive and reliable pressure relays, the sensitivity of the differential relays can be reduced to prevent undesirable operation due to inrush current (*11, 13*). Thermal-overload protection should also be provided as a third means of transformer protection.

LIGHTNING AND SURGE PROTECTION IN SUBSTATIONS

Since much of the equipment of the surface substation may be exposed, there must be protection against transient overvoltages due to lightning as well as due to switching. Overhead static wires and shielding masts (see chapter 11) are commonly used to protect substation equipment from direct lightning strokes. Two static wires can be positioned above and between the line conductors to provide shielding for overhead distribution lines. In addition, surge arresters are mandatory to limit the transient overvoltages to safe levels.

The surge arresters on the incoming lines should be located as close as possible to the transformer terminals. Station-class valve arresters should be used. Again, arrester selection should be based on the primary voltage, the effectiveness of grounding, and insulation coordination between the arrester and the transformer BIL. With dry-type transformers, the BIL is practically constant with the width of applied impulse (see chapter 12, figure 12.5). The margin of protection could then be less for the arrester sparkover voltage than for the IR-discharge voltage. It can be seen in figure 13.18 that in liquid-immersed transformers, the insulation withstand is not a linear function with the impulse width. Instead, the insulation-withstand level decreases from the front-of-wave to chopped-wave to the full-wave values (*12*). The full-wave value is the BIL rating. Standard values for oil-immersed transformers are listed in table 13.2 (*9*), and values of insulation classes below the standard are provided in chapter 11, table 11.1. As shown in figure 13.18, the BIL should be compared with the discharge voltage. Because the exposure available in substations can lead to worst case surge conditions,

Table 13.2.—Standard BIL's for oil-immersed power transformers

Primary winding phase-to-phase voltage, V	Insulation class, kV	BIL, kV
22,900	25.0	150
23,000	25.0	150
26,400	34.5	200
34,500	34.5	200
43,800	46.0	250
46,000	46.0	250
67,000	69.0	350
69,000	69.0	350
92,000	92.0	450
115,000	115.0	550
		[1] 350
		[1] 450
138,000	138.0	650
		[1] 550
		[1] 450
161,000	161.0	750
		[1] 650
		[1] 550

[1] Reduced BIL's may be applied if proper coordination is maintained with surge arresters.

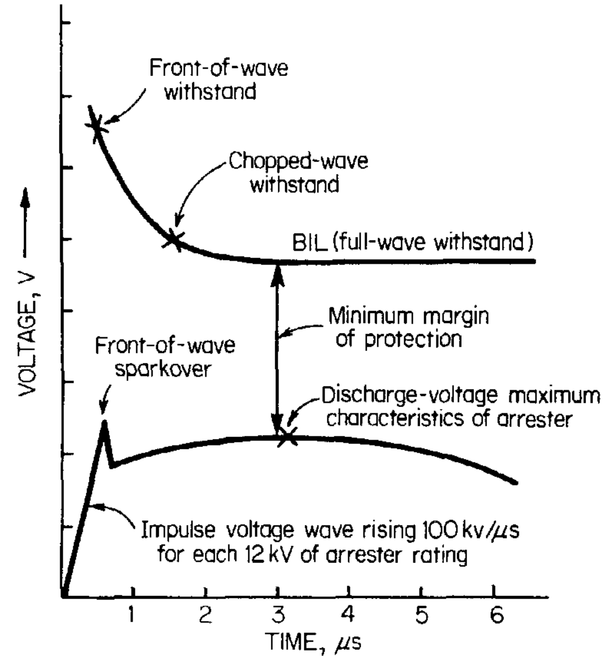

Figure 13.18.—Insulation characteristic of liquid-immersed transformer compared with the characteristic of valve surge arrester.

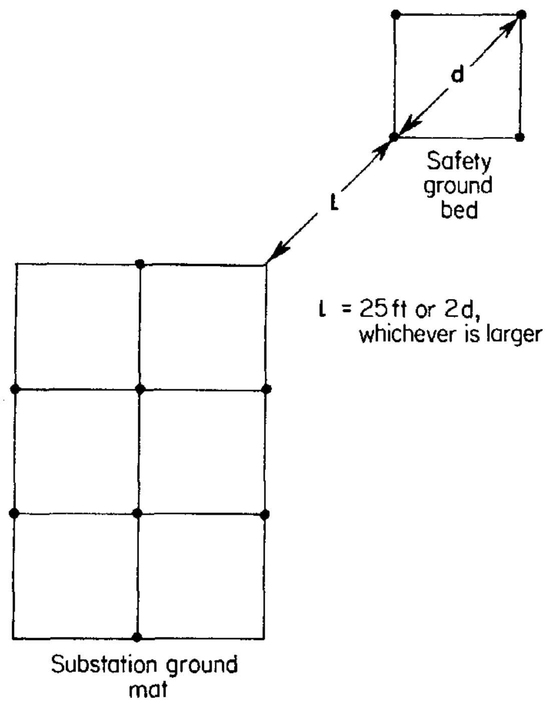

Figure 13.19.—Plan view showing locations of system and safety ground beds.

the use of 20,000-A discharge current to establish the IR-discharge voltage is probably justified. The margin of protection should not be less than 0.20.

Surge arresters on the outgoing lines should be located as close to the point they leave the substation as practical. These can be station-class arresters but are commonly intermediate class.

Since higher insulation levels are common on breakers and switches, at more remote locations it is sufficient to provide protection with arresters only. Breakers and disconnect switches are designed so that the impulse-withstand level over open switches is greater than that to ground, which reduces the possibility of flashover. As a result, a surge on these devices is likely to flash to ground and will not usually cause permanent damage (*13*).

Whenever current-limiting fuses are applied with surge arresters, the possibility exists that the arc voltage produced by fuse operation could result in arrester sparkover and damage (*13*). Station-class and intermediate-class arresters are more susceptible to damage, because of their lower protective characteristics. If fuses are located on the line side of the arresters, arrester sparkover will not occur.

SUBSTATION GROUNDING

Extensive discussion of grounding, including information about substation applications, is presented in chapter 7, but there are specific concerns that could not be covered until now. Before discussing them, a short review should be helpful.

Two separate ground beds are required at the substation: the system bed and the station bed. The station bed is usually a mesh located underneath the substation area and is sometimes called the substation ground mat. As shown in figure 13.19, the safety bed is located at a distance from the station ground bed (*3*). Surge arrester grounding conductors, static conductors, metallic frames, and fencing within the substation area are connected to the station ground bed. Only the distribution grounding conductors, including that of the grounding resistor, are tied to the safety ground bed.

Separation is mandated to prevent the voltages that might be produced across the station bed from being transferred to the safety bed. This is particularly critical when portable or mobile equipment is involved, since all equipment frames can be assumed to be at the potential of the safety ground bed (that is, the voltage drop across the bed to infinite earth). By law, the two beds must be separated by a minimum of 25 ft. However, better separation is achieved by a distance not less than two times the radial influence of the safety ground bed. This radial influence is defined as the longest straight line that can be drawn within the volume enclosed by the safety bed (*3*). The separation distance is measured between the two closest points of the system and the safety beds. To help minimize possible coupling between the two ground beds, the safety bed should be placed in a location adjacent to the corner or narrow side of the substation (fig. 13.19).

To limit the magnitude of voltage produced across a bed during current flow, low resistance is mandatory for both ground beds. The resistance value is generally defined as 5 Ω or less. To ensure that step and touch potentials are not hazardous in the grounding location, potential gradients during surging conditions must also be restricted. The design of an adequate safety ground bed that will meet these requirements is presented in chapter 8. The guidelines for the substation system ground bed are covered in the following section; this information has been abstracted from Bureau of Mines Information Circular 8835, an excellent reference on the subject (*3*).

Substation Ground Mat

The system ground bed is a series of interconnected conductors and possibly ground rods buried in the earth under the substation. The main purpose of this ground bed is to provide electric-shock protection for personnel in and around the substation during lightning strokes, short circuits, equipment failures, and many situations involving human error. This protection is provided by the multiple grounding of all accessible surfaces within the substation area, to limit step and touch potentials to safe levels. Substation equipment that cannot be grounded is placed so it is inaccessible and cannot be touched by personnel. The second purpose of the system ground bed is to limit insulation stress by conducting surges to earth.

One of the most important aspects of the substation design is the construction of the ground bed itself. The purpose of the system ground bed is to hold the voltage of the substation floor at ground potential. A common hazard within the substation occurs from touch potentials that are too high; these potentials can be held to acceptable levels by using an open grid of conductors buried beneath the surface. The maximum acceptable grid spacing depends on the available fault current, resistivity of the earth, clearing time of protective devices, and overall bed geometry. Regardless, the following guidelines should provide adequate safety in practically all situations (*3*, *6*).

The substation should be enclosed completely by a continuous buried ground conductor. This perimeter conductor should extend approximately 3 ft beyond the outer substation fence to protect any person who might contact the fence. Within the perimeter conductor, a regular grid or mesh of wires spaced approximately 5 to 15 ft apart should cover the entire substation area. These wires should be uniformly spaced and located along rows of equipment to facilitate grounding connections. Extra conductors should be added to the grid at the corners of the substation and in substation work areas.

The conductors should be bonded together at all intersecting points of the mesh, using either exothermic-welded connections or heavy clamps designed for grounding applications. All substation equipment should then be bonded to the ground mesh at two different points, preferably at points of interconnection on the mesh. The substation fence, posts, and gates should also be bonded to the grounding grid. The substation site should be chosen to avoid gas pipes, water pipes and other buried conductors if at all possible; otherwise, these ungrounded conductors must also be bonded to the grid at several points.

Borehole casings require special treatment because they can provide a low-resistance path from surface areas to underground workings (*3*). The boreholes within the substation should be bonded to the station grounding grid to protect personnel in the substation area. Protection cannot be provided to both ends of the casing; therefore, the casing extending underground must be made inaccessible for at least a 10-ft radius and labeled as dangerous. It must not be used as ground reference for any equipment or haulage system underground. If the borehole casing is outside the substation but in the vicinity of the safety ground bed (fig. 3.19), it should be coupled to the safety ground. In this case, the underground end of the borehole casing can be used as a grounding reference. The borehole supporting structure should also be bonded to the casing but isolated from the station ground so that surface personnel cannot contact both grounds simultaneously. If the borehole is outside the substation and away from the safety ground, it can be left ungrounded. Here, both ends should be avoided but isolation is not mandatory. (For additional discussion, see reference *3*, page 23.)

If a low resistance is difficult to obtain or if the area experiences severe freezing or drying, ground rods can be driven at mesh points and interconnected with the grounding grid. These rods are most effective if they are driven near the periphery of the bed, especially near the corners. However, driving the rods closer together than 10 ft has not been found effective in lowering ground-bed resistance.

All buried conductors must be corrosion resistant. Copper should never be mixed with steel or aluminum; for example, if steel clamps are used with copper conductors, they will be rapidly destroyed by corrosion unless protected.

To provide adequate mechanical strength, the size of the mesh conductors should be not less than 2/0. Prefabricated meshes are available that use numerous small conductors welded together; this is also mechanically acceptable. The conductors can be copper, copper-clad steel, galvanized steel, or stainless steel, but it is usually not a good idea to use aluminum unless an alloy is available.

The ground-bed conductors should be buried about 18 to 24 in below the substation grade. The perimeter conductor can be placed somewhat deeper to reduce the potential gradients. Closely spaced conductors near equipment manual control handles should be installed closer to the surface for maximum protection. Rubber mats or wooden platforms can be used for temporary personnel protection at control handles instead of closely spaced conductors, but these mats must not remain in the substation permanently since they are usually not effective when wet and can be a hazard in the winter. Special care must be taken to ensure that the insulated area is large enough to protect the area around the handle.

The use of a gravel surface in the substation greatly improves safety because it significantly increases contact resistance, thereby increasing tolerable touch potentials by a factor of 2 or more. The gravel should be well drained, have a thickness of at least 4 in, and extend no less than 4 ft beyond the substation fence.

The total resistance of the system ground bed is closely related to its perimeter, and a long narrow bed will have a lower resistance than a square bed of the same area. It should be noted that the total resistance cannot be lowered by decreasing the grid spacing nor by driving ground rods within the substation area. Thus the mat should be designed to cover a relatively large area with conductors arranged to reduce high-voltage gradients. Although the equipotential characteristics of the system ground bed will protect personnel regardless of its resistance to earth, it is mandatory that this resistance be low, 5 Ω or less. This limits the voltage rise during lightning discharges, thus limiting the stress on power-system insulation. A lower ground-bed resistance also tends to reduce potential gradients in the earth around the substation. For moderate to large substations, the required size of the system ground bed is usually big enough to ensure that the resistance is low. However, it may be necessary to augment the earth connections with driven rods or additional conductors in the case of small substations or substations constructed on high-resistivity soil. Figures 13.20 and 13.21 illustrate suggested system ground beds for large and small substations (*3*).

Several additional comments about substations that feed underground mines are in order. First, it is recommended that the safety ground bed be located as close as practicable to the borehole or the point where the power extends underground. This practice minimizes the possibility of damage to high-voltage circuits entering or extending underground, which can lead to hazardous conditions, and reduces the probability of lightning strokes hitting exposed high-voltage circuits extending underground. To further reduce these problems, electrical circuits should have the following protection within 100 ft of the point on the surface at which they enter an underground mine:

- Overload and short-circuit protection,
- Ground-fault protection,
- Surge protection, and
- A visual disconnect.

As all of these can be contained within the substation area, it is logical to place the entire substation as close as possible to the underground power entry. This can also prevent unnecessary duplication of components.

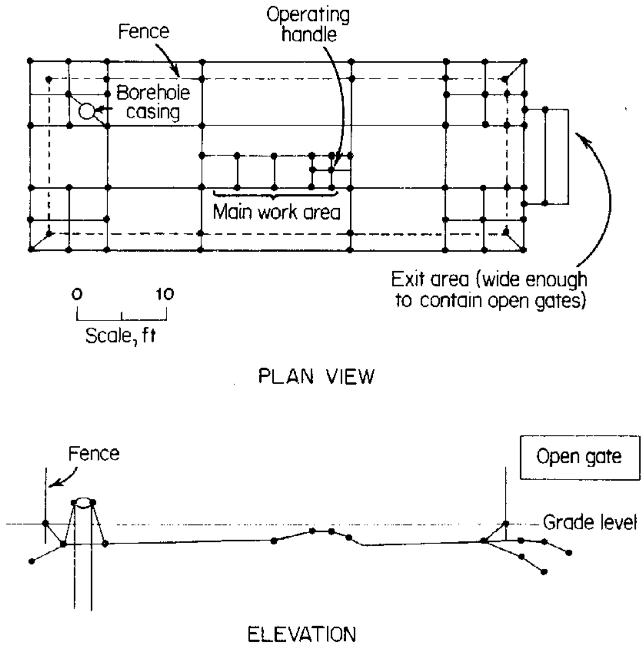

Figure 13.20.—Typical system ground bed for large substation.

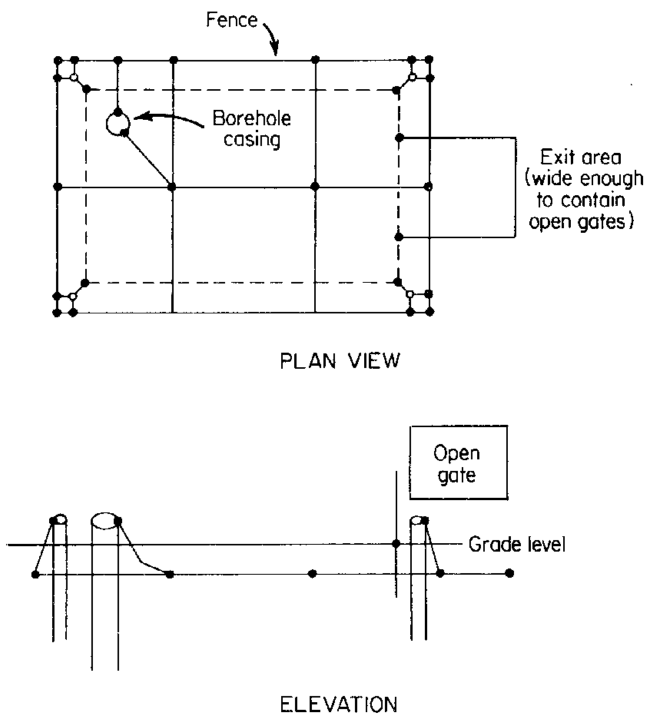

Figure 13.21.—Typical system ground bed for small substation.

Ground-Fault Protection

A resistance-grounded system is mandatory for portable and mobile equipment being fed by the substation. The ohmic value of the grounding resistor must be such that the frame potentials of the high-voltage distribution are no greater than 100 V during ground-fault conditions. For practical purposes, this restricts the maximum ground-fault current to not more than 50 A. Most modern substations serving mines use a 25-A ground current limit, however it is important that the selected current limit is greater than three times the per-phase capacitive-charging current.

Ground-fault protection at the surface substation should be by

- Zero-sequence relaying for each outgoing circuit,
- Potential relaying about the grounding resistor, and
- (Direct) neutral relaying between the grounding resistor and the neutral bushing of the power transformer or the grounding transformer.

Protective relaying to detect grounding-transformer failures is wise, but protection such as fusing should never be used if it will remove the transformer from the line or the grounding resistor. Ground-check monitoring of circuits feeding portable and mobile equipment is essential, and it may also be necessary on circuits feeding other loads.

ADDITIONAL MINE SUBSTATION LOADS

When other loads are being supplied from a mine substation, such as fans and preparation plants, grounding problems can become complex. Three fundamental concepts must still be served:

1. Provision of ground-fault protection,
2. The earth cannot be used as a grounding conductor, and
3. Isolation of the grounding system serving portable and mobile equipment.

Adherence to these criteria is mandatory where underground mines are connected with the substation. The following discussion will be pointed directly at these applications, although the comments are adaptable for any mine. Figure 13.22, illustrating a substation feeding both surface and underground loads, will be used as a focal point.

At first glance, the circuits shown in this figure would appear to be reasonably safe. The actual surface load is isolated from the main substation secondary through the transformer windings of its unit substation. Both substations have separated ground beds with all appropriate connections. Each outgoing circuit from the main substation is separately protected for line and ground overcurrents. However, consider the situation where a ground occurs in the unit substation, for instance in the incoming circuitry or within the transformer. This can establish a ground current path through the earth, the main substation safety bed, and its grounding resistor. For the time that earth current flows, the mine grounding system is elevated by a potential equal to the mine ground-bed resistance times that current. In other words, one forbidden grounding practice is shown in the figure: the earth is being used as a grounding conductor. Furthermore, the ground-fault protection used in this system portion, zero-sequence (51G) and potential (59G) relaying, must have a specific level of ground current to pick up. Earth resistivity may be large enough that detection is not made and the current persists. Even if tests are made to ensure pickup, seasonal variations might destroy all intentions.

Figure 13.23 illustrates a solution to this problem. Here, the frames of all direct loads for the main substation are connected via grounding conductors to the safety ground bed. Ground-check monitoring of these conductors to the surface facilities is preferred, but equal safety may be obtained by establishing a low-resistance ground bed on the unit substation end. The new bed must be considered as part of the mine ground bed.

Because of the increased number of components connected to the safety bed, the probability of lightning strokes and other surging conditions elevating the bed is also raised. Surge protection must therefore be added. Static lines should protect overhead power, grounding, and ground-check lines extending to the surface loads. The static lines must not be connected to the mine ground bed. Outgoing-circuit grounding of the unit substation should be separated from the mine ground bed by the same provisions as station ground beds.

The recommendations above do have exceptions. First, if the transformer used for the surface loads is in the same substation area as the main power transformer, a grounding conductor to the mine ground bed must not be used, because this could easily tie the substation system and safety beds together. In some instances, isolation of the unit substation secondary might not be practical or could create a personnel hazard. An example is a unit substation

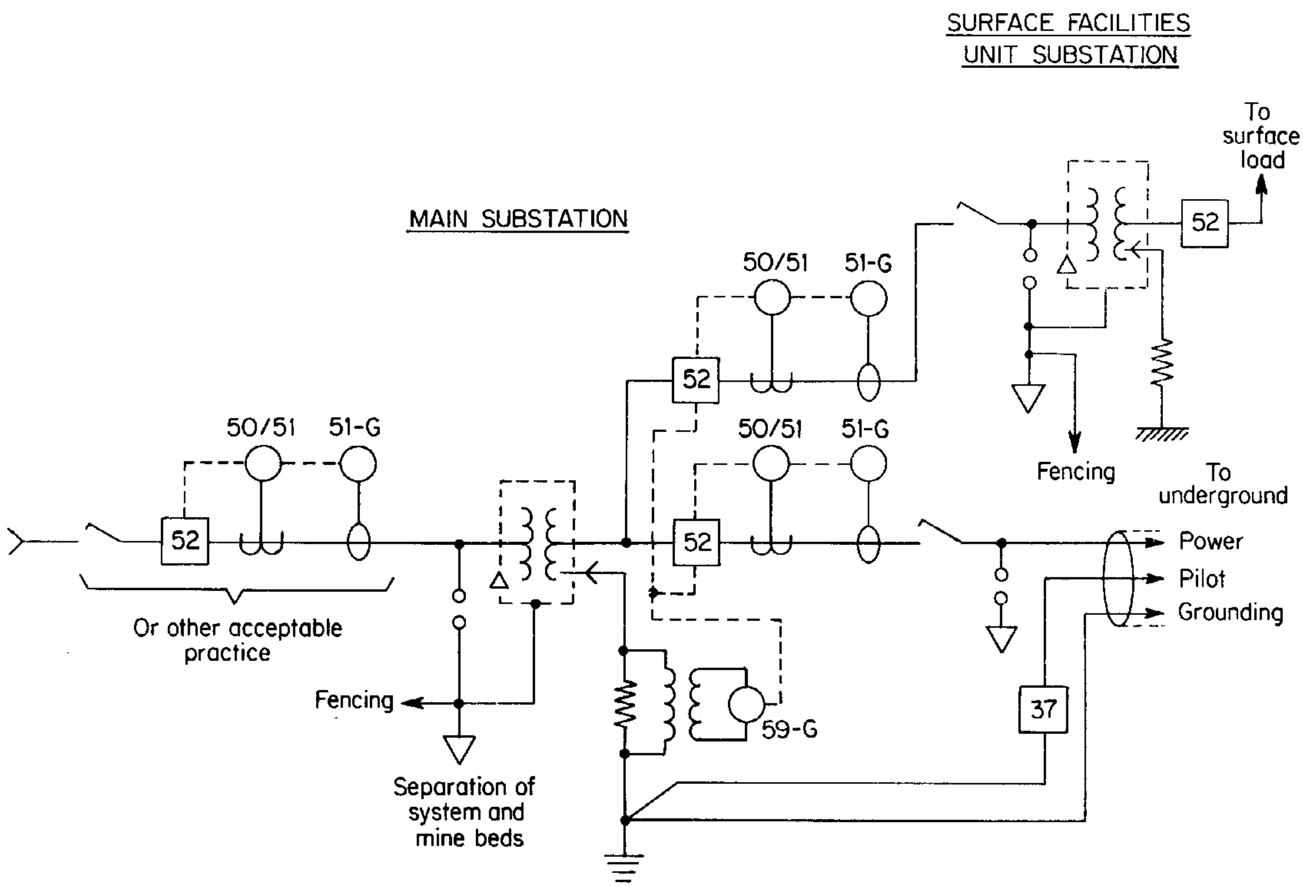

Figure 13.22.—Substation feeding both surface and underground loads (no grounding conductor).

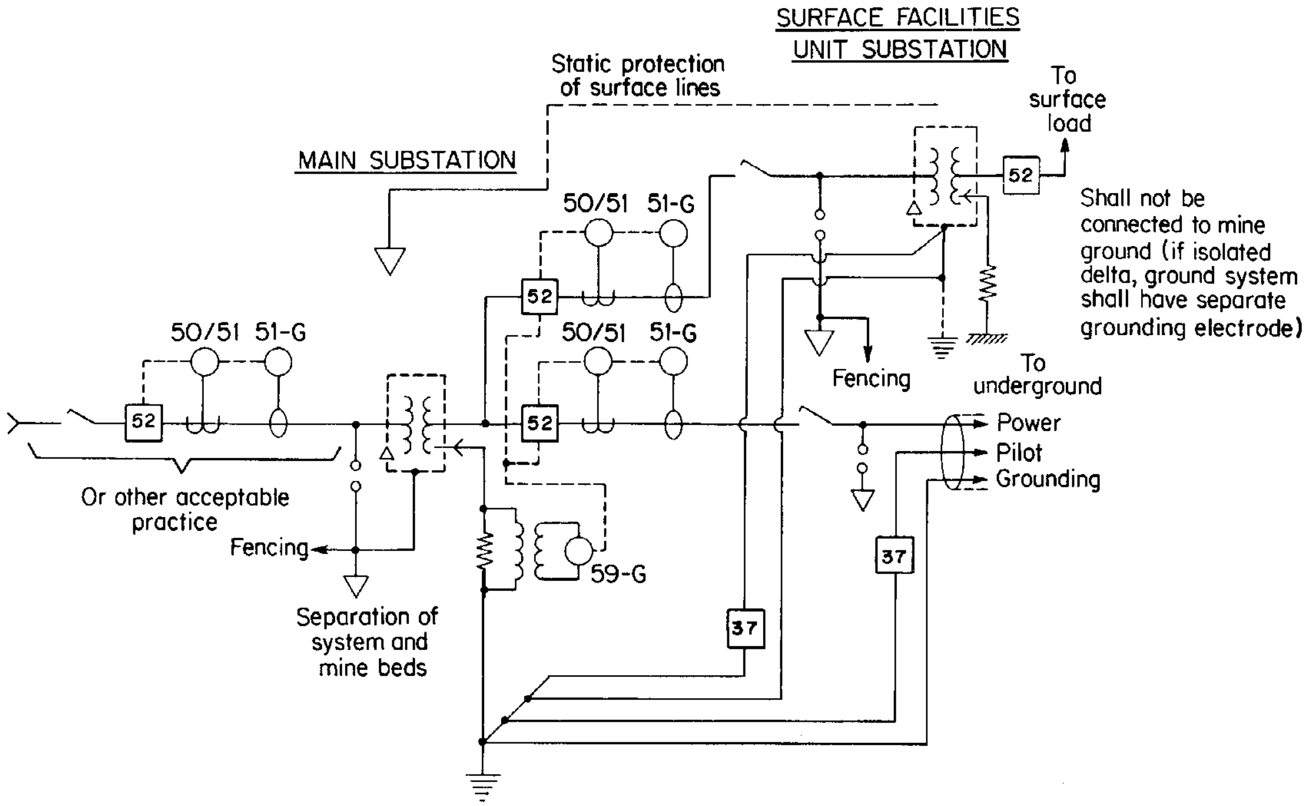

Figure 13.23.—Substation feeding both surface and underground loads.

mounted on a metallic structure associated with a preparation plant. An attempt at isolation here could result in two proximate surfaces that could have dangerous potentials existing between them. In this situation, the grounding conductor to the mine bed is still mandatory, yet exposure of the mine bed to lightning-stroke surges would be substantially increased, without the next exception.

It can be shown that if a grounding conductor is approximately 250 ft in length, its typical impedance to a surge (such as an 1.2 × 50-μs impulse) is high enough to stop the traveling wave effectively. Thus, if a safety factor of 2 is applied, any grounding conductor of 500 ft or longer, protected by a static line or otherwise, can be connected to the mine ground. This means that when isolation of the unit substation is dangerous, the mandatory grounding conductor to the mine bed should be effectively 500 ft long. In addition, if a grounding conductor is protected by static wire for 500 ft, it can be used for shielding the power conductor provided that it is properly grounded beyond that distance.

When these precautions are met, the result should be a safe substation grounding system, and yet there is still a high chance that personnel safety could be compromised through human error. Hence, maximum safety will be achieved if a separate substation is used to supply mine power. Perhaps the best but most expensive approach is to use subtransmission to feed a second substation for surface facilities, but use of an isolation transformer between the main substation and the mine provides the same benefits. This will be covered subsequently.

PORTABLE SUBSTATIONS

As mentioned earlier, portable substations have gained recent popularity for small load situations in mines. The entire substation is skid mounted and can be transported as a unit by truck. Portable substations are custom built to meet the needs of the individual mine. For some small mines, the economics are such that the portable substation is used in conjunction with a diesel-powered generator. The output of the generator may be a low voltage, such as 480 V, and here the substation transformer may step up the voltage to 4,160 V for distribution to the mine loads.

Figure 13.24 shows a situation where the utility company supplies a voltage of 12.47 kV. This particular substation is designed to operate a mine load of 750 kVA and surface loads of 150 kVA, including a conveyor belt and a ventilation fan. The surface and the mine circuits are in parallel with each other and supplied by the incoming feeder. The surface circuit can be isolated from the mine circuit by means of the load-break switch, and the primaries of both the surface and mine circuits are protected by high-voltage fuses. The surface transformer steps down the incoming line voltage to 480 V for operating the belt and fan. Each of the surface loads is protected by a molded-case circuit breaker that contains an undervoltage release powered from the utility-control transformer. The utility-control transformer also supplies power to the 240- and 120-V outlets.

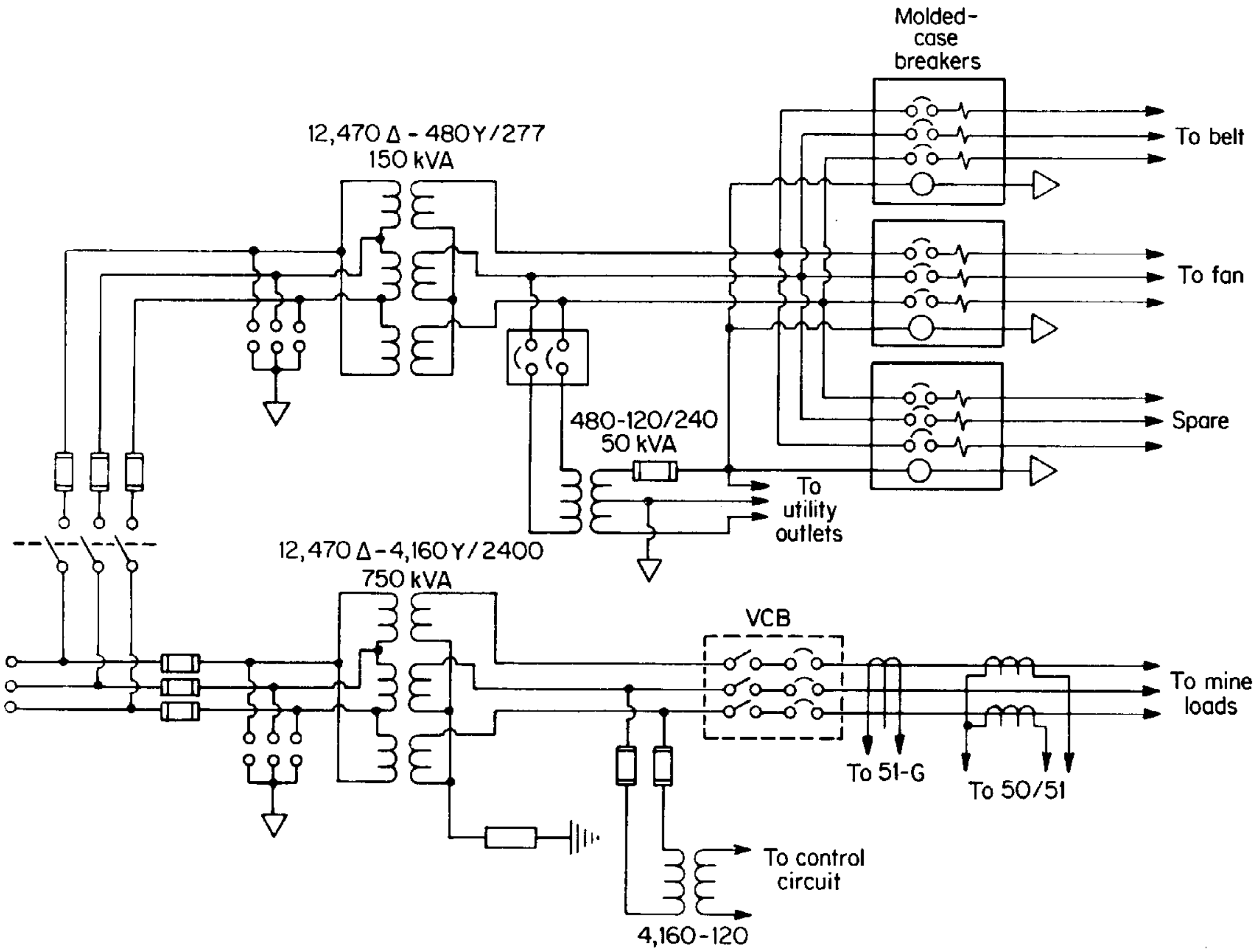

Figure 13.24.—Typical portable substation to service small mine.

The mine transformer steps down the incoming voltage to 4,160 V for distribution to the mine loads. The transformer secondary is wye-connected to permit resistance grounding and feeds the VCB, which in turn protects the distribution circuit. CT's at the breaker output supply inverse-time relays for line-overcurrent protection and ground-fault protection. The control transformer on the line side of the VCB supplies power for tripping the breaker.

As in larger permanent surface substations, individual ground beds must be provided for the station ground and the safety ground. These connections are noted by the two different ground symbols in the schematic.

UTILITY VOLTAGE AS MINE DISTRIBUTION

Some mines must purchase power at the same voltage as mine distribution, either because of economics or as the result of a utility company decision. The utility company often supplies power from an ungrounded delta transformer secondary. The mine operator must supply basically the same substation components as those discussed, but the main transformer may or may not be included, depending on the location of the utility transformer (which is the power source or source transformer) and also on the specific location where a neutral must be derived to establish mine grounding. For surface mines, this location is not always critical, but for underground mines the mine ground bed should be as close as practical to the point where circuits extend underground. Regardless, the following rules apply to all ungrounded delta sources supplying portable or mobile mining equipment, including situations where the mine owns the source transformer.

If the distance between the source transformer and the grounding location is 100 ft or less, the simple arrangement in figure 13.25 can be used. A grounding transformer derives the neutral point, a grounding resistor is inserted to limit ground-fault current, and a safety ground bed is established. Load-break switches provide visual means of disconnect, and surge protection is afforded by arresters grounded to an arrester ground bed separated from the mine ground. Protective relaying is identical to that in any substation: line-overcurrent, ground-fault, and ground-check monitoring. Here it is imperative that the recommended zero-sequence relaying be located downstream from the grounding-transformer connection, otherwise it cannot detect downstream ground faults. Zero-sequence currents terminate at the grounding-transformer terminals. From switch to switch, this additional equipment is defined as the substation for the mine.

When the distance is greater than 100 ft, the one-line diagram of figure 13.26 applies. This arrangement requires the installation of an isolation transformer, but otherwise the required substation is practically identical to that in figure 13.11. Although a delta-wye transformer is shown, a delta-delta power transformer with a grounding transformer could also be used. This system reflects the Federal regulation that requires the grounding resistor to be located at the source transformer. To be consistent with other regulations, "at" is defined as 100 ft or less, and the logic for the requirement is very sound.

Utility companies commonly supply the ungrounded delta power on overhead lines. The typical distance for one overhead span (pole-to-pole) is approximately 100 ft. Exposed power lines, beyond one span, do raise the probability of lightning striking the mine power system, but equally important, the possibility of losing a line conductor is significantly increased. One common example is a vandal's shooting an insulator or power conductor with a rifle or handgun. If one of the lines between the source transformer and the grounding transformer is lost (for example, in figure 13.25), the grounding transformer cannot find the system neutral, hence its common terminal, which is connected to the grounding resistor, will rise to almost the same potential as a remaining line conductor. With normal system operation, the problem presented here is minor since the resulting single phasing would be quickly detected by the overcurrent protective circuitry. Under no-load or low-load conditions, however, this circuitry might not detect the unbalance, and all potentials in the mine could become elevated to near line potential. Potential relays across each leg of the grounding transformer can be employed to detect such a hazard and supply tripping to the main circuit breaker. An isolation power transformer provides this same protection.

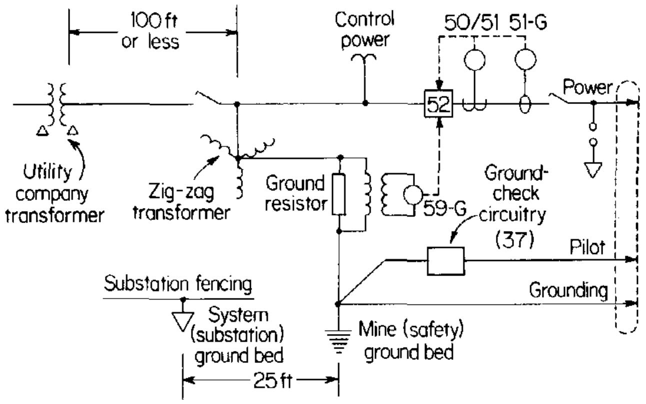

Figure 13.25.—Providing mine ground and protective relaying from utility substation.

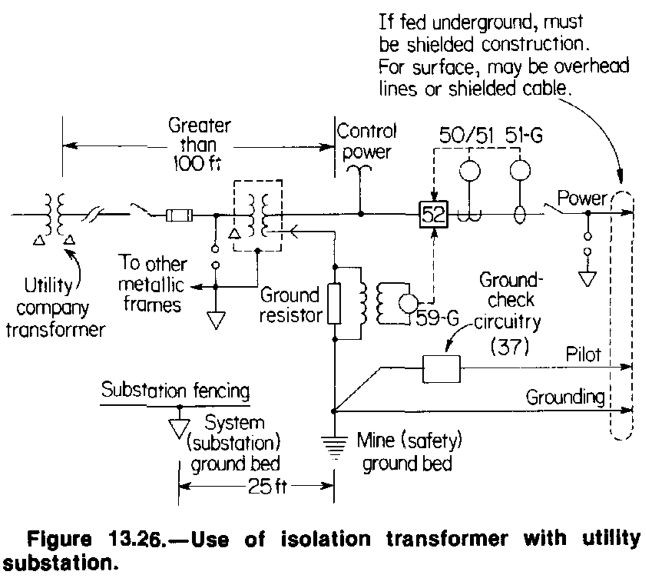

Figure 13.26.—Use of isolation transformer with utility substation.

ADDITIONAL SUBSTATION DESIGN CONSIDERATIONS

Over the years many additional design precautions have been developed through experience with permanent substations at mines. A listing of some of these follows:

- Substation conductor insulators can be of post or cap-and-pin types, and either standard strength or heavy-duty design can be used (*13*). "Extra-creepage" insulators and bushings (or one voltage class higher than required) are recommended (*7*) as they extend the time between required cleanup and minimize flashover.
- The high-voltage side should be located on the windward side of the substation (*9–10*). The prevailing winds will then help keep the substation insulators clean between regular cleanups.
- The voltage drop on the distribution system should be maintained within $\pm 5\%$ to all load concentrations (*7*). This establishes the nominal voltage and necessary taps for the secondary of the substation, and corresponds to the taps stated earlier for transformer primaries in power centers.
- Permanent substations should be located outside the influence of the mine. This is very critical for surface mines.
- Buses and bus supports should be designed to withstand the stress from the average asymmetrical short-circuit current during the first 10 cycles after a fault (*13*).
- Line tensions within the substation area should be assumed to occur under maximum wind-loading and ice-loading conditions (*13*). Structure design should also be based on these specific line tensions and wind loading.
- Substation primary voltages above 34.5 kV cannot be used for distribution-class equipment (that is, cannot be transformed directly to low-voltage and medium-voltage mine usage) (*13*). This could create problems in supplying certain surface loads.
- The utility will meter the substation power consumption on the primary or secondary, depending on the primary voltage. The substation should also contain the mine's own metering, of equivalent precision to that of the utility (*7*). This is invaluable for maintenance checks and also serves as a double-check on the utility. Metering could include kilowatt, ammeter, voltmeter, and power-factor

instrumentation for each outgoing feeder from the substation. Demand meters, for either 15- or 30-min maximum demand, may also be needed.

- The fence enclosing the substation area should be no less than 8 ft high (*1*). The fence should be provided with a door or gate, which should be locked except when authorized personnel are present. The fence should be grounded to the station ground bed. It should also extend to the ground so no one can crawl underneath. Danger signs should be posted on all sides of the substation area.
- The substation area should be maintained free of weeds, trash, and combustible material that might create a hazard to personnel or the substation components (*1*).
- All components containing liquid insulation should be mounted on pads to allow quick drainage and capture of any lost fluids. This is especially important when flammable or environmentally hazardous liquids are involved.

Chapters 12 and 13 have attempted to collate an enormous amount of information to provide a coherent presentation of typical construction practices used for mine power equipment. Some designs, like mine power centers, are so dependent upon the individual application that standard units cannot exist. At the same time, the assembly of others, such as switchhouses and substations, is rather uniform across the industry. Because the subject matter is so large, some generalization has been essential and the omission of some specialized material was inevitable. Any engineer involved with mine equipment should of course work very closely with the manufacturer to determine the best solution to the specific needs of the mine.

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CHAPTER 14.—SOLID-STATE CONTROL AND RELAYING[1]

Except for a short introduction to electronics in chapter 5, discussion of solid-state devices has been avoided up to this point. The reason is that their use is rather new to the mining industry. However, their impact has been substantial, and if one area of electrical engineering can be singled out as having the greatest probable influence on future mining operations, it is electronics. This chapter will primarily discuss two solid-state applications that are already important: motor control and protective relaying.

MOTOR CONTROL

Almost all activity and interest in the solid-state control of industrial motors have centered around the use of the silicon-controlled rectifier (SCR) or thyristor. A model and circuit symbol for this four-layer, three-junction device are shown in figure 14.1. The outer two layers act as a p–n junction and the inner layers serve as an element to control that junction. The device has three external terminals: anode, cathode, and gate.

If the anode is positive with respect to the cathode (forward biased) and if the gate is reversed biased in reference to the cathode, there exists a balance of electrical charges in the four layers, and current flow in inhibited. This process is termed *forward blocking,* and the SCR exhibits high resistance in both directions. When the gate is forward biased with respect to the cathode, the gate current upsets the electrical charge balance, and anode-to-cathode current can flow. Once this conduction starts, the gate loses all control; in other words, the gate can turn the thyristor on but not off. Gate control (or turning the thyristor off) can only be achieved by reverse-biasing the gate-cathode circuit once more and reducing the anode-cathode current essentially to zero.

The typical characteristic curve for a thyristor illustrated in figure 14.2 details the phenomenon just described. Voltages and currents are given with respect to the cathode. When the anode potential is positive and the gate is at cutoff (negative potential), the characteristics are similar to those when the anode is negative. At a specific positive potential, called the breakover voltage, the SCR will turn itself on. Here, anode current increases considerably, and the voltage drop is reduced substantially across the device. The breakover point can be altered by varying the gate current, and this is why the thyristor is so valuable in power control applications.

The common technique for initiating conduction is to apply a current pulse at the gate (*12*).[2] The pulse alone neutralizes the thyristor blocking action at the desired breakover voltage. The simplest thyristor application is in the half-wave rectifier circuit shown in figure 14.3. Breakover is determined by the gate current, and thus the average value of dc through the load can be controlled by the gate control pulse (i_G). The process of starting the thyristor is often referred to as *firing,* and the angular position or timing of each pulse with respect to the source waveform is called the firing angle (angle α). In order to control ac, the load must receive both sides of the ac waveform. Two single-phase circuits that can be used as ac voltage controllers are shown in figure 14.4. The bidirectional arrangement uses two thyristors back to back and gives a symmetrical output with appropriate firing-angle pulses for the two gates. The unidirectional circuit with one thyristor and one diode presents an asymmetrical waveform to the load. The dc component thus created is considered a major disadvantage because the source waveform must be ideal for the circuit to be of practical value (*9*).

Voltage control for three-phase loads employs three single-phase controllers as shown in figure 14.5. Although a bidirectional circuit is illustrated, unidirectional circuits can also be applied. The dc components are mostly cancelled with the three-phase waveform, but the unidirectional circuits introduce a higher harmonic content to the line currents (*9*).

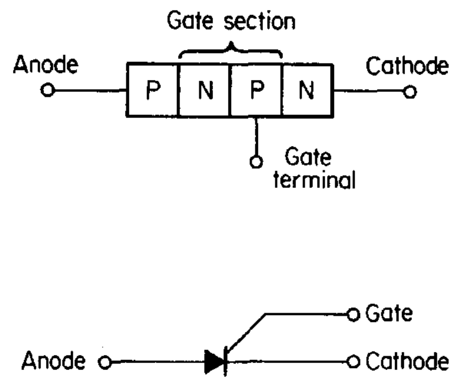

Figure 14.1.—Model and circuit symbol for thyristor.

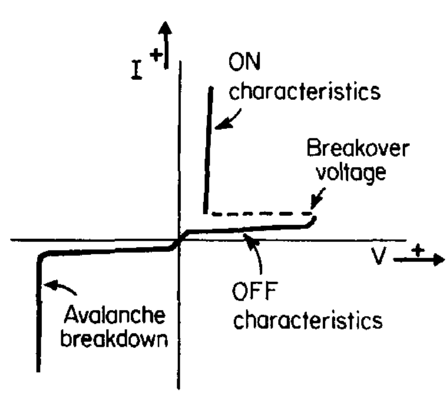

Figure 14.2.—Typical characteristic curve for thyristor.

[1] The author wishes to thank D. J. Tylavsky, assistant professor of electrical engineering, Arizona State University, who prepared the original material for the static protective relaying section of this chapter while he was a graduate student at The Pennsylvania State University.

[2] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

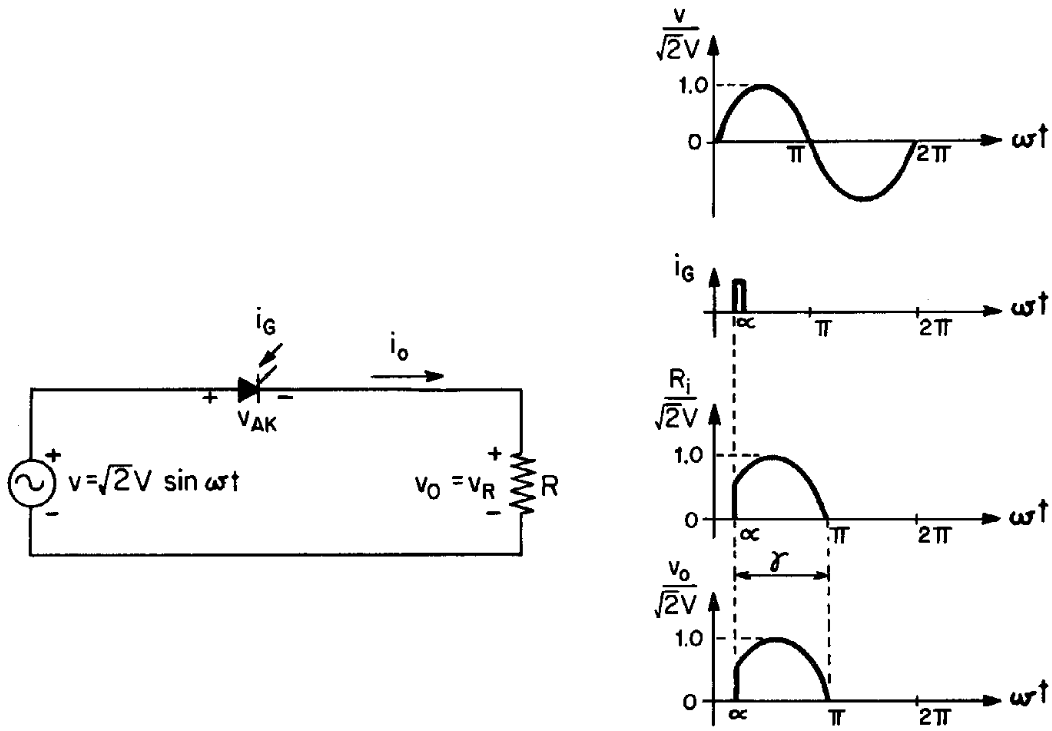

Figure 14.3.—Thyristor half-wave rectifier.

