

# The Impact of Ground Control Research on the Safety of Underground Coal Miners: 1910–2010

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**ABSTRACT:** Ground falls claimed the lives of 50,000 us coal miners during the 20th century, more than all other types of underground accidents put together. While seldom garnering headlines, ground control research has been an important focus area for the U.S. Bureau of Mines (USBM) (and now the National Institute for Occupational Safety and Health (NIOSH)). These organizations have played a key role many central developments, including:

- The transition from wood posts to roof bolts
- The requirement that every mine employ a Roof Control Plan
- The development of shield supports for longwalls
- The application of empirical pillar design methods
- The development of improved standing support systems

These and other successful interventions required that the technology, economics, and mining culture all intersect. The paper concludes with a discussion of some current safety technologies that represent the next steps forward for ground control.

## INTRODUCTION

The 20th century witnessed two revolutionary transformations in underground coal mining technology. At the beginning of the century, hand loading was nearly universal. Mechanical loading began the first great change in about 1930 (Figure 1), and this process was completed when continuous miners supplanted drilling and blasting during the '50s and '60s (Anon, no date). The rapid growth of longwall mining during the last two decades of the century was the second revolutionary development.

Neither of these new mining methods could have succeeded without equally dramatic developments in ground control technology. If roof bolting had not replaced timbers, mechanized room-and-pillar mining could not have reached its potential. Modern longwall mining is similarly inconceivable without heavy duty, self-advancing shield supports.

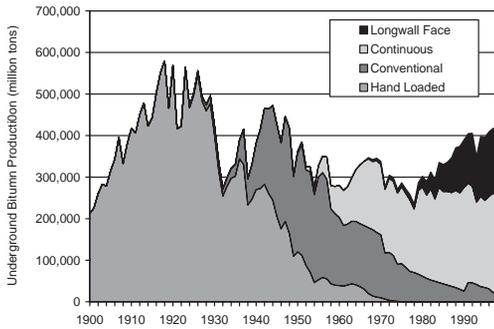
Clearly, these innovations in support technology have had an impact on mine safety as well. While roof falls once claimed hundreds of lives each year, today annual fatalities are usually numbered in the single digits (Pappas et al., 2000). The fatality rate has also improved, measured on either a per-ton or a per-hour basis (Figures 2 and 3). Yet a careful analysis of the history shows that improvements in safety did not occur in lock-step with technological

developments. The *safety culture*, which encompasses regulatory mandates, company-specific safety policies, and other social and legal aspects, has also been central because it determines the level of risk that miners take (or are exposed to) while underground.

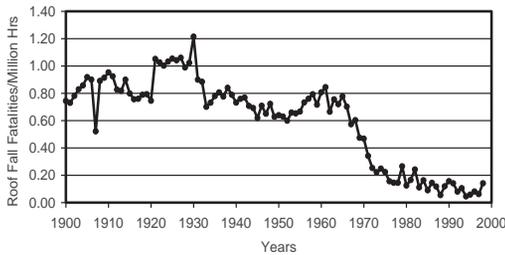
## GROUND CONTROL DURING THE HAND-LOADING ERA

At the turn of the century, roof support was considered the responsibility of each individual miner. It was his duty to “examine his working place before beginning mining work, to take down all dangerous slate, and make it safe by properly timbering the area before commencing to mine coal.” The mine operator was responsible for delivering timbering materials to each working place, and the foreman checked that it was installed properly on his daily visit (Paul and Geyer, 1928).

The miners often had considerable discretion about the amount of support they installed. In weak shale roof, posts were set 2.5 to 5 ft apart (Paul et al., 1930), but where the roof rock was strong, no posts might be set at all. An important element in early roof support systems were “safety posts,” which were set at the end of the track to protect miners while they



**Figure 1. Underground coal mining methods of the 20th Century**