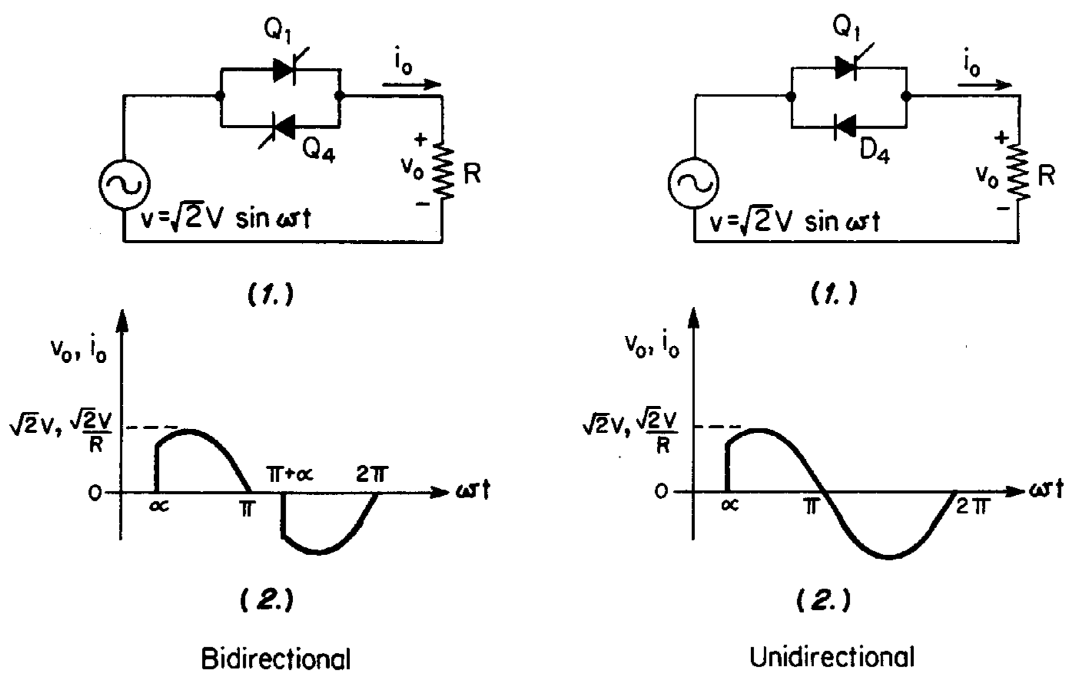

Figure 14.4.—Alternating current thyristor control.

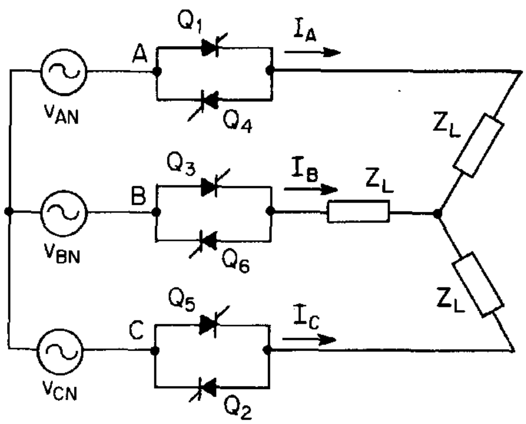

Figure 14.5.—Three-phase control with bidirectional thyristor arrangement.

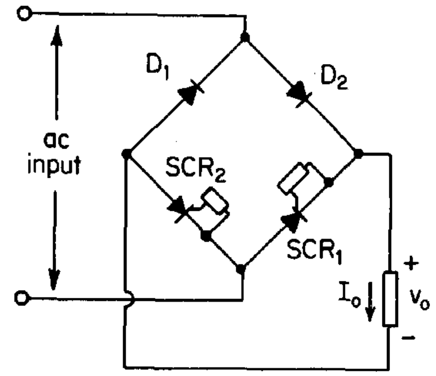

Figure 14.6.—Full-wave thyristor bridge rectifier.

Simple Motor Control

One method of speed control in dc motors has already been covered, simply adjusting the dc voltage to the machine. This would apply to loaded series-wound motors, but shunt and compound dc motors can also be controlled by two direct means: armature voltage or field voltage. The field control is interesting because field excitation is usually considered constant, but in fact motor speed can be adjusted inversely with field current. The simple circuit in figure 14.3 could be used in any of these cases when the source is ac; however, the basic circuits of figures 14.6 and 14.7 are commonly applied for single-phase and three-phase sources, respectively.

The commutation diode shown in figure 14.7 serves a specific purpose. Commutation is the transfer of current from one device to another as voltage relations change (*21*). The commutation diode conducts armature current so that it will not be transferred to the thyristors and diodes of the bridge and the thyristors can turn off at zero voltage. Without this diode, thyristor current might not reduce to zero and turnoff would not be likely. Other commutation techniques are available (*9*).

A chopper or dc-to-dc converter is another method of variable-speed dc motor control using thyristors; it is especially important because it operates between a dc source and a load. As shown in figure 14.8, the thyristors in the chopper circuitry switch the dc source on and off, supplying unidirectional voltage pulses. This reduces the effective voltage, V_o, to the load.

Variable-speed control of a standard three-phase squirrel-cage motor is possible using thyristor voltage control (for example, figure 14.5). Motor speed varies because the motor allows a greater percentage of slip at reduced voltage, then the controller increases the terminal voltage, the allowable slip is reduced, and the motor speed increases. However, three specific problems arise in this application. Reduced-voltage operation causes more heat to be generated in the motor. Motor torque is also reduced at slow speeds with lower voltage. Finally, the waveform produced by the thyristors is rich in harmonics (frequencies other than line) and this creates even more motor heat. Nevertheless, if high-quality motors are used and reduced-voltage operation is restricted to a short time, the result can be effectively controlled starting of squirrel-cage induction motors.

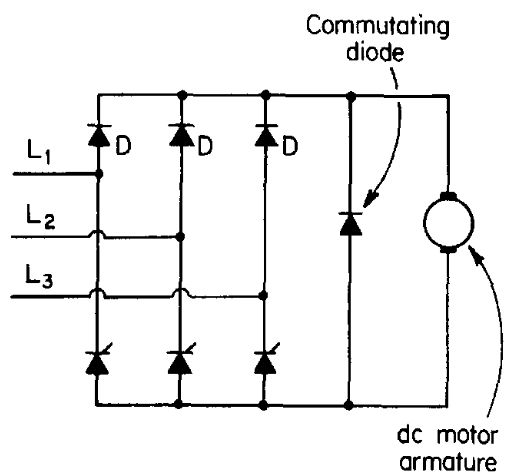

Figure 14.7.—Three-phase thyristor-controlled rectifier.

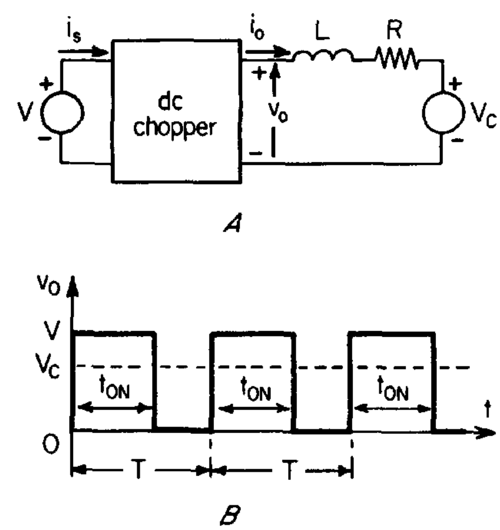

Figure 14.8.—Simplified chopper control.

Control Systems

To this point, only thyristors and their application to variable voltage control have been discussed. Other than explaining how the device is fired, nothing has been stated about the control circuitry that supplies the gate signal. Perhaps the best way to visualize power control applications with thyristors is through control-system principles. The two basic types of control systems are open loop and closed loop (*12–13*). In open-loop systems, the controlling element is unaware of the effect it produces in the controlled element, whereas in closed-loop or feedback systems, information is delivered back to the controlling element so it can adjust control of the controlled element to produce a desired result. The feedback control systems of interest here are termed *regulators* whose purpose is to maintain the result constant. Feedback systems inherently provide more precise control than do open-loop systems.

The operation of these systems can be described with the assistance of the basic block diagram in figure 14.9, where the lines in the diagram represent system variables. Each block represents a transfer function, the quantity that must be multiplied with the input to obtain the output. The comparison point in the block diagram is a position where two or more system variables are summed, and here the output is the algebraic sum of the entering variables.

The variable to be controlled, C, may be voltage, speed, or any other quantity. The input or reference variable, R, is usually adjusted manually by an operator and determines the desired value of C. The feedback element, H, supplies information about the output to the comparison point, whose function is to compare the input and feedback signal. The difference or error, E, thus obtained drives the controller, G. In the simplest system, the controller might contain the device that needs to be controlled. (Note that G usually represents the transfer functions from input to output. H denotes transfer functions in a feedback path.) A feedback system contains all these elements and variables, whereas in open-loop systems only the controller responds to the reference signal.

Elaboration of these statements can be seen in the block diagram of figure 14.10, which illustrates the main parts of a motor controller (*9*). The power circuits, which primarily contain thyristors, provide variable direct or alternating voltage output to the controlled system. The controlled system can be a rotating machine or any other driven load. Appropriate feedback is supplied back to the controlling system and could, for example, represent current, voltage, temperature, speed, and so forth. The controlling system responds to the feedback and input information, and supplies control signals to the digital circuits. Finally, the digital circuits, acting as an additional transfer function, simply switch the thyristors in the power circuit off and on at appropriate times.

Physical Characteristics of Thyristors

The two common thyristor configurations used in industrial applications are illustrated in figure 14.11. Stud-mount types are intended for smaller loads and have average forward-current ratings up to around 150 A. Heat sinking is like that shown for diodes in chapter 5 and, because thyristors have p–n junctions, the same heat-dissipation theory applies to these devices also. The larger thyristor in the figure was introduced in the late 1960's and is termed a *hockey puck, press pack,* or *disk* (*2*). It has the ability to transfer junction heat on both sides and is clamped between two heat sinks as shown in figure 14.12.

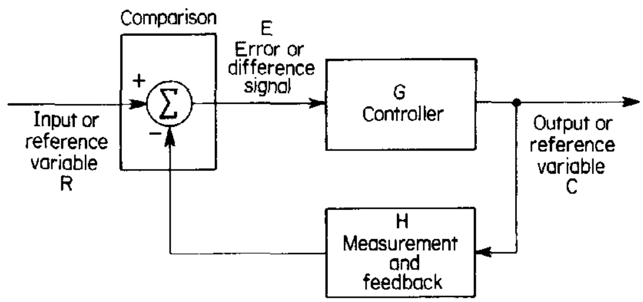

Figure 14.9.—Basic control-system block diagram.

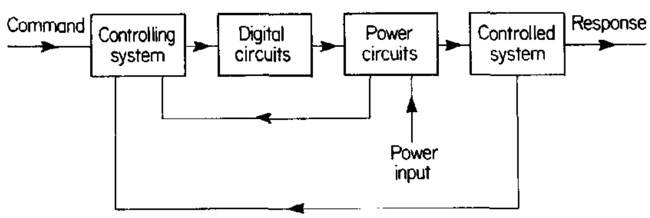

Figure 14.10.—Simplified block diagram of a motor controller.

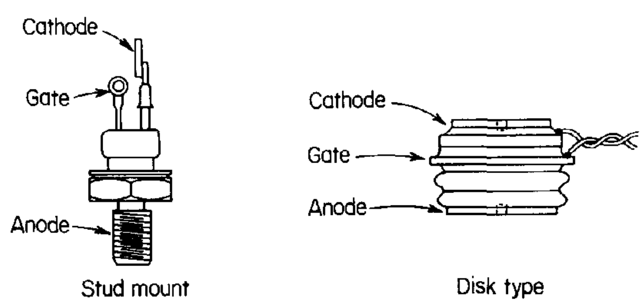

Figure 14.11.—Common thyristor configurations.

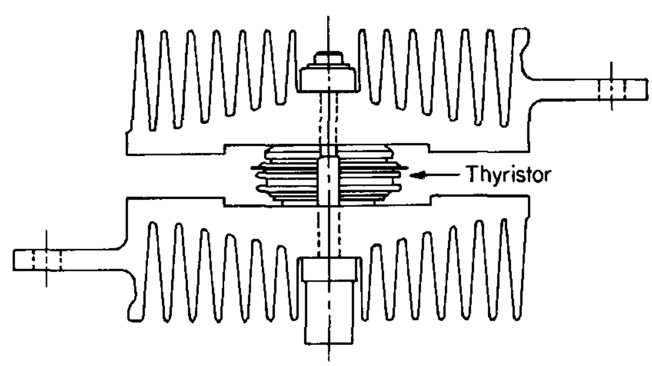

Figure 14.12.—Heat sinking of disk-type thyristors.

The heat-sink cooling can be by either air or water. Single disk thyristors presently have average forward-current ratings up to 7,500 A. Their relatively low cost has made solid-state motor control competitive in areas where prior to 1970 only electromechanical arrangements were considered applicable.

DIRECT CURRENT APPLICATIONS

There has been excellent success in applying solid-state or static control to dc equipment, especially equipment with less than 100 connected horsepower (*4, 17, 35, 39*). Examples include battery on-track and off-track vehicles as well as underground face equipment. The control has primarily used chopper circuitry, and two important advantages have been achieved: increased motor life by limiting armature current during acceleration, thereby reducing brush and commutator problems, and significantly reduced drive-train maintenance. General mechanical problems are reduced because motors have controlled acceleration. Even during plugging, which is the worst shock-loading instance, motors have controlled deceleration, reversal, then acceleration. For battery-powered vehicles, battery life is increased, since peak current demand is reduced and power is generally used more efficiently. (Applications in battery chargers are given in chapter 15.)

Static control of dc motors is also applied in surface excavators, hoists, and conveyor belts, where there are variable-speed requirements and substantial demand fluctuation (*39*). The power supplied to surface excavators, for example, can swing from 200% demand to 120% regeneration in 1 cycle.

Currently, static control in surface excavators is limited to machines of 20-yd^3 capacity or less. The form is similar to the conventional Ward-Leonard system (*20, 39*), but the motor-generator set is replaced by thyristors in a bridge configuration with feedback control. This change does improve operational performance, but it has a detrimental effect on the ac distribution system to the machine (*31*).

High-amplitude voltage transients and significant electrical noise commonly found on mine trolley systems have seriously hampered the adaptation of this technology to trolley locomotives (*39*). However, there has been some limited success with chopper control (*4*). It is interesting to note that problems have been encountered on practically all solid-state equipment receiving trolley power.

An outstanding success of static control has been the ac–dc drive in underground ac face equipment. The design maximizes power-distribution efficiency, uses the traction advantages of dc series-wound motors, and has improved the performance available from thyristor control (*35*). Diagrams for systems used on shuttle cars and continuous miners are given in figures 14.13 and 14.14. In both, ac power is supplied through the trailing cable. After the main circuit breaker, ac is supplied through conventional contactors to the pump-conveyor (hydraulic) motor, the cutting motors, and an on-board three-phase transformer. The transformer secondary supplies ac for the dc systems, involving thyristor control of the traction motors.

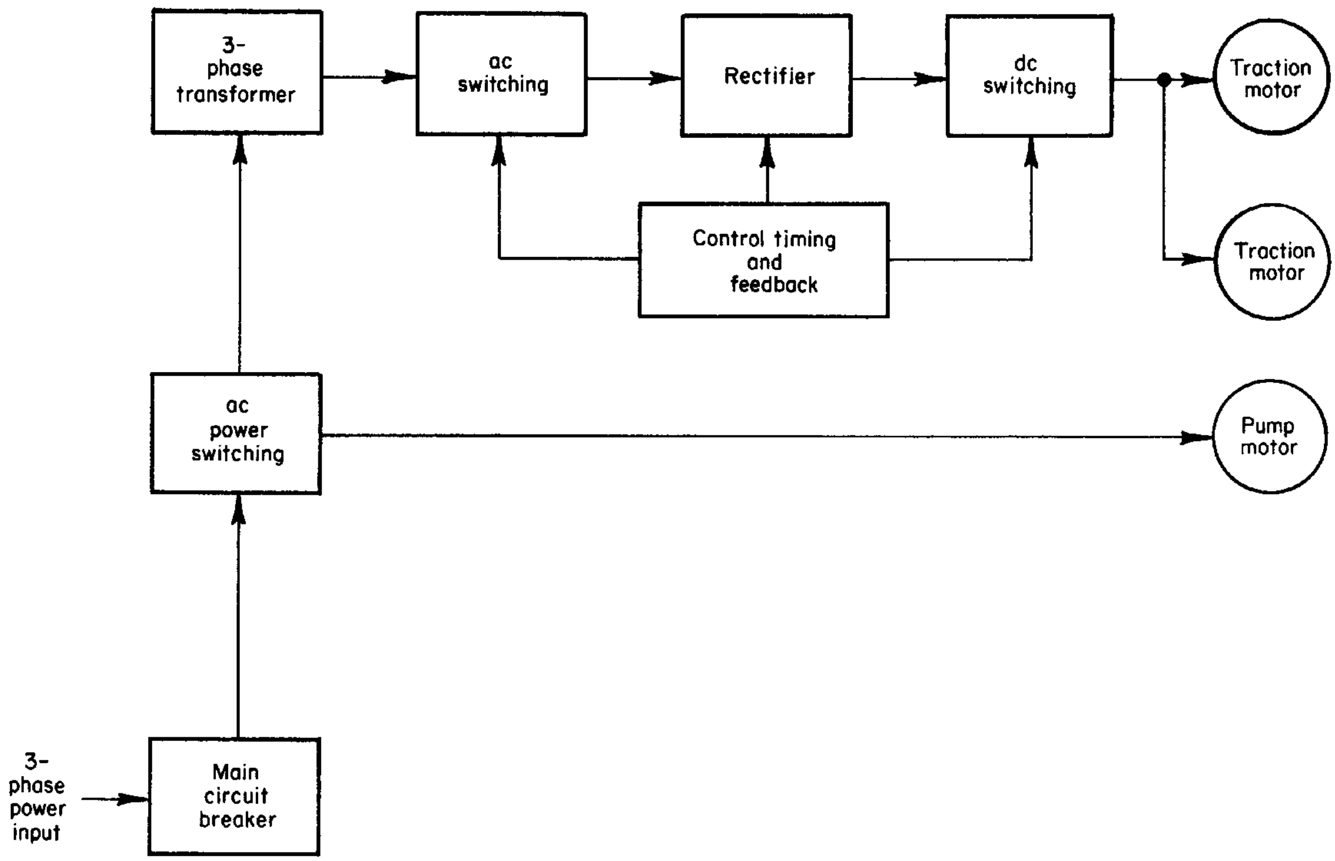

Figure 14.13.—Block diagram of ac-dc shuttle car.

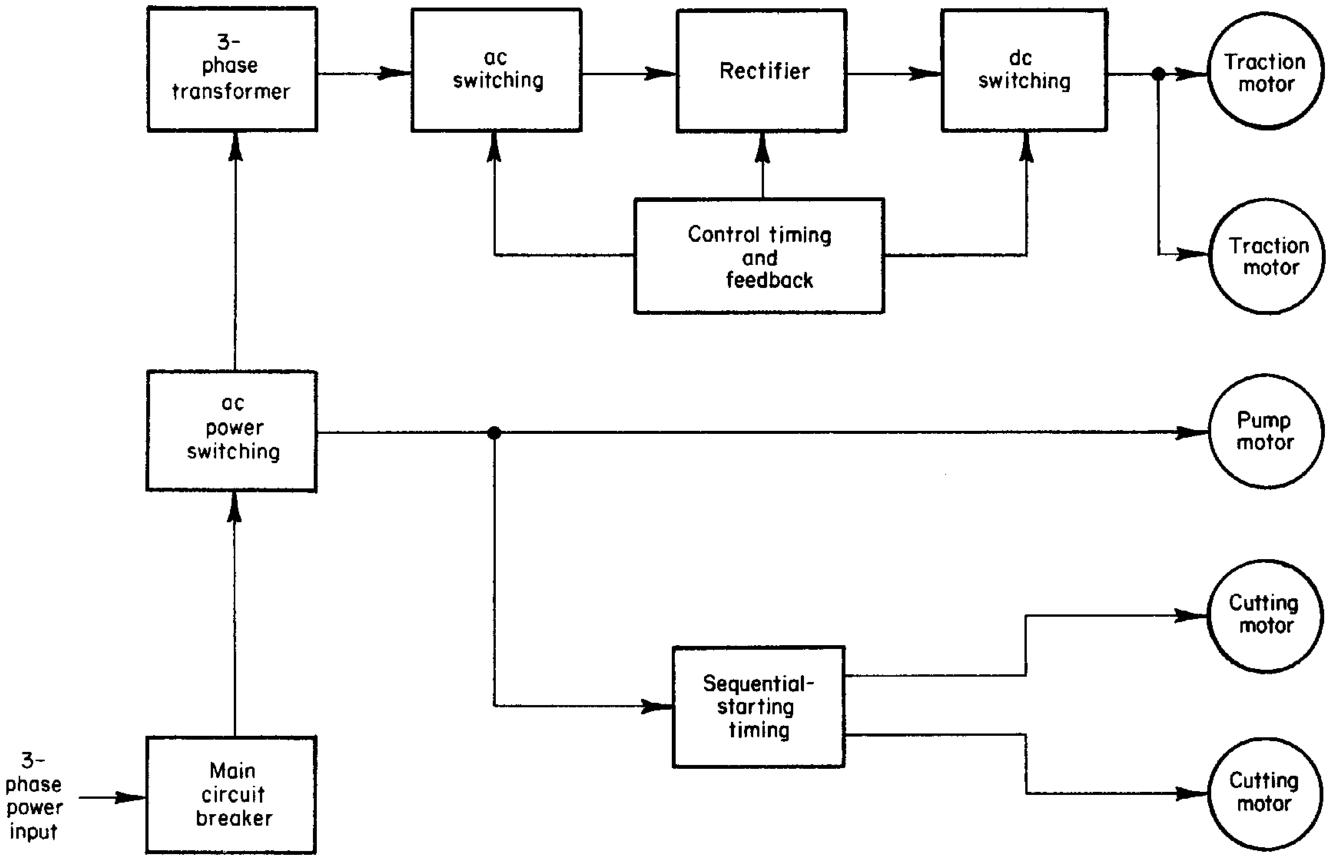

Figure 14.14.—Block diagram of ac-dc continuous miner.

ALTERNATING CURRENT APPLICATIONS

Full variable speed cannot be accomplished with straight current or voltage control on ac induction motors. However, as motor speed is a function of the applied power frequency, speed control can be obtained by varying this frequency (*12*). A simple representation for one type of variable-frequency ac drive is shown in figure 14.15, and an elementary inverter circuit is given in figure 14.16. The incoming ac is rectified then filtered by capacitance to provide a high-quality dc. An inverter then converts the dc back to ac. By controlled switching of the thyristors, an alternating square-wave signal of the desired frequency can result. This can be used directly or filtered, as in the transformer in figure 14.16, to provide a sinusoid. Three-phase ac output can be obtained by employing three inverters and firing the inverters so that 120° timing is available.

A direct adaptation of this technique has been used on production mining shovels as shown in the block diagram in figure 14.17 (*20*). This is reported to give better performance than Ward-Leonard dc motor control since machine characteristics are a function of the power electronics and not the motor. Another application of thyristor control of ac induction motors is in conveyor starters.

Across-the-line starting of three-phase squirrel-cage induction motors on in-mine belt conveyors can be very detrimental. Belt conveyor installations can call for rather high horsepowers, and the resulting high starting current

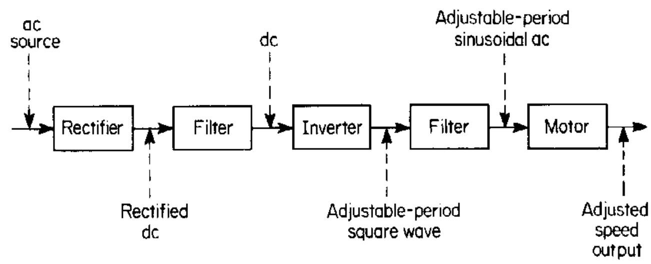

Figure 14.15.—Simple variable-frequency control.

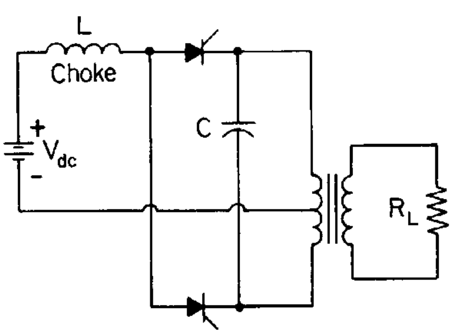

Figure 14.16.—Elementary inverter circuit.

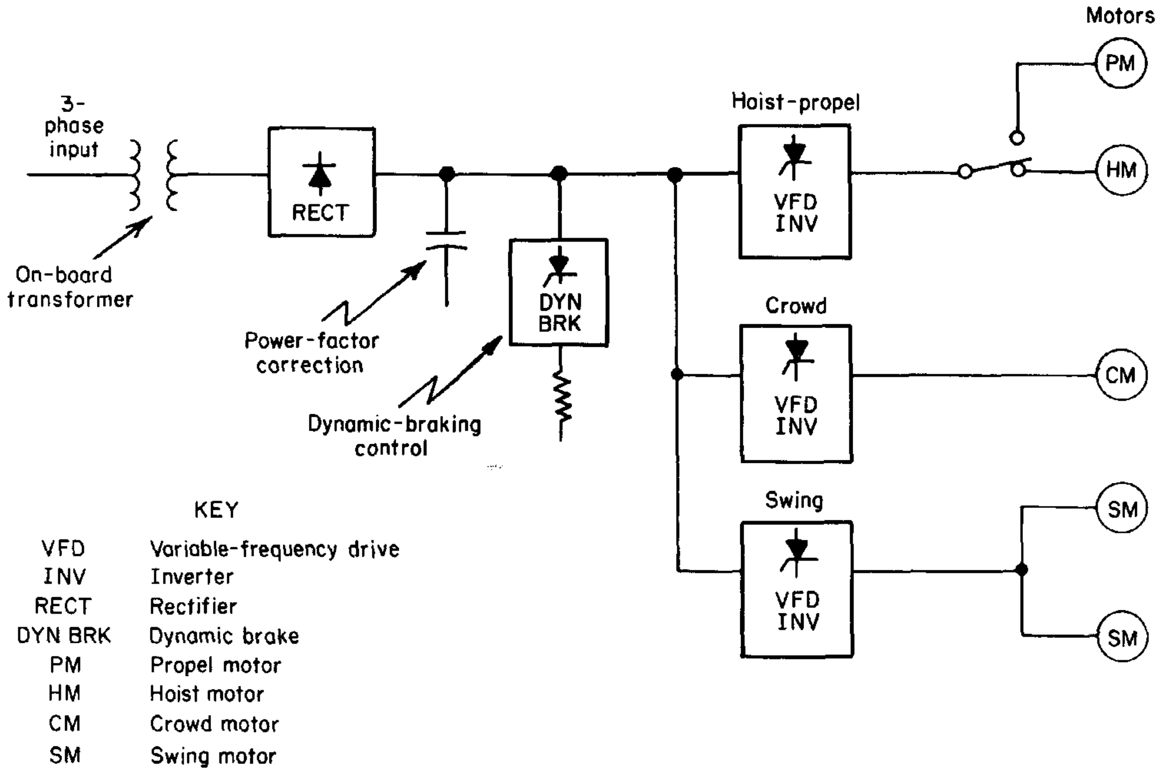

Figure 14.17.—Use of variable-frequency drive on production mining shovel.

(three to eight times full load) can produce protective-relaying problems and large voltage drops. The latter can cause a motor torque decrease, which in turn can hamper belt conveyor acceleration.

In the past, wound-rotor motors with step starters were used extensively on belt conveyors to overcome these problems. The step starter is essentially a bank of resistors that is connected to the rotor winding (see chapter 6) and allows the motor to start with a high resistance-to-reactance ratio for limited starting current and high starting torque. The external resistance is then decreased in steps, each decrease resulting in an increase in motor speed. At full speed, all external resistance is shorted out, and the wound-rotor motor operates like a low resistance-to-reactance design with corresponding high efficiency. Step starters for belt conveyor applications require large-capacity switching and contacting equipment. Historically, these electromechanical components have been maintenance problems, and the brushes and slip rings of the motor can be a continual source of difficulty.

On the other hand, controlled acceleration and limited starting current can be achieved using squirrel-cage induction motors with solid-state starters. Compared with wound-rotor motors, motor and starter maintenance is lower, and the following advantages are gained:

Reduced belt and splice tears;

Decreased stress on mechanical power-transmission components, resulting in increased life;

Elimination of some mechanical components, such as extensive gear trains, clutches, and so on;

Decreased belt slippage, thereby reducing belt burns and removing impulse stresses;

Custom design for special application; and

Increased motor life.

All these features have been validated by experience.

Control Systems

All three-phase solid-state belt starters in common use today employ reduced-voltage motor starting. The technique is based on the principle that torque developed at the motor shaft is proportional to both the rotor current and the square of the terminal voltage. Therefore, if the terminal voltage is reduced to a given level to correspond with a desired torque value, then motor current is limited. Thyristors are used to control power turn-on to the ac motor, and voltage is reduced by not firing the thyristors until some angle past the source voltage zero. This effectively reduces the average voltage across the motor terminals to a value less than the source voltage. Firing control is by one of two means: open loop or closed loop (*2, 5, 31*).

Without feedback, the open-loop systems are the simpler designs, and the terms *voltage ramp* and *current ramp* are often used to describe these systems. The controller element (G in figure 14.9) brings the voltage or current up to maximum in a predetermined time period independent of the motor conditions. The maximum value of current is not limited by the starter, but rather by the line voltage and motor characteristics. This limits the use of open-loop control to motors or power systems where the starting current does not create problems.

In the closed-loop systems, some reference signal or feedback from the motor (H in figure 14.9) is compared with a reference to adjust the controller output. The result is more control over the motor starting characteristics and allowances for loading or operating conditions. In the case of static belt-conveyor starters, and feedback signal typically corresponds to either motor line current or motor speed. The line-current feedback control schemes can be divided into current-limit or current-regulated types. Motor-speed feedback techniques are also called linear acceleration or tachometer control.

The basic current-limit scheme is shown in figure 14.18. Feedback control is used to compare an adjustable reference signal (the current-limit setting) with a signal that represents motor current. Motor current is monitored by current transformers, preferably in all three lines (*2*). The torque being developed at the motor shaft is then computed by the starter circuitry from the current being used. As a result, the current is limited to a preset value by the thyristors, and the resulting belt acceleration is nonlinear (almost logarithmic) and is a function of belt load. When the belt is empty, a full-voltage start can occur.

In current-regulated control, the problem of a full-voltage start is removed by the addition of a second reference supplied to the control-system summing point. This is an adjustable ramped reference signal, which is usually a linearly increasing voltage with time. The starter circuitry now restricts motor current to rise over a preselected time period (the acceleration-time setting in figure 14.18) to the current-limit setting. Motor acceleration is smoother than current-limit control but is still somewhat nonlinear.

In basic linear-acceleration starters, a tachometer generator is placed on the motor shaft, and its output provides either a feedback signal proportional to motor speed, which is compared with a ramped reference, or a feedback signal proportional to motor acceleration, which is compared with constant preset reference. As a result, motor voltage is adjusted to limit current, such that a specified rate of acceleration is obtained. Belt acceleration is linear and rather constant regardless of load, this is, whether the belt is empty or full. Some models combine the linear-acceleration feature with overriding current-limit and current-regulation control, as shown in figure 14.19. Thus, the starter tries to linearly accelerate the motor, but the current-regulation control keeps the rate of current rise within preset limits and the current-limit control keeps the maximum value of current below a certain level. Controlled deceleration is also available in some starters, but the deceleration time must be longer than that for the drive mechanical inertia.

The start-stop circuitry is often relay controlled. (This is not shown in figures 14.18 and 14.19.) When off, relay contacts clamp the thyristor gates and the input of the control-system amplifier. Under a start command, the relay contacts sequentially unclamp the gates, allowing the control system to start the acceleration cycle (*2*). Because the thyristors do not physically disconnect the load, a circuit breaker is also provided on the incoming line for this purpose as well as for short-circuit protection.

Control System Design Considerations

When starting an induction motor on a belt drive, three major areas need to be considered: the power system, the motor, and the mechanical equipment (speed reducers, the belt, and so forth). From the standpoint of power-system protection and optimum operation, the motor should be started with as little current as practical to

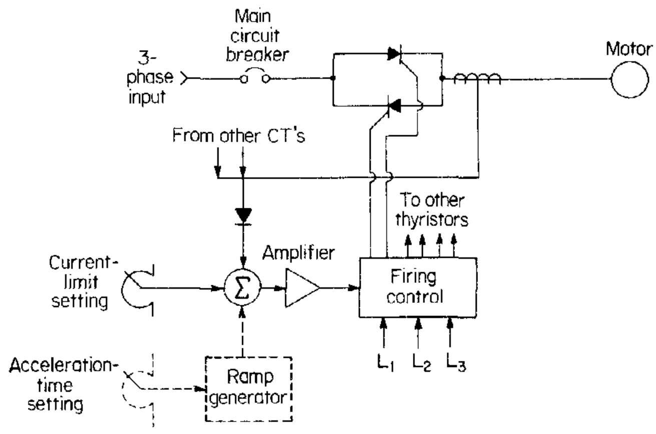

Figure 14.18.—Simplified diagram of current-regulated static belt starter.

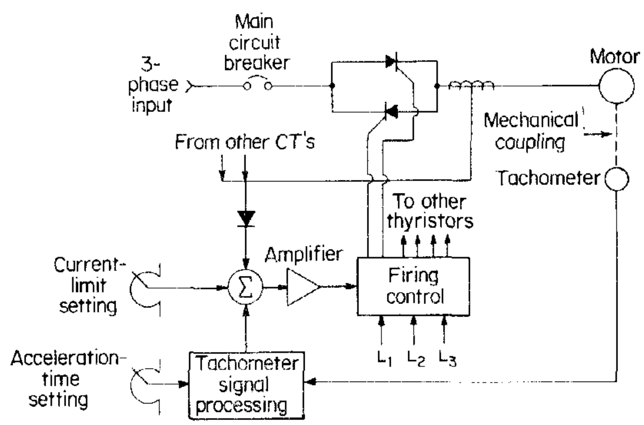

Figure 14.19.—Simplified diagram of linear-acceleration static belt starter.

minimize the overcurrent and undervoltage effects on the power system. With motor protection and operation taken as the priority, the philosophy would be to bring the motor up to full speed as quickly as possible. The reason here is to prevent insulation breakdown caused by rotor overheating. The optimum situation for protection of the mechanical equipment would be a smooth, easy acceleration of the belt. This reduces the wear and tear on the gears and the belt from excessive or uneven torque.

Unfortunately, because the design considerations are mutually exclusive, all three areas cannot be completely protected. Protection of the power system and mechanical components calls for an extended low-level starting current, whereas protecting the motor from overheating requires a rapid-rise high-level starting current. The area that is given priority helps determine which control system should be used (*31*).

When solid-state belt starters were first introduced, protection of the power system and conveyor components was the primary concern. Thus, early starters used a current-regulated or linear-acceleration scheme for smooth starting of the belt and an overriding current limit to protect both the power system from overcurrent situations and the conveyor components against overtorque problems. Either control system gave the same basic static-starter advantages given previously. Linear-acceleration controls had a special advantage in installations where belt length was constantly changed, such as for panel belts, as little or no adjustment of the starter circuitry was necessary. With current-regulator control only, adjustment may be required for any conveyor change, but current-regulator controls do have the advantage of simplicity. Most manufacturers offered both options in their equipment.

Many of these early static starters worked well, but some problems occurred because of initial design flaws. Foremost was the situation where either the current limit was set too low or the belt was overloaded. In either case, the motor would not reach full speed before the rotor overheated because of the somewhat limited acceleration imposed by the current limit, and either the thermal protection of the motor or a maximum current time limit would shut down the motor. Another common problem area was strictly associated with linear-acceleration models: high maintenance requirements of the tachometer generator and its wiring.

These problems can directly produce conveyor belt downtime and, in turn, a large loss-of-production cost for the mine. Thus, a recent trend with many manufacturers has been toward more reliability and a simpler control system, and the common design is an open-loop voltage ramp scheme without a current limit. Typically, the voltage starts at roughly 60% of the line voltage then gradually increases to 100% in a preset time limit. Because there is no current-limiting capability, thermal overload sensing in the motor is almost a necessity, and simple circuitry for tripping the main circuit breaker when the overcurrent situation has exceeded a specified time limit is usually recommended. This design is reported to have a lower breakdown rate than other more complicated systems, and it also allows for full locked-rotor torque and full locked-rotor current when needed by the conveyor. However, the simplicity does negate some of the advantages of soft starting, particularly limiting starting currents and voltage drops.

Motor Designs

There are four concerns when selecting a suitable motor and motor characteristics: providing sufficient starting torque, drawing symmetrical line current with minimum disturbance to other equipment, optimizing the thyristor characteristics, and providing an acceptable stress level on motor insulation under any condition (*39*). In terms of characteristics, NEMA design B appears to be a good match for linear-acceleration control because of its greater locked-rotor time, and NEMA C seems best suited for open-loop and current-regulated applications from its greater starting torque at a given current (*31*). However, the main consideration is thermal because the motor must accelerate to full speed for normal operation. Energy continues to be stored in the motor until the internal fan velocity can dissipate the heat faster than it is generated. Hence, NEMA C motors are often recommended for all control schemes because less energy output is needed to provide the same output starting torque. Some manufacturers have even recommended the NEMA design D because they feel the advantage of its very high starting torque outweighs the disadvantage of its inefficient full-speed characteristics.

Some motor manufacturers are producing squirrel-cage machines specifically designed for solid-state starting. They have higher quality insulation, larger fans, larger rotors, and are built on a larger frame per given horsepower. These motors can have an energy-in-heat-out balance at 75% of full speed, whereas for the standard motor, it can be as high as 90% of full speed (*39*).

Experience has shown that conventional thermal-overload relays do not provide adequate protection during the long starting times caused by static control (*2, 30*). Overtemperature detection devices installed in the motor are the best way to provide thermal protection. Of these, solid-state temperature sensors, installed internally in the motor windings, with a load-current detection backup, appear to provide the best technique (*2*).

Thyristor Configuration

Both thyristor configurations, unidirectional and bidirectional, are available for static belt conveyor starters, but there are difficulties with unidirectional control. Some problems are caused by the higher harmonic content that unidirectional control adds to line current when in the control mode. When applied to static induction motor starters, the harmonics tend to create excess heating but mainly during acceleration. If sufficient cooling time is allowed between starts and acceleration times are minimized, the heating problem is reduced considerably but not eliminated (*5*).

However, the main difficulty with unidirectional control is allowing dc to flow in the motor, and two situations illustrate the problem (*5*). During a motor ground fault, the diodes can rectify current, and dc can flow through the grounding resistor and grounding conductor, and not be detected by the zero-sequence ground-fault relaying. Thyristor failures are rare but are always in the shorted mode. If thyristor fusing is not available, there also exists a low-impedance path for dc through the motor windings and the other two diodes. Either case is a problem whether the starter is on or off.

With the harmonic and dc difficulties presented by unidirectional control, it appears that bidirectional control is better despite its high cost. Some additional advantages are gained (*39*). Two thyristors are always in series from line to line, and hence, less stress is given per thyristor by transient and long-term overvoltages. Because of symmetry, smoother control is provided under varying conditions, and finally, motor windings and cable power conductors are near neutral potential when the thyristors are off.

Firing Circuits

When discussing the basics of thyristors, a single gate pulse was shown to fire the device (figs. 14.3, 14.20*A*). However, in practice, there are many instances where a single gate pulse would fail to fire the thyristor, for example, if there is low ambient temperature or low line voltage or when the devices themselves are old (*39*). Two types of firing systems overcome this problem: sustained pulse (fig. 14.20*B*) and multi-pulse (fig. 14.20*C*). Generally, a pulse transformer is used in the multipulse system as isolation between the power circuit and the control circuit. Numerous pulses, each capable of turning the thyristor on, are applied to the gate to keep the thyristor on during

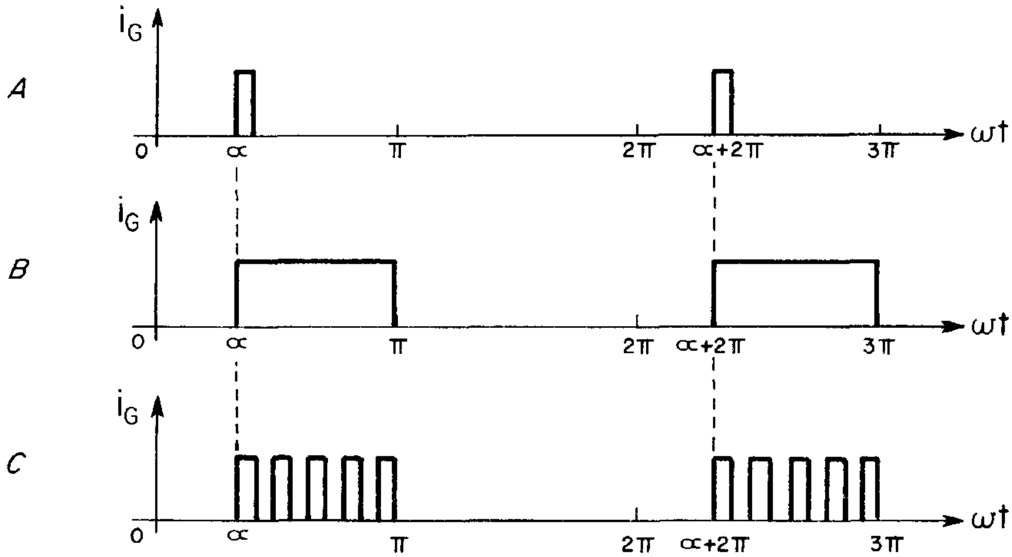

Figure 14.20.—Types of thyristor firing pulses.

its intended conduction period. The other technique, also termed the *dc firing system*, maintains a continuous gate current during the desired conduction period. This helps ensure that the thyristor will turn on, and turn on completely by continuous stimulation of the gate.

Two parameters of the firing pulse are important: the level of the current and the current duration. If the pulse is only of minimal current and duration, it may cause conduction only in a limited area of the thyristor or no conduction at all. In particular, a small conduction area coupled with a high rate of change (di/dt) of anode-to-cathode current can result in concentrated heating and possible failure of the thyristor. To ensure conduction of the whole interface of the thyristor, a technique called hard firing is used in conjunction with either dc or multipulse firing. It consists of a high-level initial pulse with a steep wave front and enough duration to operate the thyristor at near the maximum input power level of the gate. The high initial gate current floods the conduction region of the thyristor and turns it on completely. After the device is conducting, a sustained dc pulse or multipulse keeps the thyristor on even if the anode-to-cathode current approaches zero. This prevents thyristor turnoff during the critical end of the conduction cycle. Hard firing, thus, provides more consistent operation and reduced failures of thyristors, and allows the use of off-the-shelf devices (*9, 39*).

Thyristor Ratings and Protection

Thyristors are susceptible to damage from overvoltages, overcurrents, and rapid changes in voltage (dv/dt) or current (di/dt) with respect to time. Because these are rather common occurrences in mine power systems, thyristors should be protected from each of these phenomena if premature failure is to be prevented. The best way to protect the thyristors against the effects of high di/dt, which has already been mentioned, is proper design of the firing pulse. Two avenues are typically used to provide the other protection, thyristor ratings and protective circuitry, and a typical protection arrangement is shown in figure 14.21.

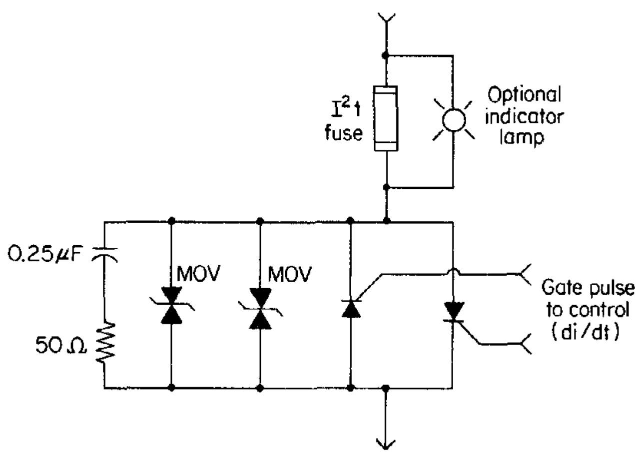

Figure 14.21.—Thyristor protection for static belt starters.

It is standard practice to select a thyristor voltage rating at 2.5 times the nominal line-to-line system voltage. For bidirectional control, this is quite adequate considering the 5-pu utilization transients discussed in chapter 11. However, the thyristors themselves are a source of abnormal transient overvoltages, and some engineers prefer to use a device voltage rating no less than 3 or 3.5 times the system voltage (*5, 39*).

Calculation of the required thyristor continuous-current rating from the load current should be easy if the thermal properties of the heat sink are known. Yet during starting, which is the worst stress case, the device is called upon to deliver much more current. The conventional approach for overload protection of the thyristor is a 30-s (overload) rating of 300% continuous current. Some engineers do not believe this is adequate, because the induction motor starting current may be as high as six to eight times the running current; hence, certain manufacturers have selected thyristors with 300% of the continuous-current requirements and a 30-s overload rating at 500%. Others feel that it is better to have an overload rating based on horsepower, 25 s at 600%, or 100 s at 300% of continuous current (*39*). Thus, the thyristor current-rating selection is not a straightforward matter, and special consideration is sometimes required for individual cases (*5*).

Some short-circuit protection is afforded by oversizing (300%) the continuous-current rating. However, past practice was to provide additional protection with semiconductor protection or I^2t (let-through energy) fuses with one fuse in series with each device. The philosophy was that, even though these fuses were expensive, the replacement cost for the thyristor was even more so. The fuse continuous-current rating was matched to that of the thyristor, but the I^2t rating was less than the thyristor I^2t capability. There were instances where the thyristor continuous-current rating was set at 150% of needed and then the I^2t fuse was matched (*39*). However, this proximity was found to lead to nuisance fuse activations, especially during acceleration.

Recent practice for overcurrent protection of the thyristors is to eliminate or oversize the I^2t fuses. When the fuses are eliminated, the thyristors are sufficiently oversized to withstand most long-duration situations, and thermal-overload sensors are used on the thyristor heat sinks. In cases where the fuses are used, the fuse current limit is set very near the failure point of the thyristor. Both of these schemes are intended to reduce starter downtime due to blown fuses, even to the point of sacrificing the main power thyristors. The general feeling among mine operators is that the production saved by eliminating false fuse tripping is worth an increased liability of the thyristors' failing. This situation has become justified because of increased production cost associated with belt downtime and the decreased cost of the thyristors.

A high rate of forward-voltage increase can turn a thyristor on, even with a zero gate current (*9*). Inductive loads, such as induction-motor belt drives, can present such dv/dt problems (*5*). The common solution is RC snubber networks across each pair of devices as in figure 14.21. Typical values for R_s and C_s are 50 Ω and 0.25 μF, but values must be selected so that the dv/dt allowed is less than the minimum specified by the device (around 100 V/μs). To ensure that transient overvoltage does not destroy a thyristor, metal oxide varistors (MOV's) across each device are sometimes specified. The maximum crest allowed by the MOV is coordinated with the maximum voltage rating of the thyristor, allowing the usual 20% safety margin.

The preceding paragraphs have not only introduced the basics of solid-state motor control but also indicated the special difficulties of applying static-starter concepts to mine belt conveyors. This is one firm example of the benefit of technology to industry. Even with the increased internal complexity, the end result has been an overall increase of belt-conveyor system reliability.

STATIC PROTECTIVE RELAYING

Most engineers define the term *relay* as an electrically controlled, usually two-state, device that opens and closes electrical contacts to effect the operation of other devices in the same or another electrical circuit (*15*). Historically, one important relay use has been the protection of people and electric circuits from electrical hazards. The operation of an electromagnetic protective relay was presented in chapter 9 but is repeated in figure 14.22 (*36*). This particular relay uses two actuating quantities (voltage and current) that directly affect the status of the relay contacts. Whenever current and/or voltage exceeds a predetermined level, the current sensor and/or voltage sensor (that is, CT and/or PT) outputs cause the relay to close its contacts through electromagnetic attraction or electromagnetic induction. The closed contacts permit current to flow through the trip coil, tripping the circuit breaker. Electromagnetic-attraction relays in common use are the solenoid, clapper, and polar types (*38*). The typical electromagnetic-induction relay is the induction disk.

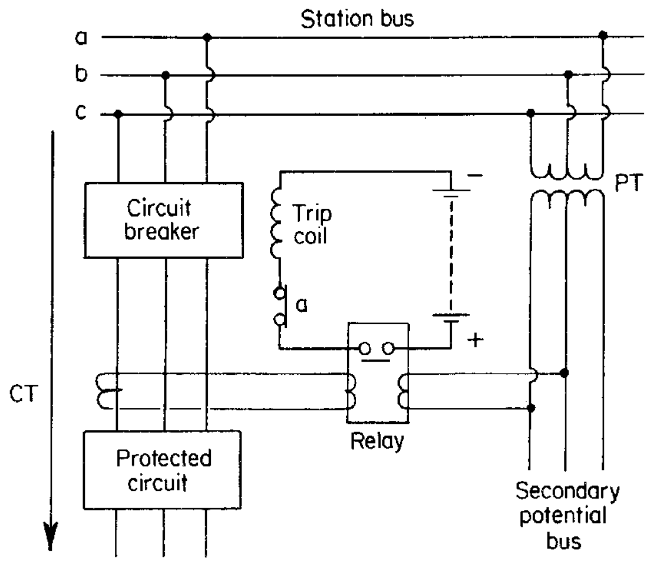

Figure 14.22.—Protective-relay connections.

The basic concept in the design of solid-state relays (again also called static relays) is to replace the mechanical contact device with a solid-state device. The solid-state device is inserted in the trip coil circuit and controlled by the sensor circuit. When unactuated, the solid-state device acts as a very large resistance in the trip coil circuit, limiting the current through the trip coil to a very small value (known as leakage current), which is incapable of tripping the associated circuit breaker. When actuated, the solid-state device acts as a very small resistance, which allows ample trip coil current, thus tripping the circuit breaker. The solid-state devices commonly used are the transistor, the thyristor, and the triac.

OPERATION OF SIMPLIFIED SOLID-STATE AND HYBRID RELAYS

The electromagnetic relay of figure 14.22 is represented schematically by figure 14.23, where the current and voltage input have been replaced with a manual push button, and the trip coil (load) circuit is supplied with a 120-V, 60-Hz power source. The static relay differs from this in that the contents of the dashed box of figure 14.23 are replaced by the semiconductor device shown schematically in figure 14.24 (*16*). The load current now flows through the common terminal of the semiconductor device, which is common to the trip (contact) and sensor (control) circuits.

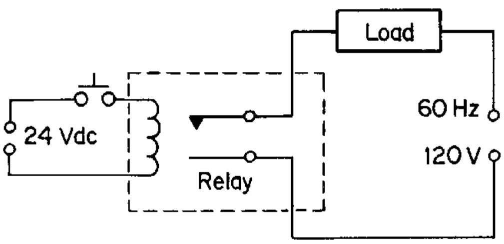

Figure 14.23.—Simple electromechanical relay.

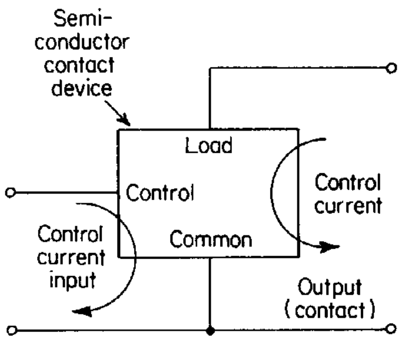

Figure 14.24.—Simple static relay.

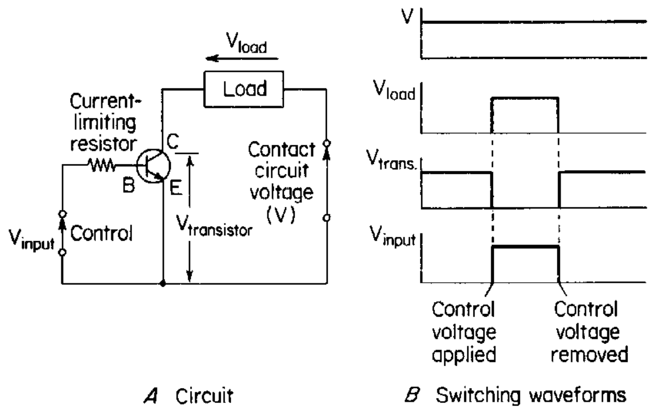

Figure 14.25.—Transistor used as relay.

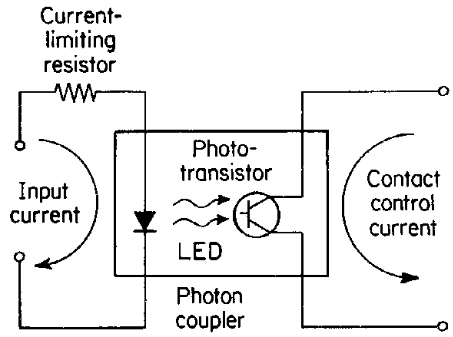

Figure 14.26.—Optical transistor as relay.

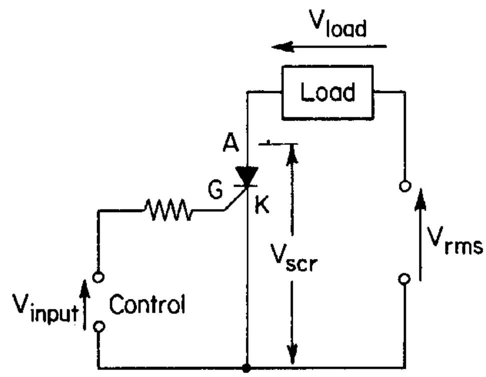

Figure 14.27.—Thyristor used as relay.

If the static device is an npn transistor, the circuit of figure 14.25*A* results. Since the transistor is a current-controlled device, if zero input (control) voltage is applied to the transistor base, no current will flow into the base. Because base current is required for current flow from collector to emitter, no current will pass through the load. Hence, the contact-circuit voltage supply will not be dropped across the load, and the voltage must therefore be across the collector-emitter terminals of the transistor. A positive input voltage applied to the transistor base causes a positive current to flow into the base. This base current is the controlling factor that permits a large collector-to-emitter current to flow through the transistor. When sufficient base control current is supplied, the collector current will increase until essentially all contact-circuit voltage is across the trip coil (load). Thus, the switching characteristics of figure 14.25*B* are obtained.

A drawback to the circuit in figure 14.25*A* is the lack of electrical isolation between the control and the trip circuits. One isolation method is to use the photon-coupler circuit shown in figure 14.26 (*16*). Here, the transistor performs the same function as that in figure 14.25*A* except that it is light controlled, with a high-intensity light (from the LED) acting as a large base current. The light impinges on the phototransistor base region, allowing current to flow from collector to emitter. Without an input to the LED, no light is produced and consequently no collector-to-emitter current flows.

Replacing the transistor with a thyristor results in the circuit of figure 14.27 (*16*). This device has been described earlier in the chapter; thus, its operation here should be clear.

The triac, which is a contraction for triode ac semiconductor, is a bidirectional solid-state device that acts like two thyristors connected back-to-back as shown in figure 14.4*A*. The triac provides full-wave voltage control in one solid-state structure with only one gate control, as shown in figure 14.28*A*. The load-voltage characteristics, as shown in figure 14.28*B*, are very much like those exhibited by the thyristor. The single structure has heat-dissipation limitations, restricting the triac to small-current applications (*9*).

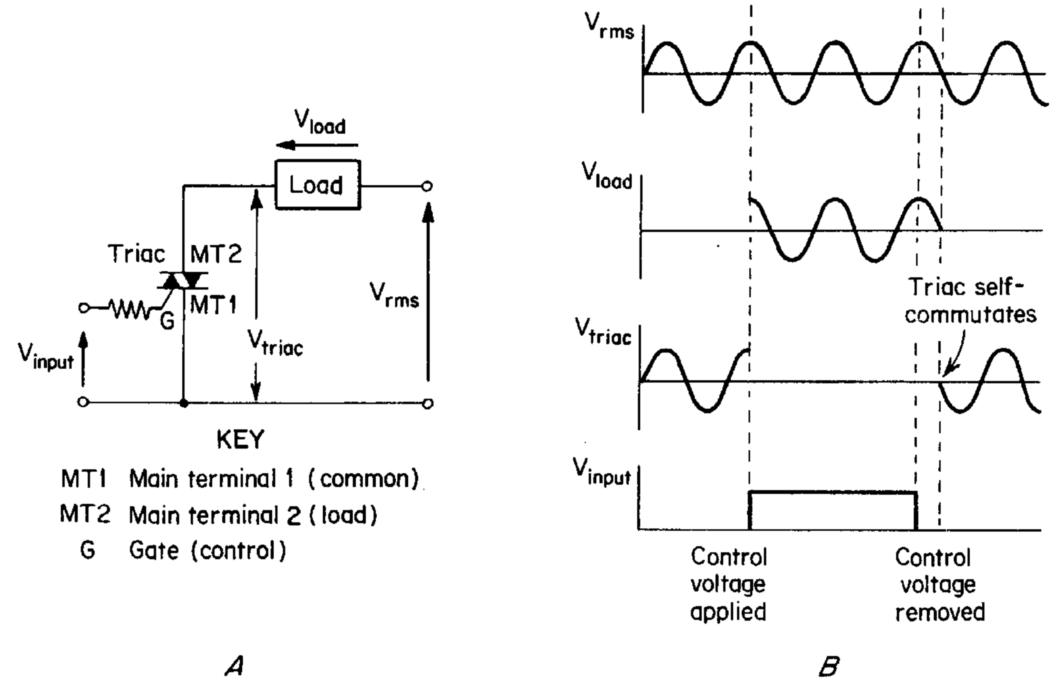

Figure 14.28.—Triac used as relay.

The output and input circuits of the triac and thyristor relays again have a common terminal between them. Where input-output isolation is required and a solid-state relay is also desired, hybrid relays are commonly used. Hybrid relays employ a relay with a low-power operating coil in either the input or output stage and a solid-state device functioning in the other stage. The hybrid relay shown in figure 14.29*A* has the reed relay in the input stage (*8*). Switching the reed relay activates the gate-control input, which fires the triac. The hybrid circuit of figure 14.29*B* uses a solid-stage input stage that can react to the power network conditions. When a predetermined threshold is reached, the solid-state stage sends a current through the relay coil that will pick up the relay contacts.

Reed relays alone, when compared with static devices, have the advantages of simplicity, low cost, high reliability, and low maintenance. The best reed contacts are mercury wetted and provide bounce-free operation. Dry reed relays with heavy-duty silver contacts will carry 2 kW with a maximum of 30 A. They can withstand 5 G (50 m/s^2) when mounted correctly. The characteristics of electromagnetic (attraction and induction types), reed, transistor, and thyristor (including triac) relays are summarized in table 14.1 to allow a quick comparison (*37*). This table shows that static relays outperform electromagnetic relays in almost every case.

Before a detailed comparison of static and electromechanical relays can be made, it is necessary to determine whether static relays are capable of performing the same functions as electromechanical relays. The latter are usually classified according to the actuating quantity (see chapter 9), and the most common types used in mining are overcurrent, overvoltage, and undervoltage. The overcurrent or overvoltage relay is designed to trip when a predetermined current or voltage threshold value has been exceeded. The undervoltage relay is designed to trip whenever the measured voltage falls short of a predetermined level. Static relays can be designed to respond to these actuating quantities as well as many others (*37*).

As an example of static operation, consider the overcurrent relay shown in figure 14.30. The ac from the CT is forced through a resistor, which produces an alternating voltage across the input rectifier bridge. This voltage is rectified by the bridge and applied to the series RC

Table 14.1.—Typical electromechanical and static relay characteristics

Parameter	Electromechanical		Static	
	Electromagnetic	Reed	Transistor	Thyristor
Input ....W..	1–3	0.1–3	0.010	0.040
Switching capacity ....W..	30	≤20	50	1,200
Power gain	10–30	7–200	5,000	30,000
Continuous carrying current ....A..	5	1.0	1	5
Delay ....ms..	10	1–2	0.020	0.050
Number of contacts	≤6	≤6	1	1
Maximum operations	10^7	10^7	([1])	([1])
Ambient temperature range ....°C..	-5–70	-5–55	-20–100	-20–100
Affected by vibration	Yes	Little	No	No
Affected by corrosive atmosphere	Yes	No	No	No

[1] No limit.

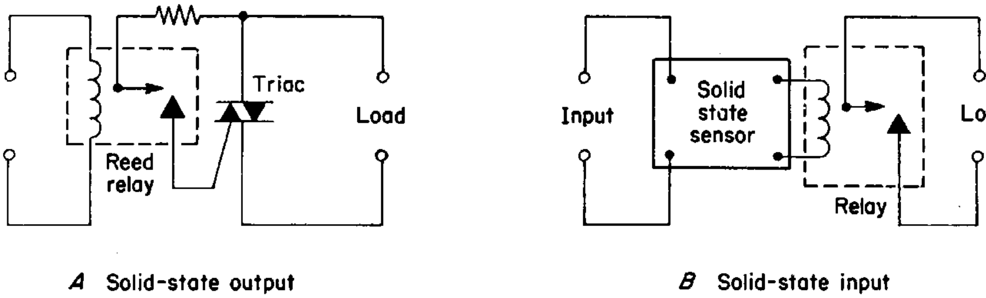

Figure 14.29.—Hybrid static relays.

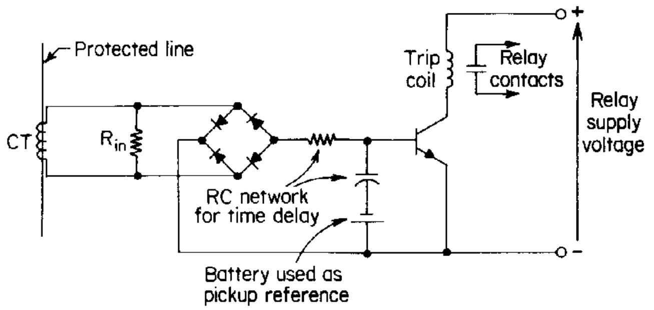

Figure 14.30.—Simple overcurrent static relay.

elements in the base-emitter circuit. When the capacitor voltage exceeds the battery voltage in the base-emitter circuit by a sufficient amount, the transistor begins to conduct current. Thus current picks up the trip coil, tripping the circuit breaker. A time delay is achieved in this circuit by virtue of the series RC time constant associated with the base-emitter circuit. The other actuating quantity operations can be described in a similar manner.

STATIC AND ELECTROMECHANICAL RELAY COMPARISON

A large amount of literature has been devoted to the advantages and disadvantages of static relays. The following detailed comparison summarizes this literature and discusses the advantages and disadvantages associated with the differences between static and electromechanical operation. A listing of the references used follows each topic of the discussion.

Relay Speed

Table 14.1 shows that static relays are much faster than electromechanical relays. This speed difference is because there is no mechanical motion in solid-state devices. High speed is advantageous since it gives the static relay the capability to turn on at zero-voltage crossing. This eliminates much of the radio frequency interference (RFI) caused by capacitance switching or prestrike that is found with mechanical opening and closing of circuits with large capacitance values. Controlling RFI is important because digital circuits are easily affected by such interference. RFI is not present when static devices open, since for example, the thyristor and triac always turn off at zero current. The variability in the closing and opening time precludes this type of control in electromechanical relays. Reset times are also much shorter for static relays. An extremely inverse electromechanical relay may require 15 to 25 s to reset, while a static relay resets in a few cycles (*1, 8, 18, 22, 26, 37*).

Relay Power Requirements

As shown in table 14.1, static relays require far less input power than electromechanical relays. Input power delivered to any one static device may come from a wide range of voltages, including ac and dc. Electromechanical relays are normally limited in the voltage range they can accept for proper operation and require different coils for ac and dc operation. Static relays are also capable of using a range of ac and dc power supplies in their tripping circuit (*3, 18*).

Heat production in the solid-state relays results from the 1.0- to 1.5-V drop across the device in the *on* position; thus, power consumption is limited by the current required for tripping operation. Static relays used for protective-relaying applications usually require only 5 A, which is insufficient to produce enough heat for concern. Relays carrying between 5 and 25 A, usually referred to as *power relays*, may require an auxiliary heat sink or special mounting arrangements. Above 25 A, a relay becomes a contactor, even though its function does not change; hence, a static relay above 25 A is considered a solid-state contactor. Electromechanical relays require no heat sink since, in an underrated condition, their contact resistance measures in the milliohm range (*1, 8, 18*).

Temperature

Typically, the maximum allowable continuous junction temperature for thyristors and triacs is 120° C, while electromechanical relays may be used in ambient temperatures that exceed this value. Temperature has little effect on the electromechanical characteristics; however, solid-state devices exhibit different characteristics at elevated temperatures. If subjected to these temperatures for long periods, their characteristics may even change permanently. Short-range thermal compensation can be accomplished by the use of thermistors in the design or, more commonly, differential-stage construction. The long-range effects of elevated temperature are guarded against by preassembly and postassembly heat soaking and testing (*1, 18, 37*).

Power Transient Response

The term *power transients* refers to transients on the power system being monitored and transients on the trip circuit control power. Static relays are fast-responding devices, and they rarely trip on power-system transients unless set to do so. Electromechanical relays, however, may trip erroneously owing to either ratchetting or overtravel. As was discussed in chapter 10, mechanical inertia causes overtravel of the induction-disk or induction-cup relay even after the actuating quantity has been removed. If the relay is not set properly, overtravel may cause the undesired tripping of a backup relay, thus isolating a sound part of the power system being protected. Static relays have negligible overtravel, which precludes this type of error and makes it possible to have closer time settings on backup relays to provide faster protection. Table 14.2 shows a comparison of the time-margin calculations for a backup time-overcurrent relay. These calculations assume that the circuit breaker operates within 3 to 8 cycles after pickup has been exceeded in the primary relay. A tolerance margin is allowed for the normal variation in the time-current curves. The smaller tolerance margin used in the minimum time calculation corresponds to establishing the trip time by testing, while the larger margin assumes a time selected using the time dial settings on the relay. For practical purposes, a 0.3-s margin is used for the electromechanical and a 0.2-s setting for the static relay. The difference is due to the negligible overtravel of the static device (*19, 22*). Ratchetting is the accumulation of overtravel due to successive faults or sequential starting of motors. Static relays do not exhibit ratchetting because of their negligible overtravel and short resetting time (*22*).

Table 14.2.—Time-margin comparison between electromechanical and static relays, seconds

Operation	Electromechanical		Static	
	Min [1]	Max [2]	Min [1]	Max [2]
Breaker time	0.05	≤0.13	0.05	≤0.13
Overtravel	.10	≤.10	.01	≤.01
Tolerance margin	.07	≤.17	.07	≤.17
Total	.22	≤.40	.13	≤.31

[1] 3 cycles. [2] 8 cycles.

Trip circuit control-power transients may cause false operation of an electromechanical relay if that device relies on the control power to keep it closed under normal circuit conditions. Voltage spikes on the control-power circuit can also cause unwanted tripping of static relays from two separate situations. If the transient is of sufficient magnitude, the solid-state device may break down, permitting conduction. This problem has to a large extent been overcome by triacs with blocking voltages of 1,000 V and higher. However, if the voltage spike has a high rate of rise, the inherent capacitance of the thyristor or triac will allow a momentary current to flow (that is, i = Cdv/dt). With sufficient magnitude, this current will act as a gate current and turn on the device. This dv/dt problem is similar to that discussed for static belt conveyor starters and is being overcome by including filters capable of absorbing the energy contained in these transients. On the other hand, dv/dt current is not a problem with electromechanical relays. Breakdown voltages of these relays are normally many times their operating voltage because of the air gap between the mechanical contacts (*1, 18, 26, 30*).

Mechanical Nature

Electromechanical units have contacts that close mechanically and electrically, while static units close only electrically. The differences that result from the construction of the two types of relays can be described by comparing the relays when the contacts are open or closed, and when the contacts are in the act of opening and closing.

Unlike static relays, which are usually encapsulated in epoxy, open electromechanical contacts are exposed to the surrounding atmospheric conditions. Settling dust can act as an abrasive agent and cause premature contact failure, and chemical contaminants promote oxide-layer production, which can also increase contact resistance. However, the physical separation of the open contacts provides a very large resistance in the trip coil circuit, which cannot be matched by a solid-state device. The leakage current that flows through the solid-state device when in a high-impedance state is on the order of 10 to 20 mA. Static relays should not be used where this current magnitude cannot be tolerated (*16, 18, 26, 29*).

When electromechanical contacts are closed, they normally have only a few milliohms of resistance to control-power current, while thyristors or triacs provide a 1.0- to 1.5-V voltage drop. If the contacts become so degraded that contact resistance causes excessive heat generation, the contacts may weld closed. Moderate overloads can be tolerated for short periods. On the other hand, when solid-state relays are underrated for a particular application, they will fail catastrophically. Where more than one contact closure is necessary, additional contacts are inexpensive and easy to add to electromechanical relays, but they can only be added to static units by a costly duplication of solid-state circuitry portions. Similarly, interlocks and remote indicators are also more expensive in static relays; in fact, these tasks may be performed more economically by using digital logic on the control side (*8, 18, 26, 29*).

The input-output coupling of electromechanical relays provides a very large isolation resistance between input and output. Hybrid relays can match this resistance, while straight static devices can approach a value of between 10^{10} to 10^{11} Ω if the photon coupler isolator shown in figure 14.26 is used.

Although a contact is in the act of closing or opening for only a small portion of its life, this movement is the essence of the difference between static and electromechanical relays. It adds the problem of potential arcing to the operation of electromechanical relays: arcing does not occur in static relays because of the nature of the solid-state contact. High-inductive circuits will usually cause contact arcing during separation, and arcing can pit the contact surfaces, giving a higher value of contact resistance. Arcing is particularly dangerous in explosive environments. Closing contacts are subject to contact bounce, which can also cause arcing (*8, 16, 18, 26, 29*).

Testing electromechanical relays by listening for contact closure is a simple method of inspection. Early static units had no equivalent test procedure, but more modern static relays have pushbutton testing that uses a target (flag) to signal proper operation (*8, 26, 34*).

Both moving and stationary mechanical contacts are subject to nuisance tripping due to external vibrations, and this is especially bothersome with the portable power equipment used in mining. The mechanical nature and diverse mechanical configurations of electromechanical relays make their response difficult and costly to predict.

The epoxy resin covering, physical structure, and structural arrangement of static relays make them much less susceptible to seismic disturbances (*15, 29*).

Versatility

If there is any one characteristic, other than reliability, that ensures an ever-increasing role for static relays, it is their versatility. They can duplicate the time-current characteristics of electromechanical relays described in chapter 9 and also provide functions that heretofore were impossible. For instance, static directional-overcurrent relays are feasible and provide unique characteristics for improved operation (*11*); new static negative-sequence relays are capable of tripping at only 10% negative-sequence component content, where electromechanical relays are limited to tripping levels of 18% negative-sequence component (*22, 32*). Many static relays are compatible with digital logic (*8*), and it is expected that the capability and versatility of this mode will push static relaying in directions not even considered previously. More will be said about this when the future of static relaying is discussed.

Current Transformer Burden

CT burden has been defined as the external load applied to the secondary of a CT. As noted in chapter 10, high CT burdens can cause core saturation because of the magnitude of the voltage developed at the CT secondary terminals. Saturation causes a burden reduction since, under deep saturation, the burden approaches its dc resistive value. This burden change modifies the relay characteristics and can cause incorrect operation (*36*).

The possibility of CT saturation is of particular importance when low-ratio CT's cause large secondary currents (large secondary voltages) and when large burdens from additional secondary loads cause large secondary voltages from even moderate secondary currents. Hence, relays presenting lower burdens are clearly advantageous. Table 14.3 compares the burdens presented by a typical electromagnetic-induction overcurrent relay with its static counterpart when set on the minimum tap setting. The static relay burden includes that of the relay current-measuring portion and the power-supply portion because the relay receives its operating power from the CT. The burden of the current-measuring portion of the static relay is actually much less than that for the electromechanical, but note that the burden is greater than the electromechanical burden at pickup. The lower static burden has the advantages of allowing:

- More relays to be connected in series,
- The lowest tap of a multiratio CT to be used,
- A step-up auxiliary CT to be used for greater sensitivity, and
- More auxiliary burdens to be connected without CT saturation.

Another advantage is that the low CT burden will permit the use, without saturation, of a less expensive CT that has no relay classification (*19, 34*).

Accuracy

In general, static relays are more precise than their counterparts for reasons other than those already mentioned. Common induction-disk relays are adjustable through 11 discrete tap settings in one range only. A typical static time-overcurrent relay can have an additive tap-block arrangement, which permits numerous tap settings in more than one range. At low multiples of pickup, typically below 1.5 times pickup, induction-disk relays produce very low torque levels, which can make time settings unpredictable. Static relays are reported to work reliably down to 1.1 multiples of pickup. The frequency content of the signal applied to a relay is another source of error. Static relay response in the band of interest is uniform, while induction-disk relays show a distinct variation of response with frequency. The uniformity of static relay frequency response should make the performance of static relays more predictable (*19, 22*).

STATIC RELAY MINING APPLICATIONS

An outstanding application of solid-state relaying to mining has been in ground-check monitoring, especially when connected to portable or mobile face equipment. Because the success of these devices has been adequately covered in chapter 9, nothing more will be said here. Static protection relays have been applied successfully in U.S. mines for the following common ac distribution areas (*7, 10*):

- Line short circuit and overload (device 50/51),
- Undervoltage (device 27),
- Phase loss and phase sequence (device 47),
- Zero-sequence ground fault (devices 50G and 51G), and
- Grounding-resistor overvoltage (device 59G).

In all these instances, the replacement of electromechanical devices with static devices has led to greater environmental resistance, easier adjustment and testing, more sensitive ground-fault protection, and lower maintenance time (*10*).

Table 14.3.—Comparison of induction-disk and static time-overcurrent relay burdens to a current transformer, magnitude of impedance in ohms

Relay	Relay range,[1] 0.5 to 4 A				Relay range,[1] 1.5 to 12 A			
	1X	3X	10X	20X	1X	3X	10X	20X
Electromechanical:								
Inverse [2]	22.0	10.80	5.00	3.66	1.45	0.65	0.32	0.24
Very inverse	4.15	4.15	2.90	2.20	.59	.58	.40	.25
Extremely inverse	1.60	1.60	1.60	1.60	.17	.17	.17	.17
Static: Inverse, very inverse, extremely inverse	6.42	1.5	.42	.31	.72	.174	.046	.033

[1] Coil current is expressed as a multiple of maximum coil range.

[2] The inverse induction-disk relay was not available with a range of 1.5 to 12 A. The burdens given in that column are for relay range of 2 to 16 A on the 2.0–A tap. Comparison is still possible because the static-relay burden is constant regardless of the tap setting.

Of all the kinds of protection that can be provided by static relays, the use of zero-sequence relaying in high-voltage resistance-grounded systems is likely to give the most benefit to the mining industry. The increased sensitivity and decreased burden should remove many of the problems that were covered in chapters 9 and 10. The other major distribution protective-relaying concern is short circuit and overload, but induction-disk relays have already proved themselves to be effective and reliable here. For these relays to be replaced by static devices, lower cost and higher reliability must first be effectively demonstrated to the industry.

A protection-relaying problem that has plagued the mining industry for many years is the occurrence of faults on dc trolley circuits. The situation can be divided into two main subjects (*6*).

1. For short-circuit protection, the dc overcurrent relay (device 76) often works off a shunt. When using large locomotives, its pickup setting is on the order of 3,000 to 4,000 A. Testing with such large values to trip electromechanical relays can be dangerous and the resulting accuracy questionable.

2. Because normal load currents can exceed 3,000 A, conventional electromechanical relaying cannot detect arcing or high-impedance faults, which may, for instance, lead to a mine fire.

The best avenue for solving these problems appears to be through solid-state relaying. Static overcurrent relays afford safe, fast testing as required by Federal regulations (*33*), with inexpensive hand-held devices (*6–7*). For item 2 above, many techniques are presently under demonstration, and they were covered in chapter 9.

When conventional molded-case circuit breakers are used for trailing-cable protection, the precision of short-circuit settings can always be questioned because of the mechanical nature of the trip elements. Overload adjustment is not available and, if possible, requires an exchange of thermal elements. Mining-duty molded-case circuit breakers are available with static trip elements for both short circuit and overload. Here the conventional thermal-magnetic trip elements are replaced by CT's and solid-state circuitry. The CT's proportionately reduce the line currents to a level that can be used as input to the solid-state circuitry. Breaker tripping is initiated when a low-power flux-transfer shunt trip or UVR is activated from the output of the solid-state circuitry. The overload rating can be altered by simply changing a small rating plug on the front of the breaker. The instantaneous trip range is specified as a multiple of the overload-current setting. Operation of the static trip elements has shown their increased accuracy, reliability, and repeatability over their mechanical counterparts.

It is interesting to note that in the United Kingdom static relays are used extensively for overload, short-circuit, and ground-fault protection of distribution and utilization equipment in mines (*28*). Two of the techniques used there deserve presentation: the sensitive earth-leakage system and the phase-sensitive short-circuit system. Recent research under Bureau of Mines funding has been investigating the adoption of similar systems to U.S. mine power systems (*23–25*).

Sensitive Earth-Leakage System

In an attempt to reduce the dangers of incendive arcing from damaged trailing cables, the U.K. National Coal Board developed the sensitive earth-leakage system (SEL) for ground-fault protection (*32*). The system also substantially reduces the chance of an electrocution by limiting ground-fault currents to extremely low values (*27*).

A simplified diagram of the SEL system is shown in figure 14.31. The neutral-grounding impedance limits the maximum ground-fault current to 750 mA. Fault detection is by the zero-sequence scheme (see chapter 9), but a solid-state amplifier increases sensitivity to as low as 90 mA versus 4 to 6 A with typical electromechanical devices on low-voltage systems in the United States. Currents below 90 mA are allowed to flow continuously, but currents up to 750 mA are permitted for about 0.02 s. To provide selectivity, a 0.4-s time delay is introduced at the

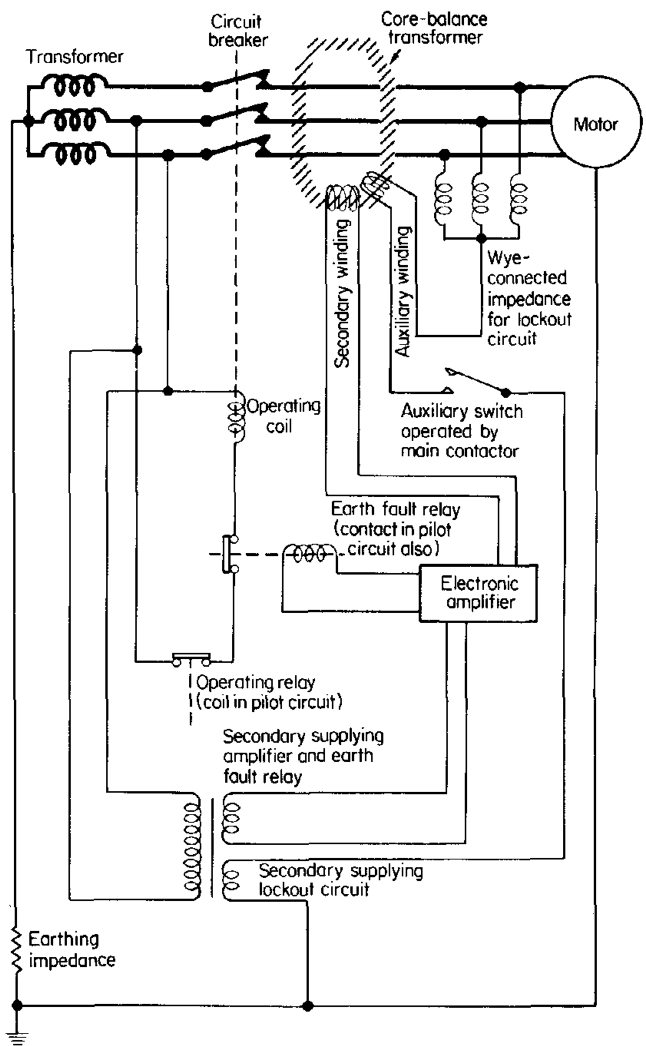

Figure 14.31.—Simplified sketch of the SEL system.

main circuit breaker. The power factor of the neutral-grounding (earthing) impedance is normally specified from 0.65 to 0.75 to avoid limiting the current to a level that cannot be detected because of system capacitance. The circuit is simply a solid-state amplified zero-sequence relay. Because of the high sensitivity, grounded shields are placed over the CT to reduce electromagnetic interference that can cause nuisance tripping.

To prevent the circuit from being reset until a ground fault is cleared, the SEL system has an auxiliary circuit connected to a second winding on the CT. Upon ground-fault pickup, an auxiliary contact is closed by the relay, which in turn causes the auxiliary CT winding to be energized. This induces a voltage on the other CT winding, which creates a lockout.

The other ground-fault method in use in the United Kingdom is the multipoint SEL. Here a false-neutral transformer, which is impedance grounded, replaces the zero-sequence transformer. The source-transformer secondary is isolated from ground across a spark gap (fig. 14.32). When a ground-fault occurs, a potential is developed across the wye-connected impedances (false-neutral transformer), and current flows through the grounding impedance. The voltage developed across the impedance is amplified, causing the relay to pick up. As with the preceding system, an auxiliary changeover contact provides lockout until the fault is cleared.

The multipoint system, however, has several disadvantages: the technique is indiscriminate and limits the number of units that can be utilized at a gate-end box (utilization center). The maximum number of units for a 550-V system is 37 and for a 1,100-V system, 18. Ground-fault current is again limited to 750 mA, but this time all relays see the fault current produced and will pick up. This

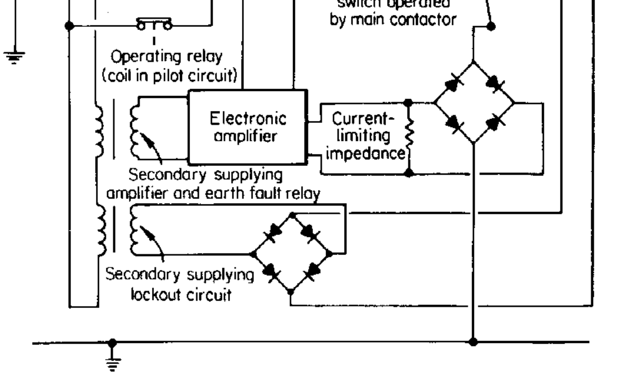

Figure 14.32.—Simplified sketch of the multipoint SEL system.

is a definite drawback, even though the unfaulted units may be reset at once and the faulted unit will be locked out. In the United Kingdom, automatic circuit-breaker resetting is allowed on this system to restore operation to the unaffected portion.

The sensitivity of the multipoint system is excellent: pickup is about 3 and 6 mA on 550- and 1,100-V systems, respectively. The high sensitivity is desirable in the case of two simultaneous ground faults on separate lines (*27*). Here, with a motor at full load, only 50% of system voltage is available to drive the ground-fault current.

Phase-Sensitive Short-Circuit Protection

Because both motor starting and fault conditions result in large current flows of comparable magnitude, it is often difficult to adjust standard relays to differentiate between the two and to provide interruption only when a fault occurs. Phase-sensitive relaying is able to distinguish between these two conditions by sensing the phase angle between the current and voltage. It utilizes the fact that the induction-motor power factor at starting is approximately 0.5, while typical faults have power factors of about 0.9 (*14*). Actual in-mine use of the method has been very promising, revealing that nuisance tripping on motor starting can be eliminated, while short-circuit trip settings can be reduced to half the value of those required for standard instantaneous relays (*28*). Two techniques can be used to provide the protection: a diode bridge and an electronic comparator.

The circuit shown in figure 14.33 is a simplified diagram of one phase of a three-phase phase-sensitive system employing a switching diode bridge. In this circuit, the CT and diodes act as a standard full-wave bridge-rectifier circuit to create the voltage across the bridge (V_b). The PT secondary may be modeled as the secondary of an ideal transformer in series with a resistor as shown in figure 14.34. The voltage in the secondary of the PT, V_v, will be a sinusoid, and the resultant voltage across the transformer secondary resistances V_r is $V_b + V_v$. Current flow in this resistance will be proportional to V_r, as will the resultant current flowing through the CT and the load resistance R_L. The trip signal will then be the voltage appearing across R_L due to the current flowing through it. Thus, the trip signal ultimately is proportional to $V_v + V_b$.

If the voltage and current are in phase, V_v and V_b are in phase, and they add in phase to produce V_{trip}. If V_b increases because of an increase in current flow, the peak value of V_{trip} will increase. The instantaneous-relay unit connected across R_L could be adjusted so tripping would occur when V_{trip} reaches a certain magnitude that corresponds to a short-circuit condition on the system. If the voltage and current are out of phase by 60° (0.5 pf), relating to motor starting, V_{trip} is less than for the in-phase situation. Even if the current were increased significantly (as in motor starting), V_{trip} would still be lower than the voltage produced by a system fault, and the instantaneous unit would not trip.

A block diagram of an electronic-comparator circuit that could also be used to provide this protection is shown in figure 14.35. The voltage and current comparators will each output a "1" (high logic level) when their respective inputs are above a specific threshold level. The AND gate

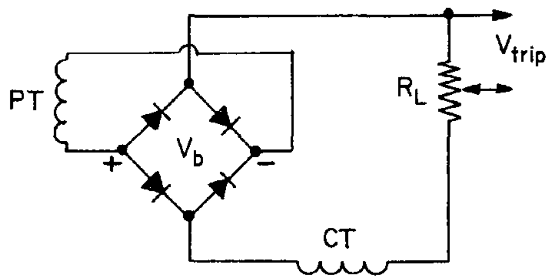

Figure 14.33.—Diode-bridge phase-sensitive protection.

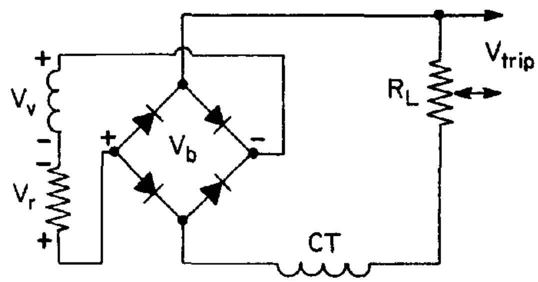

Figure 14.34.—Equivalent model of figure 14.33.

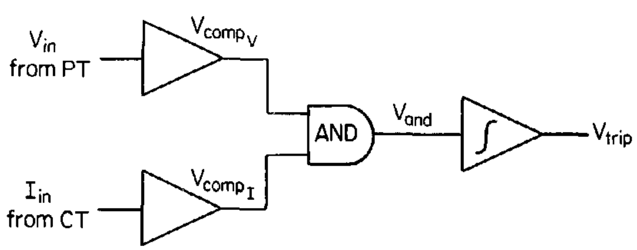

Figure 14.35.—Electronic-comparator method of phase-sensitive protection.

will then output a "1" as long as the outputs of both the current and voltage comparators remain high. This AND-gate output signal is then integrated by the integrator circuit as shown, causing V_{trip} to reach a level that would operate a circuit-interrupting device if the AND-gate output remained high too long. The integrator can be designed with enough leakage so V_{trip} will not reach the tripping level during short-time intermittent overvoltages or overcurrents.

If the voltage and current are in phase and a high-current magnitude exists (a fault condition), both V and I are above the comparator threshold levels, causing the comparator outputs to be high during the voltage and current peaks. Moreover, the voltage and current are in phase so the AND-gate output is high during these peaks as well. The AND-gate output is integrated, and V_{trip} increases to the level where tripping occurs. It can be seen that normal current levels, even if they are in phase with the voltage, would not cause tripping because they would never cause the current comparator output to reach a high state.

During motor starting, I and V are still above the comparator threshold levels, but they are out of phase, resulting in a narrower pulse at the output of the AND gate. Although the AND-gate output is integrated, the narrow pulse width allows the integrator to return V_{trip} to zero before the next pulse occurs. This prevents short-circuit tripping for a motor starting condition on the system. Indeed, a separate relaying circuit would still be necessary for overload protection.

This system protection scheme, which can be implemented either electromechanically or electronically, promises to contribute toward meeting both of the conflicting objectives of coordination: protection and selectivity. Tests have shown that electronic phase-sensitive protection tripped at much lower levels and in shorter times than standard short-circuit devices. While typical instantaneous relays may need a pickup setting of 7 to 10 times normal full-load current to prevent nuisance tripping on motor start, it has been found that phase-sensitive relaying eliminates spurious tripping during motor start, even when the pickup is as low as 3 times full-load current (*14*).

SOLID-STATE RELAYS IN THE FUTURE

The future development of static relays is indicated in many theoretical papers on the subject. Basically, new devices can be divided into three groups: continuous, digital-controlled continuous, and digital. Continuous relays are the types in which the power system is sensed continuously and relaying becomes activated from the sustained existence of a malfunction. A digital-controlled continuous technique has been illustrated in figure 14.35, and figure 14.36 relates its use in timed-overcurrent relaying. The full-wave rectifier and CT continuously sense power-system operation, and here the particular relaying characteristics are provided by the function generator with time delay provided by the linear-ramp generator. The pickup-level and ramp-level detectors essentially function as digital control devices since they operate in a *go* or *no-go* manner (*19*).

Digital relays rely on discrete sampling of current and/or voltage waveforms, directing this data through an algorithm and decision process, which ultimately decides whether a relay is to be actuated. The ultimate in digital relays of the future will probably use microprocessors distributed throughout the power system to analyze circuit conditions on a real-time basis. Each microprocessor could be responsible for making decisions for its system portion with control relinquished when demanded by a larger computer or when power-system conditions exist that are beyond the analytical capability of the particular microprocessor in question.

SUMMARY

The solid-state applications discussed in this chapter provide a viable alternative to their electromechanical counterparts. Indeed, most of them are decidedly superior in carrying out conventional functions, and in some cases solid-state relays exhibit characteristics and can perform functions that are not available with other devices. A major drawback with static belt starters and relays is their

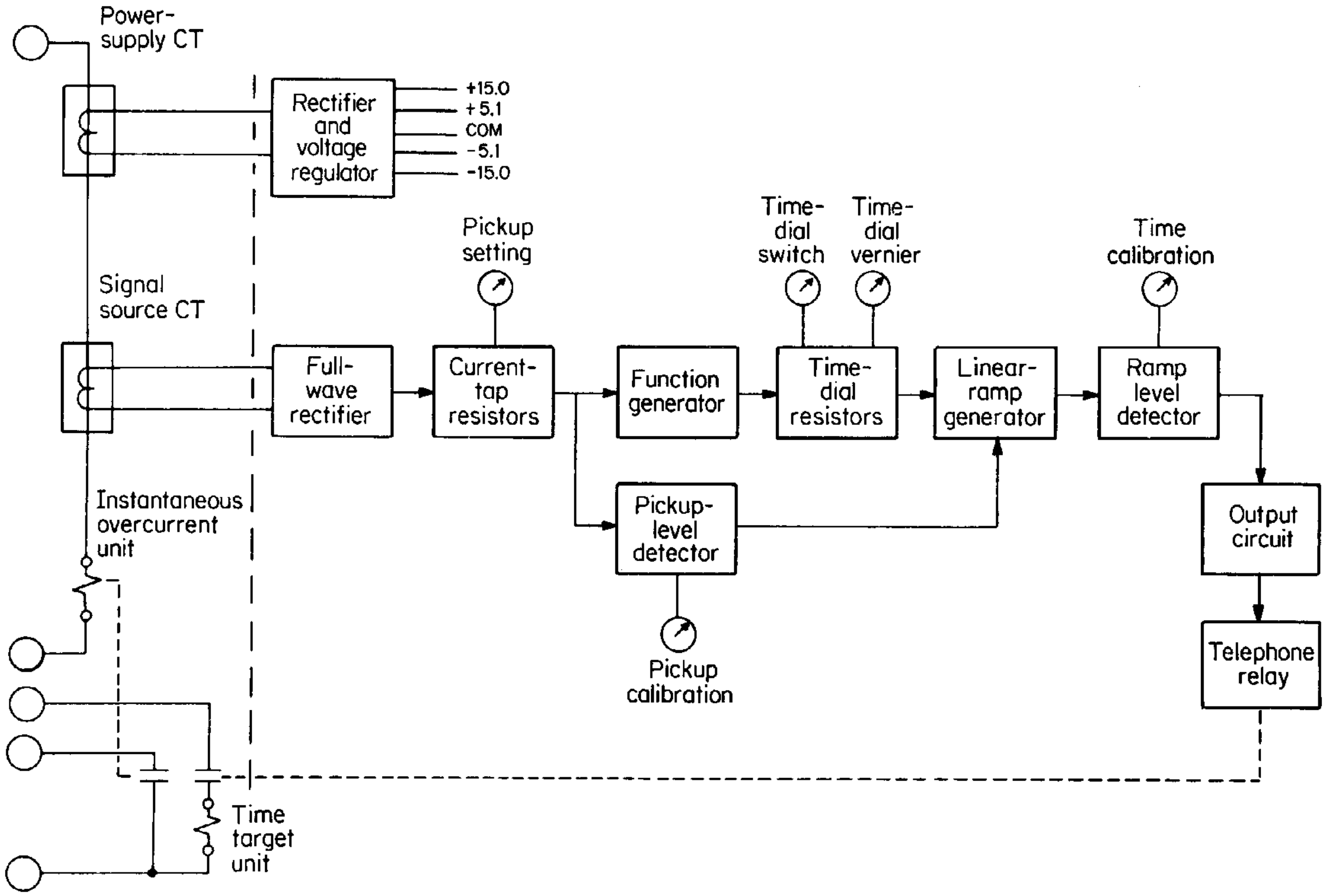

Figure 14.36.—Digital-controlled continuous static relay used for timed overcurrent.

cost, and the major factor in the cost breakdown is reliability. Belt starters have proved their desirability through a decade of in-mine experience, whereas the reliability of static relays has not been well documented to date, especially in mining. This problem is due to their recent introduction and is compounded by the fact that older models have been replaced by newer designs before life test data could be compiled (*33*). Static relays presently occupy only a small portion of the protective-relaying market. Currently, the main benefits they provide for the mining industry are decreased burden problems, increased sensitivity for zero-sequence relaying in resistance-grounded systems, and their use as solid-state tripping elements in molded-case circuit breakers. When their reliability is more fully documented and is expressed in terms of cost effectiveness, they may well become more competitive with electromechanical units and expand their role to include overload and short-circuit protection.

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CHAPTER 15.—BATTERIES AND BATTERY CHARGING

Although the storage battery had a variety of applications in the 1800's, its successful use for traction purposes was not achieved until the turn of the 20th century. Early mining batteries were used to power gathering locomotives, and to a certain extent they replaced mules in nongassy mines where open lights were used. From these beginnings, the battery-powered vehicle has gradually increased in popularity to become an important part of many underground coal mines, both for rail and off-track haulage.

The first haulage applications had little concern for safety, but the storage-battery locomotive was soon recognized as having inherent safety advantages over trolley locomotives and cable-reel locomotives. As early as 1919, Appleton noted that batteries reduced the chance of fire from short circuits and arcing (*1*).[1] Ilsley (*15*) said of the battery locomotive:

> That its energy is self-contained and limited to the immediate zone of the locomotive is a safety factor of great importance. In the trolley type of equipment one necessarily uses the track return, and the danger zone from the return current may extend through the mine. Poor bonding or no bonding may force the return current back toward the face. A storage battery locomotive does not use or need the dangerous overhead trolley with its constant shock menace and fire hazard, and with the possibility of trolley or feeder circuits becoming a factor in the ignition of gas or coal-dust.

Storage batteries have had a relatively good underground safety record.

However, early battery locomotives had open controllers, open motors, weak battery covers, crude exposed wiring, and battery jars that were prone to breakage. To help correct these problems, the Bureau of Mines in 1919 issued Schedule 15, which set standards for permissible battery-locomotive equipment (*30*). Even with this regulation in force, several incidents involving battery gathering locomotives occurred that raised further questions concerning battery safety underground. For instance, a major mine explosion in Everettville, WV, claimed the lives of 97 miners, and the resulting investigation traced the ignition to a battery locomotive (*5*). A report by Owings (*21*) contained a description of another accident where the mine operators felt strongly that the cable-reel locomotive was much safer than battery power in gaseous mines. However, Owings examined the facts pertaining to both incidents and concluded that battery locomotives were relatively safer than the other available gathering-locomotive types. The Everettville explosion was created by a nonpermissible locomotive and could just as easily have occurred from the use of other nonpermissible equipment. The second incident was later discovered to have resulted from a faulty cell that had accidentally been reversed, causing the cell to emit hydrogen, which exploded and dislodged the box cover. Owings concluded that the occurrence was very rare and that the risk of fire was greater when trailing cables were involved.

[1] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

The shuttle car, introduced in 1938, was initially battery powered, but cable-reel shuttle cars soon became much more popular than the battery cars, probably because of the low-capacity per-unit weight of the batteries then available. In the United Kingdom, trackless, battery-powered vehicles were widely believed to be unsafe. As a result of explosions caused by storage batteries at Weetslade and Eppleton collieries, battery-powered trackless vehicles were prohibited from nearly all British mines and the use of battery locomotives was restricted to types approved by the Government (*2*). In the United States, however, permissible battery-powered equipment is allowed inby the last open crosscut.

Although batteries are not currently used to power shuttle cars, they have a variety of other uses in coal mines, including the provision of cap lamp power. There has been a recent increase in popularity of battery-powered vehicles because the efficiency of batteries has been greatly improved. One prominent manufacturer reported a 70% increase in ampere-hours per cubic foot and a 39% increase in watthours per pound over earlier methods (*12*). In addition, improvements in plate and grid design have increased the average service life of motive power batteries. The trend in underground mining is toward the use of conveyor belts for coal transport and rail systems for personnel and supply movement, but mines that have ac face equipment and dc trolley lines can be subject to nuisance tripping of ac circuit breakers from stray dc ground currents. These currents, which often result from poor track bonding, ineffective trolley-line insulators, or inadequate dc-to-ac ground-system isolation, can be eliminated when batteries are used to power the locomotives.

Batteries are also used to power articulated ram-dump haulers, tractor-trailer units, or scoops (front-end loader tractor units). These battery-powered face-haulage units eliminate problems involved with shuttle car trailing cables, and in small conventional mines, the operator can be spared the cost of a loading machine and a separate machine for cleanup and supply haulage. For moderate-to-large production operations, the extreme mobility of the tractor-trailers and scoops has made them invaluable ancillary equipment for cleanup and supply in practically all longwall and many continuous operations. This chapter will examine the application of batteries to power such equipment.

Some inherent hazards in battery use remain: batteries emit hydrogen, an explosive gas, while charging; batteries and battery chargers are capable of delivering a fatal electric shock; and batteries can be a potential fire hazard. A number of less catastrophic hazards may also be encountered, ranging from acid burns from spilled electrolyte to pinched fingers from careless handling.

BASIC BATTERY AND BATTERY-CHARGING THEORY

A storage battery can be defined as a battery in which the electrochemical action is reversible (*7*); that is, after an output of electrical current (discharge), the battery can be returned to the original state (recharged) by passing current through it in the opposite direction. The basic unit of the battery is the cell, which simply consists of positive plates, negative plates, and electrolyte. One or more cells are connected together to form the battery. The connection is usually in series, although parallel and series-parallel combinations are sometimes used. The battery voltage, often given as an open-circuit value, is the sum of the series cell voltages. The battery capacity is commonly expressed in ampere-hours or kilowatthours and is mainly dependent upon the plate size (surface area).

Two types of storage batteries have been employed in underground traction, alkaline and acid. The nickel-iron or Edison cell is an alkaline cell because of the type of electrolyte used. The plates for this battery are constructed of nickel oxide and iron, immersed in an electrolyte of potassium hydroxide and lithium hydroxide. Edison batteries were once popular in the mining industry because of their high reliability and low-maintenance characteristics, but lead-acid batteries have now entirely replaced the Edison type, as a result of their high energy per unit volume and high power capability.

The basic lead-acid cell utilizes a lead peroxide (PbO_2) positive plate and a sponge lead (Pb) negative plate. These plates are suspended in a solution of dilute sulfuric acid (H_2SO_4) (fig. 15.1) (*18*). When a circuit is completed between the positive and negative plates, the following reaction occurs:

$$PbO_2 + Pb + 2H_2SO_4 \underset{\text{charge}}{\overset{\text{discharge}}{\rightleftarrows}} 2PbSO_4 + 2H_2O + 2e^- \qquad (15.1)$$

The battery must be recharged when a large portion of the PbO_2 and Pb is in the form of $PbSO_4$ and H_2O. Charging a battery consists of supplying electricity to drive the reaction as shown. When fully charged, the positive-plate active material is all lead peroxide, that of the negative plate is all sponge lead, and the specific gravity of the electrolyte is at a maximum (*7*).

The lead-acid cell has the highest voltage of any commercial battery cell. The nominal voltage is generally referred to as 2.0 V, but the actual value varies depending upon the electrolyte specific gravity and whether the cell is being charged or discharged. The open-circuit voltage is around 2.12 V per cell, with a full-charge specific gravity of about 1.280, which is a common level for motive storage batteries (*7, 18*). However, because of the effective internal resistance, the cell voltage decreases as soon as discharge commences. As illustrated in figure 15.2 the voltage continues to decrease with discharge, and the rate of voltage decrease is connected to the discharge current rate (*7*). The final voltage is the point where the battery is no longer effective for its application. Because this value also changes with the discharge rate, a typical final voltage of 1.75 V is often assumed. A standard discharge time is taken as 8 h. When the discharged battery is placed on charge, cell voltage immediately increases and continues to rise as the charging processes (fig. 15.3) (*18*).

The number of times a lead-acid battery can be recharged is a function of the discharge level of the battery during its working cycle, the method used to charge the battery, and the quality of battery maintenance. Each parameter is independent of the others. The effect of the discharge level on N, the number of times a battery can be recharged is given by

$$N = \frac{k}{D^x}, \qquad (15.2)$$

where k and x are constants of the particular battery, and D is the percent of total battery energy removed during a typical discharge. The equation states that the theoretical number of times a battery can be recharged is inversely proportional to the discharge level raised to the power x. Battery manufacturers recommend that lead-acid batteries should be recharged when they reach 80% of their capacity. Because it is difficult to determine the extent of battery discharge until the battery is dead, a battery should be recharged whenever the machine it is powering begins to show signs of sluggishness. The discharge should

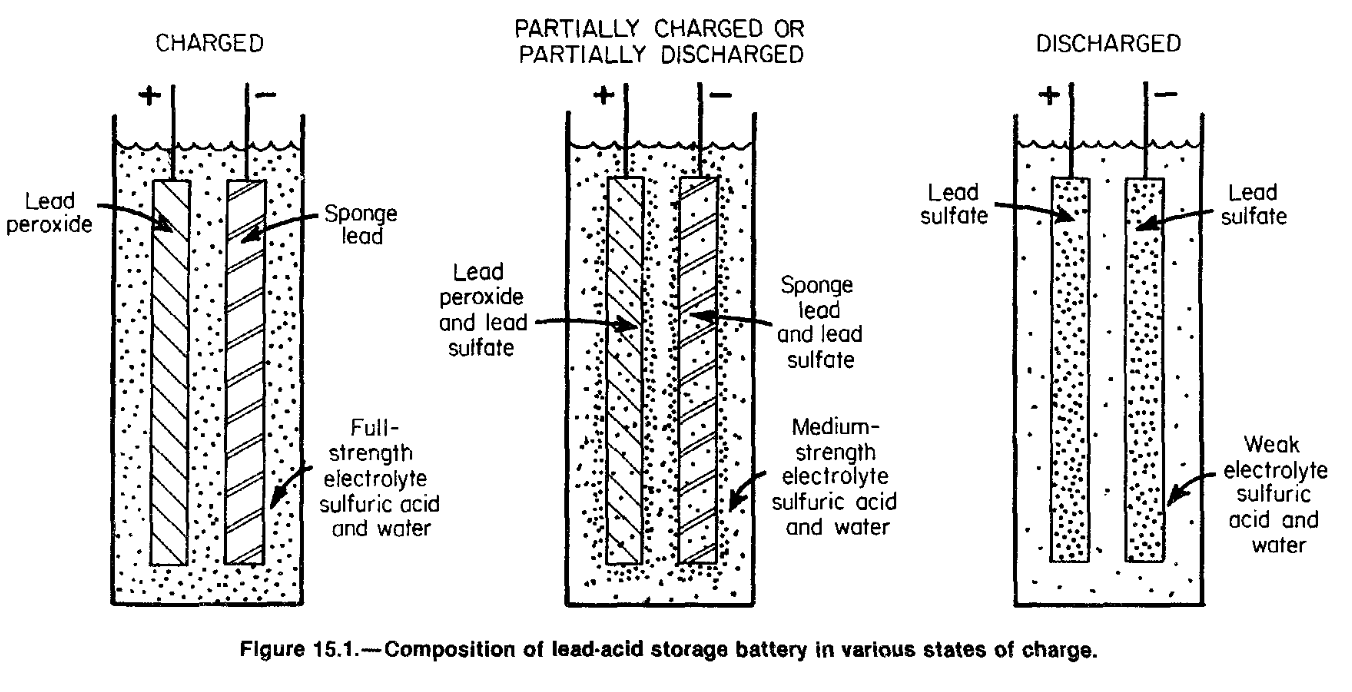

Figure 15.1.—Composition of lead-acid storage battery in various states of charge.

not reach the point of cell exhaustion or where voltage drops below a useful value (*7*).

About 110% of the ampere-hours discharged must be returned to fully charge a lead-acid battery. The rate at which charge is restored is an important consideration in attaining the maximum number of charge cycles and maximum life of the battery. Generally, any current level is acceptable, provided that temperatures above 115°F are not produced (125°F for short periods) and excessive gassing does not occur (*18*). Manufacturers may publish a normal or finish rate; this is also the current level at which the battery can be safely charged any time charging is required. The charge cycle is usually 8 h long.

Most of the electricity supplied to a cell being charged is used to transform water and lead sulfate into sulfuric acid, lead, and lead peroxide, but some of the current causes electrolysis, breaking the water down into its constituents,

$$2H_2O \rightarrow 2H_2 + O_2, \tag{15.3}$$

which is called gassing. The rate of gassing increases dramatically at a cell voltage of 2.37 V when increasing

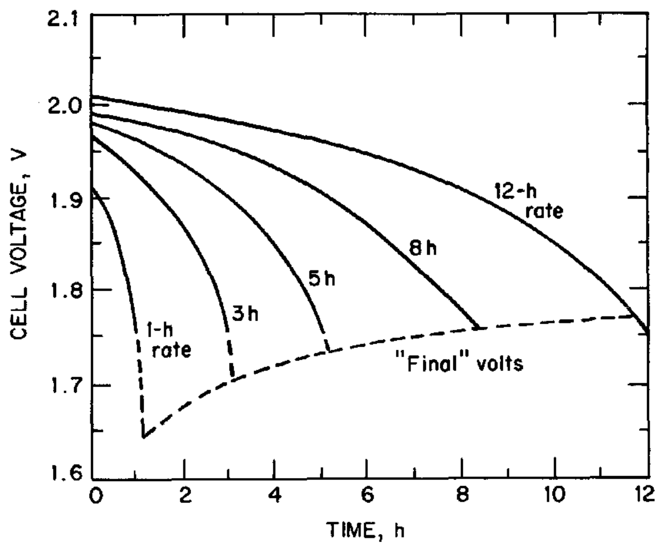

Figure 15.2.—Voltage per cell of a typical lead-acid battery with varying continuous rates of discharge.

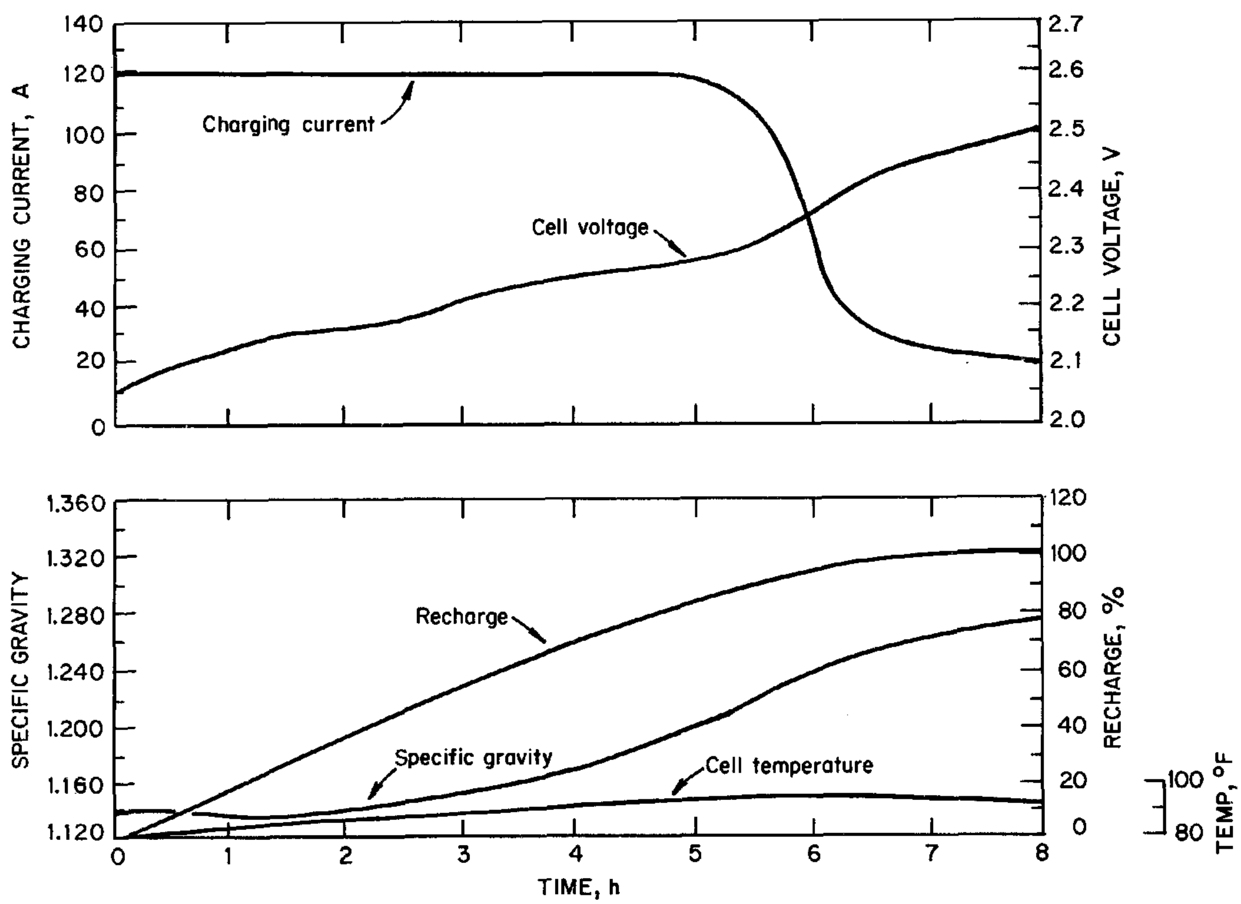

Figure 15.3.—Typical charging process of cell from 18-cell, 725-Ah battery. Ambient temperature, 77°F. Specific gravity temperature adjusted.

quantities of current become available for electrolysis because of the higher state of charge in the cell.

A certain amount of gassing is a necessary consequence of a good charge, which explains why water must periodically be added to batteries, but excessive gassing or overcharging causes damage to the plates, excessive water consumption, and excessive hydrogen emission. For this reason, the rate of charging current must be controlled as the battery charges. The large amounts of H_2 and O_2 released during excessive gassing cannot be detected by mine personnel. However, the H_2 and O_2 are sometimes accompanied by amounts of H_2SO_4 released into the mine atmosphere, which is easily identified by smell.

Undercharging a battery is also harmful, even if it is only practiced occasionally. Insufficient charging leads to a gradual sulfation of the negative plates, which eventually causes a reduction in battery capacity and life (*7*). A similar situation can occur when batteries are left standing in an uncharged state for long periods (*18*).

BATTERY MAINTENANCE

Proper battery maintenance has a very significant influence on battery life. Battery manufacturers normally include a recommended maintenance program with their batteries, based on specific gravity levels and equalizing schedules. Equalizing is the process by which all the cells in a battery are brought to the same voltage.

In lead-acid batteries, the electrolyte specific gravity is a function of the state of battery charge. Consequently, a plot of electrolyte specific gravity versus discharge depth for a particular battery is important. All lead-acid batteries require periodic equalization, but excessive equalization can cause unnecessary battery deterioration.

The following guidelines can be used to develop a good battery maintenance program, but see also references *3-4*, *6*, *8-9*, and *14*. There are three groups of activities in the program: those that should be performed daily or during each charge period, those that should be performed weekly, and those to be performed approximately once every 3 months. Accurate records should be kept of all maintenance activities for each battery as these are a convenient way to monitor individual battery performance. Deteriorating battery conditions can thus be detected before the battery becomes a safety hazard or the source of costly downtime.

Daily battery maintenance activities should include the monitoring of one battery cell, which is called the pilot cell. Any battery cell can be used as the pilot. The following cell characteristics should be recorded during each charge: specific gravity before and after charging, electrolyte temperature before and after charging, and the water level in the cell. If any of the pilot parameters falls outside those specified as acceptable by the manufacturer, all cells should be checked and corrective action should be taken. Other daily maintenance activities should include checking the battery for physical defects such as cracked cell plugs and ensuring that the charger output voltage is correct.

The weekly maintenance program includes checking all battery cells for proper water level (the water consumption of a good battery is generally equally distributed among the individual cells) and routine cleaning of the battery tops. Every 3 months, it is good practice to take a complete set of cell voltage and specific-gravity readings at the end of an equalizing charge to ensure that these parameters meet manufacturer specifications.

CHARGERS

The current supplied to a cell must be dc. It would be possible to obtain this current from a trolley-distribution system, but this source has a serious drawback: it is difficult to obtain the precise current requirements for charging. As a result, most mine batteries are charged from the ac distribution system, using a transformer-rectifier combination. Mercury arc rectifiers were used previously (*10*), but selenium or silicon rectifiers are now universally employed, and silicon diodes are considered to be the industry standard. Transformers generally have isolated secondaries and are most often three phase, although single phase can be used. Rectifiers are usually in a full-wave bridge.

Several methods can be used to control the rate of charge. Most are designed to initiate the charge at a fairly high current or starting rate, commonly 20 to 25 A per 100-Ah capacity. The level is then tapered off to the finish rate as the battery charge is restored, for example, 4 to 5 A per 100 Ah, as shown in figure 15.3. Some chargers reduce the starting rate to the finish rate in one step when the charge is about 80% complete, but control devices that taper the charge rate are more common in mining. Taper chargers are either active or passive, and in both types the charge is usually stopped automatically when full charge is reached (*7*).

The most popular passive system for taper charging employs some value of ballast resistance placed in series with the battery (*32*). This resistance limits the initial charge current and gives a relatively flat current-versus-time curve throughout the charging cycle. The advantage of this method (sometimes termed *modified constant potential*) is its simplicity. Its disadvantages include the fact that the ballast resistor dissipates a rather large amount of energy and the charge rate does not coincide with that considered optimal for lead-acid batteries. A variation of the above system uses a timer to switch additional resistance in series with the battery, thereby reducing the rate of charge at some point in the charge cycle.

A characteristic common to all active systems is control of the charging current by a feedback system that samples the battery voltage during charging. The simplest active system in taper chargers consists of a voltage-controlled relay that switches additional resistance in series with the battery at a cell voltage of 2.37 V. This and all active systems must have different voltage thresholds for batteries with different numbers of cells.

Another active method, shown in simplified form in figure 15.4, utilizes a saturable reactor type of voltage transformer to feed the rectifier. The saturation level of the transformer core is controlled by a dc applied to a winding on the core so that the secondary voltage is regulated by voltage-controlled current feedback from the battery. Output regulation usually begins at a battery voltage of 2.37 V per cell and is varied to zero current at the end of the charge cycle (*9*).

A relatively new method uses thyristors to replace half of the silicon diodes in the full-wave bridge, as shown in figure 15.5. The firing angle is determined by a feedback circuit that senses battery voltage (*17*). The circuit given is for a single-phase charger. Three-phase chargers would have three SCR's in the full-wave bridge with three firing boards, but the control circuitry would be much like that for single phase. Although many manufacturers and users have had success with full-wave rectifier configurations, in some designs a phase reference for firing the

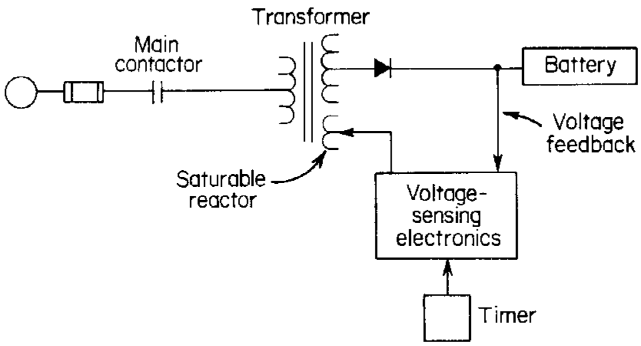

Figure 15.4.—Simplified schematic of saturable-reactor charger.

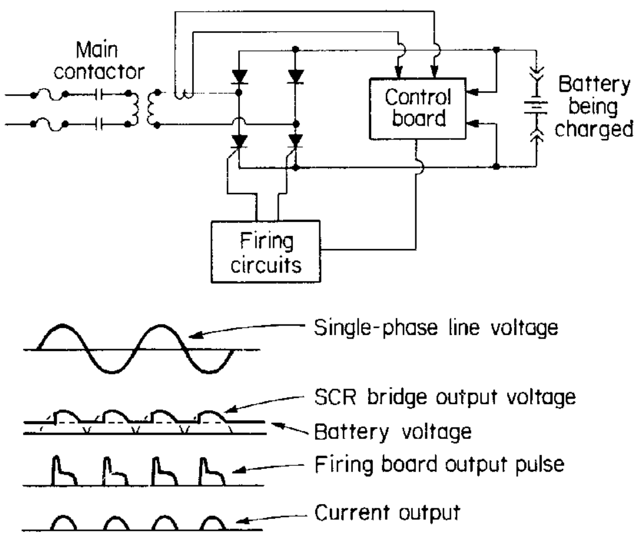

Figure 15.5.—Simplified schematic of single-phase thyristor charger.

SCR's cannot be obtained with a full wave because of negligible ripple. As a result, a half-wave must be used and this gives less efficient rectification and sometimes produces a chopped waveform, which is not entirely suitable for battery charging.

The last active method employs a ferroresonant transformer and bucking coils (*13*). For the conventional transformer, as modeled in figure 15.6, any change in primary voltage produces a corresponding change in secondary voltage, which is maintained until saturation commences, the knee shown in the transformer magnetization curve of figure 15.7. This is an advantage for most power applications since transformer operation approaches ideal when used in the linear portion of the curve (to the left of the knee). However, there are many instances where fluctuations in the secondary voltage due to normal primary-voltage changes are unwanted, and here the secondary voltage must be regulated, or remain reasonably constant under a specific load condition. A more constant but distorted secondary voltage could be obtained by driving the transformer into the saturation region (to the right of the knee in figure 15.7). Here, any change in the primary-voltage magnitude would cause only a small change in the

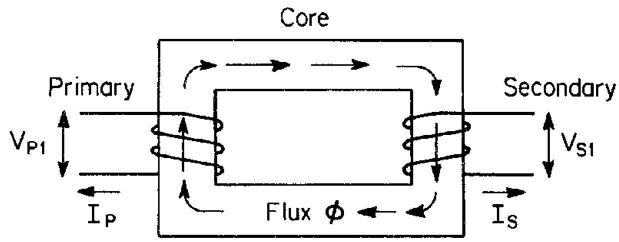

Figure 15.6.—Two-winding transformer model.

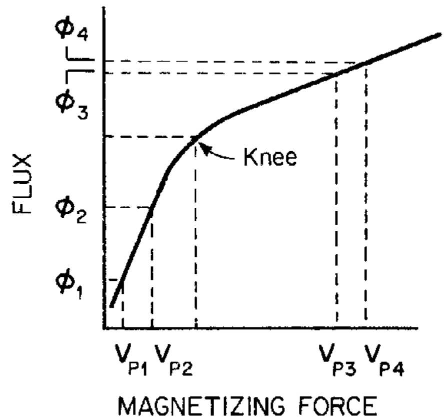

Figure 15.7.—Representative transformer magnetization curve.

core flux and thus a small change in induction in the secondary winding, but this operation is unwise because primary current becomes excessive. Nevertheless, regulation can be obtained with normal primary current if the transformer is modified so the secondary winding is exposed to a saturated core portion while the primary operates under unsaturated conditions. This is the basic principle behind a ferroresonant transformer.

A model of a basic ferroresonant transformer is shown in figure 15.8 (*13*). The leakage block of magnetic material provides a shunt path to bypass part of the flux produced by the primary winding, with the air gaps limiting the amount of flux bypass. The flux not bypassed causes induction in the resonant winding. The inductance and capacitance of this winding are selected so that additional flux is produced that is in phase with the primary flux when there is no load on the secondary winding. This increases the flux in the right core portion to about the knee of the magnetization curve. As a result, the right transformer portion will operate under saturated core conditions while the primary sees an unsaturated core.

In the basic ferroresonant transformer, an increase in primary voltage will still produce a slight increase in secondary and resonant winding voltages. To offset any increase in secondary voltage, a compensating or bucking coil, consisting of a few turns wound directly over the primary, can be connected in series with the secondary winding but with opposing polarity (fig. 15.9). Any increase in primary voltage will produce a proportional change in V_b, which will offset any increase in secondary voltage and produce a rather constant output voltage over a specified range of primary voltages.

Figure 15.10 shows the adaptation of a ferroresonant transformer to a battery charger. Although a single-phase unit is illustrated, the technique can also be used in three-phase units. Some chargers do not use the bucking coils for reasons discussed later in the chapter.

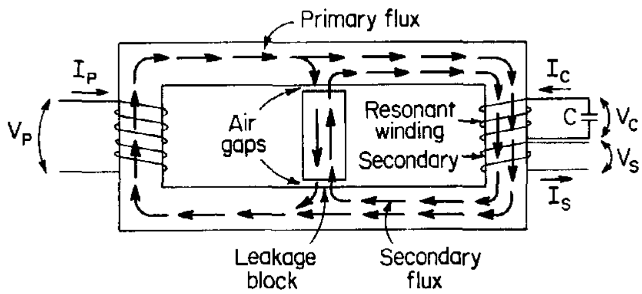

Figure 15.8.—Ferroresonant transformer model.

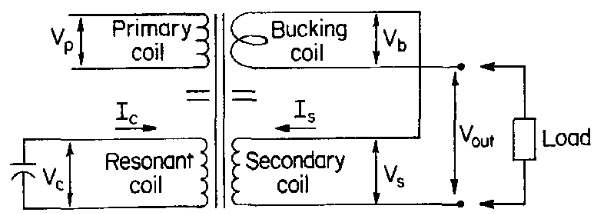

Figure 15.9.—Ferroresonant transformer.

Active charging systems have an advantage over passive systems in that feedback techniques can be used to give more accurate control of the charge rate. The most popular chargers for mine vehicle batteries use saturable reactors, thyristors, or ferroresonant transformers. Charge termination can be achieved by a timer or by monitoring the cell voltage or its rate of change to determine when the battery is fully charged.

Although the success or failure of the battery-powered mine transportation system is largely a function of the operator's ability to get maximum life from the batteries, it is also a function of the safety factor involved in battery usage. The basics of battery safety will therefore be discussed in some detail in the following sections.

CHARGING STATIONS

Special charging stations are required in underground mines to charge vehicle batteries. These stations must be designed and constructed to meet specific ventilation requirements, but because mining methods and plans vary widely, it is extremely difficult to define a rigorous set of guidelines. Hence, this section presents the principles that underly charging-station construction and explains the requirements that must be met.

The first problem that must be addressed at battery charging stations is dissipation of the gas produced by the charging operation. It has already been stated that hydrogen gas is liberated at the close of the charge cycle. Although modern chargers are designed to prevent excessive gassing by automatically dropping the charge rate to a very low value when a specified cell voltage is reached, it is impossible to charge a battery properly without

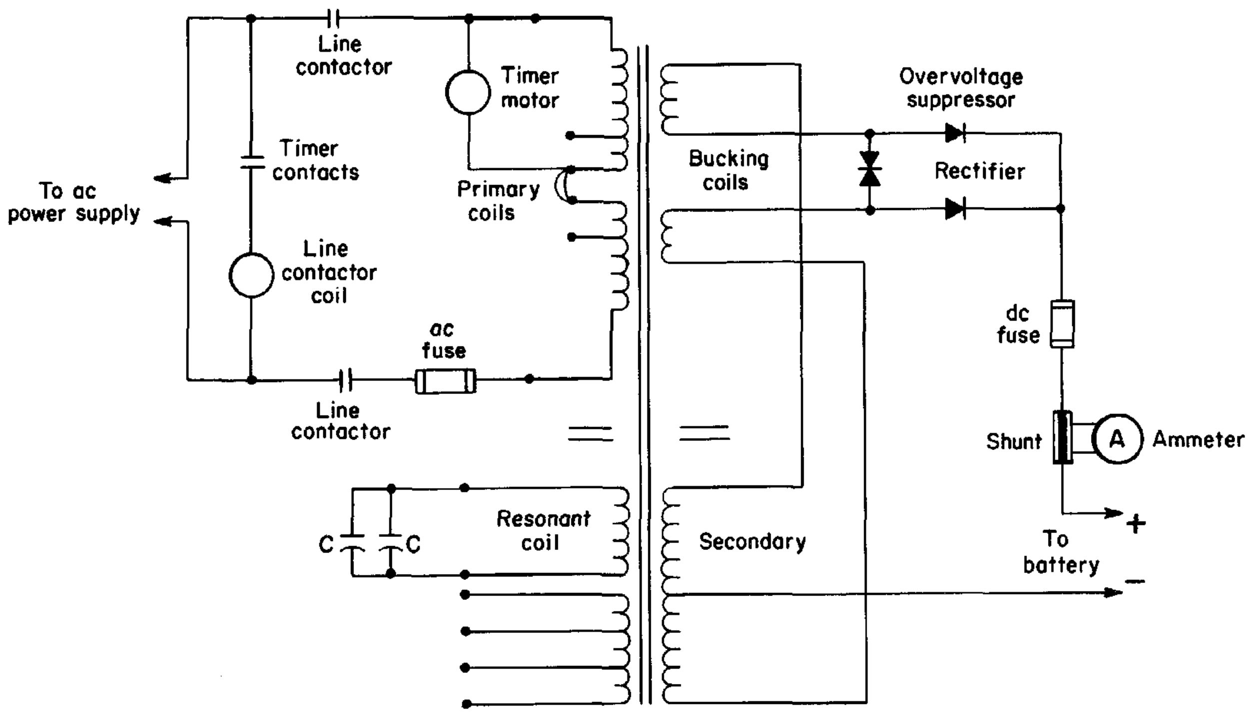

Figure 15.10.—Ferroresonant battery charger.

producing some gas. This gas is explosive and must be diluted to render it harmless in the mine atmosphere. The traditional method of achieving this is by forced ventilation of the charging room. The hydrogen concentration must be kept below its lower explosive level of 4%. Naturally, because of the catastrophic nature of underground explosions, hydrogen concentrations are closely controlled by Federal regulations. These limit the permissible concentration of hydrogen in coal mine atmospheres to 0.8% by volume, which provides a safety factor of 5 (30 CFR 75.301.5). This limit is analogous to a maximum allowable concentration of one-fifth of the lower explosive level of methane.

In order to maintain these requirements, it must be possible to monitor or estimate the evolution of hydrogen in the battery. By assuming a typical charge characteristic, manufacturers have made it possible to estimate hydrogen evolution from the number of cells and rated ampere-hour capacity. Table 15.1 lists such formulas from three different manufacturers. It can be seen that the H_2 evolution calculated from these equations is fairly consistent, ranging from 0.0024 to 0.0028 ft^3 per cell ampere-hour. These figures are applicable to the latter stages of the charge cycle at cell voltages of 2.37 V and greater. These estimated values have been confirmed by comparison with actual concentrations and charging room airflow rates obtained by in-mine survey (*19*).

Table 15.1.—Formulas to estimate hydrogen evolution

Manufacturer	Formula for ft^3 H_2 liberated in last 3 h of charge [1]	H_2 evolution, ft^3/h per cell Ah	References
C & D	(number of cells) (Ah) (0.0024)........	0.0008	*3–4*
Exide	(5) (Ah/100) (number of cells) (0.016)	.0008	*6*
KW	(number of cells) (Ah) (0.002948)....	.0007	*17–18*

[1] For KW, last 4 h of charge.

The second requirement for charging stations that is stipulated by Federal law is that all underground battery charging stations must be "housed in fireproof structures or areas." To render the station fireproof, it is common practice to line the charging area with corrugated metal siding so that all exposed coal (which has previously been rock-dusted) is covered. Concrete block is used as a lining in some instances but is more costly.

The third Federal stipulation is that "air currents used to ventilate structures or areas enclosing electrical installations shall be coursed directly into the return." This implies that the charging station must have a separate split of fresh air. To comply with this ventilation requirement, stations are frequently located in unused crosscuts immediately adjacent to return-aircourse entries. A small opening is made in the ventilation blocking so that fresh air passes over the station and dumps immediately into the return. Figure 15.11 illustrates a charging station that meets Federal regulation. Many other configurations are possible.

When each charging station is designed, it must be verified that the hydrogen concentration is below the maximum allowed. A simple but effective approach is to establish a worst case airflow quantity for each battery on charge, then substitute the safe condition by an actual airflow measurement of the charging station. This measurement can be made with an anemometer.

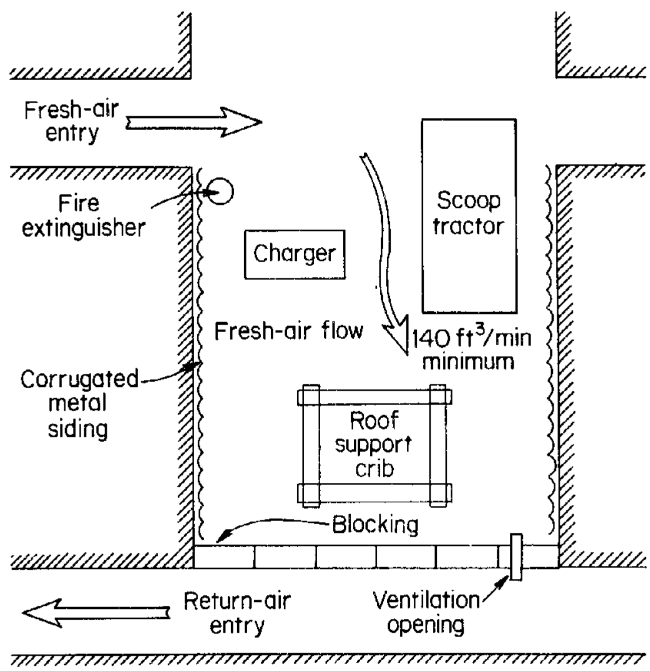

Figure 15.11.—Plan of underground charging station.

From table 15.1 it can be seen that batteries in the final stages of charge evolve hydrogen to the following approximate level:

$$(5)\,(\mathrm{Ah}/100)\,(\text{number of cells})\,(0.016) = \mathrm{ft^3/h}\ H_2.$$

Accordingly, dilution requirements can be calculated by:

$$Q(\mathrm{ft^3/min}) = \frac{(5)\,(\mathrm{Ah}/100)\,(\text{number of cells})\,(0.016)}{(0.008)\,(60)}$$

$$= \frac{(5)\,(\mathrm{Ah}/100)\,(\text{number of cells})\,(0.016)}{60}, \qquad (15.4)$$

where $Q(\mathrm{ft^3/min})$ is the minimum quantity of air necessary to dilute the H_2 produced to 0.8%, not allowing for dilution by room volume. To approach a worst case, it could be assumed that all batteries are 120-cell, 700 Ah, which is the present upper limit in battery size. Thus, each battery in the charging station requires about 140 ft^3/min of air for good ventilation.

This air quantity assumes that the battery lids remain open or removed during charging. Although this is mandatory, it is a practice that is sometimes neglected, especially in low coal. For charging, vehicle storage batteries are placed in strong enclosures (trays) with removable lids. If these lids remain closed during charging, dangerous accumulations of hydrogen are likely to accumulate because of the small space above the battery top. A charging battery can be expected to liberate 60 ft^3/h per 25,000 cell ampere-hours. The result is an explosive concentration unless forced ventilation is applied, and this is considered impractical by many. Enlarging the venting slots significantly is not an answer to the problem since it would weaken the battery box structure. The only solution is to ensure that the lids are open or removed.

BATTERY BOX VENTILATION

The ventilation of the traction battery enclosure is not only important while charging but also during operation. Following the charge cycle, the covers are closed and the vehicle is placed in service. From chemical reactions and entrapped gas within the cells, lead-acid batteries continue to emit gas for several hours after receiving a charge, be they open-circuited or on discharge (*22*). Considering the ignitability of hydrogen, it is possible that a dangerous air mixture will accumulate. Hence it is necessary to provide adequate ventilation for the evolved hydrogen in the closed box while the battery is in operation.

Prior to 1945, there was little mention in the literature of possible gas emissions from lead-acid batteries during discharge. In fact, many early publications stated categorically that there were no emissions under normal conditions (*33*), and this statement was repeated in mining literature (*15*). However, the possibility of gas emission did receive much attention from authorities responsible for high-capacity battery installations in such confined spaces as submarines. Robinson (*22*) quoted a report revealing that lead-acid batteries in confined spaces are always liable to emit considerable quantities of hydrogen and oxygen for the first few hours after charge. Although the actual quantities varied greatly from battery to battery, an 80-Ah cell, standing idle at 80°F (26.7°C) with 1.26 specific-gravity electrolyte, would probably emit 5 to 20 mL/h H_2 for about 12 h after charge. The approximate rate was found to be

- Directly proportional to the battery capacity,
- Doubled for each 15°F (9.5°C) rise in temperature, and
- Doubled for each 0.050 unit increase in electrolyte density.

As a result of antimony contamination of the negative plates, the open-circuit emission also increased with cell age. Upon discharge, additional hydrogen was evolved from the negative plates, which was thought to be a release of gas entrapped during charging. Combining the open-circuit and discharge emissions on a worst case basis, it was postulated that a lead-acid battery cell could release up to 500 mL (about 0.2 ft^3) of hydrogen per hour at the end of its useful life.

An advisory committee on coal mining, formed in the United Kingdom during the mid-1940's, investigated such problems related to underground battery vehicles (*28*). The committee recommended that traction battery enclosures be properly ventilated to prevent a hazardous accumulation, and accordingly, a regulation for U.K. coal mines was promulgated in 1949 (*27*). In the United States, the Bureau of Mines subsequently incorporated a similar requirement in Schedule 2G, stating that "battery boxes shall be adequately ventilated" (*31*).

Corresponding to the 1949 regulations, a test procedure was made statutory for the United Kingdom (*22, 29*) in which

- Hydrogen generation was calculated at 3.0 ft^3/h per 25,000 cell ampere-hours;
- Tests were to be made in still air;
- Maximum hydrogen concentration within the box under these conditions could not exceed 2.0% with tolerance.

The emission calculations assumed that all factors contributing to the hydrogen-emission rate, such as cell temperature, acid specific gravity, and discharge rate, were simultaneously at worst case. Still air was selected to simulate a vehicle traveling at the same velocity as the mine ventilation so that the container's natural ventilation would receive no assistance. The maximum specified concentration provided a safety factor of 2.0 below the lower flammable level of hydrogen (4.0%).

In his research, Robinson examined simple arrangements of container vents that would provide sufficient natural ventilation to meet these ventilation requirements (*22*). He used the fact that hydrogen has the highest coefficient of diffusion into air of any gas; because of its extreme mobility, hydrogen is very difficult to retain within a leaky enclosure. The investigation used a simulated battery box containing a dummy 56-cell, 288-Ah lead-acid traction battery. Hydrogen liberation for discharge was calculated at 2.0 ft^3/h by the foregoing relationship. While hydrogen was pumped in at this steady rate, hydrogen concentrations were measured in the space between the battery top and cover (volume of 6.1 ft^3). Eleven different venting arrangements in the enclosure top were used, ranging from 40 to 140 in^2, and sealed top vents were also investigated. End-plate venting was available in all tests.

Robinson's findings were as follows:

1. Equilibrium between the battery rate of hydrogen emission and rate of hydrogen escape through the vents was established within 1 h after each test commenced. Thereafter, the hydrogen concentration remained almost constant.
2. Maximum concentrations with natural top venting ranged from 1.3% to 2.8%, related inversely to vent area and following a near logarithmic curve.
3. When the top vents were sealed and the side vents open, hydrogen concentration rose sharply to 5.3%, achieving 4.2% in 8.0 min. However, with 31 ft^3/min of forced ventilation through the side vents, the 5.3% maximum dropped to 1.7% H_2. It was suggested that normal haulage speeds and mine ventilation flow rates would create a high factor of safety.

The overall conclusion was that a battery box could be readily vented to meet U.K. requirements and forced ventilation was not needed.

In 1959, Titman (*26*) reported on gas emissions from lead-acid batteries for the 45-min period after charging, supplementing his earlier study for alkaline cells. It was then suspected that gas emission rates immediately after charge might be greater than those emitted by the battery after a standing period. Titman also investigated the parameters causing variations in emissions, such as acid strength, discharge rate, and increased cell temperature. The experiments employed a new 6-V, 309-Ah traction battery. The conclusions of the research were

1. Total gas emission for similar cells varied as much as 15%.
2. Hydrogen emission rate doubled for an increase of 0.050 in acid specific gravity (1.260 to 1.360). (Note: this increase could easily occur if the battery electrolyte was not topped up (*19*).)
3. The rate doubled for a 12.5°C increase in electrolyte temperature.

4. Overall rates were similar to those for alkaline cells.

5. Immediately after charging, hydrogen rates up to 5.0 L/h per cell were observed (corresponding to 14.30 ft^3/h per 25,000 cell ampere-hours).

6. After 45 min, the cells were found to liberate as much as 1.3 L/h per cell (3.72 ft^3/h per 25,000 cell ampere-hours).

7. The minimum emission rate was always reached within 5.0 to 8.0 min with the battery standing after charging.

As can be seen, the findings for specific gravity were similar to those reported by Robinson. However, hydrogen emission rates were considerably higher than the 3.0 ft^3/h testing standard: 376% higher immediately after charging and 24% higher after a brief standing period.

Using these emissions rates as a basis, Titman (*25*) further investigated the effect of high hydrogen emission rates on typical battery box ventilation. The enclosures investigated were basically the same as those used by Robinson (*22*). The findings included

1. A 2.0% H_2 concentration was not exceeded until the emission rate was 10 times the 3.0-ft^3/h standard rate.

2. After ½ h, the hydrogen became uniformly distributed, with no tendency to accumulate at a specific place.

3. For hydrogen concentrations of 2.0% or less, the enclosure concentration varied as the two-thirds power of the emission rate. (This was also verified theoretically.)

4. With the high emission rates, acid-drainage holes assisted enclosure ventilation.

This research has been presented in detail to demonstrate the reasoning behind battery box venting requirements and to emphasize its importance. A direct comparison of hydrogen emission rates versus the top venting area of the enclosure would be advantageous, yet no direct relationship has been found. In practice, however, it has been discovered that about 25 in^2 of top vent area will allow 1.0 ft^3 H_2 to escape per hour, while meeting the U.K. testing requirements (*20*). As an alternative, the use of catalyst battery caps appears to be an excellent solution for difficult venting situations.

These battery caps are a fairly recent introduction and are not yet used widely in mining, although they have been in use for some time in equipment ranging from torpedo batteries to personnel carriers in salt mines. They have also had some limited use in metal-nonmetal mining, such as uranium mines. In all these applications they have proved to be reliable and effective.

A catalyst battery cap converts any emitted hydrogen and oxygen back into water (*23*). The caps have the dual function of preventing the escape of any hydrogen from the cells and restraining the loss of cell water. The technique has considerable safety advantages. No additional ventilation is required for the tray or for the charging station itself. Battery lids do not have to be opened for charging, which is a major advantage in low coal. The explosion hazard associated with batteries is practically eliminated. Since watering of the cells is not required, no electrolyte can be spilled on the battery top, and the possibility of surface leakage (see below) is minimized. Tray corrosion from spilled electrolyte is similarly reduced. One problem with catalyst battery caps, however, is that the palladium catalyst may be destroyed if the cap is turned over. Special care must be taken not to do this during maintenance.

BATTERY SURFACE LEAKAGE AND FAULTS

A battery is designed to be an electrically floating system, insulated from its tray, which is at machine frame or ground potential. Yet three situations can occur that could connect the battery with its tray.

1. Current may leak across the battery surface to the steel tray; this is referred to as *surface leakage.*

2. A poorly insulated or damaged cable, bushing, and so forth may contact the tray, causing a fault condition.

3. A rock fall or collision may force the battery box cover down onto the battery terminals, possibly shorting one or more to ground.

Since lead-acid traction batteries are quite heavy, often weighing several hundred pounds, the only material presently available for tray construction is heavy-gauge steel. Steel is a reasonable conductor of electricity, a disadvantage when it is used to support an isolated system such as a battery. After a battery has been in operation for some time, dust mixed with spilled electrolyte can collect on the battery top. If these contaminants are permitted to accumulate, a number of low-resistance paths may form between the various cell terminals and the tray. Since the terminals are at different potentials, currents tend to circulate across the battery top and through the tray, and may cause the following problems:

1. Currents circulating in resistive loops cause heating. This heating is proportional to I^2R, where I is the leakage current and R is resistance of leakage path. When R is low enough to permit a substantial current to flow, a smoldering fire may ensue. This hazard is compounded by the often explosive hydrogen concentrations that occur in the battery above the electrolyte level.

2. The presence of paths from the cell terminals to the tray can be a shock hazard for mining personnel. A miner leaning against a battery box and touching an exposed terminal could be shocked. Terminal-to-tray conducting paths present an especially serious hazard when the battery is charging if the charger design or a fault within the charger permits dc to flow in the frame or ground.

3. Circulating currents waste battery power and may reduce the amount of time a battery can be used before recharging is necessary. They also tend to increase tray corrosion.

Another potentially hazardous situation can be caused by a low-resistance ground fault, for example, produced by a damaged cable in contact with the battery tray. Since the battery is theoretically a floating system, a single such fault should cause no current flow, but the effect of surface leakage paths can result in a certain level of current, depending upon path resistance. Two simultaneous low-resistance ground faults, however, would permit extremely high currents to flow. Depending on the contact resistance of the fault, currents on the order of 10,000 A may exist for a short duration because of the low internal resistance of lead-acid batteries. A very hazardous situation could result because batteries cannot be deenergized; hence, a fire caused by faults would be difficult to extinguish until the battery was discharged.

The surface leakage problem has been recognized by battery manufacturers for some time, and they constantly expound the virtues of keeping battery tops clean. An example is the following excerpt taken from a typical maintenance article (*11*):

> If the battery tops become wet and dirty, or if tray corrosion is visible, give the battery a soda wash. Mix a handful of bicarbonate of soda (baking soda) in a bucket of water. Pour this solution over the top of the battery, using one full bucket per tray. Be sure vent caps are in place. Water will dry, leaving some dry soda on the battery top. It is good practice to give batteries a soda wash once a month. If a battery is accidentally flooded (acid spilled on cell tops) due to overfilling cells, give soda wash as soon as possible.

Indeed, if mine batteries were always kept clean and dry, the surface leakage problems would be greatly reduced. Unfortunately, cleaning is often neglected in the mining industry, especially in low coal where batteries are so difficult to access. It would therefore be advisable to find some alternative way of reducing this problem.

Most batteries now in use underground have a thick coating of paint on their trays. This reduces leakage somewhat, but the paint is prone to deterioration by chipping, abrasion, and attack by battery acid. Recently, some manufacturers have been coating their battery boxes inside and out with a tough vinyl compound sold under the brand name Plastisol. This product is readily available and can be sprayed onto any properly prepared steel surface. It appears to reduce surface-leakage problems significantly.

Battery safety could be increased further if the exposed intercell connectors were insulated in some manner. Unfortunately, this is not a straightforward matter since any insulating coating applied to the connectors reduces their heat-transfer capabilities and the intercell connectors remove about 80% of the battery heat.

As an option on their mining batteries, leading battery manufacturers now produce a dead-top battery with totally insulated intercell connectors. This system should greatly reduce the potential for surface leakage and also remove the possibility of dangerous arcing if a tool is dropped across the battery top.

Since surface leakage and ground faults are potential hazards, a method is needed to detect and isolate them. Recognizing this, Statham and Littlewood (*24*) developed a fault-detection system designed to be fitted to the battery or battery charger. This system, shown in the circuit in figure 15.12, is a simple way to detect ground faults that might occur between the battery and its load. One problem with the circuit is that a large portion of the battery itself cannot be protected: for instance, a fault occurring at point A in figure 15.12 would produce no current flow through the current relay. Figure 15.13 is a plot of relay current versus fault position on the battery; it is assumed that the current-relay shunt is set at 1,000 Ω and that a 200-V traction battery is used. In order to increase the sensitivity of their circuit to battery surface leakage faults, Statham and Littlewood proposed installing switches between point X and frame and between point Y and frame (fig. 15.12). These switches would be alternately opened and closed by a mechanical or solid-state device. The upper dashed lines in figure 15.13 show the new position-sensitivity curve resulting from the modified circuit. The entire battery could be protected in this manner, although sensitivity still varies with fault position for surface leakage faults.

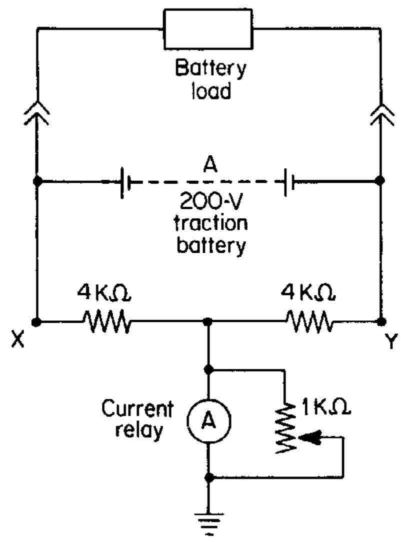

Figure 15.12.—Circuit for detecting faults in batteries.

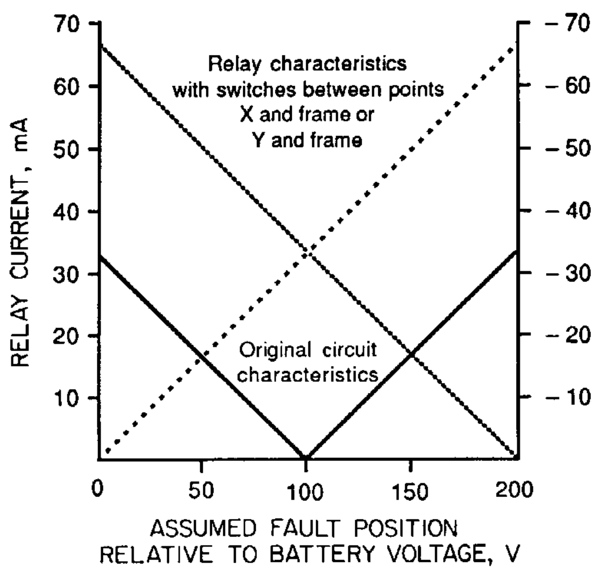

Figure 15.13.—Curve of relay current for various fault positions on battery.

Virr and Pearson (*34*) devised an electronic ac injection system that would provide protection sensitivity independent of fault location. They utilized a system of red and green lamps to indicate a *not safe* or *safe* condition. Although Virr and Pearson recommended mounting the device on each battery, this would not be necessary for trackless battery-powered vehicles since a single device on the charger would be sufficient to prevent an unsafe battery from being charged. Since the development of surface leakage presumably takes place gradually with the accumulation of conducting material on the battery top, continuous monitoring of each individual battery is probably unnecessary.

There is an additional hazard when using lead-acid batteries in any environment: an explosive hydrogen-air mixture can be available inside the cell. This is true even with catalytic caps. If ignition energy is sufficient within the cells or even close to the vent cap, the internal mixture can explode, possibly spraying acid and blowing bits of the cover toward personnel in the vicinity. Internal faults can produce this kind of explosion. Virr and Pearson related that a common source of such events also occurs when electrolyte leaks from a cracked cell, producing a spark when the acid level falls below the bottom of the plates. There is no known method of preventing or suppressing these internally initiated cell explosions. However, ground-fault protection will detect electrolyte leakage.

Low-resistance faults to the tray or vehicle frame caused by damage to cables, and so on, represent a different problem. As previously mentioned, no provision is currently available for deenergizing a faulted battery, which would continue to discharge until its stored energy was dissipated. Some form of circuit breaker between the cells of the battery would provide a way to sectionalize the battery in the event of a fault. In conditions of excessive current flow, the intercell connectors might themselves act as protective fuses, melting down when their current-carrying capacity was exceeded.

Proper isolation of the battery electrical system from the mine ground system is a prerequisite for safe battery use. The reliability of battery isolation can be greatly enhanced by following the maintenance and design suggestions given in this chapter. Total electrical safety for any battery installation is, however, a function of charger characteristics as well as battery isolation.

BATTERY-CHARGING HAZARDS

A brief review of four representative accidents sheds some light on the hazards associated with battery charging: [2]

1. A scoop operator was electrocuted when his body came in contact with the frame of a scoop tractor that was being charged. During the subsequent investigation, it was found that a potential difference of 260 V existed between the tractor frame and mine floor when the charger was energized. The cause was a low-resistance surface-leakage fault between the battery and battery tray, which caused the tray and the tractor frame on which it was resting to become energized. Faulty insulation between the primary and secondary windings on one arm of the three-phase transformer permitted secondary current to flow in the ground. The glaring error here was the failure of mine personnel to ground the steel frame of the scoop tractor properly while it was being charged.
2. A utility worker received a fatal electric shock when his body contacted the frame of a battery charger. A fault occurred within the charger that caused 210 Vdc and 114 Vac to exist between the charger frame and earth. Despite the fact that the primary cause of the electrocution was a worn bushing that failed to insulate the timer circuit from the charger frame, a proper frame ground could again have prevented the fatality.
3. An electrician was fatally injured when he came in contact with a bare conductor on the charging leads of a battery charger while connecting the charger to the vehicle. The electrician was standing in water while attempting to connect the battery. Although the charger switch was in the off position, a primary-to-secondary fault in the charger transformer circumvented the switch (which interrupted only one primary conductor) and caused the charger leads to become energized.
4. A surveyor was electrocuted when he contacted the battery ground clamp that was attached to the charger frame. The charger frame was energized because of a fault within the charger. The investigation showed the accident to be due to inadequate safety grounds on the frames of all the electrical equipment in the mine.

These accidents occurred using chargers containing the basic circuitry discussed earlier in this chapter, and with the possible exception of the enclosure, they were similar to chargers used by other industries. This implies that additional design concepts or components must be included in battery chargers in order to ensure safety in the mine environment.

Poor grounding and component failure (most especially the power transformer) have been the most notorious contributors to mine charger accidents and electrocutions. They have caused the ac source power to be impressed on the dc charging circuit. Consider an instance where a primary-to-secondary power transformer fault elevates the charging circuit by the primary potential. If this occurs simultaneously with a fault that energizes a battery tray, and the grounding is unsatisfactory, the battery tray will be a shock hazard. Of vital importance is adequate grounding because an intact grounding system ensures that frame potentials do not exceed reasonably safe levels, regardless of the contribution of other factors to the hazardous condition.

A list of other problems causing accidents has been made through an analysis of battery charger accidents and input from operators, manufacturers, and State and Federal regulatory agencies:

- Bad charging-cable insulation;
- Unconnected, exposed charging couplers (plugs) that are still energized;
- Poorly designed charging couplers that have exposed ungrounded metal parts where the cables are attached and allow cable damage;
- Charging couplers with poor contact as a result of frequent damage or inadequate maintenance;
- Battery surface leakage and internal faults;
- No overcurrent protection for the ac power input;
- Faults causing the control circuitry and components to have an elevated potential;
- Personnel connecting a charger to a battery of wrong polarity;
- Repair work performed by unauthorized personnel; and
- Charging stations in abnormally wet locations.

In view of this poor safety record, it is advisable to consider the safety features that would reduce the potential for accidents and electrocutions when charging batteries:

1. The charger input cable should contain a monitored ground to ensure that the grounding conductors to the charger frame are intact.
2. The charger should be equipped with panel interlocks that deenergize the charger at the outby source when access panels are removed.

[2] MSHA reports of fatal coal mine electrical accidents.

3. The charger should have an emergency off switch located in a conspicuous place on the charger frame. This switch should not be spring-loaded, thus requiring resetting after use.

4. The power transformer should electrically isolate the battery being charged from the power source, with primary and secondary windings being so arranged to eliminate hazardous interwinding faults.

5. A separate grounding conductor and ground-check monitor circuit should be provided for each battery tray serviced by the charger.

6. The dc couplers should be of the type that interrupts the ground-check circuit before the charging circuit.

7. The dc connection between the charger and battery box should consist of a single cable with appropriate grounding and ground-check conductors.

8. The charger should contain battery surface leakage detection circuitry that prevents a leaky battery from being charged.

9. The power transformer secondary and all dc circuits and components should be isolated from the frame ground of the charger.

10. The charger should have a meter or similar device that indicates the state of battery charge.

11. The charger should have overload and short-circuit protection on both the input and output.

12. The charger should contain circuitry that prevents a battery of the wrong voltage from being charged; alternatively, the charging connector must be keyed or sized such that no connection can be made with a battery with fewer cells than that for which the charger is designed.

Figure 15.14 is a diagram of a typical solid-state controlled taper-rate charger. All the desired electrical components specified in the above list are signified by the letters in parentheses. These and other features will be discussed in the following paragraphs. It should be noted that in some instances there are alternative practical methods that could be employed to protect personnel.

System Grounding

Because of its ac-to-dc application, the charger illustrated is considered to be portable electrical equipment and is fed by a resistance-grounded low-voltage or medium-voltage ac system. Under Federal regulations for coal mining, a grounding conductor and ground-check monitoring of this grounding conductor are required between the power source (usually a power center) and the charger frame. The portion of the ground-check circuit within the charger is indicated at *a*. The panel interlock switches (*b*) and the emergency off switch (*c*) are shown in series with the ground-check circuit. Here, a pilot-type monitor is implied. If the circuit is opened in any manner, the circuit breaker at the power source is tripped. After tripping, the circuit breaker must be reset manually at the source and cannot be reset until the ground-check circuit is restored.

Electrocutions involving contact with the charger frame have taken place when grounding practices were unsatisfactory and a fault existed between the frame and some charger internal component. One method for preventing this type of hazard would be to use an insulation coating to isolate the enclosure completely from possible contact with live circuits. There are, however, several shortcomings to this method of protection. Most battery chargers designed for underground use are mounted on skids for easy mobility. In rough mine use, any insulation would be quickly worn off, especially on the skids. Insulation coatings would be difficult to apply everywhere.

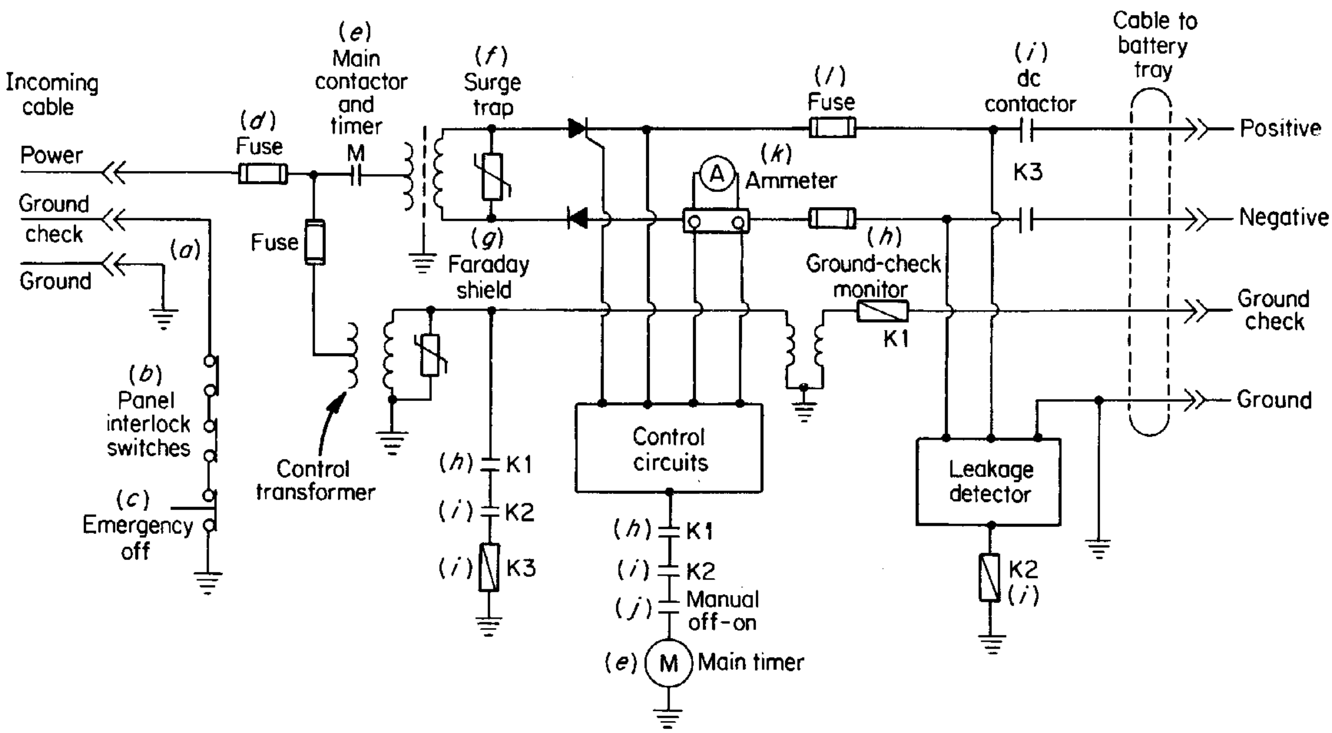

Figure 15.14.—One-line diagram of desired charger features.

Devices mounted to the cabinet would have to be coated for 100% protection, and bonding all parts together and bonding components like timers and switches to the exposed panels would pose a considerable problem. Furthermore, any insulating compound deteriorates with time. Thus insulation coating(s) might both be impractical and result in a false sense of security. As a result, present enclosure grounding standards may well be a more effective safety approach.

Panel Interlocks

The main reason behind the panel interlock switches is to prevent authorized or unauthorized personnel from unknowingly contacting live parts inside the charger. The emergency stop or "panic button" provides definite safety advantages. In case of an emergency, it provides a quick power stoppage at the charger. Without such a switch in an underground mine, a miner would have to go back to the power center, which could be as far as 500 ft away. The switch has a large red button for easy identification; pushing the button breaks the upstream ground monitor circuit. The button must be pulled out manually before the outby circuit breaker can be reset and so provides a double check on reenergization. This is particularly important for maintenance personnel since it prevents power from being accidentally restored to a circuit while they are working on it.

A justifiable complaint against panel interlocks is that they must be *defeated* for maintenance and may be left in that condition, rendering the system dangerous, from a false sense of security. However, experience with panel interlocks on other mine power equipment has shown that these safety devices rarely remain in a *defeated* condition. Remember that any protection system can be knowingly or unknowingly rendered useless, and it is the responsibility of training personnel to make sure this is minimized. There are other ways to discourage unauthorized entry; for example, barriers or partitions can separate energized components within the charger, and special opening tools and attachments can be used on many covers. But even in these cases, panel interlocks would give an extra safety margin for any panel that is opened or removed for normal adjustments.

Transformer Failures

A Faraday shield is indicated at *g* in figure 15.14. This prevents primary-to-secondary (interwinding) faults in the power transformer and has the secondary benefit of isolating any high-frequency transients on the incoming power from the charger where they could interfere with the control circuitry. If the amplitude of these transients was high enough, they could destroy solid-state devices. The Faraday shield probably exceeds the desired safety requirements for secondary-circuit isolation.

Axial displacement of the secondary and primary windings on the transformer core plus separation of primary and secondary circuits is also acceptable instead of the grounded shield. Again, the goal is to accomplish electrical isolation of primary and battery circuits.

Transient overvoltage conditions on mine power systems, caused by switching or in some cases lightning, apply large electrical stresses to power transformer insulation. To protect the transformer and also to provide backup for the personnel protection given by the Faraday shield, the transformer insulation should be able to withstand a peak of five times the nominal line voltage peak. Ventilated dry-type transformers designed to IEEE standard 462–1973 (*16*) for low-voltage chargers have a 10-kV BIL, which is well within the transformer transient-protection needs. Accordingly, the voltage ratings of all other charger components are coordinated with the maximum anticipated overvoltage. The transformer secondary, and thus all circuits connected to it (rectifiers, charge-rate control, and timer circuitry) are further protected from transients by surge traps (metal oxide varistors) *f* in figure 15.14.

Outgoing Cables

The cable used to charge the batteries contains grounding and ground-check conductors as well as the two charging power conductors. The battery tray couplers should have four pins to accommodate the cable. Obviously, the general safety requirements for any low-voltage coupler should also be adhered to: for example, each contact should be able to continuously carry the maximum current in the circuit for which it is designed; all exposed uninsulated metallic parts should be grounded to the grounding conductor; and the coupler-cable interface should be designed to prevent cable insulation damage.

The grounding and ground-check conductors are grounded on separate welded studs inside the battery tray. If only one stud is used and this is knocked loose, the ground-check circuit could be intact but the battery tray could be ungrounded.

A logical safety item is provision of insulated strain relief for all cables associated with a charger. At least one previous fatality could have been prevented if cable strain relief had been provided. To increase protection against cable damage, insulated glands should be provided for any cable passing through the charger frame, and there should be adequate storage on the outside of the enclosure for all permanently attached cables. This would alleviate damage caused by leaving charging cables and couplers on the mine floor.

Outgoing Ground-Check Monitoring

The ground-check circuit designated by *h* in the figure monitors the grounding connections to the battery tray: one circuit is provided for each battery on charge. The monitor shown is a simple current-sensing pilot or loop monitor. If the loop formed by the charger cable grounding and ground-check conductors, the coupler contacts, and the battery tray grounding studs is opened, relay K1 will be deenergized and trip the charging power with contactor K3. A serious problem with ground-check monitoring on battery charging systems is the potential for development of parallel grounding paths. For this and other reasons, the monitor must be designed to meet all the guidelines discussed in chapter 9.

Surface Leakage Detection

The charging power will also be tripped if the leakage detector circuit (*i*) senses surface leakage or a ground fault from the battery to the tray. Following work performed by Virr and Pearson (*34*) in the United Kingdom, a total battery leakage resistance threshold of 1,000 Ω appears satisfactory. This value should not have nuisance tripping problems caused by too-sensitive relaying, but should substantially reduce the gas ignition and electrical shock hazards created by such faulting.

Charging Circuit Isolation

The goal of completely isolating the transformer secondary and dc charging circuitry from ground is aimed mainly at preventing shock hazards caused by charger-battery grounding problems. With either polarity grounded, a low-impedance source of dc voltage is available between the frame of the vehicle being charged and the ungrounded battery side. A person touching the vehicle frame and any battery intercell connector could then receive a dc shock. The same would be true for anyone standing on a wet mine floor and touching any ungrounded terminal connected to the charging circuitry. If the positive or negative charger terminal is grounded, this would also promote excessive corrosion of the battery tray.

State-of-Charge Indication

An ammeter (*k*) is provided so that at any given time it is obvious what part of the charge cycle the system is operating in. Strictly speaking, this is not a safety feature, but having knowledge of the status of the charge cycle is an aid to good operating procedures, which in turn can contribute to safety.

Another valuable feature would be a fail-safe means of preventing overcharge. The principal goal here is to protect against excessive battery gassing, and as a first line of defense, almost all manufacturers use timing circuitry to reduce the charging current substantially through the last part of the charge cycle. However, there are common occurrences that call for additional protection. For instance, consider that the charge cycle is interrupted during charging, say by a charger component failure, a power failure, or someone disconnecting the charger from the batteries and then reconnecting them. The bad component could cause the charger to continue the charge at the starting rate for an indefinite period. In the last two cases, the main timing circuitry could reset to zero, and an unknowing individual might manually restart the charger for a full charge cycle. Considering the real world, a "fail-safe" timer would need to deenergize the charger with the occurrence of any failure mode that could lead to overcharge. Many of these failures are perceivably beyond the control of practical timing circuits; thus, a more reasonable safety requirement would be having the charger time circuitry operate to minimize battery overcharge as much as practical.

One means of affording protection against overcharging would be to have a redundant timing device that would override the main timing circuitry and shut the charger down after a set period. The timers (*e*) in figure 15.14 perform this function: their dc motors are tied to the charging power and have automatic reset capability. The time deenergizes the charger after a maximum operational period measured from its initial setting.

Overcurrent Protection

Fuses (*d*) and (*l*), in series with each ungrounded incoming and outgoing power conductor, provide overload and short-circuit protection. A circuit breaker could also be used for the transformer primary circuit, but this is probably not suitable for outgoing-circuit protection. Other relaying could also be included for semiconductor protection, as discussed in chapters 12 and 14.

Additional Features

The preceding discussion covered the minimum number of features that should be included in a mine charger, but some additional items are also desirable. One of these is a contactor (*i*) for both the positive and negative outgoing conductors, which should be located near the point where charging power leaves the enclosure. This would remove battery power from any uninsulated component within the enclosure if the charger is opened and the charging cable is still connected. Accordingly, the contactor would trip any time the ac ground-check circuit is broken. Because the contactor load contacts might still be energized upon entry, the device should have an insulated cover with a warning notice.

There have been many instances where charging couplers have been removed during the charge cycle, with the charger not turned off. The energized plug has been left on the mine floor, at times lying in water. The outgoing ground-check circuit will prevent this, as well as any arcing that might be encountered during plug removal. As a backup to power tripping, it would also be advantageous to have the manual on-off device simultaneously reset to *off*. This could be a simple switch tripped by the dc ground-check monitor or a manual timer that is capable of being automatically reset to zero (component *j* in figure 15.14). Automatic restarting of the charger (for example, when the battery plug and receptacle are reengaged) is inadvisable as it could pose a hazard to the unwary miner. Manual restarting is preferred as this necessitates a deliberate act by the user before the charger can be energized.

A very serious and quite obvious hazard can also occur if an energized charger is connected to a battery of wrong polarity. It is essential that the control circuitry be able to sense incorrect polarity and if it exists, prevent energization of charging power. Many maintenance personnel have received burns from battery-energized couplers during repair, maintenance, or replacement. Hence it is necessary that the battery terminal connections be constructed so power to the couplers can be removed.

This chapter has scanned the subject of batteries and battery charging in mining. Some significant points have been covered. A good battery-maintenance program is essential for successful battery use. Adequate ventilation of the typical charging station requires little more than common sense. The batteries must be located in the mainstream of airflow during charging, and provisions should be made to deliver 140 ft^3/min for each battery on charge. Battery lids should be removed during charging. Catalyst battery caps might be considered as a viable alternative to ventilation and lid removal for handling hydrogen accumulation problems. Safe battery chargers can be made and should be used to provide a larger safety factor in the battery charging and usage process.

The Federal requirement to deenergize the mine electrical power system in the event of a mine ventilation failure raises an unresolved question concerning battery employment. System deenergization should include all nonpermissible battery-powered systems at the source. Because batteries are independent and cannot sense a mandatory power shutdown, a "dead-man" type of device may be needed on battery-powered vehicles. This device would be timed to interrupt the battery power, perhaps 20 min after the machine is stopped. The operator would be required to engage an interlock before the machine could be reactivated. Such a device could be interlocked through the battery tray coupler, thus deenergizing the receptacle contacts when the plug is not present.

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CHAPTER 16.—PERMISSIBILITY AND HAZARD REDUCTION

Any industrial area in which flammable or explosive gases, vapors, and dust can be encountered is designated as a hazardous location. Since the occurrence of a hazard depends upon the presence of an ignitable mixture, an ignition source, and contact between them, the chance of ignition is always present when electrical apparatus is used in hazardous atmospheres. The probability of ignition cannot be brought to zero in any portion of an underground mine, nor in selected parts of surface mines and surface facilities such as preparation plants. Hence electrical hazard reduction techniques must be applied in these areas to protect both personnel and equipment.

TERMINOLOGY

Important measures used to reduce incendive hazards in underground mines include provision of adequate ventilation, control of flammable coal dust through mandatory rock dusting and watering, and the regulation of equipment. The last item is the one of greatest interest here. The regulation of electrical face equipment in underground mines is specified in 30 CFR 18 (*37*).[1]

The responsibility for ascertaining compliance with this document is assumed by the U.S. Department of Labor, Mine Safety and Health Administration (MSHA), Approval and Certification Center. It is important that the terminology for Part 18, "Electric Face Equipment," is clearly understood.

Approval: This term applies to completely assembled electrical machines and accessories. *Accessories* mean associated electrical equipment such as a distribution or splice box that is not an integral part of the machine. *Approval* means that a formal document has been issued by MSHA, which states that the machine or accessory has met the applicable requirements of the regulation. An *approval plate* is then attached to the approved machine or accessory identifying it as suitable for use in hazardous locations. Such equipment is subsequently referred to as *permissible equipment*. This process is mandatory for all electrical equipment used inby the last open crosscut and in return air of an underground coal mine.

Certification: This term applies to an electrical *component*, that is, an integral part of an electrical machine or accessory that is essential to the functioning of the machine. *Certification* means that a formal written notification has been issued by MSHA, which states that the component complies with Federal requirements and is suitable for incorporation in a permissible machine.

Acceptance: This term applies to flame-resistance requirements and to auxiliary equipment such as a cable, hose, or belt. *Acceptance* means that written notification has been received from MSHA designating the equipment as meeting requirements for flame resistance. *Acceptance marking* is the identification that appears on the equipment.

Intrinsically safe: This identifies equipment that is incapable of releasing enough electrical or thermal energy under normal or abnormal circumstances to cause ignition of a flammable mixture.

[1] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

Hazardous locations in surface mines and surface portions of underground mines are often classified by guidelines specified in the National Electrical Code (NEC), article 500 (*23*). In this classification system, the nature of the hazard and the degree of hazard are the main considerations, and a location is specified by a class, group, and division designation. The class refers to the generic nature of the hazardous material, and the following are of specific importance in mining:

- *Class I*: Locations containing flammable gases or vapors that may be present in the air in sufficient quantity to produce an explosive or ignitable mixture.
- *Class II*: Locations having combustible dust in quantities that can cause a hazard.

The group designation is a subclassification and refers to the nature of the hazard. Each group contains a listing of materials that present the same general hazard. The groups of interest in coal mining could include

- *Group B*: Atmospheres containing hydrogen or gases or vapors of equivalent hazard such as manufactured gas.
- *Group D*: Atmospheres containing gasoline, hexane, naptha, benzine, butane, propane, alcohol, acetone, benzol, lacquer-solvent vapors or natural gas (methane).
- *Group F*: Atmospheres containing carbon black, coal, or coke dust.

When more than one hazard is involved, the class and group describing the most serious situation usually applies.

The division defines the probability of a hazardous material being present in an ignitable concentration:

- *Division 1*: Locations where the hazards exist continuously, intermittently, periodically, or where they may exist during maintenance or equipment failures.
- *Division 2*: Locations where hazards are presumed to exist only under abnormal conditions.

An example of a division 2 location is an area rendered nonhazardous by forced-air ventilation that could become hazardous if the ventilation failed. Another instance would be a location where dust layers could accumulate and might interfere with proper and safe heat dissipation from electrical equipment, thereby causing a dust-layer ignition.

The NEC, article 501, contains the general rules applicable to electrical wiring and equipment in class I locations, and article 502 applies to class II locations. Electrical equipment used in preparation plants commonly follows design criteria for hazardous locations established by organizations other than MSHA, but MSHA has jurisdiction. The National Electrical Manufacturers Association (NEMA) defines criteria for explosion-proof and dust-ignition-proof enclosures, motor classifications, and insulation classifications (*22*). Underwriters' Laboratories, Inc., lists motor and control standards for hazardous locations (*35*).

HAZARD-REDUCTION METHODS

Hazard reduction is a practice in low probabilities, specifically low incremental probabilities, and relies on the fact that a safe electrical installation in a hazardous location does not significantly raise the probability of fire or explosion above that existing without the equipment (*14*). The methods adopted to reduce hazards are based on the principle that the occurrence of a hazard depends upon the presence of an ignitable substance, an ignition source, and contact between them. The methods focus on one or another of these interrelated factors.

Among hazard-reduction methods used in or around mines, explosion-proof and dust-ignition-proof containers are the most important. Here, internal ignition is possible but the resulting combustion is so well controlled and contained that hazard is prevented. Since explosion-proof containers are so widely used in the mine environment, they will be discussed in the greatest detail in this chapter, but first, other, less common methods will be outlined and the topic of intrinsic safety will be covered. Two systems used in other industries, lamina and labyrinth, are variations of explosion-proof enclosures and will not be discussed.

Increased safety or protection type "e," as defined by the International Electrotechnical Commission, is an approach in hazard reduction where special design considerations are given to ensure an extremely low probability of electrical or mechanical breakdown that could produce a spark or temperature rise (*8*). As in *intrinsic safety*, the principle is to remove the ignition source. The technique is widely used in the Federal Republic of Germany and is frequently applied in mine lighting fixtures. Increased safety designs include such features as large spacings and creepage distances between live parts, protection against hot spots, large rotor-stator clearances for motors, superior amounts and quality of insulation, and special enclosures and fastenings to prevent unauthorized entry (*8*).

Immersion systems rely on controlled environments to reduce or remove hazardous atmospheres from ignition sources. In sealing or potting methods, granular material such as sand is used to encapsulate the potential hazard. This does not actually isolate the source but quenches any incipient flame, thus preventing any effective contact between the source and the hazardous atmosphere. Purging or pressurized systems use liquid (oil) or inert gas to provide protection (*14*). The pressurized approach is rarely used in mining applications (*30*).

The basic problem with most of these hazard-reduction methods is their complexity. The overriding considerations for electrical hazard reduction in mining are robustness, interchangeability, and mobility; hence, there is no indication that bolted explosion-proof enclosures will lose their popularity in the near future.

As previously defined, intrinsically safe equipment is incapable, under normal or abnormal conditions, of releasing sufficient energy to cause an ignition of the most ignitable methane-air atmosphere (*39*). The principle applies to complete electrical circuits and not to individual components. The advantage is that safety is inherent in the design and is difficult to defeat. But safety can be compromised easily by maintenance.

Normal conditions are taken to include the effects of extreme power-supply and environmental variations (within the equipment specifications) as well as the opening, faulting, or grounding of all conductors leading to the apparatus (*14*). Abnormal conditions involve all failures of internal components and wiring.

The design of intrinsically safe circuits is a subject too vast in scope to cover adequately in this chapter, but some of the most important considerations are

- Maintenance of suitable spacing between uninsulated conductors (only if firmly tied down);
- Provision of additional insulation, barriers, or partitions (isolation by construction);
- The use of isolated transformer windings (for example, axial displacement or grounded shields between primary and secondary windings);
- Isolation by potting or encapsulation;
- Limiting circuit capacitance and inductance (thus, the available stored energy);
- Use of extensive circuit analysis and fault calculations to ascertain available energy under normal and abnormal conditions.

A prime concern with the last item is that with the failure of a single component (or subsequent failures resulting from the failed component) the device should remain intrinsically safe or fail without creating a safety hazard (*37*). The fundamental design concept usually applied is that two unrelated failures, each independently detectable but occurring simultaneously, must not compromise safety. Hence, the unsafe condition could occur only with a third failure. Two safeguards always exist between the safe and unsafe conditions (*14*).

References *14*, *31*, and *40* can be consulted if additional information is required. Also, 30 CFR 18 contains extensive requirements and tests for approval of intrinsically safe equipment (*37*).

EXPLOSION-PROOF ENCLOSURES

The explosion-proof enclosure used in U.S. mines complies with the applicable design requirements of 30 CFR 18, subpart B. It is able to contain internal explosions of methane-air mixtures without undergoing damage or excessive distortion of its walls or covers, without causing an ignition of a surrounding methane-air mixture, and without discharging flame from the inside to the outside of the enclosure (*37*). Outside the United States, the same definition often refers to flameproof enclosures, but the words *discharge of flame* are omitted (*7*). Although the terms *flameproof* and *explosion-proof* commonly have the same connotation, flameproof enclosures are not presently allowed in U.S. underground coal mines unless they also meet the 30 CFR 18 requirements.

Explosion-proof enclosures are found on all electrically powered face mining machines in U.S. coal mines and all motor applications in class I, division 1 locations in all U.S. industries. These enclosures have heavy-walled cast or welded construction and bolted or threaded close-fitting flanges. However, they are not necessarily vapor tight. Ignitable gas may enter a properly secured explosion-proof enclosure in several ways (*11*). Enclosures on machines allowed to stand for several hours in a gas-filled place can become completely filled with that gas from diffusion through openings, even though those openings are very small. Another process, *breathing*, results from the expansion of the enclosure atmosphere during operation and contraction as it cools at rest. Air is forced out during expansion, and the atmosphere outside, including any gas present around the machine, is drawn in when cooling. Gas may also enter the enclosure when covers are removed for inspection and repair.

A methane-air mixture in the proper proportions will explode if an adequate ignition source is present. The approximate explosibility range of methane in air is 5% to 15%, with about 9.8% being the critical point for the most violent explosion in terms of maximum pressure, highest rate of pressure rise, and highest temperature. Methane-air mixtures above or below this range will burn but not explode. The lowest autoignition temperature (ignition without additional energy) is about 550°C *(41)*, but a methane ignition requires that a sufficient gas volume is maintained at or above the autoignition temperature for a period of time. In other words, there is a critical ignition energy that must be injected to sustain combustion *(14)*. Empirically, this minimum has been found to be 0.25 mJ for flammable methane-air mixtures *(13)*.

Ignition of the explosive gas by electrical means can be triggered in several ways. Arcs of sufficient energy, termed *incendive*, can result from normal or abnormal operation of electrical devices. Normal incendive-arcing modes can be attained during contact closure or even between fixed electrodes in a capacitive circuit, by opening contacts in inductive circuits, and by opening or closing contacts in resistive circuits *(14)*. In the first two cases, energy stored in the capacitance or inductance is released to the arc as the contacts close or interrupt current, respectively (see chapter 9 on arcing and chapter 11 on energy storage for details). Examples of abnormal incendive arcing could involve faulted components or intermachine arcing, as discussed in chapter 17. Another source of ignition is contact of the gas with hot surfaces, but this depends on the surface's heating sufficiently above the autoignition temperature *(14)*. One instance of a hot surface would be excessive current through fine conductors, such as in a damaged cable.

Explosion Transmission

Since an internal explosion is a possible occurrence, escaping gases generated in the explosion must not have sufficient energy to propagate the explosion to any hazardous atmosphere surrounding the enclosure *(14)*. Conventionally, these hot gases are allowed to escape only through specially designed openings in the explosion-proof enclosure that provide long, narrow quenching distances. It is important that the gases be allowed to escape, otherwise internal pressures capable of rupturing the enclosure might develop.

Although explosion-proof enclosures have been researched since around 1906, little concrete information has been gained about the phenomena that make them effective, but it is accepted that cooling and inhibition are two of the more probable mechanisms *(14, 27)*. The hypotheses can be explained with reference to figure 16.1, which shows a cross-sectional sketch of a typical explosion-proof enclosure. An explosive methane-air mixture exists both inside and outside the chamber. After an explosion is initiated inside the enclosure, a flame front propagates toward the chamber walls, burning the available combustible material. This raises the temperature and pressure inside the chamber, and unburned gas, then heated burned gas, is forced through the flange gap. The jet of heated gas is cooled first by heat transfer within the flange gap. As it exits from the enclosure, it may be further cooled by adiabatic expansion into the surrounding atmosphere; during this process, combustible gas from the outside methane-air mixture is also entrained in the jet. Although the additional gas provides fuel for combustion, it further cools the escaping gases. If the sum of this cooling does not lower the jet temperature below the gas ignition temperature, the explosion will propagate to the surrounding atmosphere. However, if the jet entrains an excessive amount of combustible gas in this last phase, the heat supplied by the jet will be less than that lost through cooling, and no ignition of the outside gas will occur.

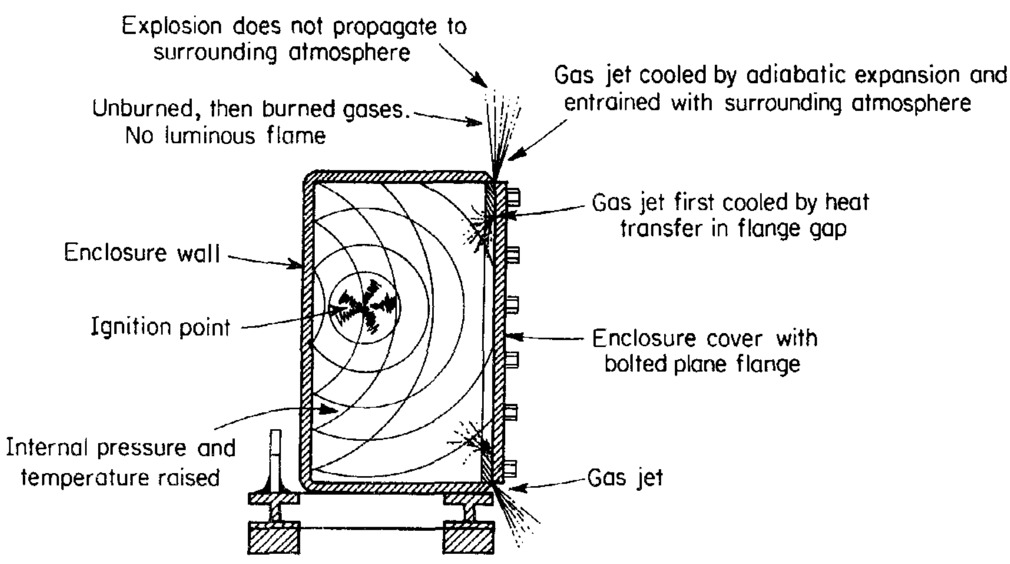

Figure 16.1.—Cross-sectional sketch of typical explosion-proof enclosure.

The action within the flange gap is believed to go beyond cooling. Combustion of a gas-air mixture does not proceed with a single chemical reaction but rather a chain of reactions. If the chain carriers or active molecules from a preceding step are inhibited, for instance by contact with the flange wall, then combustion stops. This theory is also used to explain the protection properties of flame safety lamps.

The properties of the explosion-proof enclosure that reduce hazards have been discussed using the flange gap as an illustration, but other enclosure openings such as pressure and relief vents provide the same protection phenomena (*6*). From this discussion of the theory it is obvious that the design criteria for joints and vents are also crucial in the construction of explosion-proof enclosures.

Enclosure Joints

Explosion-proof enclosures must have as few openings as possible to minimize the number of flame paths. It is also essential that limited access be provided for inspection and replacement of parts. The joints formed where covers are placed over the required openings are either threaded or machine surfaced. Flanges formed by machining may be flat, cylindrical, or a combination of both. The most common types of machine joints are the plane flange and the step flange; tongue-and-groove is uncommon. Step flange joints are often a combination of flat and cylindrical flange surfaces, as in the joint between end bells and the frame of explosion-proof motors. Typical cross sections of these joints as well as other popular types of entrances into enclosures are illustrated in figures 16.2 through 16.5. The maximum and minimum dimensional requirements are given in these figures and in table 16.1. Allowable clearances are 0.004 in for a plane flange joint and 0.006 in for a step-flange joint in enclosures with volumes exceeding 124 in^3. A simple check of the adequacy of such joints would be to run a 0.005- or 0.007-in feeler gauge (respectively) along the joint; it should not fit in the gap.

Table 16.1.—Structural gap dimensions for explosion-proof enclosures as specified by 30 CFR 18

Type of joint	Dimension	Empty enclosure volume					
		Less than 45 in^3 (738 cm^3)		45 to 124 in^3		Greater than 124 in^3 (2,032 cm^3)	
		in	mm	in	mm	in	mm
Flanged joints in 1 plane	Minimum width of joint	0.5	12.7	0.75	19.0	1.0	25.4
	Maximum gap (clearance)	.002	.05	.003	.076	.004	.10
Step-flange joints in 2 or more planes (cylinders or equivalent).	Minimum width of joint [1]	.375	9.5	.625	16	.75	19.0
	Maximum gap:						
	Plane portion	.006	.15	.006	.15	.006	.15
	Portion perpendicular to plane [1]	.008	.20	.008	.20	.008	.20
Cylindrical joints other than shafts	Minimum length of flame path	.5	12.7	.75	19.0	1.0	25.4
	Maximum radial clearance	.0015	.04	.002	.05	.003	.076
Shafts with sleeve bearings	Minimum length of flame path	.5	12.7	.75	19.0	1.0	25.4
	Maximum radial clearance	.003	.076	.004	.10	.005	.13
Shafts with ball or roller bearings	Minimum length of flame path	.5	12.7	.75	19.0	1.0	25.4
	Maximum radial clearance	.010	.25	.0125	.55	.015	.38
Bolts	Minimum diameter	.25	7	.25	7	.375	10

[1] If perpendicular portion is more than 1/8 in but less than 1/4 in, maximum radial clearance shall not exceed 0.006 in. Neither plane nor radial portion shall be less than 1/8 in.

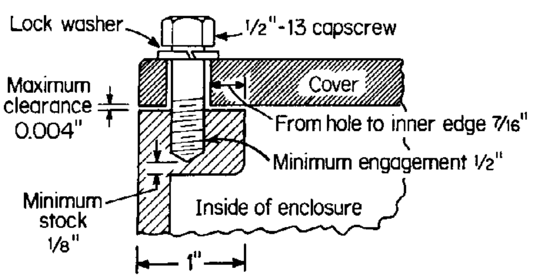

Figure 16.2.—Typical plane-flange joint; enclosure internal volume larger than 124 in³.

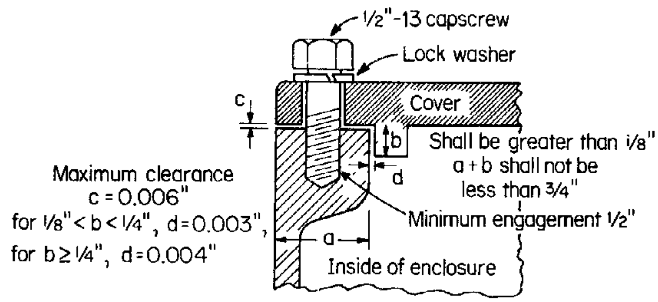

Figure 16.3.—Typical step-flange joint; enclosure internal volume larger than 124 in³.

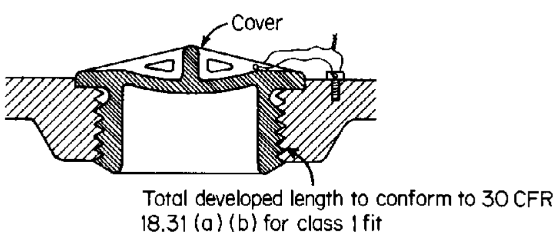

Figure 16.4.—Threaded joint.

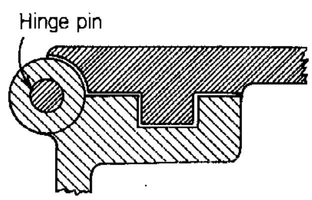

Figure 16.5.—Tongue-and-groove joint.

Bolting holes must not penetrate to the interior of the enclosure; otherwise, the omission of a bolt would provide entrance to the chamber. Through-holes must be blind or bottomed as shown in figure 16.6. The maximum spacing between fasteners for joints all in one plane is 6 in. The maximum spacing between fasteners for joints, portions of which are on different planes, is based on the size and configuration of the enclosure, the strength of the materials, and explosion test results. The bolts or other means of clamping the joints should be proportioned to minimize stripping of threads and to give adequate strength from the stress developed during an internal explosion Minimum bolt diameters are also given in table 16.1.

The minimum distance from the enclosure interior to the edge of the bolt hole is important as it must provide an adequate flame path length. The minimum Federal requirements (considering just flange width) are (*37*)

Joint size, in	*Distance* *in*	*mm*
½	⅛	3.2
¼	3/16	4.8
1 or over	7/16	11.1

Figure 16.3 shows how these measurements are applied.

The Maximum Experimental Safe Gap

To help understand the joint criteria in conjunction with the theory of explosion-proof enclosures, a presentation of past research is in order. The external ignition-suppression properties of flange joints have been an active research area for several years, with extensive investigations carried out in the United States, the United Kingdom, and the Federal Republic of Germany. Most research work has resulted in similar conclusions, and the general international consensus is that design standards for explosion-proof enclosures are imperative.

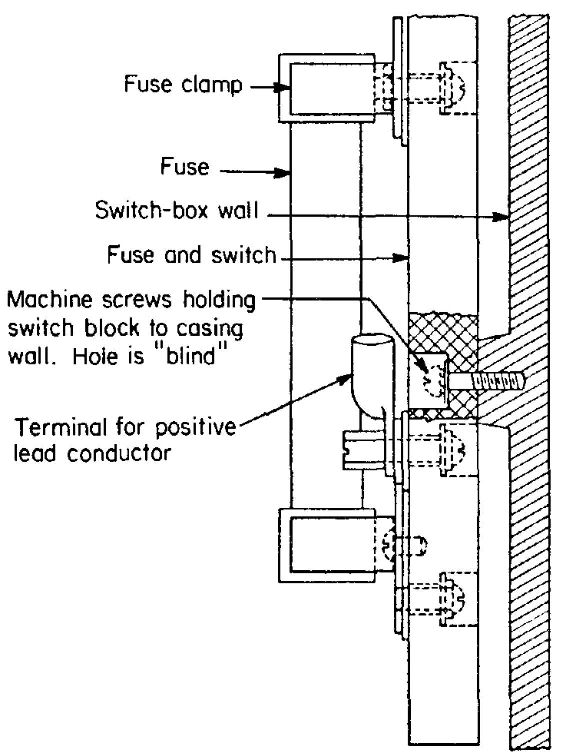

Figure 16.6.—Blind screw hole.

The maximum experimental safe gap (MESG) is a standard often used to determine the explosion-transmission properties of a flame path. The MESG is measured by igniting a flammable gas mixture inside a test system and observing if it ignites a surrounding gas mixture outside the enclosure (*14*). The MESG is the largest gap for a flame path length that does not permit ignition outside. Relatively speaking, a higher MESG implies a safer situation. Note that a luminous flame is permitted to pass through the flange, provided that it has insufficient energy to ignite the surrounding atmosphere. This does not coincide with the Federal requirements given earlier, since U.S. regulations do not allow passage of any flame (*37*). A gap that will not allow ignition is 7 to 12 times larger than a gap that quenches visible flame (*14*). The following general statements can be made about past research findings (*14*, *25–26*); a parenthetical comment is made where these conflict with present Federal regulations.

1. The MESG increases as the enclosure volume decreases because thermal losses predominate in small enclosures. Accordingly, smaller flange widths with the same gap can be allowed for smaller enclosures.
2. It is not likely that gaskets or O-rings used to make explosion-proof containers weathertight have any significant effect on enclosure safety, as long as they are external from the flange surface. It can be noted that O-rings are not permitted under 30 CFR 18 (*37*).
3. There is no evidence that metallic materials are essential for the flange construction. (Federal regulations generally require metal-to-metal joints except for headlight and meter enclosures. Metal-to-metal flanges usually provide a better heat-sinking effect.)
4. Surface finish is not considered critical by most authorities. (Federal regulations require that flat surfaces be planed to within one-half the maximum clearance allowed, and finished to not more than 250 μin.)
5. The apparent safe gap between bolted flanges, the normal commercial construction, is considerably smaller than would be predicted from immovable or nonbolted flange tests.
6. Evaluating the effectiveness of explosion-proof enclosures or studying flange-gap quenching is still highly empirical.

Certain researchers have dealt with other specific subjects pertaining to the MESG. Torry (*33*), for example, established that no effect can be ascertained in the MESG over the humidity range of 0% to 50%, but the MESG increases as humidity increases from 50% to 100%. Consequently, the MESG can be measured with confidence when humidity is under 50%. By varying the methane-air mixtures inside and outside, in addition to in the flange gap, Titman and Torry (*32*) found the most ignitable outside methane-air mixture was 6.5%, with a 9.5% methane-air mixture inside the enclosure. James (*10*) varied the temperature in several plane-flange experiments. Using a 1.0-in (25.4-mm) joint, he found the gap that would just permit propagation to the outside was 0.045 in (1.14 mm) at 80°F (25°C), and 0.035 in (0.89 mm) at 500°F (260°C). He stated that with a safety factor it is unwise to exceed a gap of 0.020 in (0.51 mm) for a 1.0-in flange width. Furthermore, a gap of 0.02 in or larger cannot be applied to plane flanges without a consideration of enclosure material strength, since one part of the flange is a plate that could spring or bend during an internal explosion, thus increasing the gap. James also showed that explosion-proof enclosure safety is influenced more by slight changes in gaps than by changes in the flame path.

Several researchers have discovered that doubling the flange width, with other conditions remaining constant, increases the MESG by about 1.3 (*14, 25, 32*). This statement is limited to flange widths 1.0 in and less because few data are available for wider flanges, which are uncommon. As the flange width nears 1.0 in, the relative effect of widening decreases. Phillips (*25–26*) determined quantitatively that the MESG should be smaller if the initial enclosure temperature is raised, and furthermore, that turbulence outside the enclosure increases the MESG. He substantiated empirically the enclosure-protection theory presented earlier, and most of his work has been further substantiated by the experimental results of other researchers.

The standards established in 30 CFR 18 (*37*) and by Underwriters' Laboratories, Inc. (*36*) do not permit passage of flame and specify almost identical flange-gap and width values. This means that the allowable gaps found from MESG research are 8 to 10 times larger than the maximum allowable gap in the United States. Some foreign countries specify the gap values as a fraction of the MESG, and the question has been asked as to why the United States does not allow such larger gaps. The U.S. requirement can be readily justified by the need for a safety factor that will accommodate the problems encountered with deterioration resulting from exposure to the mine environment or neglect. In fact, Short (*28*), in discussion with various foreign testing authorities, has found that none of them will accept a product with gaps as large as those indicated by MESG research. In practice, many countries permit gaps two to four times larger than those specified in the United States.

Pressure Vents

Some manufacturers add special pressure-venting devices that supplement the conventional release of pressure through flange and shaft gaps. The concept is illustrated in figure 16.7 (*6*). Properly sized pressure vents can reduce the internal pressure during explosion to 12 to 20 psi (*6*). The vents exhibit a large effective area for gases from an ignition but provide cooling and inhibiting to arrest the flame. A prototype vent developed under Bureau of Mines contract is shown in figure 16.8 (*6*). Here the escaping gas is filtered through flame-arresting material.

Cable Entrances

The explosion-proof enclosure must also, obviously, have openings through which electrical connections can be made. These openings are particularly important because of the frequency with which trailing cables must be replaced on permissible mobile equipment. The most popular type of cable entry incorporates a packing gland or stuffing box, which may have straight-through entry or angled entry. Figures 16.9 through 16.11 show cable entries with slip-fit stuffing boxes, where the box is a separate component with a cylindrical projection that fits into the enclosure wall (*37*). Specifications for enclosures exceeding 124 in^3 include a flame path length of 1.0 in and a radial clearance between the wall and box not to exceed

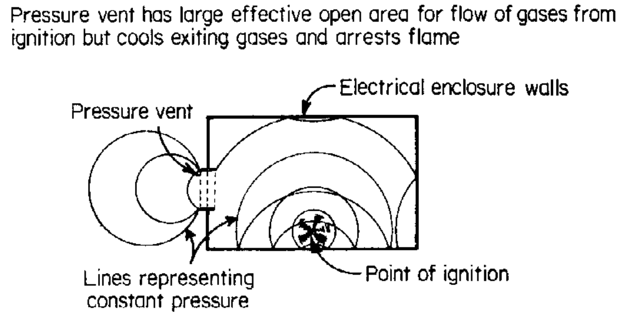

Figure 16.7.—Pressure vent limiting pressure buildup during internal explosion.

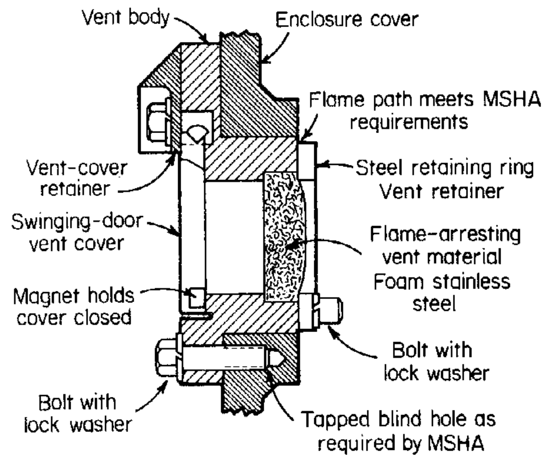

Figure 16.8.—Pressure vent assembly using metal-foam material.

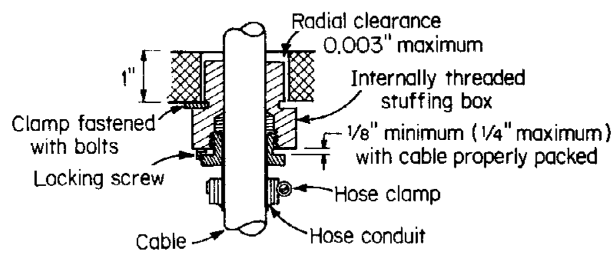

Figure 16.9.—Typical slip-fit straight stuffing box and packing-gland lead entrance.

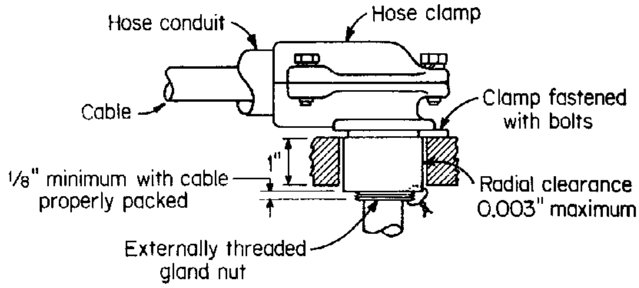

Figure 16.10.—Typical slip-fit angle stuffing box and packing-gland lead entrance with hose clamp.

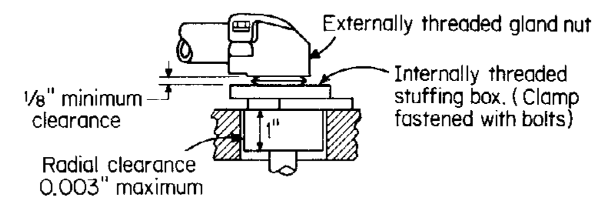

Figure 16.11.—Typical slip-fit angle stuffing box and packing-gland lead entrance.

0.003 in. Figure 16.12 shows how a spare cable entrance hole must be plugged, and figure 16.13 illustrates a packing gland that is an integral part of an enclosure (*37*). A stuffing box is usually constructed so that when the packing-gland nut is threaded into the opening and tightened, it forces the stuffing material against the cable, making a very tight joint. The packing material is usually untreated asbestos or an MSHA-accepted asbestos substitute material. When compressed, there must be at least ½ in of packing along the length of the cable, and the clearance between the packing-gland nut and the stuffing box must be no less than ⅛ in (*38*).

Repacking a stuffing box in the mine can be extremely awkward and time consuming. A recently introduced method that uses a tapered polyurethane flame-resistant grommet in place of the asbestos (fig. 16.14), greatly simplifies the assembly (*6*). The elastomeric grommet has high compressibility so that one size can accommodate a limited range of cable sizes.

The requirements for leads that pass between explosion-proof compartments separated by a common wall are not as rigid as those for leads passing through an exterior wall. One type, an insulated stud entrance, is shown in figure 16.15. Here the conductor does not actually pass through the wall but is connected to a finished brass casting stud that is isolated from the enclosure wall by an insulated tube and washers.

Windows and Lenses

Federal regulations state that MSHA may waive window and lens material testing except for headlight lenses (*37*). All window and lens material must be sealed in place or provided with correct flange joints and must be protected from mechanical damage, either by guarding or inherently through location and structural design. If the exposed material area exceeds 8.0 in^2 (51.6 cm^2), the window or lens (excluding headlight lenses) must be protected with guarding or the equivalent. Both thermal-shock and impact performance tests are outlined in the Federal regulations.

Enclosure Mechanical Strength and Internal Pressures

The maximum internal pressure developed during the ignition of an explosive air mixture in an enclosure is directly related to the amount of venting through flange gaps and any auxiliary pressure-relief devices. Tests of mechanical strength and internal pressures provide important parameters used to design enclosures that will not transmit the explosion to a surrounding atmosphere. Measurements of explosion pressure are made to prove that the container strength is adequate. It is particularly important to ensure against flange-gap distortion.

Plug shall be secured by spot welding or brazing. Weld may be on plug, clamp, or fastening bolt.

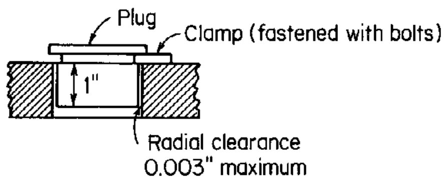

Figure 16.12.—Typical plug for spare lead-entrance hole.

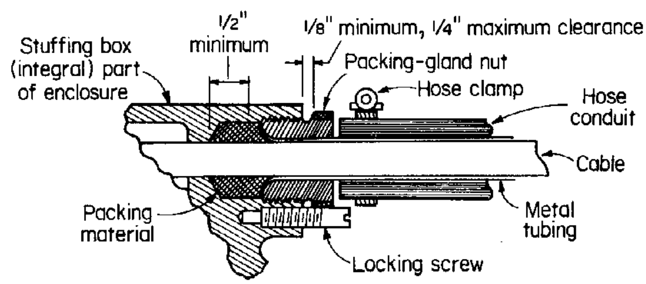

Figure 16.13.—Typical threaded straight stuffing box and packing-gland lead entrance with provision for hose conduit.

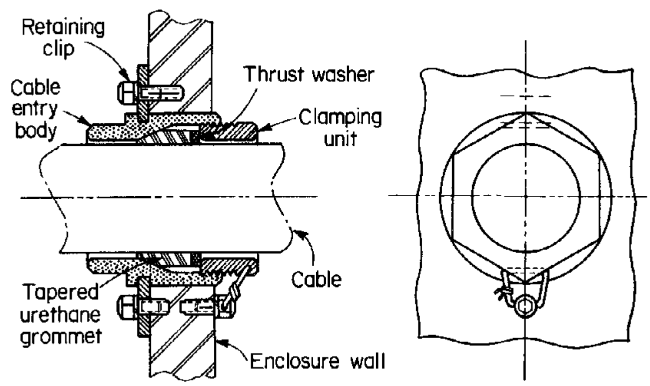

Figure 16.14.—Prototype trailing cable entry with polyurethane grommet.

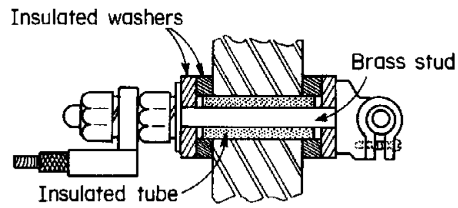

Figure 16.15.—Insulated-stud lead entrance.

30 CFR 18.31(a) specifies that cast or welded enclosures must be designed to withstand an internal pressure of 150 psig (1.04 MPa) and that casting must be free from blowholes. Welded joints forming an enclosure must be continuous, gas-tight welds in accordance with American Welding Society standards. Minimum allowable thicknesses for enclosure walls, flanges, and covers are also outlined. Again, a review of relevant research findings assists in understanding these regulations.

Nagy (*21*) conducted extensive tests on methane-air and coal-dust explosions to determine pressures with varying enclosure geometries. A range of 38 tests was performed on enclosures with internal volumes from 0.043 ft^3 (101.5 cm^3) to 905 ft^3 (25.63 m^3), and methane-air mixtures from 6.0% to 14.1%. He found that with an ambient internal pressure of about 14.2 psia (98 kPa), the maximum pressure developed for all vessels was 119.6 psia (0.825 MPa). When the ambient internal pressure was at 17.6 psia (121 kPa), the maximum was then 143.3 psia (0.988 MPa). Both maximums were obtained from examination of a 1.0-ft^3 (2,359-cm^3) vessel with 9.4% methane-air mixtures. From 39 other experiments, using the same enclosure but a Pittsburgh coal-seam dust, a maximum of 119.3 psia (0.823 MPa) occurred with a concentration of 0.6 oz/ft^3 (601 g/m^3) in an 0.32-ft^3 (7.55-cm^3) vessel. Nagy concluded that

- Vessel size does not affect the maximum pressure level if heat loss is neglected;
- The rate of pressure rise decreases as the vessel size is increased;
- The vessel shape does not affect the maximum pressure if heat loss is neglected;
- A change in initial pressure produces a proportional change in the maximum pressure and the rate of pressure rise;
- The changes in the initial temperature produce an inverse change in the maximum pressure and have little effect on maximum rate of pressure rise.

Considering that the maximum pressure developed in an explosion is a function of venting, the closed-system enclosure tests carried out be Nagy could be taken as the worst case. The Federal regulations require enclosures to withstand 164.7 psia (1.136 MPa) assuming standard atmospheric pressure, or two times the maximum pressure recorded during any tests without any pressure limit (*37*). The maximum permissible flange gap is rather small and therefore restricts venting. Consequently, the Federal specification does provide some margin of safety.

Magison (*14*) suggests that one of the most significant general facts about the performance of explosion-proof containers is that higher maximum pressures and rates of pressure rise are associated with small flange gaps (for the same flange width with identical explosive gas mixtures). Referring to the work performed by the Safety in Mines Research Establishment (*34*), he states that the venting effect of the MESG in an 8-L sphere (488 in^3) reduces maximum pressures to a few pounds per square inch (ignition of 9.8% methane-air mixture). Magison indicates that, when considering heat loss, the actual maximum pressure achieved in an enclosure will depend upon the location of the ignition source, the container size and geometry, plus the arrangement of inside equipment. Theoretically, enclosure size has no explicit effect on the maximum pressure, but research has shown that size does have an effect for lean and rich mixtures (*40*). Here, the explanation given is that a longer reaction time occurs in larger vessels and allows for more heat loss.

MSHA specifies twofold pressure tests to ascertain conformance with strength requirements. The first test is mainly to determine the explosion-proof characteristics, specifying that an explosive mixture within the enclosure will be ignited electrically and the subsequent explosive pressure will be recorded. The test must be repeated a minimum of 16 times to accommodate various ignition points, different atmospheric composition, and the operating status of the motors. No part of the enclosure must rupture as a result of these tests, nor have permanent distortion exceeding 0.04 in per linear foot. If 125 psig (0.862 MPa) is exceeded during any test, the enclosure can be rejected unless it is reconstructed to reduce the pressure or, alternatively, is able to withstand a dynamic pressure twice the highest value recorded in the initial test. A second or static-pressure test is applied only when MSHA's Approval and Certification Center decides that visual inspection is inadequate to reveal casting or welding defects. The pressure applied in this test is 150 psig or 1.5 times the maximum value recorded during the explosion tests, whichever is greater.

The external surface temperature of all permissible mechanical or electrical components for use on electrical face equipment must not exceed 150°C under normal operating conditions in U.S. mines (*37*). Underwriters' Laboratories, Inc., has a similar specification but it applies only to dust-proof enclosures, which will be discussed later. The regulation is concerned with the possibility of causing an ignition through surface contact between the enclosure and any of the four potentially hazardous mixtures: methane-air, coal-dust-air, combined methane and coal-dust mixtures in air, and collected coal dust on the enclosure surface. The lowest autoignition temperature must obviously form the basis of the regulation.

Of the methane-air mixtures, a 2.9%-methane mixture had the lowest autoignition temperature of about 550°C, which can be eliminated immediately (*41*). Measurements to establish the minimum ignition temperatures of coal-dust clouds and dust layers have been performed by Nagy (*20*) on several U.S. coals. The dust cloud experiments on 22 different coals gave an average minimum ignition temperature of 617°C. The lowest ignition temperature was 440°C for a Colorado high-volatile coal. Tests performed on layered dust of 16 U.S. coal samples showed an average minimum ignition at 222°C, with the lowest temperature being 160°C for a high-volatile Illinois coal. Ten of the layer tests resulted in minimum ignition temperatures of 190°C or below. Hence, layered coal-dust ignition temperature is the limiting factor for the maximum allowable surface temperature of explosion-proof enclosures and justifies the 150°C requirement of the Federal regulation. The requirement is particularly significant for equipment in underground mines and preparation plants where dust accumulates readily.

Enclosure Hazards

In recent years, some mining experts have questioned the inherent safety of explosion-proof enclosures. The concern is that when an explosion is triggered because of an arc or short circuit within the enclosure, gases can be generated that would defeat the containment properties of the enclosure. A number of failure modes are known to be possible when a high-voltage short circuit occurs within the enclosure: these include burning through the enclosure wall, ignition transmission by hot gases or flame, particle ignition transmission, and enclosure deformation or bursting.

The possibility exists that once an arc is established within an enclosure, it may jump across to the enclosure wall, and the intense heat and power might then cause the arc to burn through the wall. Killing and Tielke (*12*) have demonstrated this possibility quite convincingly.

If organic insulation is used within the enclosure, combustible hydrocarbons may be present under fault conditions that could lead to defeat of an ignition-containment requirement that has been designed only for methane.

Arc formation between two metal conductors (usually copper) within an enclosure can also cause ignition of the surrounding atmosphere by hot particles expelled through the flange gap. Heat from the arc causes metal particles to be separated from the conductors and subsequently forced through the gap by pressure buildup. These hot metal particles are capable of igniting a combustible gas-air mixture. Hence, the MESG must be small enough to prevent passage of such particles, further justifying the small gaps required by U.S. regulations.

However, if enclosure gaps are made small enough to ensure that ignition of exterior gases is not caused by expulsion of hot gases, flame, or hot particles, the possibility then exists for high pressures to build up within the enclosure under fault conditions. Covers could then be blown off, with subsequent ignition of the surrounding atmosphere. Davidson and Lord (*5*) cited half a dozen such incidents in the Federal Republic of Germany, one in Canada, and two in the United Kingdom. The flameproof enclosures that burst in Canada and the United Kingdom had gap specifications similar to those required in the United States. In a typical instance, a steel cover was thrown several meters after shearing apart fourteen ½-in.-diameter securing bolts. In all cases there was severe electrical fault damage within the enclosure, including charring of organic electrical insulating material. Ciok (*4*) reported that high-voltage short circuits within flameproof enclosures have become a matter of concern in Polish coal mines. Such short circuits have occurred several times, and the covers closing the equipment chamber have been blown off.

These high pressures within the enclosures appear to be partly due to the evolution of gases from organic insulating materials incorporated in the enclosure construction. Simon (*29*) described experiments where organic insulating materials within enclosures were subjected to heating by an electric fault arc. He showed that in these tests, gases evolved that were capable of causing sufficient pressure to rupture the enclosure. The evolved gases (volatization products, consisting primarily of hydrogen, carbon monoxide, nitrogen, and methane) ignited spontaneously upon contact with the external atmosphere. If the flange gaps or pressure vents would allow all the volatization products to escape, no enclosure failure would occur, although the nature of the escaping hot gases must be considered.

Another possible contributing factor is an effect investigated by Brown (*3*) and Bossert (*2*). It is known that hydrocarbon-air flames produce free ions that cause an electrical current to flow when the flame front bridges two points of opposite electrical polarity. This current may be sufficient to form an arc discharge; that is, an initial flame might produce ions that contribute to additional faults within the enclosure. The situation might be aggravated by the presence of organic insulating materials: the initial fault may produce hydrocarbon gases through thermal insulation breakdown, and the burning of these gases might induce additional arcing. This possibility would increase as voltage levels were raised, triggering a chain-reaction effect that could culminate in deformation or bursting of the enclosure.

Materials that apparently contribute to overpressures are those that give off gases when subjected to heating. The greatest hazard is from materials that evolve combustible gases, here, insulating materials and accumulated water. When water is subjected to an electrical current, electrolysis takes place and hydrogen and oxygen are evolved. This constitutes a very undesirable situation; hence, water accumulation within enclosures should be avoided as much as possible, for not only does it have the potential of creating faults, but when a fault occurs it may contribute significantly to increased enclosure pressures.

When organic insulation materials are subjected to electrical arcing, tracking or heating, decomposition takes place that yields various gases, both combustible and noncombustible. Over the past 30 yr, electrical equipment manufacturers have shown a tendency to replace traditional insulants (cellulose, natural fabrics, asbestos, mica, porcelain, glass, etc.) with newly developed organic polymers. Yet at the same time, traditional insulants are still in common use. As a result it is possible to find almost any known insulation within an explosion-proof enclosure. The numbers are so great that it is extremely difficult to categorize the many types and variations of insulating and plastic materials that are likely to be found. However, a short summary of insulants that are known to evolve dangerous gaseous products follows (*19*).

Cross-Linked Synthetic Polymers. This group contains the basic synthetic resins commonly utilized in the manufacture of plastic materials. Of these, the phenolic, epoxy, and silicone resins appear to offer the greatest hazard potential because of their high yield of combustible volatile products. The amino (melamine) polyester, polyurethane, and isocyanate resins appear to have a lower yield of combustible volatile products. Materials made with these resins should therefore exhibit a lower, though still quite significant, hazard potential.

Linear Synthetic Polymers and Elastomers. This group contains many of the well-known types of insulators. The worst potential hazards (similar in magnitude to those for the cross-linked group) appear to be associated with polypropylene, polymethylene, polystyrene, polyethylene, neoprene, and polymethyl-methacrylate. Other materials such as nitrile butadiene (NBR) synthetic rubber, natural rubber, styrene butadiene (SBR) (GR S) synthetic rubber, Dacron polyester fiber, Teflon fluorocarbon polymer, polyvinylchloride, polyurethane, polyvinyl formal, and nylon show less though still very significant hazard potential.

Other Possibly Dangerous Materials. Insulating oils should also be considered high hazard materials. Even though the gases formed are initially dissolved in the oil, upon saturation they may eventually evolve, causing an unsafe situation. A similar statement can be made about oil-impregnated paper. Other materials, such as cellulose and cotton, which do not appear dangerous at first glance, could also contribute to a hazardous condition because of the amounts of water and carbon monoxide evolved.

Safe Materials. The electrical insulating materials that should be used whenever possible in explosion-proof enclosures are the electrical porcelains, ceramics, glasses, asbestos, and mica. These materials are the most resistant to the production of gaseous products when exposed to heating or arcing. Note, however, that these recommendations concerning materials consider only the possible contribution of evolved gases to increased enclosure pressures; toxicity is not considered.

Although it does not appear feasible to eliminate all potentially hazardous materials from enclosures, the use of any materials that can evolve gaseous products under fault conditions should be avoided wherever possible. Of course, if a sustained arc occurs, even the conductors are vaporized, so all such statements are relative. Perhaps the best recommendation should be that the materials to be avoided are those that are readily volatilized, particularly those that evolve large amounts of combustible gases.

PERMISSIBLE EQUIPMENT

As defined at the beginning of the chapter, the term *permissible equipment* is applied to completely assembled electrical machines or components that have received official approval from MSHA. The term *completely assembled* means all equipment portions from the protection at the power source to all internal and external components of the machine, including the trailing cable. Permissibility requirements have been mentioned at various places in other chapters; details of grounding requirements were given in chapter 7, trailing cables and components in chapter 8, protective devices in chapters 9 and 10, battery equipment in chapter 15, and explosion-proof enclosures in this chapter. The aim here is to demonstrate how this information is tied together, by giving an overview of the procedures used by MSHA to investigate prospective permissible equipment for safety. The overview is followed by a discussion of procedures recommended for checking equipment after it has been placed in service, and for maintaining explosion-proof enclosures and permissible equipment in proper condition.

Permissible Equipment Schedule

This is based on information in a 1954 Bureau of Mines Information Circular (*9*), as updated by MSHA's Approval and Certification Center. As already stated, the published regulations are contained in 30 CFR 18. Other pertinent regulations are found in 30 CFR 19 through 29 for electrical equipment and 30 through 36 for mechanical equipment. Each of these parts is often termed a *schedule.* Schedules are revised from time to time to conform to equipment development and to permit as much freedom as possible without lowering standards. Thus, some of the details in the following information may become outdated, but the general nature of the requirements will not change.

Investigations are carried out by MSHA to determine the permissibility of such equipment as continuous miners, shuttle cars, battery-powered vehicles, pumps, distribution boxes and so on, and for certification of components such as explosion-proof enclosures, connectors, and battery assemblies. The investigations are divided into four major consecutive parts: review of drawings to verify that the design meets the requirements, detailed inspection of the equipment, tests of the equipment or internal components in explosive gas-air mixtures and/or adequacy tests where appropriate, then a final inspection of the tested accessories in the completely assembled unit or machine for which approval is requested. Although investigations for the certification of components may follow the same process, only the first three steps are usually necessary.

To initiate the procedure, a written application must be made to MSHA, accompanied by a set of detailed drawings, wiring diagrams, specifications, descriptions, and any related material. Any intrinsically safe components must be stated.

When approval is being considered, only those components that have a bearing on permissibility are studied; only one motor, controller, protective device, or unit of a given design is required. The investigation starts with a check of the drawings and specifications in order to determine compliance with the applicable regulations, and then a detailed check of all parts against the drawings, to see that they coincide. Exact measurements are performed on the dimensions of joints, bearings, pressure vents, and other possible flame-arresting paths in enclosures. For explosion-proof enclosures, for example, the examination determines any unnecessary through-holes, the adequacy of design and construction of cable and lead entrances, the adequacy of electrical insulation and clearances between live parts and between live parts and the enclosure, any weaknesses in welds or flaws in casting, any distortion of enclosures before tests, and the adequacy of the fastenings, including their size, spacing, security, and possible bottoming. The quality of design, material, and workmanship receives careful scrutiny. Only equipment adhering to the following statements will be accepted for further investigation (*37*):

1. "Electrically operated equipment intended for use in gassy mines shall be rugged in construction and shall be designed to facilitate inspection and maintenance."
2. "Only electrical equipment that is constructed of suitable materials, is of good quality workmanship, based on sound engineering principles, and is safe for its intended use" will be tested by MSHA.

The testing phase emphasizes explosion-proof characteristics and component properties. The general nature of the explosion testing has been covered earlier. No less than 16 internal explosions are made at various ignition points, using a methane-air mixture for all, with bituminous coal dust added for some tests. The actual internal electrical equipment, or dummies of equal dimensions, must be in place for a prescribed number of tests. Motor rotors are tested in both stationary and rotating modes. An enclosure can be rejected with the occurrence of any of the following conditions:

- Discharge of flame from any joint or opening,
- Ignition of an explosive mixture surrounding the enclosure,
- Development of afterburning (gas drawn into the enclosure by the vacuum created by the explosion, then ignited within the enclosure),
- Rupture of any part of the enclosure or any panel or divider within the enclosure,
- Permanent distortion of the enclosure exceeding 0.04 in per linear foot.

Other tests and examinations are made to determine the adequacy of components for the intended use. Some of these are performed at the discretion of MSHA investigators.

• Where the durability of a component is in doubt, mechanical tests will be performed to ascertain whether any points need to be strengthened.
• Battery boxes are examined for ventilation, electrical clearances, insulation, drainage, and suitability for specific service.
• Switches, circuit breakers, or contractors intended to function as switches are checked to see if they are capable of interrupting the maximum current permitted by the circuit's automatic protection device.
• Cables, conveyor belting, and hoses are tested for flame resistance.

At the end of the investigation, a final inspection is made of completely assembled new machines or of machines that were previously approved but have since undergone substantial modification. The aim of the final inspection is to uncover any unsafe features, and the inspection includes such items as

• Compliance with joint, lead-entrance, or other pertinent requirements;
• A check of wiring between components and the adequacy of cable clamping and mechanical protection for cables; survey of the positioning of cables, particularly those in proximity to hydraulic components;
• Determining the adequacy of protection against damage to headlights, push buttons, and other vulnerable locations;
• A check of the settings of overload and short-circuit protection;
• Ensuring that there is a suitable means of connecting and protecting the trailing cable.

Finally, MSHA has the option to have a staff engineer check the first machine produced, preferably at the factory where it is built.

When approval is granted, MSHA issues a formal notice of approval, which is sent to the manufacturer. The notice is accompanied by a photograph of an approval-plate design. Plates reproducing the design and required information are mounted in a conspicuous place on the machine, serving to identify the machine or accessory as having met the applicable requirements of 30 CFR 18.

Maintenance of Permissible Equipment

For equipment to retain permissibility, it must be maintained in the same condition as that approved by MSHA. This is the advantage of familiarity with the schedule requirements: the essential details may be extracted to establish a routine maintenance and inspection program that will be in full legal compliance.

Safety can be compromised easily during maintenance procedures. One of the most common problems with explosion-proof enclosures is that small unapproved openings, which can constitute an immediate danger, can appear because of incorrect maintenance. Such openings are rarely caused by a manufacturing error and usually result from negligence or a lack of understanding on the part of the mine maintenance crew. One of the most common of such violations is the citation "open box—0.005," which means that a plane flange has been found to exceed 0.004 in (*16*). Other typical problems that cause dangerous enclosure openings are bottomed bolt and screw holes that have been drilled through, holes drilled through when plates or components have been attached, loose or improperly assembled cable packing glands, or undue wear on bearings where shafts enter an explosion-proof enclosure (*11*).

The following precautions and procedures are recommended in the maintenance of permissible equipment (*11, 14, 16, 38*), but reference should be made to the Federal regulations for precise compliance. Items 1 through 11 concern explosion-proof enclosures specifically, while items 12 through 19 relate to permissible equipment covered in more detail in other chapters. Items 20 through 23 list specific schedule requirements.

1. Before any apparatus is examined, the power source should be deenergized, locked out, and tagged. This procedure is particularly important before an explosion-proof enclosure is opened, although it should be obvious practice for any electrical equipment, permissible or not.
2. All joint clearances should be examined regularly with feeler gauges to determine that the maximum allowable clearances given in table 16.1 are not exceeded.
3. All pressure vents should be examined for cleanliness.
4. Any missing bolts, lock washers, or ineffective fasteners must be replaced. All threaded inspection covers must be secured.
5. All cover flange and thread surfaces must be treated with great respect. They must not be handled roughly, and anything, including tools, that might possibly scratch or mar a joint or thread surface must not be allowed to come into contact with it. If a cover thread is damaged, it must be replaced. If a joint is scarred by any means, the equipment must be removed from service.
6. If an enclosure is opened, the cleanliness of joint and thread surfaces must be maintained so that foreign matter does not enlarge a gap. Hence, all joints and threads should be carefully cleaned before reassembly, and if necessary, a thin layer of lubricant should be applied. Care must be taken that the joint does not become recontaminated. When reassembled, all joint clearances should be rechecked.
7. Frequent examination should be made to determine any corrosion on flanges, threads, shafts, bearings, and any other flame-arresting path. Corrosion inhibitors and lubricants may be used, but if any corrosion is found, the enclosure should be taken out of service and sent for repairs since corrosion products cannot be removed adequately from equipment in operation.
8. Enclosures should be examined regularly for burned holes.
9. All cable packing glands should be examined to see that the cable fits tightly, and the clearance between the gland nut and housing should be checked to see that it is adequate.
10. Headlights should be checked for loose or broken lenses, loose packing glands, missing or broken parts, and improper assembly.
11. As much as practical of the accumulated coal dust should be removed.
12. Portable or mobile equipment must be properly frame-grounded or provided with equivalent protection. Ground and ground-check conductors must be attached to separate studs that are not attached to a removable panel. If a separate grounding conductor is used on dc machines supplied by trolley power, the return (which is usually negative) and the grounding conductors must be attached to the rail or other grounded conductor with separate clamps.

13. All electric components must be solidly attached to the machine frame. Light fixtures must be grounded with a separate grounding conductor.

14. No splices are allowed in the external wiring on permissible equipment, except for intrinsically safe circuits.

15. All conduit hose must be flame resistant and MSHA accepted. The condition of mechanical cable protection such as guards, conduit hose, and clamps should be examined regularly. Conduit hose should not be spliced. Worn or cut conduit hose may be repaired using MSHA approved flame-resistant cable-jacket repair material.

16. Trailing cables should adhere to Federal regulations in type, size, length and condition (see chapter 8). A temporary splice must not be made within 25 ft of the permissible machine, except in reeled applications. Short-circuit protection must be provided for all ungrounded power conductors, either by correctly adjusted circuit breakers or properly sized dual-element fuses (see chapters 9 and 10). Overload protection is recommended.

17. All trailing cables must be provided with effective strain relief at the entrance to equipment.

18. Machine cable reels and spooling devices must be insulated with flame-resistant material. Rollers, sheaves, and reel flanges must be maintained so as not to damage cables. Reels should maintain a positive tension on the trailing cable during reeling and unreeling. Reel collector rings should be examined for any deterioration that would cause a high-resistance contact.

19. All circuit breakers and overload protection on the machine must be maintained in working order. A main circuit breaker, contactor, or disconnect switch must be provided on the machine and be capable of deenergizing all power conductors on board the machine except the methane-monitor power supply. Headlights and floodlights must have a separate two-pole switch to deenergize the power conductors.

20. All wheel-mounted equipment must be provided with brakes, unless the design of the driving mechanism prevents accidental movement when parked.

21. If a mobile transportation unit travels faster than 2.5 mph, headlights and red reflecting material are mandatory on both the front and rear of the vehicle. Such vehicles must have an audible warning device.

22. Guards and safe-off devices for push buttons must be maintained in working condition.

23. The approval plate must be attached to the equipment.

Any unauthorized changes to the equipment not contained in the approval will render the equipment nonpermissible. Furthermore, no machine that has been changed from that approved may be placed in service until a field modification to the approval has been reviewed and approved by MSHA.

COAL DUST HAZARDS

Coal dust can pose an ignition hazard of two types: a dust layer ignition, or dust cloud ignition. A dust cloud concentration above the lower explosive limit can be ignited by an arc, thus presenting a severe dust explosion hazard. The dust cloud is nonhomogeneous, and ignition is dependent upon the volatile content (the chemical composition), the moisture content, and the ash content, as well as the size and shape of the cloud (*14*). When the dust is composed of material containing less than 8% volatile matter, as some anthracites and coke, there is almost no explosion hazard, but a fire hazard may still exist. Bituminous coals and lignite, which commonly have volatile contents ranging from 30% to 40%, present a serious dust explosion hazard.

Bureau of Mines research (*20*) has determined the parameters required for the ignition of a bituminous coal dust that is minus 200 mesh dry (less than 5% moisture):

Minimum energy	mJ..	5
Minimum autoignition temperature	°C..	617
Lower explosive concentration	g/L..	0.05
Upper limit range	g/L..	2–5

It can be seen that the energy required for ignition is many times greater than that required for the ignition of methane-air mixtures. However, this energy level exists at a concentration of about 0.2 g/L and above. Concentrations can be measured quite readily, but the rule-of-thumb measure widely accepted among miners is that a dust cloud does not exceed an explosive concentration if a person can see his/her outstretched hand in front of his/her face (*20*).

Layered dust that accumulates on equipment can also be a safety hazard. The parameter of most concern is the autoignition temperature, which depends mainly on the layer thickness and also the particle size related to the surface temperature of the equipment. Table 16.2 gives the minimum autoignition temperature for various thicknesses of layered coal dusts (*20, 24*). The maximum surface temperature of 150° C specified in the Federal regulations for permissible equipment was based on this problem: A 150° C surface temperature would allow dust thicknesses up to 100 mm (about ½ in) without incurring autoignition, but at that thickness there would be almost no safety factor. If a 1.25 safety factor is applied to a 150° C equipment surface temperature, table 16.2 indicates that no more than 20 mm of dust should be allowed to accumulate. With a 1.50 safety factor there should be no more than 5 mm of dust. By comparison, the Federal Republic of Germany requires that the maximum surface temperature be 75° C below the autoignition temperature for a 5-mm dust layer (*24*). The moisture level is relatively unimportant since moisture will quickly evaporate from any surface exceeding 100° C, whether or not it is covered with dust.

Table 16.2.—Minimum autoignition temperatures (in degrees Celsius) versus layer thickness for bituminous coals.

Thickness, mm	Low-volatile coal	High-volatile coal
5	300	240
20	250	190
50	ND	175
100	ND	160

ND No data.

Classification of Dust Locations

All areas where coal dust is a hazard are defined as class II locations. The decision flowchart in figure 16.16 can be used to assign the class II locations as either division 1 or division 2. It is based on information from the

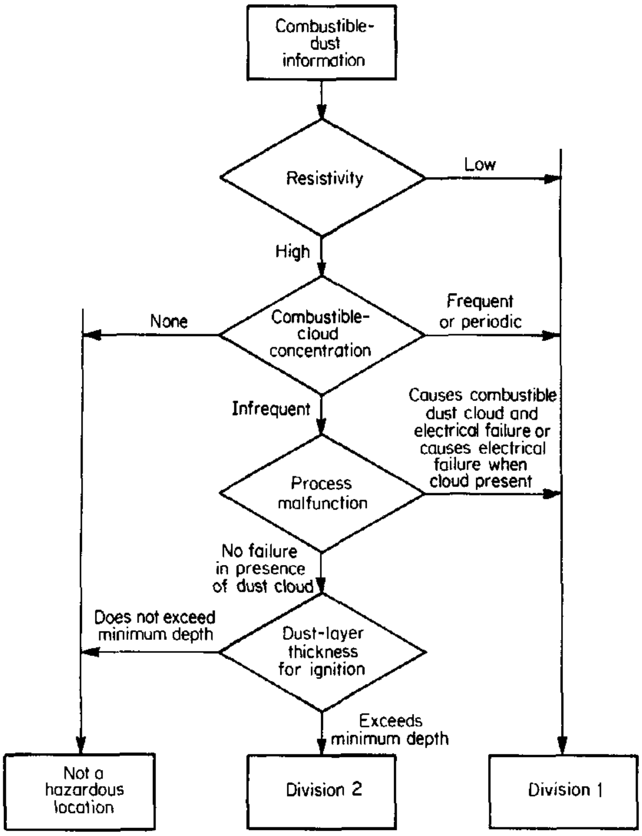

Figure 16.16.—Decision flow chart of class II, division 1 and 2 hazardous locations.

NEC (*37*) and the literature (*17, 23*). Division 1 locations occur with

- A low-resistivity dust,
- A high-resistivity dust in a frequent or periodic combustible cloud, or
- A plant malfunction that allows formation of a combustible cloud and electrical equipment to spark or overheat.

Division 2 locations are defined when

- The thickness of layered dust exceeds the minimum depth to propagate flame, or
- A high-resistivity dust infrequently forms a combustible cloud.

An additional division 2 location should be noted: when the layered dust interferes with heat dissipation—but Magison (*14*) suggests that this condition is implicit in item 1 above for any combustible dust.

Low resistivity has been defined as values less than 100 Ω-cm; a conductive dust is one that breaks down under an electric field of 1,000 V/cm or less. A direct hazard exists with these dusts because they can provide conductive paths between live parts and are susceptible to arc formation (*14*). Moodie (*18*), however, has found that coal dust has high resistance and is nonconductive, even when contaminated with mine drainage water. Hence, hazardous locations appear to exist when a cloud concentration exceeds 0.05 g/L or when the dust layered on electrical equipment exceeds a thickness of about 20 mm. These values have been extracted from the current literature and should be taken as a guide rather than a specific rule.

Reducing Dust Hazards

In class II hazardous locations, the types of enclosures in common use are dust-ignition-proof and dust-tight. Pressurized and intrinsically safe systems are used less frequently (*14*). The objective of these enclosures is to keep dust away from ignition sources and to prevent ignition of layered dust. Dust-ignition-proof enclosures meeting Underwriters' Laboratories, Inc., requirements conform to both objectives (*36*). The requirements are similar to those for explosion-proof containers but less severe. Joints, for example, must not be less than 3/16 in wide, with maximum gaps of 0.0015 in. With wider joints, the maximum clearances are raised proportionately but must not exceed 0.008 in. Gasketing is allowed as an alternative between mated surfaces, but the surface and gasket width cannot be less than 3/8 in. Such gaskets must not deteriorate under normal use and cannot be glued to the surface. The goal is to provide an enclosure that will prevent hot particles from escaping and dust from entering (*14*). The maximum allowable temperature for the surface of such equipment is 150°C for equipment that can experience overload and 200°C for equipment that will not usually overload. Dust-ignition-proof motors are suitable for both division 1 and division 2 locations.

Dust-tight enclosures are intended only for division 2 locations. The standards for their dust-tight construction are less restrictive than for equipment in division 1 locations, but they undergo the same tests by the Underwriters' Laboratories, Inc. (*35*).

Hazardous Locations in Preparation Plants

Most of this chapter has been projected at hazard reduction in underground coal mines. The reason should be obvious: all portions of underground coal mines inby the last open crosscut and in return airways are considered hazardous locations. Hazardous locations in surface mines and surface facilities of mines are not so easily defined. The following information on coal preparation plants is intended to provide an example of areas that can be hazardous.

Coal preparation plants commonly experience three conditions that are classified as hazardous:

- High levels of coal dust suspended in the plant atmosphere,
- Significant accumulation of coal dust settled on electrical equipment and other surfaces,
- Dangerous accumulations of readily ignitable gases, such as methane released from coal being processed.

The extent of these hazards depends on the characteristics of the coal being handled, the preparation plant design, and the steps taken to modify or control the hazardous condition.

The unit operations and plant locations susceptible to class II hazards include (*1*)

- Transfer points in the materials-handling system, such as conveyor-to-conveyor and bin-to-feeder locations;
- Coal-crushing and rotary-breaking systems, including operations that create new particles due to coal friability;
- Coal wetting, sizing, and sorting operations;

- Plant cleanup systems;
- Manual picking tables;
- Thermal dewatering systems.

Coal dust accumulates in these locations, particularly inside dust covers and covered conveyors and chutes where effective cleanup is difficult.

Class I locations where methane can accumulate are mostly operations involved in particle size reduction and locations that have both limited ventilation and stored or slow-moving coal, such as silos, storage bins, pressure discharge chutes, and the tunnels and chutes leading to these locations (*1, 15*).

The purpose of the chapter has been to introduce hazard-reduction techniques that are used in and about coal mines. Portions of coal-mine operations can be rendered dangerous without diligent adherence to the procedures and regulations presented in the foregoing paragraphs. Anyone desirous of gaining more knowledge in this subject should read reference *14*.

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CHAPTER 17.—MAINTENANCE[1]

The ultimate goal of the power engineer is to maximize system availability without compromising personnel safety. It has already been seen that protective circuitry with its relays, circuit breakers, and surge arrestors plays a vital role in achieving this goal. Yet system protection extends beyond protective circuitry: a good maintenance program can be equally important in providing a safe reliable mine system. Knowing the principles behind each type of maintenance and when and where to apply them can save both time and money as well as improve overall system safety.

There are three types of maintenance: emergency, preventive, and predictive. While maintenance is a familiar topic for most engineers, this familiarity rarely extends much beyond emergency repairs or routine maintenance. Typically, a component such as a motor fails, and in the heat of the production environment, maintenance crews make repairs as quickly as possible. Maintenance becomes an exercise in troubleshooting.

Preventive maintenance (PM) is practiced by few mining operations, although its concept is as old as industrialization itself. PM is traditionally taken to be the periodic performance of various tasks that will extend the life of a component: an example is the regular lubrication of bearings. More generally, PM is taking measures that will prolong the useful life of a component, and then when failure is imminent, replacing it at a time when the minimum downtime and personnel hazard will be incurred. This is the definition that will be used in this chapter.

Techniques of predictive maintenance are rarely applied in the mining industry, although the technology is routinely practiced in several other industries. Predictive maintenance is the prediction of component failure through measurement of key parameters and observation of any changes that occur over an extended period of time. An example is the measurement of insulation resistance and comparison with previously recorded values. When the recorded curve approaches a critical value, failure is impending and predictable. Advances in electronic technology have opened the door for increasingly sophisticated prediction techniques, which, like the more common predictive techniques, can be included as part of a comprehensive maintenance program.

In this chapter, various aspects of preventive and predictive maintenance programs are discussed, maintenance techniques are outlined, and finally, some of the elusive problems that can plague the mine electrical system are explored. But first, a few definitions are in order.

The study of existing conditions in a variety of mine power systems indicates that a properly designed and maintained power system is characterized by its safe, reliable, and economic operation; conversely, poorly designed or maintained systems tend to be unsafe and experience low availability (*14*).[2] Such conclusions are not surprising and are well known by power engineers working in surface transmission and distribution (*13*). What is significant is the close relationship between safety and availability. When the goal is to obtain the maximum availability of equipment, system safety is also maximized.

Availability can be defined as the mean time of component operation, that is, the ratio of total operating time to the total time in which the component could have been operated (*9*). Mathematically, availability is a function of reliability and maintainability, though it is impossible to define the precise balance of reliability to maintainability that produces a specific availability. In order to understand and improve availability, it is necessary to examine the factors of reliability and maintainability more closely.

Reliability can be defined as the mean time between failures. It is primarily affected by the component design, although poor reliability can be caused by abuse or improper application of equipment; for example, a portable cable rated for 8,000 V can be used on a 15-kV system, but the life and reliability of the cable will be considerably reduced. However, although unreliable components are sometimes found, and although there remain a small number of engineers who do not understand the principles of good system design, in general it can be stated that mine power-system reliability is essentially fixed. While manufacturers could try to improve the reliability of individual components, the results would probably not be commensurate with the effort and cost expended. Furthermore, attempts to improve component performance often lead to increased complexity, which, paradoxically, can further degrade reliability. Based on this argument, it can be surmised that any significant improvement in the availability of a mine power system will have to come through increased maintainability.

The U.S. Department of Defense has defined *maintainability* as the quality of the combined features and characteristics of equipment design that permits maintenance to be accomplished under operating conditions by personnel of average skills (*9*). As implied, this factor is affected by the original design, but the skill and integrity of the maintenance personnel are also important factors in the maintainability of a component or of the full system. Another factor that is often overlooked is the ratio of PM to emergency maintenance. For instance, the maintainability of a trailing cable will be poorer if it is repaired only after a failure, as opposed to being repaired when signs of wear are discovered during a PM shift. In such a case, the reliability of the cable is also affected; this statement is substantiated by the fact that splices made on production shifts tend to be less reliable than those made on PM shifts.[3]

To increase availability, the problem is then to improve maintainability. The initial responsibility for improvement would appear to lie with the manufacturer, but once a system is operational, manufacturer changes rarely impact on total availability. Consequently, the burden lies on the maintenance effort at the mine. Emphasis must be switched from repair and emergency work to PM. This of course is easier said than done, and experience indicates that PM programs at the mine are generally haphazard and neglected (*16*).

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[1] The author wishes to thank J. L. Kohler, assistant professor of mining engineering, The Pennsylvania State University, who prepared the original material for this chapter.

[2] Italicized numbers in parentheses refer to items in the list of references at the end of this chapter.

[3] Personal communication from R. H. King, Department of Mineral Engineering, The Pennsylvania State University, July 1980.

MINE MAINTENANCE PROGRAM

At any mine or plant, a maintenance program must be justified before the company will release personnel, materials, and money. This is an unfortunate reality. The quantification of the costs and benefits of a PM program is always difficult, especially when maintenance and production departments have the misguided notion that they have opposing goals. It is easy for a plant superintendent to visualize the program costs, but it is more difficult to see the long-range savings that will be reaped from a well-conceived PM program. Similarly, it is easy for a supervisor to compute the lost production costs when a machine is shut down for an hour of PM, but it is frustratingly difficult for the maintenance people to compute long-term savings due to increased machine availability. Unfortunately, the prevailing attitude among managers, from the faceboss to the superintendent, is that preventive maintenance is worthless unless proven otherwise.

Economic Justification

A good economic analysis is difficult to obtain if there are insufficient supportive data on component failure. At many operations, maintenance records of mine equipment are incomplete, at best. Consequently, it is necessary to make reasonable assumptions about the probable effects of a good PM program. Manufacturer data as well as data from related mine operations can be very helpful, but in the absence of this information there are some general guidelines that can be used to present a strong justification for a PM program.

Three ratios are available for a first-order approximation (*25*):

$$\frac{\text{cost of component repair or replacement}}{\text{cost of component PM}}; \qquad (17.1)$$

$$\frac{\text{cost of downtime}}{\text{cost of PM}}; \qquad (17.2)$$

$$\frac{\text{safety hazards}}{\text{cost of PM}}. \qquad (17.3)$$

If the value of any one of these ratios exceeds 1.0, then PM can be justified. Although the application of these ratios should be obvious, an example of each is given.

If a motor bearing fails, there is a good chance that the motor will be damaged and short out. The repair costs then become substantial because the motor has to be pulled and rebuilt. The PM costs in this case would be very small, consisting of monthly vibration measurements on the bearing. Ratio 17.1 is applicable to this case and, depending on the downtime involved, ratio 17.2 could also be used.

The second ratio is more applicable to situations where the cost of the failed component is insignificant, but the failure causes a significant amount of downtime. For instance, if a contactor fails on the starter of the wound-rotor induction motor that drives the main conveyor belt, the contactor replacement costs will be minor compared with the cost for idling a major portion of the mine.

Ratio 17.3 cannot be evaluated in dollars, at least not in good conscience. Although some industries, most notably the automobile industry, have placed dollar values on human lives, this cannot be condoned. Instead, the goal should be to practice PM to the point of removing any reasonable chance of personnel injury.

Other techniques, such as investment analysis, can be performed in the absence of extensive historical data. This method is probably best approached by considering the investment to be labor and overhead, and then conservatively estimating the effects of the PM program on the cost per unit of output. In the mine, the cost might be in dollars per raw ton, whereas in a preparation plant, it would be in dollars per clean ton. When such an estimation is carried out, the PM investment can be profit-adding rather than just profit-maintaining. However, it should also be noted that PM does follow the law of diminishing returns; carried to excess, it is not cost effective.

Preventive Maintenance Program Implementation

When the need for PM has been established, it is important that the top level of company management be involved in the decision to implement a suitable program. Since the returns from PM will initially lag behind its cost, such management support is crucial. Whenever company profits fall and operating budgets must be slashed, the PM program is usually the first to suffer, but this is just the time when it is needed most. The backing of management can alleviate this industry-wide problem.

The composition of the PM crew should be planned carefully. Experience in other industries indicates that at least half of the work crew should be seasoned mechanics who have a working knowledge of all the electromechanical system they will encounter. Circumstances may warrant specialists. For example, as solid-state equipment becomes more prevalent in and out about the mine, an electronics technician would be a valuable addition to the PM crew. An established policy to have all PM personnel as salaried workers might be the most satisfactory.

The maintenance superintendent typically assumes responsibility for the PM crew. However, this is not necessarily appropriate in the mining industry, where ideally the job should be filled by a separate individual very knowledgeable in electrical or mechanical engineering (preferably both), who has several years of mining experience. The PM engineer should report directly to the plant or mine superintendent. Regardless of the crew makeup, a policy should be established that the PM personnel do not carry out normal repair maintenance.

When implementing a PM program, an initial decision must be made concerning the equipment to be included and the priorities to be followed. First priority should perhaps be given to new or expanding facilities. Maintainable equipment and installations do not just happen. From the planning stages through equipment purchasing to installation, a maintenance specialist should be involved in all the decisions. In fact, a recent study found that when this procedure was followed, the time required for a new preparation plant to reach capacity operation was reduced from an average of 3 yr to less than 18 months (*23*).

When selecting the equipment for PM coverage in an existing operation, the components whose failure would have the greatest impact on production or safety should be considered first. In an underground coal mine, these might be surface substations, ventilation fans and belt-drive motors, after which, rail-haulage motors and rectifiers, face machinery, and dewatering pump motors could be added. The economic ratios presented earlier can be used to review the types of system components and assess different levels of coverage when designing the PM program.

The next decision, and the one with the greatest impact on the success or failure of the program, concerns the organization of the data. The choice is basically between some type of card-filing system and a computerized system. Card files are rapidly becoming outdated; computers are becoming the normal operating method. Microcomputer systems now have extended capabilities and improved graphics and provide a convenient and flexible alternative that is adequate for handling the PM data of most companies. A wide variety of off-the-shelf software is available that can be readily tailored to individual company requirements. Such computerized card-filing systems have built-in analytical routines that simplify record keeping; they can print out summary reports of various types, print out daily calendars of work to be accomplished, and supplement data with graphs and charts. With minimal training, the PM crew can record data directly on to the system on a shift-by-shift, daily or weekly basis.

TECHNIQUES OF PREVENTIVE MAINTENANCE

The following sections discuss a variety of electrical and mechanical tests that will provide the data necessary for operating an effective PM program. To these could be added a variety of other procedures; such simple tasks as measuring basic temperatures, clearances, or runout, for example, can provide the maintenance engineer with valuable information on the condition of equipment. In large transformers, spectroscopic analysis of the oil provides sufficient data to detect many potential problems. Analysis of lubricating oils and greases can also be useful. The presence of a few parts per million of metal can indicate excessive wear or other problems. In oil-filled distribution transformers, for example, partial discharge on the windings is indicated by the presence of a few parts per million of copper (*10*). Many other techniques could be mentioned, but the objective of this chapter is to outline the measurements that would form the backbone of the PM program, and these are electrical and mechanical tests. In practice, it is important that the engineer not lose sight of the interrelationships between electrical and mechanical components in the overall system. It has been estimated that up to 75% of all electrical failures in motors can be caused by the initial failure of a nonelectrical component.

Basic Electrical Measurements

The main reason for taking measurements on the mine power system is to discover impending failures. Many potential problems can be detected with a simple voltmeter. Voltages on machines and at key points around the electrical system should be checked periodically to detect excessive voltage drops. Voltage drops of 20% to 30% under machine rated voltage are too common around mines, and the ramifications of such undervoltage can be serious, as stated in chapter 6. The voltmeter is also a valuable tool for troubleshooting the solid-state circuits that are becoming more common.

Another useful voltmeter test is measurement of the voltage differences among phases at a given location, particularly at a transformer that supplies power to motors. Motor damage can result if the voltage differences among phases are more than a few percent. In fact, one research group has found that the negative-sequence current caused by a 2% voltage difference can create enough heat to halve the life of a motor (*25*).

The measurement of current is a simple task that can be performed for ac with the voltmeter and a CT or for dc with a resistance shunt. Current measurements taken periodically on a motor at no-load and full-load can provide valuable information on its condition. Although this type of measurement is better suited to stationary motors, as on belts or pumps, there is no reason why it cannot be used on mobile equipment. One specific application for current measurements is in the periodic checking of dual-motor belt drives. Here, it is normal for the load to be shared unequally by the two motors, for example, 52% to 48%, but it is not normal for the ratio to change by more than a few percent with time. If a significant change occurs, there may be an electrical or mechanical problem, such as a coupling misalignment. Resistance readings can also be invaluable. Although some of these readings must be made with a megohmmeter or bridge meter, many can be made with an inexpensive volt-ohm-milliammeter (VOM): resistor-bank values, open relay coils, and so on. In some cases, such as contact-resistance measurements, more accurate results can be obtained using a Kelvin bridge, although VOM measurements have some practical utility.

Insulation Measurements

Insulation is the only material used in electrical components that begins to deteriorate the moment it is manufactured. Strictly speaking, insulation should be called a dielectric, since it is simply one application of dielectric material (*6*). There are three primary stresses that can cause dielectric deterioration in the mine power system: mechanical, thermal, and electrical. Electrical stresses result from an overvoltage in excess of the rated withstand level of the dielectric. This may cause immediate failure or gradual deterioration, depending on the level of the disturbance.

Thermal and mechanical stresses are closely related. Most dielectrics do not have significant mechanical strength, so actual or circumferential stress may cause a failure, and they usually do not have much resistance to fatigue. Thermal stress, resulting from an operating temperature in excess of rated, causes a change in the physical properties of the insulator, which usually changes its electrical and mechanical properties. The integrity of insulation is eroded as temperature increases, and several factors combine to cause a decrease in resistance or increase of current flow. The primary cause is the increased carrier mobility at elevated temperatures (*6*). Some types of insulation, especially some of the enamels used in motor wire, tend to become brittle, and as they expand and contract under the changing temperature conditions, they crack. The resulting leakage currents will be larger than those in insulation with decreased resistance.

Thermal stress can also manifest itself as a mechanical stress. This effect is more likely to occur where two or more dielectrics are sandwiched together, as in a portable cable. Since each insulator has its own particular coefficient of thermal expansion, the expansion for each material will not be the same. This is particularly critical when an underlying material is being constrained, since mechanical stresses will result. Strain may then occur, resulting in voids or weakened areas. When failure eventually takes place, it can appear to be mechanical or electrical in nature.

Insulation is the material most likely to fail in a motor, and periodic measurements are a very valuable means of detecting incipient failure. The model of a dielectric shown in figure 17.1 indicates the parameters that can be measured to determine the integrity of the insulation (*6*). The two obvious ones are resistance and capacitance. Insulation resistance is measured directly: a perfect insulation would have $R_p = \infty$ and $R_s = 0$. Capacitance is not measured directly. Instead, θ, the phase angle between the voltage and the current, is measured, as shown in figure 17.2. The tangent of δ (or the cotangent of θ) is known as the *dissipation factor*, and the cosine of θ is called the *unloaded power factor* (*6*). At power angles close to 90°, the two factors are nearly equal. The voltage-current relationships for a sound dielectric will be almost purely capacitive, and the unloaded power factor and the dissipation factor should be very low or very close to zero. As insulation deteriorates, large leakage currents will tend to increase these factors significantly (*24*). The dissipation factor is used frequently in technical literature, whereas the unloaded power factor is used more frequently by manufacturers when specifying component tolerances (*6*). Hence, the latter might be more useful in preventive maintenance than the dissipation factor.

Regardless of the factor selected, readings should be taken at periodic intervals, and records should be kept of each reading so that any trends can be detected. Typically, a manufacturer will specify the maximum acceptable unloaded power factor, usually 2% or less. If this value is exceeded, one of two problems is usually indicated: either the insulation is severely deteriorated or it is water soaked. In either case, failure may be imminent, although allowing the insulation to dry would correct the latter problem.

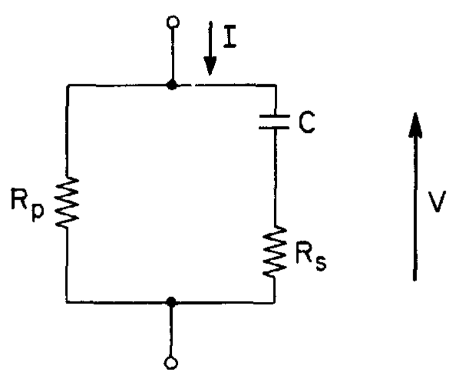

Figure 17.1.—Circuit modeling a dielectric.

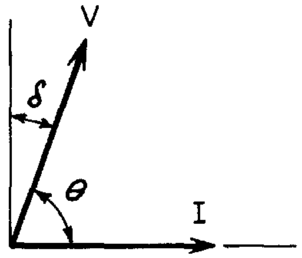

Figure 17.2.—Current-voltage chracteristics in a dielectric.

A suitable instrument for making these measurements is a capacitance and dissipation-factor bridge. Since the temperature affects the power-factor readings, a correction factor must be applied. These factors are dependent upon the particular dielectric type and should therefore be obtained from the manufacturer. The instrument connections to the apparatus being tested are identical to those of the megohmmeter tests, which are discussed in the next section.

The shunt resistance of the leaky capacitor, R_p, shown in the insulation model of figure 17.1, can be measured and used to predict insulation failure as well as to observe some measure of the insulation integrity. In practice, this measurement is usually performed with a megohmmeter. The current that flows because of the potential impressed on the dielectric is composed of two parts: a leakage component across the insulation surface and a current actually through the insulation. When the detection of deterioration is of interest, the superficial leakage current must be minimal, otherwise the insulation resistance reading will be artificially low. Consequently, it is necessary to examine those factors that can cause incorrect measurements. They are

1. Surface conditions: abrasion, foreign material, moisture;
2. Moisture;
3. Temperature;
4. Method of test: test-instrument potential, duration of test;
5. Residual charge.

The surface condition of the insulation normally has the largest effect on the superficial current component; for instance, an abraded surface will collect airborne dust. If the dust is from a highly conductive material such as from carbon brushes, the possibility of establishing a conductive path on the insulation surface is rather high. Even dust with a much lower conductivity (such as coal dust) can allow enough current to flow so a significant error is introduced. A nonabraded surface will have a tendency to collect some dust, but here the problem is not as severe.

A conductive path can also be created if the dielectric surface is moist, and foreign material on the surface will aggravate the problem considerably. The surface moisture problem can usually be minimized by taking the measurements only when the temperature of the component is above the dewpoint. In the case of machines, surface moisture is of no significance at the normal operating temperature, but some researchers have expressed concern over the effect of high relative humidity during testing (*4*). While it is true that experiments conducted in a controlled environment do reveal a decreased insulation resistance with increased relative humidity, the changes are not troublesome if the data are correctly analyzed. More will be said about this later.

The second factor is moisture other than surface moisture. Some insulating materials are hygroscopic; they absorb moisture. This effect is entirely different from that of surface moisture and presents a more serious problem because it compromises the dielectric integrity. Although a dielectric may exhibit a greater propensity for absorbing water as it ages, this characteristic is not a reliable method of determining aging, at least not outside the laboratory.

The hygroscopic moisture content can be reduced by bringing the component to normal operating temperature for a few hours before making any measurements. If a measurement is low and moisture absorption is suspected, caution should be observed when energizing the component, to prevent failure. In some cases, such as with large motors, it may be worthwhile passing a current equal to load current through the windings at a low voltage. The heating that occurs will dry out the windings.

The third factor, insulation temperature, has a significant effect on the measured resistance. If the insulation temperature is known, a normalized resistance can be calculated:

$$R_c = R_t K_t \qquad (17.4)$$

where R_c = insulation resistance corrected to 40° C, Ω,
R_t = measured insulation resistance, Ω,
and K_t = temperature coefficient.

The temperature coefficient factor is obtained from a graph such as that shown in figure 17.3, which is valid for rotating machines (*11*).

The measurement of insulation temperature is sometimes difficult, but neglecting this factor can be risky. A suitable compromise can be reached in many cases; for example, taking the measurements consistently before or after the component is at operating temperature may stabilize the readings. The ambient temperature may vary over a small enough range to ignore its effect on the preoperating temperature of the component. Of course, steps such as these must be evaluated for each situation, and what might work well for a large motor could be meaningless in the case of a circuit breaker.

The fourth factor, method of test, is important but presents no real application problems. The test potential of a megohmmeter is usually 500 to 1,000 V. Measurements on a given component should always be taken at the same test potential. The basic test format for a given component is fixed and will be suggested shortly. Beyond this, the test measures and procedures should always be duplicated precisely. Since the purpose of the test is to analyze a trend over time, small errors introduced by nuances of the test procedure will not pose a problem, provided that the methodology is consistent.

Residual charge is the final factor. The leaky capacitor of the dielectric model must be discharged before a resistance measurement is taken. This is achieved by shorting to ground the outer skin of the dielectric and the enclosed conductor. The short should remain in place for a few minutes. When multiple readings are taken on the same insulation system, the dielectric should be shorted for at least 3 min, assuming the standard 3-min test (*11*). In specialized megohmmeter tests, where parameters such as the time and rate-of-rise of resistance are analyzed, this factor is critical.

Megohmmeter Tests

As was described in chapter 5, a megohmmeter is a portable instrument with an integral voltage source and a meter calibrated in megohms. The voltage source is either a handcrank generator or a battery pack. The type of instrument most suitable for routine insulation resistance measurements has only two probes, much like a voltmeter. Tests are conducted by placing the probes across the insulation system and measuring the resistance. Component manufacturers generally provide instructions for megohmmeter testing of their components. Three different tests are performed: spot reading, time resistance, and multivoltage.

Spot Testing

Readings for the spot test are taken consistently at 60-s intervals, since the resistance in a good dry insulation will always increase with time, as shown in figure 17.4.

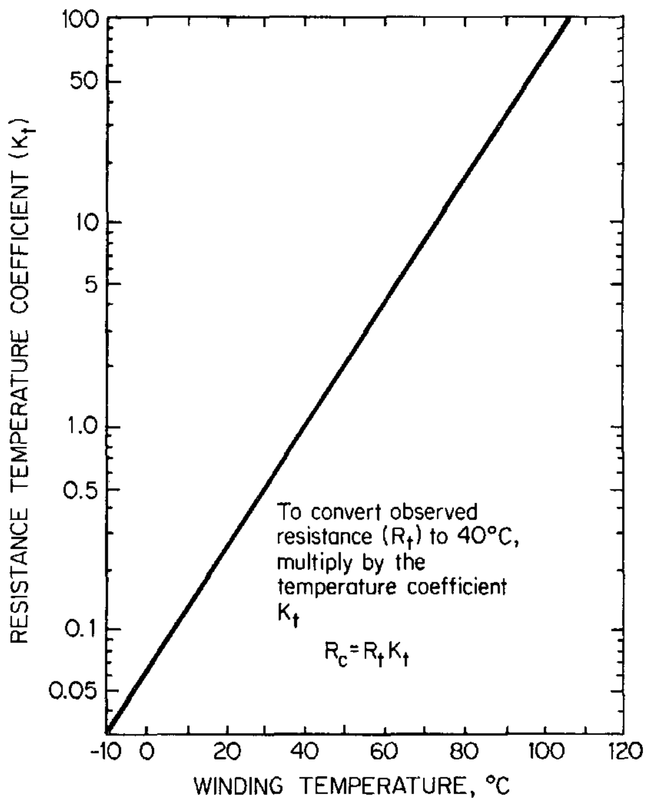

Figure 17.3.—Graph relating approximate insulation resistance variation with temperature for rotating machines.

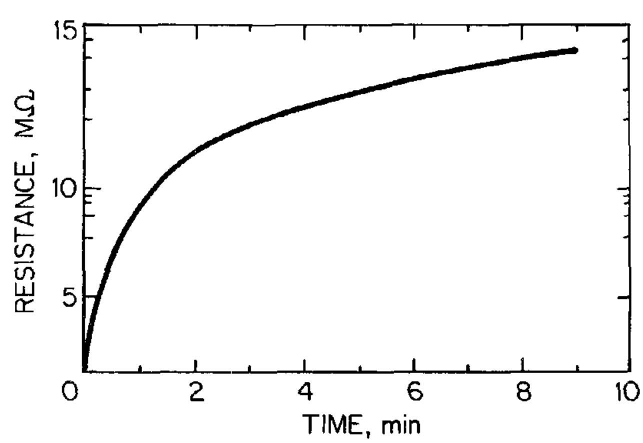

Figure 17.4.—Insulation resistance versus application time of test voltage.

Spot readings are taken periodically on system components and recorded so that any persistent downward trends that indicate potential failure can be detected.

Conceptually, the most straightforward spot test is on cables, where one power conductor is checked at a time, with all other cable components shorted together and to ground. The connections for checking the line A conductor of a cable are shown in figure 17.5. After the connections are made, the test voltage is applied for 1 min and the final resistance value is recorded.

Megohmmeter testing of motors is similar, but this test checks only the insulation system to ground and not the turn-to-turn insulation system. Figure 17.6 shows the test connections for line A testing of an ac motor, and figure 17.7 shows the dc motor connections. The minimum value for the spot resistance test should always be 1.0 MΩ, plus 1.0 MΩ for each 1.0 kV of nominal voltage rating for the equipment. Consequently a low-voltage mine motor should have a minimum resistance of 2.0 MΩ. Typical spot resistance test records for motors are shown in figures 17.8 through 17.10.

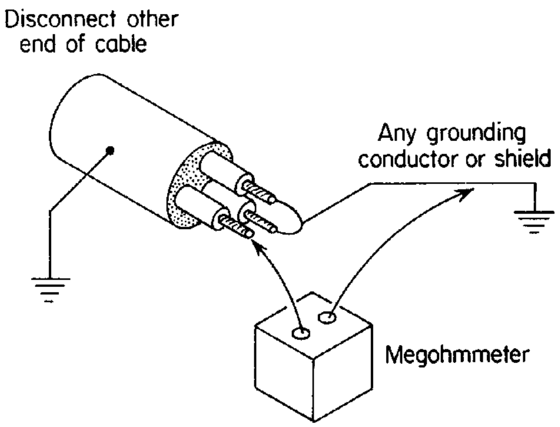

Figure 17.5.—Megohmmeter test connections for checking cable insulation in line A.

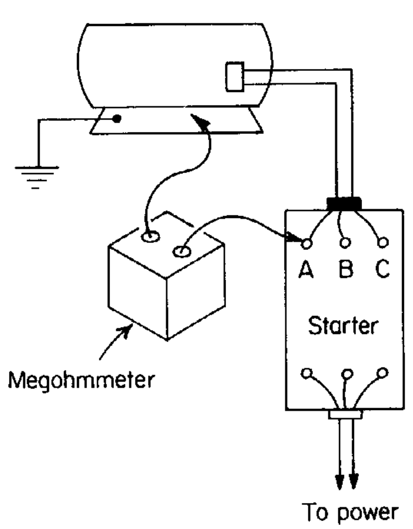

Figure 17.6.—Megohmmeter test connections for ac motor.

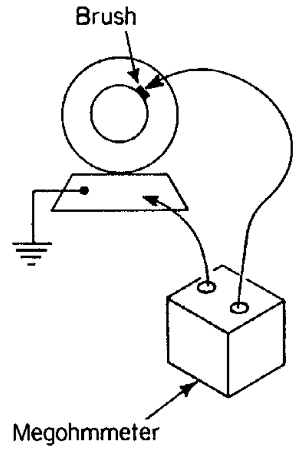

Figure 17.7.—Megohmmeter test connections for dc motor.

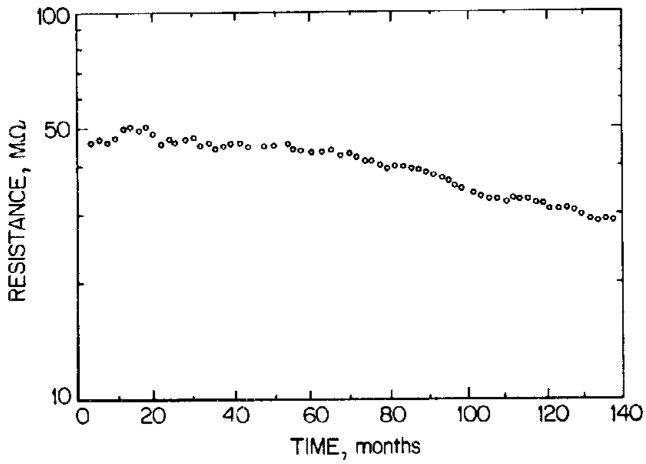

Figure 17.8.—Spot resistance curve for normal motor.

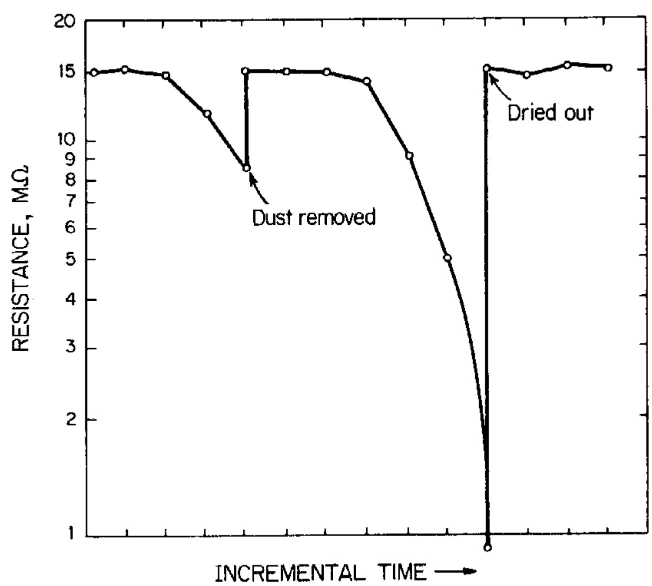

Figure 17.9.—Spot resistance curve showing effects of dust and moisture.

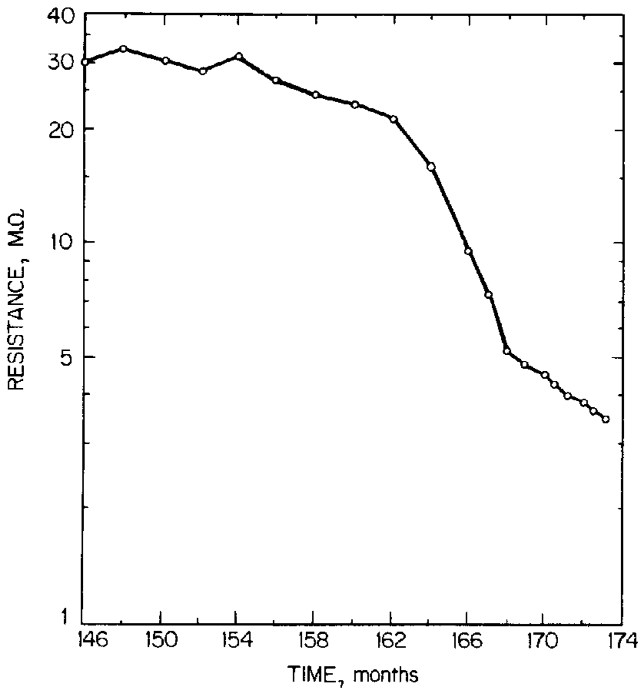

Figure 17.10.—Spot resistance curve for defective motor.

When spot-testing a transformer, the core iron must be grounded and the resistance connections or solidly grounded connections must be removed. All windings except the one under test must then be shorted together and grounded. The test connections are shown in figure 17.11. Similar testing procedures exist for starters, control boxes, relays, and circuit breakers, to name a few.

Time-Resistance Tests

With the same connections, time-resistance tests can be performed. Here successive readings are taken at specific time intervals to form time-resistance curves as shown in figure 17.12. Of particular interest is the point where the curve begins to level out, since a good insulation will have a continual increase in resistance with time (*26*). Figure 17.13 shows curves for an actual deteriorating motor.

The polarization factor can be calculated by plotting the resistance values at 1 and 10 min:

$$P = \frac{R_{10}}{R_1}. \tag{17.5}$$

The computed factor should have a minimum value of 2.0, as shown in figure 17.14. The polarization factor is independent of temperature and equipment size, which makes it very convenient for some applications. A polarization factor curve for a deteriorating motor is shown in figure 17.15.

Multivoltage Tests

Multivoltage tests are particularly valuable for assessing the efficiency of high-voltage components. Insulation resistance measurements are repeated at different

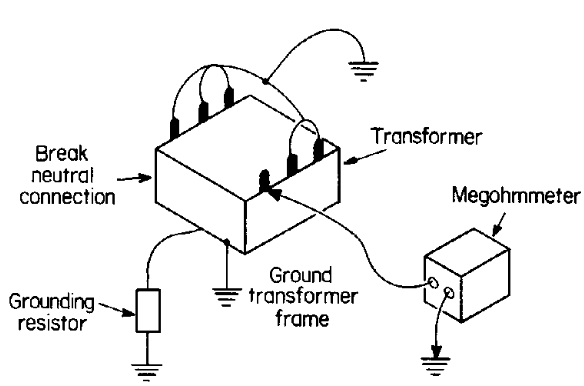

Figure 17.11.—Megohmmeter test connections for transformer.

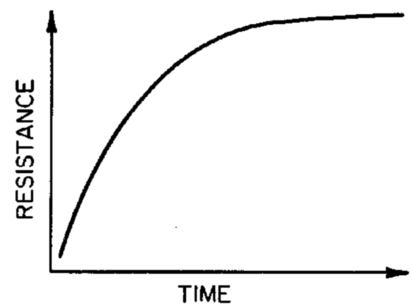

Figure 17.12.—Time-resistance curve.

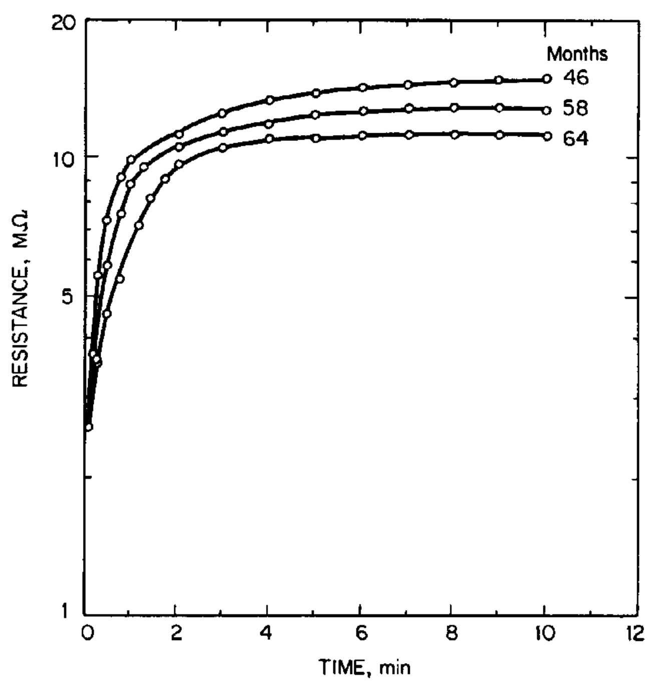

Figure 17.13.—Three time-resistance curves for deteriorating motor.

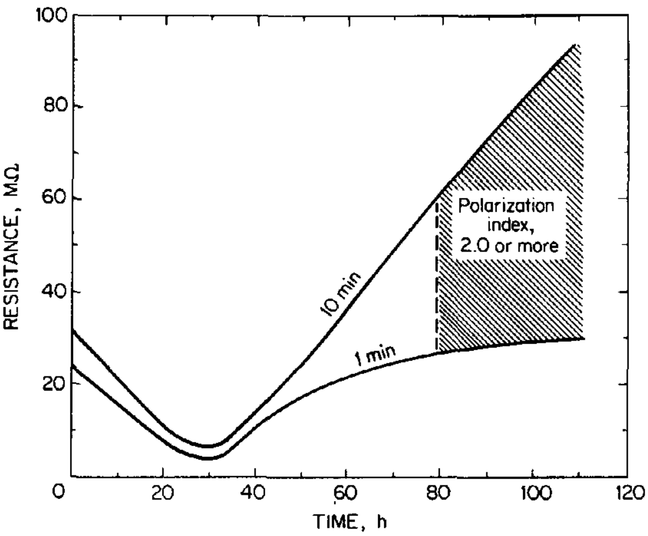

Figure 17.14.—Time-resistance curves showing polarization for hypothetical motor.

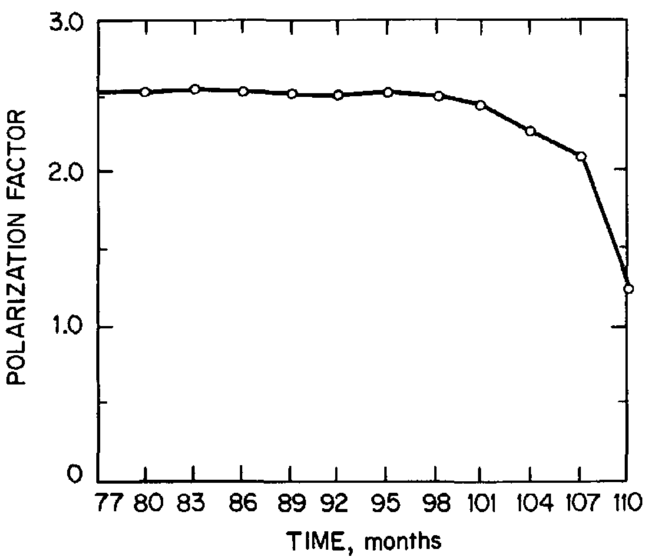

Figure 17.15.—Polarization factor curve for deteriorating motor.

voltage levels, such as 100, 250, 500, and 1,000 V, and the resistance found at the higher voltage should always be equal to or greater than the resistance at the lower voltage. A large reduction in insulation resistance with measured voltage is indicative of insulation weakness, as shown in the curve for a deteriorating motor in figure 17.16 (*26*).

Harmonic Analysis

As insulation deteriorates, a small initial current that is rich in harmonics can be produced from electrical discharge across the voids (*26*). The mechanism producing the harmonics is probably partial discharge, which will be discussed later in this chapter. The presence of harmonic current provides a method of detecting insulation failure,

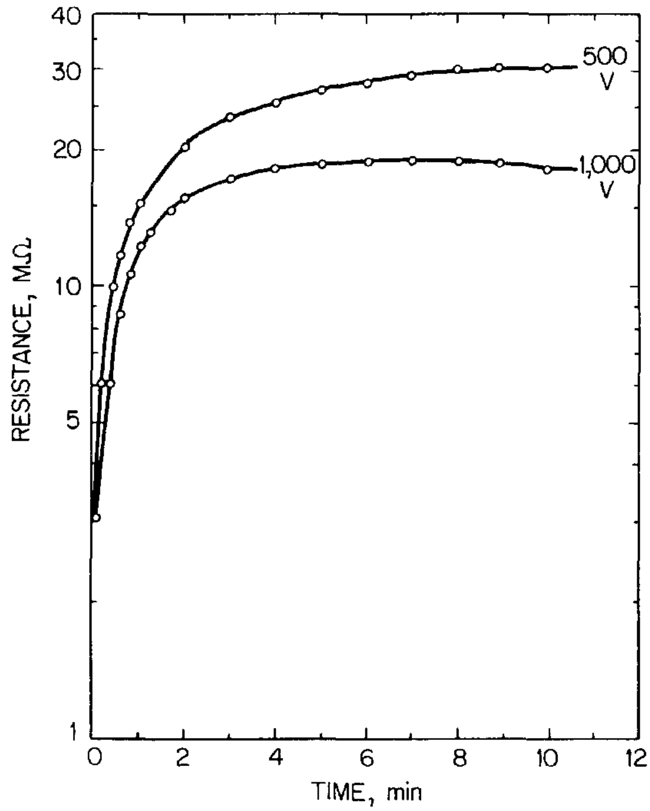

Figure 17.16.—Multiple voltage curves for deteriorating motor.

since researchers have shown that in new insulation the frequency component of a current is primarily that of the source, whereas in old insulation the current is composed of many frequencies (*26*).

Harmonic analysis is carried out by a spectrum analyzer, which performs a Fourier analysis on a signal and displays the amplitude of each frequency component. An instrumentation arrangement to carry out the analysis is shown in figure 17.17, where current is flowing from a 60-Hz source. The CRT display of the spectrum analyzer immediately indicates insulation of questionable integrity. The method has proved to be effective for cables, motors, and transformer-insulating oils.

Power Factor Versus Voltage

In this test the unloaded power factor is plotted against various voltage levels. For perfect insulators the resulting curve is a straight line, but in deteriorating dielectrics the curve rises with increased voltage to form a parabola, as shown in figure 17.18 (*26*). This is because voids in insulation contain entrapped air, which tends to ionize as voltage approaches the breakdown point, thus producing the extra loss that causes divergence of the plotted curve. A description of a simple power factor meter for in-mine measurements can be found in reference *26*.

Infrared Testing

Although it is not an electrical test directly, the observation of infrared emissions with a portable instrument can detect abnormal hot spots in operating electrical

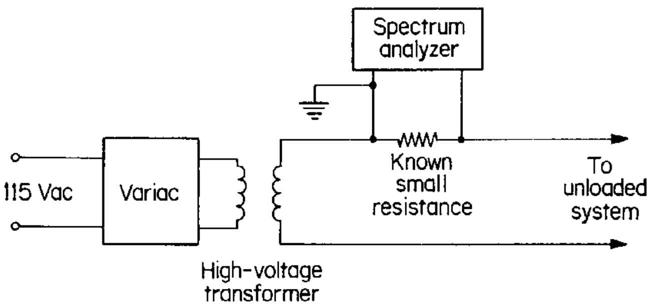

Figure 17.17.—Circuit for harmonic tests.

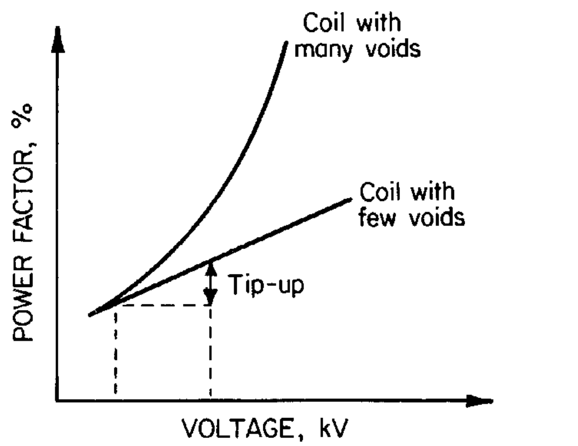

Figure 17.18.—Power-factor versus voltage curves showing tip-up.

equipment. At any temperature above absolute zero, all bodies radiate energy. The radiation in the infrared region is closely proportional to the body temperature. Excessive temperature can be caused by broken conductor strands or excessive insulation leakage in cables, defective coils in motors, or excessive leakage flux in transformers (*26*). When using routine infrared observations as part of the maintenance plan, any sudden temperature increase can indicate trouble.

Real-Time Computer Analysis

The two greatest problems concerning PM in the mining industry are probably the need to depend on personnel to make the necessary measurements and the inadequacy of existing techniques for predicting many of the failures that occur. The search for ways to alleviate these problems has been an ongoing process (*18, 20*). One recently developed method, real-time computer analysis, eliminates the human link in the data collection and analysis process. The method is also sensitive to deterioration processes that cannot be detected by more conventional tests. The technique is based upon the classification of a deterioration matrix composed of the values of electrical parameters, such as power-frequency harmonics and symmetrical components, that are collected continuously in a real-time environment. Economic analyses indicate that the cost of such a system could be amortized within 2 yr. However, practical implementation of the method is for the future. In the meantime, much can be gained through using the conventional tests described here.

MECHANICAL MEASUREMENTS

The failure of simple mechanical devices on a machine, such as a bearing, can lead to a catastrophic electrical failure. Electrical problems can cause excessive vibration (as with open-end rings on rotor-bar circuits), which in turn can cause electrical deterioration as commutator damage, insulation cracking, and squirrel-cage rotor damage. In fact, a variety of electrical problems can mimic mechanical deterioration. The only way to pinpoint these things is to include mechanical measurements in the maintenance program. Indeed, in the case of rotating machines, it is the most applicable method.

Vibration

Machine vibration is an important symptom of electrical or mechanical problems. Electrical problems that can cause vibration include the presence of a subharmonic voltage component (15, 30, or 45 Hz) or shorted turns in one phase of a motor. Vibration can also be caused by such mechanical problems as unbalance, misalignment, or faulty parts.

Unbalance can occur in a motor shaft or coupling as the result of improper installation, mishandling or defect. It may be static, occurring in a single plane, or dynamic, occurring in more than one plane. Whatever the cause, the vibration should be eliminated to prevent commutator damage, winding-lead fatigue or cracking of insulation. Some devices, such as fans, require balancing both before use and at periodic intervals. Coupler misalignment can also lead to vibration problems, especially when the motor coupling to a pump or air compressor is not in perfect alignment. Faulty parts can cause vibration. For example, when a bearing begins to fail, its vibration level will increase. If vibration levels are recorded routinely, for example, every 60 days, any significant increase will be detected, incipient failure can be predicted, and problems avoided.

The measurement of vibration is not too difficult. The transducer used is typically a linear accelerometer, which for convenience is often attached to a magnet or probe, as shown in figure 17.19. The output signal is connected to an instrument that gives a readout of acceleration, velocity (by integrating the acceleration signal), and displacement (the second integral of the acceleration signal). These instruments are also available with an oscilloscope display for examining the measurements in real time and in time or frequency domains.

Although analysis of vibration data is beyond the scope of this chapter, a few comments are in order. First, taking readings involves engineering judgment. In the rather simple case of the motor and pump, readings would be taken along each axis at points A through D, as shown in figure 17.20. Careful analysis is required to avoid various pitfalls; here for example, an unwary reader might confuse cavitation with vibration. Table 17.1 gives a brief summary of some of the more common causes of vibration, and figure 17.21 provides some insight into the severity of different vibration levels (*27*).

Table 17.1.—Common causes of vibration

Cause	Amplitude	Frequency	Phase	Remarks
Unbalance	Proportional to unbalance. Largest in radial direction.	1 × r/min	Single reference mark	Most common cause of vibration.
Misalignment of couplings or bearings and bent shaft.	Large in axial direction. 50% or more of radial vibration.	1 × r/min usual; 2 and 3 × r/min sometimes.	Single, double, or triple	Best found by appearance of large axial vibration. Use dial indicators or other method for positive diagnosis. If sleeve bearing machine and no coupling alignment, balance the rotor.
Bad bearings—antifriction type.	Unsteady—use velocity measurement if possible.	Very high, several × r/min.	Erratic	Bearing responsible, most likely the one nearest point of largest high-frequency vibration.
Eccentric journal	Usually not large	1 × r/min	Single mark	If on gears, largest vibration in line with gear centers. If on motor or generator, vibration disappears when power is turned off. If on pump or blower, attempt to balance.
Bad gears or gear noise	Low-use velocity measure if possible.	Very high, gear teeth × r/min.	Erratic	No data.
Mechanical looseness	No data	2 × r/min	2 reference marks. Slightly erratic.	Usually accompanied by unbalance and/or misalignment.
Bad drive belt	Erratic or pulsing	1, 2, 3, and 4 × r/min of belts.	1 or 2 depending on frequency. Usually unsteady.	Strobe light best tool to freeze faulty belt.
Electrical	Disappears when power is turned off.	1 × r/min or 1 or 2 × synchronous frequency.	Single or rotating double mark.	If vibration amplitude drops off instantly when power is turned off, cause is electrical.
Aerodynamic hydraulic forces.	No data	1 × r/min or number of blades on fan or impeller × r/min.	No data	Rare as a cause of trouble except in cases of resonance.
Reciprocating forces	..do	1, 2 and higher orders × r/min.	..do	Inherent in reciprocating machines; can only be reduced by design changes or isolation.

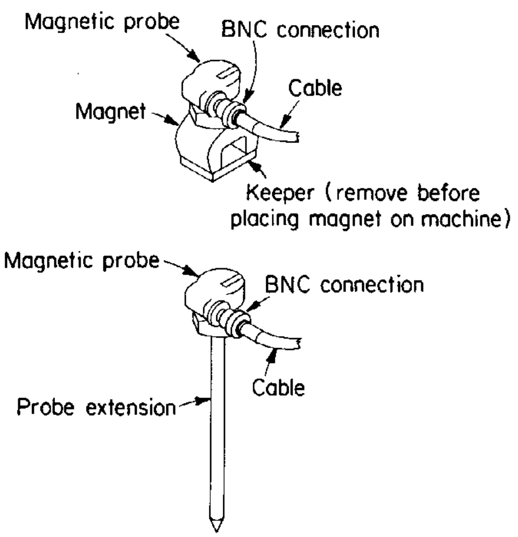

Figure 17.19.—Mounting techniques for two vibration transducers.

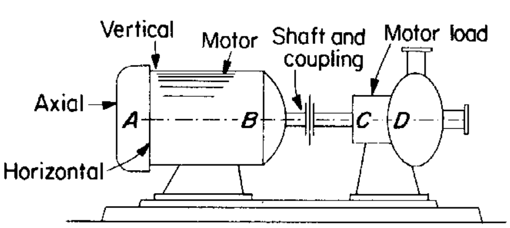

Figure 17.20.—Four typical vibration measurement points.

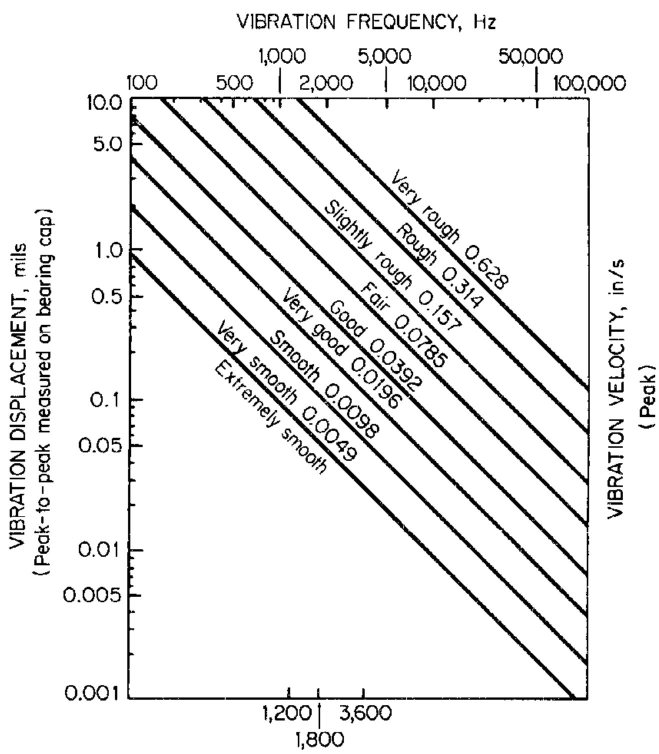

Figure 17.21.—Typical vibration severity chart.

Acoustic Emission

All rotating machines produce wide-band acoustic emissions that can provide considerable insight into the operating efficiency of a component. In fact, acoustic-emission analysis can predict most incipient mechanical failure, ranging from shaft defects and bearing problems to bad welds, many days in advance of the failure. It is a relatively new technology that has not yet been widely adopted in the mining industry. Portable instruments are available for measuring and displaying acoustic signals, using either contact or noncontact probes at specific locations on the machine. Figure 17.22 illustrates the technique (*12*).

The distributions of greatest interest are the frequency and the amplitude, and both play a role in emission analysis. A crack propagating in a bearing, for example, generates a narrow acoustic-emission pulse with a flat frequency distribution that is almost identical to friction noise in the bearing assembly. In this case, the frequency emission alerts the engineer to a potential problem. Subsequent analysis of the amplitude distribution will identify the specific problem, since the emission pattern for a propagating crack and that caused by friction are readily distinguished.

Acoustic emissions divide conveniently into low-frequency emissions, ranging from about 5 to 1,000 Hz, and high-frequency emissions that start around 65 kHz and rise into the megahertz range, but 1,000 kHz is considered the practical cutoff point. Although low-frequency signals can be used to diagnose many machine malfunctions, they have the disadvantage of containing many frequencies caused by normal phenomena, and these complicate interpretation of the readings (*12*). High-frequency emissions do not have this problem with background noise, and furthermore, they are more defect oriented. The signals in the high-frequency range are rapidly attenuated, which makes it easier to locate the source of the emission.

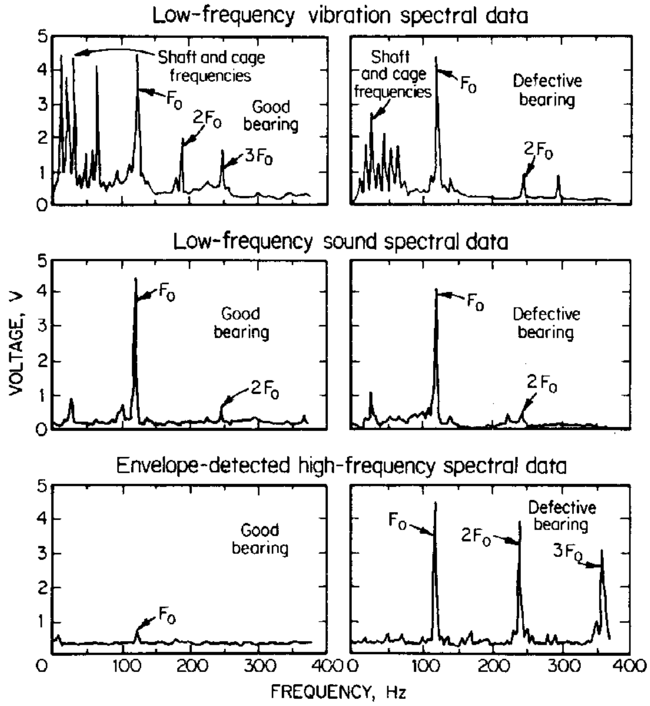

Figure 17.22.—Comparison of acoustic-emission techniques for detecting failing roller bearing.

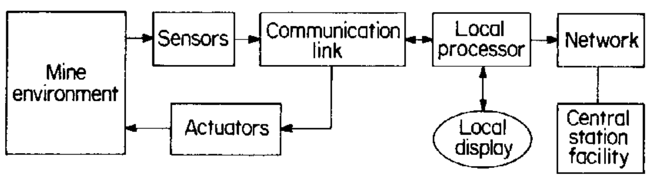

Figure 17.23.—Conceptual diagram of generalized mine monitoring and control system.

CONTINUOUS-MONITORING SYSTEMS

Recent research in predictive techniques has focused on the development of reliable continuous-monitoring systems that will automatically identify disturbances in patterns of normal operation and thus give warning of impending problems. The systems employ contact and noncontact sensors located at strategic locations on equipment and linked via a communication network to a central processing facility. Research designers favor distributed systems with outstations or local data processors that collect and display information and serve as a filter, passing only important data through to the central station. The outstations may have limited decision capability and may trigger actuators that are part of a system control network. These features are diagrammed in figure 17.23. Remote sensing and control systems have the potential to reduce downtime, thereby increasing productivity, and it is anticipated that the use of such systems will become routine in mines.

Esoteric spectral analysis and remote sensing and control can seem far from the day-to-day routines of the electrical maintenance engineer who is concerned with basic repairs and troubleshooting defective equipment. Few tasks can be more daunting to the young, inexperienced engineer, armed only with theoretical knowledge, than being faced with a downed system while surrounded by an impatient production crew. Troubleshooting is a skill—some would claim an art—that is acquired through experience. Many of the techniques outlined earlier in this chapter are employed to diagnose equipment problems, and these together with troubleshooting tips and layouts provided by equipment manufacturers are usually sufficient to identify the cause of the trouble. There are, however, some instances where the problems are more puzzling and defy routine analysis; among these are corona and partial discharge. These topics will be discussed in the remainder of this chapter.

CORONA

Corona is the name given to very small transient discharges that occur as part of the process of localized gaseous ionization associated with dielectric materials. Corona is most prevalent in high-voltage systems, and when the process reaches critical levels in regions of high electrical stress, the byproducts of ionization can cause degradation of insulation and lead to system failures. Awareness of the problem became widespread in the mining industry in the early 1970's when 12.47-kV distribution systems were adopted in several underground mines. The increase in system integrity expected to result from the high voltage did not materialize; instead, the change was accompanied by anomalous failures in couplers, cables, and stress cones. Ensuing research added greatly to the understanding of partial-discharge phenomena and the practical implications for mine power systems. Corona is now recognized as a concomitant feature of high voltages. In this section the conditions for the inception of corona, its subsequent behavior, and the effects on components in the power system are outlined.

Figure 17.24 shows a general graph of conduction effects in a gas such as air. These effects can be explained in terms of the ionic theory of conduction (*28, 30*). Up to point *A* in the graph, Ohm's law is valid; at point *A*, saturation occurs because of the space-charge density. After the potential is increased to point *B*, the gas begins to ionize and the field changes from a subdischarge field to an ionizing field. With any further increase in potential there will be complete breakdown; in other words, a discharge field is attained.

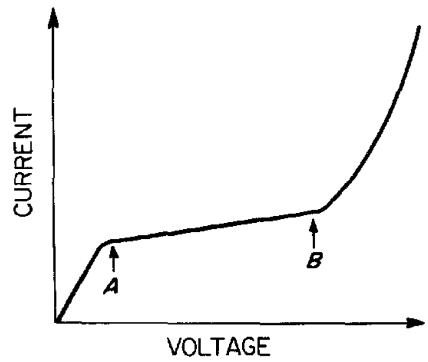

Figure 17.24.—Conduction in gas.

The proportionality between current and voltage up to point *A* occurs because the electric field is so weak that it ionizes very few gas molecules. Hence, the number of ions and free electrons is very small compared with the number of gas molecules. Multiplication effects are minimal, so in general the current depends only on the mean speed of the ions and their relative numbers. The formation of ions and free electrons is approximately equal to the number of recombinations, and the current will depend on the rate of this progress. Saturation between points *A* and *B* occurs when the electron density reaches such a level as to decrease the field intensity; the flux of ions and electrons, thus current, remains constant. A further increase in potential causes the ions and free electrons to acquire sufficient velocity that, when they collide with neutral molecules, enough energy is imparted to split the molecules into ions and free electrons. This process increases exponentially; thus, the current also increases exponentially. This *ionization by collision* is the most important mechanism of conductivity in gases.

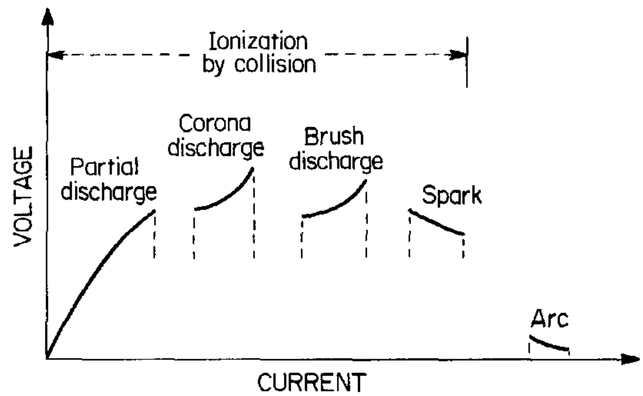

Figure 17.25.—Discharge sequence in an ionizing field.

The most complete breakdown in the process of ionization by collision gives rise to a spark. Essentially, a spark consists of a quantum of electrical energy traveling through the gas, with the associated current limited only by the source. It is very unstable because the passage of current lowers the voltage. Below complete breakdown, there are three other types of discharge that are known collectively and rather loosely as *corona*. These are the discharges of interest here. In descending order of magnitude they are:

- Brush discharge: These discharges are often considered an anomaly of corona and consist of a very small number of sparks ending in air.
- Corona discharge: The onset of this discharge is signaled by a glow or halo ("corona" is the Latin word meaning "crown").
- Partial discharge: This discharge cannot be detected visually; chemical changes occur in the gas.

In this chapter the term *corona* will be used to describe the phenomena of partial, corona, and brush discharges.

Figure 17.25 is a plot of the ionizing field for gases showing the different types of discharges. In any given situation, the sequence of discharges will always progress in the order shown, but the discharges can begin at any point in the sequence (*28*). The arc in the figure is included for completeness, but this discharge is not a product of ionization by collision; it is an unstable condition that occurs when the electrodes are hot enough to supply electrons for the current, thus vaporize the anode, and create positive particles that heat the cathode by impact (see chapter 9).

Any system where corona may be initiated is known as a *corona source*. The specific area of discharge, the *corona site*, is the source of transient currents that pulsate for a few nanoseconds with a current too small to be measured directly. When the corona pulses occur at regular intervals for several minutes, the phenomenon is known as *continuous corona*; where the periods succeed each other at increasing intervals, it is known as *intermittent corona*. The frequency may vary from 1 pulse per minute for dc, to 100,000 pulses per second for 60-Hz ac (*8*). The shape of the corona pulse varies widely, being affected by the corona current at the site.

Discharges are a response to electrical stress. Mathematically, the conditions for the occurrence and maintenance of the discharge are given by Townsend's continuity theorem (*28*):

$$\int_0^{\ell} \alpha[\exp \int_0^{x} (\beta - \alpha)dx]dx = 1, \tag{17.6}$$

where α = ionizing coefficient of positive ions,
β = ionizing coefficient of negative ions,
and ℓ = path of ionization (e.g., wire).

A very important part of Townsend's theory shows that discharges occur only in nonuniform electric fields; these are also the points of greatest electrical stress concentration. Common geometric configurations conducive to high stresses and thus to the formation of corona are shown in figure 17.26.

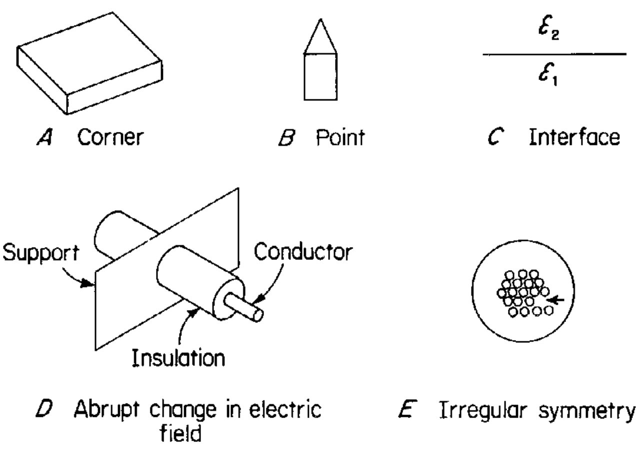

Figure 17.26.—High-stress geometrics.

Corona Behavior

The inception and behavior of corona from an ac potential can be illustrated by considering a single bare conductor that is parallel to a ground plane. In this case, the dielectric or insulator is air. A high-voltage test set is connected to the conductor, and as the voltage of the conductor is increased, the gradient between the conductor and the ground plane rises. When the gradient reaches a critical value, the air molecules at the conductor interface are ionized. The voltage at this point is referred to as the *corona inception level.*

The ionization rate is a function of the air temperature, pressure, and the potential gradient. Higher gradients result in higher ion and electron velocities, hence, greater energies. When these particles strike other molecules, the energy transfer is sufficient to cause ionization, and there will be an exponential increase in ionized gas. The ionization process will continue outward until the ionized particles no longer have sufficient energy to split any other molecules. During this process, the conductor will have the characteristic violet halo of corona discharge, extending outward for two or three times the conductor diameter.

Closer examination of the conductor will reveal a group of bright beads, evenly spaced and superimposed on the otherwise uniform halo. The beads are negative corona, which occur on the negative side of the sign wave; the uniform glow is positive corona, which occurs on the positive half. The inception level for negative corona is higher than that for positive corona and negative corona is believed to be more destructive (*22*). Sensitive detection equipment has been used to determine the macroscopic ionization frequency, which varies from 1 to 10,000 Hz (*8*).

If the line voltage is reduced to the corona inception level once more, the corona will be significantly reduced but it will not disappear. In the case of the bare conductor, a surface discharge, the corona extinction level is slightly below the inception level. For a discharge in a void, the extinction level is 15% to 20% less (*24*).

In the literature, particularly the early literature, researchers tended to be indiscriminate in their use of corona terminology. Two terms are commonplace: *threshold corona level,* meaning the point at which it occurs, and *visual corona level,* meaning the point at which it can be seen. Since corona is now detected and quantified using sophisticated instrumentation that includes ultraviolet light imperceptible to the human eye, the interpretation of visual corona level is subject to confusion, and great care must be exercised when comparing data given in the literature.

The characteristics of dc corona are basically the same as those of ac corona, for their respective polarities. However, dc corona usually occurs intermittently and at a slightly higher potential. Since the dc conductor will not change polarity, unlike the ac conductor, a surface charge is deposited by the initial discharge, which must leak away before another discharge can occur. This decreases the frequency of discharges. The ac corona is often considered to be the mean of positive dc and negative dc coronas (*15*).

Harmonics due to corona can range into several thousand, but the third harmonic is the chief development (*21*). There is a dual relationship between harmonics and corona: harmonics are caused by corona, and there are also corona losses associated with the harmonics. In transmission systems, the latter are the most important.

In 60-Hz systems, corona occurs during part of each half cycle, and the corona discharge is pulsated at twice the line frequency. This causes a cyclic change in line admittance, which results in a modified or distorted waveform (*6*). The sinusoid has considerable harmonic content, particularly the third harmonic; the fifth, seventh, and ninth harmonics are often present. Fourier analysis of the symmetrical components of the current shows that the third harmonic must be composed only of zero-sequence currents or voltage; therefore, if the lines are connected in a grounded–wye configuration, the triple-frequency (third-harmonic) currents caused by corona will flow through the lines and into the ground loop. Zero-sequence currents do not flow in delta-connected systems; instead, a triple-frequency voltage pulsates between lines. The appearance of other harmonics as currents or voltages is easily determined by knowing the sequence of the harmonic in question (*18*).

Ionization during corona causes radiation of radio frequency noise. Below the visual corona level, radio frequency noise is negligible, on the order of 10 μV. The radio interference voltage increases rapidly up to 100 or 200 μV with the occurrence of visual corona (*17*).

When air is ionized during a surface discharge, ozone and nitrogen oxides are the principal byproducts. If moisture is present, the nitrogen oxides combine with the water to form nitric acid, which causes gradual deterioration of the insulation. Ozone is very unstable and changes quite rapidly to harmless diatomic oxygen, but a thin layer of ozone persists in the vicinity of the insulation. This can cause the insulation to harden, to become brittle; and if the site is under flexure stress, it will develop large cracks.

The effects of corona in small or microscopic voids within insulation can cause more serious damage because the site is not visible to external inspection. Virtually all commercially available insulated conductors contain air within small random dielectric voids. Such voids usually occur as bubbles with diameters ranging from 0.1 to 0.01 mm. Despite excellent quality control, present economics and technology prohibit the manufacture of insulated conductors that do not contain some voids. An air film may also exist between the conductor-semiconductor interface, or between layers of insulation, and occluded gases have also been observed in the interstices of the dielectric, as shown in figure 17.27 (*24*). Voids can also result from careless handling or poor splicing practices in the mine.

Little is known about the air spaces within a given insulation. The size, location, distribution, pressure, and gas content of the voids are all unknown, making corona evaluation difficult. The boundary between the dielectric void and the insulation can be represented as two materials having dielectric constants E_1 and E_2 with a potential across them (fig. 17.26*C*) (*24*). This series combination of the two materials results in an effective dielectric constant, E_k, which is less than either E_1 or E_2. From electromagnetic theory, it is known that this effect is due to the increased electric-filled intensity at the interface, caused by the discrete change of the dielectric property (*7*). The breakdown potential of the contained air is lower than that of surrounding air; thus in a void, corona can occur at a voltage that is below the rated corona extinction level.

Degradation of insulation by corona can be explained by two separate mechanisms that result in the chemical decomposition of the dielectric. Electron bombardment is believed to be the primary mechanism; chemical degradation is a secondary mechanism. As explained earlier, ionization by collision depends upon the continuation of energy transferal during collision. Depending on the potential gradient, the ionization extends outward for some finite distance. However, in a dielectric void, the contained gas will be ionized, and then the ions and free electrons strike the molecular structure of the dielectric. This energy transfer is usually sufficient to break the weak bonds and produce volatile products with a lower molecular weight (*5*). As the void begins to deteriorate, the electrical stresses increase, causing a snowball effect, until the surface of the insulation is pierced. Depending on the location of the punchthrough and the specific cable application, a line-to-line or a line-to-ground fault may occur. The corona site may not result in an immediate failure but nevertheless remain active, continuing to deteriorate the dielectric material.

During surface ionization, the chemical products of the process are solely responsible for dielectric damage, but in voids chemical degradation can be considered a secondary mechanism. Volatile byproducts of ionization, which include some very caustic acids, enhance the deterioration process during ion bombardment.

Corona Detection

Numerous tests have been devised to determine the presence of corona, but the tests have been designed primarily for checking corona in commercially produced cables and, hence, are of limited use in the mine. Nevertheless, the principles underlying these devices could be applied with slight modification for mine use. Since the current of the corona pulse is too small to be measured, its effect on a traveling wave is monitored. The change in the traveling wave can be related to the apparent charge of the corona pulse, which is proportional to the damage caused by the corona. An apparent charge greater than 4 pC is usually considered harmful.

A basic system for detecting corona discharge (fig. 17.28) consists of a partial discharge detector and display, power-separation filter, power supply and voltmeter, and high-voltage transformer. The power supply is used to deliver a voltage high enough to initiate corona. The 60-Hz signal and its harmonics are filtered out using the power-separation filter, leaving only the high frequency due to corona discharge. The detector contains the electronics necessary to integrate the pulses and determine the peak pulse values. Various circuits are used to perform these operations. Practically all detectors work on this principle, though variations are sometimes incorporated to obtain results under special or adverse conditions. The technology in this area is still changing rapidly.

A fundamentally different technique that uses ultrasonics has possible application to mine power systems. This method is based on the fact that corona breakdowns cause both audible and ultrasonic pressure waves at the

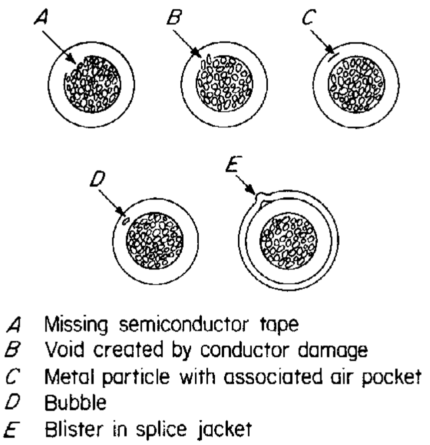

Figure 17.27.—Typical dielectric voids in cables.

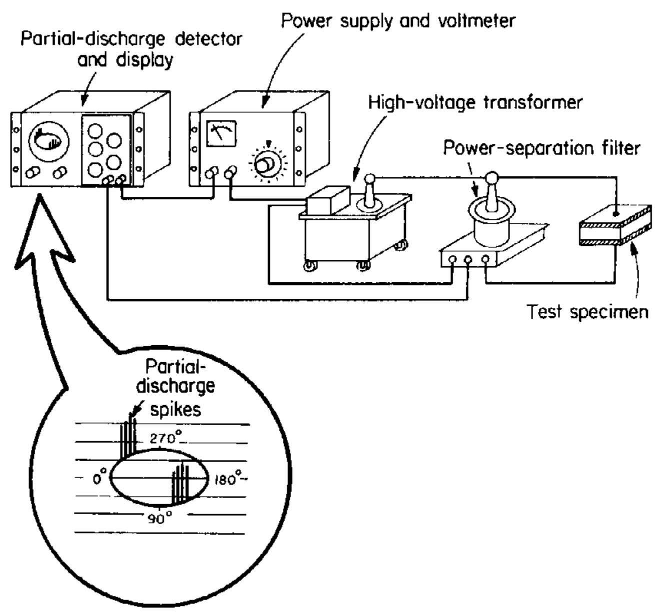

Figure 17.28.—Block diagram for corona-detection system.

corona source. If the medium containing the corona source is in free-moving air, these pressure waves can be detected by an ultrasonic transducer. Successful tests using a barium titanate transducer have been reported (*2*). The advantages of this method are the portability of the equipment and the relative simplicity with which the presence of discharges is determined. Disadvantages include the need to add complex equipment to obtain quantitative data. When traveling through a solid, the pressure wave will follow the path of least resistance, thereby increasing the time of travel and making it difficult to determine the exact location of the corona site. An additional problem is that noise in transformer cores, known as magnetostriction noise, renders the ultrasonic detector useless around transformers.

Partial-Discharge Problems in Mining

In mine power systems, some corona destruction goes undetected until failure occurs. Subsequent analysis of failed cables, couplers, or stress cones frequently identifies the culprit as partial discharge, the lowest level of discharges associated with corona. In other cases, partial discharge is the suspected cause but a lack of corroborative data precludes a direct correlation. Instead, such failures can be attributed to anything from "bad cable" to "transients," which may not be entirely accurate.

In high-voltage systems that require a 15-kV insulation class, problems with partial discharge are common in cables and couplers. Most transients do not have sufficient energy to cause failure in good cable, but when a cable has been weakened by partial-discharge degradation, it becomes susceptible to failure or at least to a higher rate of deterioration because of the increased stress. Ultimate failure can come from a single large energy transient or from a number of smaller transients over a period of time; hence, it is important to minimize transients in the mine power system.

Partial discharge can be initiated by improper cable handling, particularly bending the cable sharply or passing cable directly over metal parts or through insulation. Any abrupt change in the electric field along the cable length causes sufficient stress to initiate partial discharge. The subsequent chemical degradation can eventually reach the conductor surface and terminate with a fault. Cable applications should therefore be investigated carefully before they are installed in power centers, distribution transformers, or other units. A specific installation can be partial-discharge resistant in one case, but susceptible in another.

The severe discontinuity in the electric field that can occur at the termination of a high-voltage cable is a prime site for partial discharge, and various stress-relief systems are employed to prevent its inception. Some of these methods are shown in figure 17.29 and were discussed in detail in chapter 8. Similar problems are found in the confined spaces of high-voltage couplers, where stress relief is provided by preformed filler moldings or stress-control tape. With both methods, it is extremely important to eliminate air voids, since partial-discharge site can be created by any manufacturing or mounting defect.

Couplers can be subjected to other damage caused by partial-discharge byproducts as the result of careless handling in the mine. A coupler with a folded insulator tube, for example, can create a restricted area with lower surface resistivity at the fold. Ozone and nitric acid can form at this site and cause rapid deterioration of insulation inside the coupler housing.

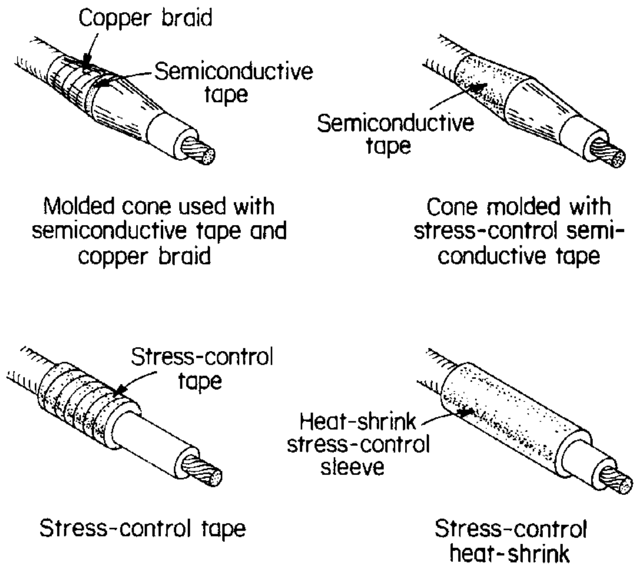

Figure 17.29.—High-voltage cable terminations.

Recent research into discharge phenomena within high-voltage couplers identified typical sites for the formation of partial discharge and demonstrated the importance of adopting impeccable standards of workmanship when repairing or reassembling couplers (*19*). It was shown that the smallest void, cavity, or discontinuity can become a partial-discharge site; examples are

- Small nicks or cuts in cable insulation.
- A void at the termination of extruded semiconducting tape.
- A service loop in the grounding strap that is too long and is located too close to the uninsulated power conductors when assembled within the shell.
- An uninsulated void between the pin and the original cable insulation (fig. 17.30).
- Nuts on conductor pins that, when turning, have caused conductor strands to untwist, forming voids.
- Voids in insulation caused when pinned conductors are inserted into insulator tubes and the middle section of the coupler shell. (A well terminated coupler end might have discharge levels below 3 pC but may increase to 30 pC or more when inserted into the insulator tubes.)
- Voids at the bottom of insulator tubes (fig. 17.31).
- Voids formed when potting compounds cure and shrink away from the coupler shell.

The use of a detector such as that diagrammed in figure 17.28 was found to be essential when attempting to isolate partial-discharge problems in failed couplers.

The extinction level of partial discharge is an important parameter in all parts of the mine power system. The critical value for initiating or extinguishing partial discharge is the potential to the ground plane. Since the neutral in high-voltage mine distribution systems establishes the ground plane, the main concern is line-to-neutral voltage. Obviously, the extinction level must be safely above the nominal line-to-neutral system voltage;

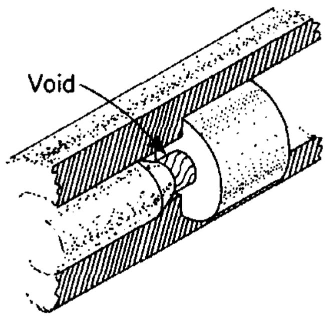

Figure 17.30.—Major insulation void sometimes found in high-voltage coupler terminations.

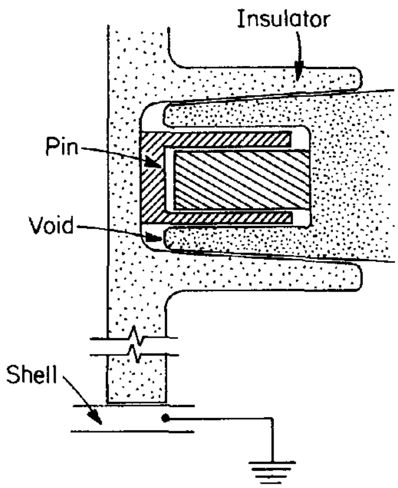

Figure 17.31.—Possible stress site in high-voltage coupler insulators.

otherwise, partial discharge initiated perhaps by a transient overvoltage would continue. A safety factor of 25% above the maximum line-to-neutral voltage is consistent with power-engineering practice (*3*). This converts to an 11-kV extinction level for the 15-kV couplers used on mine distribution systems.

In general, the recommendations for reducing partial discharge in the mine system are

- Minimize high-voltage transients.
- When installing cables inside power centers, belt transformers, and so on, take care not to bend or stretch the cables unless they have been designed for that purpose.
- Pay special attention to eliminating insulation voids and possible partial-discharge sites in high-voltage systems.

INTERMACHINE ARCING

Intermachine arcing, another maintenance problem, refers to electric arcing between the frames of underground electrical face equipment that is of sufficient magnitude to ignite explosive methane-air mixtures. Wolf (*29*) has described the sources and corrections for ac intermachine arcing on mobile face equipment, and this section summarizes his work.

When the 1969 Coal Mine Health and Safety Act specified that all low-voltage and medium-voltage trailing cables for ac mobile equipment must contain an insulated conductor for the ground-monitoring circuit, mine operators could not comply immediately because available cables did not meet the specified requirements. The demand prompted conversion from three-conductor type G round and flat cables to a newly developed type G–GC. In this cable, one of the grounding conductors is insulated to serve as a pilot conductor, and the size of the remaining grounding conductors is increased. The result is a cable with asymmetrical cross section. In 1971, mining companies began replacing their mining equipment trailing cables with this new type, but soon after, sparks were observed arcing between equipment frames. Following exhaustive tests by the Mine Enforcement and Safety Administration (MESA, now MSHA), it was concluded that this sparking was caused by an induced voltage in the grounding conductors of continuous miner trailing cables that originated from the cable asymmetry. Subsequent investigations confirmed that the energy released by the arc was incendive.

A review of the conditions associated with this phenomenon indicates that the problem increases with trailing-cable length, the power demand of the machine, and the asymmetrical geometry of the cable. The presence of the insulated ground-check conductor in the cable causes a physical asymmetry that aggravates the problem. Such is true whether the cable is shielded or nonshielded.

There are four basic methods for solving the induced-voltage problem and the subsequent arcs, which will be discussed in turn:

1. Transposition of the phase conductors,
2. Use of symmetrical-cable types,
3. Use of diodes to suppress arc currents, and
4. Use of saturable reactors to suppress arc currents.

Transposition means to change the power-conductor position with respect to the individual neutral conductors at specific locations along the cable length. Induced voltages in the grounding conductors can be cancelled by dividing the cable precisely into thirds, and cutting and resplicing it as shown in figure 17.32. This method is based on the fact that if a grounding conductor is located the same distance from the three line conductors, the voltages induced by the line currents sum vectorially to zero. Equal transposition of the power conductors along the cable length has the same effect.

Although this method is both simple and effective, its disadvantages inhibit its use in mining. It shortens cable life and physical strength, and in actual practice equal transposition lengths are difficult to maintain, and perhaps most important, correct transposition can be easily lost during subsequent splicing.

The use of symmetrical cables is an alternative solution. In theory, symmetrical cables exhibit no induced-voltage or arcing-current phenomena. The common symmetrical cables are the type G round and type G+GC round (see chapter 8). However, a symmetrical cable that has been spooled or one that has been physically abused exhibits considerable distortion and is not truly symmetrical. Further, any unbalanced power-conductor load or conductor defect will alter the cable symmetry, and the resulting induced voltage will be impressed on the machine frame. Despite these problems, the use of symmetrical cables appears to be the best long-term solution to the problem of intermachine arcing. It should be noted that induced grounding-conductor voltages can be negated only if the continuity of each individual neutral conductor is assured through monitoring. This is obviously impossible with SH–D cable types.

Another approach to solving the problem is to insert a nonlinear impedance in the grounding circuit. The device must have a high effective impedance at low induced voltages, but a low effective impedance to the higher voltages and currents that are available during a ground fault. Any such device must have a continuous-current rating of at least 25 A and have a short-circuit capability equivalent to the cable grounding conductors.

Diodes connected in a bridge arrangement, as shown in figure 17.33, apply this concept. The center diodes conduct at all times while the other diodes conduct every alternate half-cycle. The devices that are usually used conduct about 1.0 A with a voltage drop of 0.6 V. Since an incendive arc occurs with about 1.0 A, the danger of explosion can be eliminated with a sufficient number of diodes. For instance, with an induced voltage of 6.0 V, 10 diodes in series are needed, which means that a total of 12 diodes are required in the bridge arrangement. Because simultaneous line-to-neutral faults on different lines at different system points are possible, if the diodes fail, it is essential that they fail short, not open.

A saturable reactor is a nonlinear element, which can limit arcing current without affecting fault current. The saturable reactor has several advantages over diodes, mainly, that there is no junction of semiconducting material subject to damage. The device is simply an iron-core inductor that has a high impedance to the point where the core is saturated. The inductance of the reactor is much less after saturation than before. This means that the device will limit induced voltages and arcing currents to a safe value, while operating as a linear device. For fault currents and voltages, the saturable reactor operates in both linear and nonlinear regions; in other words, the currents are not sinusoidal, but the effective impedance will be low because the greater part of operation is in the nonlinear region. Thus, the grounding system is still effective.

The selection of a suitable saturable-reactor characteristic is somewhat similar to that for diodes. The main input is the amount of induced voltage present. Once this is known, a reactor can be chosen that will operate in a linear region for the induced voltage, limiting arc current to 1.0 A or less. The reactor should saturate for a slightly higher voltage and thus be effective in limiting machine frame potentials during faults. An acceptable saturable-reactor characteristic is shown in figure 17.34.

Saturable reactors can be damaged by high separate-line, simultaneous-fault currents, and the winding conductor can be fused by the large current if its capacity is not great enough. As with diodes, the device must fail short in these cases. Another problem is that saturable reactors can store energy, which lowers the atual incendive current. Such energy storage can be restricted by selecting a low quality factor (X/R ratio).

Saturable reactors and diodes are equally effective if they are installed between the machine frame and the grounding conductor or in the power center between the grounding conductor and frame ground. But when they are installed in power centers, the grounding pin in metal-enclosed cable couplers must be isolated from ground; otherwise, the device will be shorted out. (This is the same precaution as that recommended in chapter 9 for wireless ground-check monitors.)

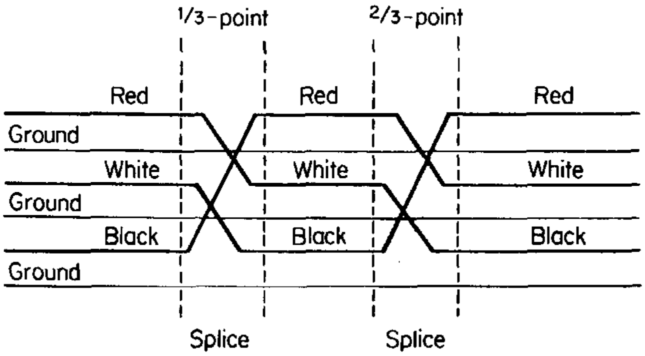

Figure 17.32.—Power-conductor transposition on three-conductor type G cable.

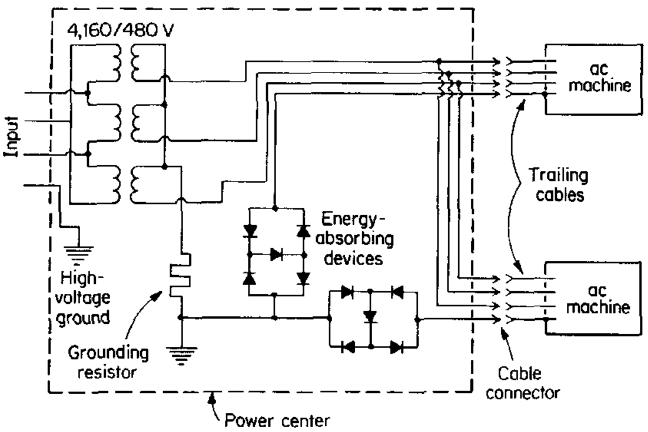

Figure 17.33.—Application of diode-suppression bridges in power center.

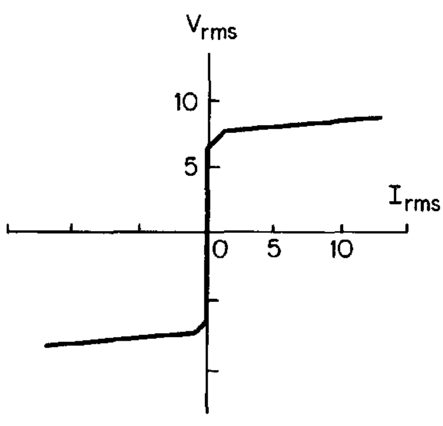

Figure 17.34.—Typical saturable-reactor characteristic.

GROUND DIRECT CURRENT OFFSETS

The dc offsets in grounding can be a serious problem wherever mixed ac–dc systems are used. Intermachine arcing can result from stray dc as well as from ac induced voltage on grounding conductors. The presence of dc ground currents on the ac grounding system can also cause protective-relaying malfunctions with ground-fault relays and ground-check monitors. This section discusses the methods available to eliminate these offset currents, while information on the sources of the currents can be found in chapter 7.

A discussion of corrective methods cannot be restricted to dc offset currents alone since they are only one anomaly caused by dc superimposed on the ac ground in a bipower ac–dc system. All major problems, including ground-bed deterioration by electrolysis, must be considered. The analysis can be conveniently divided into three parts:

1. Feasible power-system modifications,
2. Grounding-system construction, and
3. Water-distribution systems.

Power-System Modifications

The dc finds its way onto the ac ground by direct contact between equipment frames and the mine floor and also through two-wire dc shuttle car supplies where the negative lead is tied directly to the frame of the shuttle car and rectifier. In the first case, stray current from the mine's trolley system flows through the mine floor, roof or ribs, then moves through an ac equipment frame and into the ac safety-ground circuit. Stray current tends to flow in the mine floor because here it finds a lower resistance path back to the rectifier through the track. The resistance is usually higher in the track because it is difficult to bond track systems satisfactorily underground. Some bonding procedures are hard to follow and lead to improper bonds, while other methods are easier to use but result in a bond that is physically unsound and easily damaged. Even if a good bond is achieved, it can be destroyed by heavy rail usage or even the type of abuse caused when moving continuous miners into the section over the tracks.

A parallel feeder, bonded at each rectifier and also bonded to the track at 100-ft intervals, should greatly attenuate the amplitude of stray dc flowing in the mine floor. Stray dc from trolley sources should present little problem when this arrangement is used in conjunction with careful ac equipment and rectifier placement, for example, positioning equipment a minimum of 25 ft from the track or, if this is impossible, placing ac equipment on insulating mats.

The use of isolated or floating dc feed for shuttle cars would help to alleviate the problem with shuttle car loads. A two-wire negative-ground system is cheaper than a three-wire arrangement, but this represents a serious electrical compromise, as does diode grounding. A two-conductor type G shuttle car trailing cable is advocated to prevent offset problems. Although it is not in practice, ground-check monitoring of the dc cables is worthwhile.

Grounding-System Construction

The dc offset problems are minimized with correctly constructed grounding systems. Construction of a ground mat or grid is governed by three needs: to achieve a low value of earth resistance, to control potential gradients, and to prevent corrosion. Low earth resistance is achieved by placing a sufficient length or surface area of metal in intimate contact with the soil. If the soil has a high conductivity, less metal is required than if soil resistivity is high. For safe grounding, a very low resistance is required between the soil and the buried metallic grid. Potential gradients are controlled by the depth of burial beneath the surface and the placement of the grounding conductors in relation to one another. The manner in which a grounding grid responds to the flow of current through it depends on the magnitude and duration of the loading. Generally, all metal structures should be tied together to eliminate potential-gradient hazards. However, corrosion protection necessitates the isolation of underground metallic structures from the corrosive effects of the soil. Detailed information on these subjects is provided in chapter 7.

Water-Distribution Systems

There is no doubt that underground water-pipe systems can serve as excellent sinks for any stray currents moving through the mine floor. Water pipes should be isolated from current sources as much as possible. Where stray currents are present, nonconductive pipe sections could be inserted in the waterline at intervals, which would serve to limit the amount of current carried by the pipe. However, a problem remains because of the water inside the pipe. Although distilled water is an insulator, the presence of dissolved ions renders it conductive, and most mine water is rich in impurities. Hence, the best protection is to maintain isolation (as much as practical) of the water-distribution system from possible stray ground-current sources.

The suggestions given here on dc offset ground currents must be general in nature because no two mine systems have identical problems. Hence, guidelines are more effective than specific rulings in this instance. When this material is combined with that in chapter 7, a thorough list of correction methods for reducing the effects of ground dc offset currents can be assembled.

SUMMARY

The foregoing sections in this chapter have covered many aspects of maintenance, including justification, measurements, and planning, as well as some specific problems related to the subject. Many other areas fall under this general title and its supervision in the mining industry. These aspects are major portions of chapters 6 through 16, where they are presented in detail. Using this knowledge as background, the main objective of chapter 17 has been to integrate additional information into a concept for a feasible PM program.

The importance of adequate PM cannot be overemphasized. Through its employment, equipment availability can be maximized while personnel hazards are minimized. On the other hand, poorly done PM is worse than worthless, and managers who have experienced this are very opposed to any approach to PM in their mines.

Chapter 17 brings to a conclusion this publication. The objective has been to assemble a comprehensive engineering reference on mine power systems. Because not all aspects of mine electrical systems could be included and because this area of mining is rapidly changing and growing, the aim has been to collect as much significant information as possible that will provide the basic tools needed to continue a knowledgeable involvement in mine electrical applications. Such an involvement is important today, but in the future it will have even greater significance as the use of sophisticated electrical systems expands.

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APPENDIX.—ABBREVIATIONS AND SYMBOLS

UNIT OF MEASURE ABBREVIATIONS[1]

A	ampere
Ah	ampere-hour
A/m	ampere per meter
A/m^2	ampere per square meter
A/µs	ampere per microsecond
As	ampere-second
A/s	ampere per second
A/V	ampere per volt
A/Wb	ampere per weber
C	coulomb
°C	degree Celsius
cm	centimeter
cm^3	cubic centimeter
C/m^2	coulomb per square meter
cmil	circular mil
C/s	couloumb per second
C/V	coulomb per volt
°C/W	degree Celsius per watt
deg	degree
F	farad
°F	degree Fahrenheit
F/m	farad per meter
ft	foot
ft^3	cubic foot
ft^3/h	cubic foot per hour
ft·lb	foot pound
ft·lb/V	foot pound per volt
ft^3/min	cubic foot per minute
ft/µs	foot per microsecond
g	gram
g/L	gram per liter
g/m^3	gram per cubic meter
H	henry
h	hour
H^{-1}	reciprocal henry
H/m	henry per meter
hp	horsepower
Hz	hertz
in	inch
in^2	square inch
in^3	cubic inch
in/ft	inch per foot
in/s	inch per second
J	joule
J/K	joule per kelvin
J/s	joule per second
K	kelvin
kA	kiloampere
kg	kilogram
kg/m^3	kilogram per cubic meter
kΩ	kilohm
kΩ/V	kilohm per volt
kPa	kilopascal
kV	kilovolt
kVA	kilovoltampere
kVA/hp	kilovoltampere per horsepower
kvar	kilovar
kV/ft	kilovolt per foot
kV/µs	kilovolt per microsecond
kW	kilowatt
kWh	kilowatthour
kWh/°C-kg	kilowatthour per degree Celsius-kilogram
L	liter
lb	pound
lb/ft	pound per foot
lb/Mft	pound per thousand feet
lb/mi	pound per mile
lb/yd	pound per yard
L/h	liter per hour
m	meter
m^2	square meter
mA	milliampere
µA	microampere
µF	microfarad
mH	millihenry
µH/ft	microhenry per foot
min	minute
µin	microinch
mJ	millijoule
mL/h	milliliter per hour
mm	millmeter
mm^2	square millimeter
MΩ	megohm
mΩ	milliohm
µΩ	microhm
MPa	megapascal
mi/h	mile per hour
ms	millisecond
m/s	meter per second
m/s^2	meter per second squared
µs	microsecond
MV	megavolt
mV	millivolt
µV	microvolt
MVA	megavoltampere
MV/µs	megavolt per microsecond
N	newton
N·m	newton meter
Ω	ohm
Ω/°C	ohm per degree Celsius
Ω-cm	ohm-centimeter
Ω/cm^2	ohm per square centimeter
Ω-cmil	ohm-circular-mil
Ω-cmil-ft	ohm-circular-mil-foot
Ω-cmil/ft	ohm-circular-mil per foot
Ω/cmil-ft	ohm per circular-mil-foot
Ω-ft	ohm-foot
Ω-in	ohm-inch
Ω-m	ohm-meter
Ω/Mft	ohm per thousand feet
Ω/mi	ohm per mile
Ω/V	ohm per volt
oz/ft^3	ounce per cubic foot
pC	picocoulomb
pF	picofarad
psi	pound per square inch
psia	pound per square inch, absolute
psig	pound per square inch, gauge
rad	radian
rad/s	radian per second

[1] Standard IEEE format (rather than Bureau of Mines format) is used for unit of measure abbreviations in this publication.

r/min	revolution per minute
r/s	revolution per second
S	siemens
s	second
S/m	siemens per meter
T	tesla
ton/ft²	ton per square foot
V	volt
VA	voltampere
V/A	volt per ampere
Vac	volt, alternating current
var	voltampere reactive
V/cm	volt per centimeter
Vdc	volt, direct current
V/m	volt per meter
V/Mft	volt per thousand feet
V/mi	volt per mile
V/μs	volt per microsecond
Vs	volt-second
W	watt
W/A	watt per ampere
Wb	weber
WbA	weberampere
Wb/A	weber per ampere
Wb/m²	weber per square meter
Wh	watthour
W/(m·°C)	watt per meter degree Celsius
wt %	weight percent
yd³	cubic yard
yr	year

OTHER ABBREVIATIONS AND ACRONYMS

ac	alternating current
ACSR	aluminum conductor steel reinforced
AMSW	ammeter switch
AWG	American Wire Gauge
BIL	basic impulse insulation level
BS	breaking strength
cemf	counterelectromotive force
CFR	Code of Federal Regulations (U.S.)
ckt bkr	circuit breaker
CRT	cathode-ray tube
CSP	chlorosulfonated polyethylene
CT	current transformer
dc	direct current
DS	disconnect switch
emf	electromotive force
EPR	ethylene propylene rubber
FA	forced air
FET	field-effect transistor
FOA	forced oil and air
GCR	ground-check relay
GCS	ground-check system
GND	ground
GTR	ground-trip relay
h.s.	high-strength (guy grade)
IC	integrated circuit
ICEA	Insulated Cable Engineers Association
IEEE	Institute of Electrical and Electronics Engineers
LCD	liquid-crystal display
LED	light-emitting diode
MCC	motor control center
MCM	thousand circular mils (wire gauge)
MESA	Mine Enforcement and Safety Administration (U.S.)
MESG	maximum experimental safe gap
m-g	motor-generator
MOS	metal oxide semiconductor
MOV	metal oxide varistor
MSHA	Mine Safety and Health Administration (U.S.)
NBR	nitrile butadiene rubber
NC	normally closed
NEC	National Electrical Code
NEMA	National Electrical Manufacturers Association
NESC	National Electrical Safety Code
NO	normally open
NTIS	National Technical Information Service, U.S. Department of Commerce
OA	over air (self-cooled)
OCB	oil circuit breaker
OL	overload relay, overhead line
OTR	overtemperature relay
PCB	polychlorinated biphenyl
pf	power factor
PIV	peak inverse voltage
PM	preventive maintenance
PT	potential transformer
pu	per unit
PVC	polyvinyl chloride
RC	remote control
RFI	radio frequency interference
RM	rotating machinery
rms	root-mean-square
SA	surge arrester
SBR	styrene butadiene rubber
SCR	silicon-controlled rectifier
SEL	sensitive earth-leakage (system)
SF	safety factor
SI	International System of Units
s.m.	Siemens-Martin (guy grade)
TBS	two-breaker skid
TDR	time-domain reflectometer
tpdt	three-pole double throw (switch)
UV	undervoltage
UVR	undervoltage relay
VCB	vacuum circuit breaker
VOM	volt-ohm-milliammeter
VR	voltage regulator
VTVM	vacuum-tube voltmeter
WVDC	working volts, direct current
XLP	crosslinked polyethylene

ELECTRICAL AND ELECTRONICS SYMBOLS

A	anode
	area
a	turns ratio of transformer (chapter 3)
	unit vector e^{j120} (chapter 4)
	radius of rod, spacing between electrodes (chapter 7)
B	susceptance, magnetic field flux density
	area (chapter 7)
	base (chapter 8)
C	capacitance
	collector (chapter 5)
	capacity (chapter 8)
	controlled variable (chapter 14)
D	electric flux density (chapter 5)
	diameter, in feet or meters (chapter 8)
	diode (chapter 9)
d	conductor diameter; coil diameter, in inches (chapter 2)
	distance (chapter 11)
E	electric field strength
	emitter (chapter 5)
	potential (chapter 7)
	difference or error (chapter 14)
	dielectric constant (chapter 17)
F	magnetomotive force
	factor of safety (chapter 13)
f	force, frequency (chapter 6)
	safe bending stress (chapter 8)
G	conductance
	cable code: grounding conductor
	gate, galvanometer (chapter 5)
	ground-fault device (chapter 9)
GC	cable code: ground-check cable
H	magnetic field strength
	feedback element (chapter 14)
h	thickness
H_T	total operation time for motor (chapter 6)
$\underline{I}$	current
$\bar{I}$	current phasor, total circuit current
i	instantaneous current
	current density at electrode surface (chapter 7)
	current through circuit breaker, system current (chapter 11)
$\bar{I}^*$	conjugate of complex current (chapter 3)
j	imaginary operator, $\sqrt{-1}$ (chapter 2)
K	Boltzmann constant (1.38×10^{-23} J/K) (chapter 5)
	torque constant, proportionally constant (chapter 6)
	reflectance factor (chapter 7)
	winding series capacitance (chapter 11)
	cathode (chapter 14)
	relay (chapter 15)
k	coefficient of coupling (chapter 3)
L	inductance
	total length, distance, in feet or meters (chapters 7-8)
l	length, in inches (chapter 2)
	path of ionization (chapter 17)
LF	actual average power consumed divided by rated average power (chapter 8)
M	mechanical moment
	mutual inductance (chapter 3)
	machine contractor (chapter 9)
	voltage-sensing relay (chapter 12)
MP	cable code: mine power
MPF	cable code: mine power feeder
MP-GC	cable code: mine power feeder with ground-check conductor
N	north
	number
	turns of the coil (chapter 2)
n	number
	semiconductor with excess negative charge (chapter 5)
	armature speed (chapter 6)
P	power, permeance
p	instantaneous power (chapter 2)
	semiconductor with excess positive charge (chapter 5)
	number of magnetic poles presented by stator (chapter 6)
Q	charge, reactive power
q	electrical charge, stored electrical charge in capacitor
R	resistance, reluctance
	reference variable (chapter 14)
r	radial distance or moment arm (chapter 6)
	equivalent radius (chapter 7)
S	apparent power
	south
	distance (chapter 8)
	transformer capacity (chapter 11)
s	displacement (chapter 2)
	motor slip (chapter 6)
$\bar{S}$	complex power
SH	cable code: shielded
SH–C	cable code; see tables 8.2, 8.3
SHC–GC	cable code; see tables 8.2, 8.3
SH–D	cable code; see tables 8.2, 8.3
SHD–GC, SHD+GC	cable codes; see tables 8.2, 8.3
SS	substrate
T	circuit configuration (chapter 2)
	junction temperature (chapter 5)
	torque (chapter 6)
	time (chapter 7)
	tensile strength, tension (chapter 8)
t	temperature
	time
	taper (chapter 8)
Ta	ambient temperature (chapter 5)
U	velocity of propagation (chapter 11)
$\underline{V}$	voltage, electromotive force, potential difference
$\bar{V}$	voltage phasor
v	instantaneous voltage
W	energy, work
	weight
	wind (chapter 8)
	cable code; see tables 8.2, 8.3
w	work done (chapter 2)
X	reactance
x	distance (chapter 11)

Y	admittance
y	wye circuit configuration
Z	impedance
$\|Z\|$	magnitude of impedance
z	burial depth
α	temperature coefficient (chapter 2)
	no-load position, angular position (chapter 6)
	firing angle (chapter 14)
	ionizing coefficient of ions (chapter 17)
β	α divided by $(1-\alpha)$
γ	conductivity
Δ	delta circuit configuration
δ	soil density (chapter 7)
ϵ	permittivity
η	efficiency (chapter 3)
θ	phase angle
	maximum allowable soil temperature rise (chapter 7)
	power-factor angle (chapters 4-5)
	protective angle (chapter 11)
	thermal resistance
$\varkappa$	relative permittivity
λ	soil thermal conductivity (chapter 7)
μ	permeability (absolute)
μ_r	relative permeability
ρ	resistivity
σ	conductivity
	soil specific heat (chapter 7)
Φ	potential difference, magnetic flux
	flux direction (chapter 6)
ϕ	phase angle (chapter 2)
	flux (chapter 15)
ψ	electric flux
ω	angular frequency

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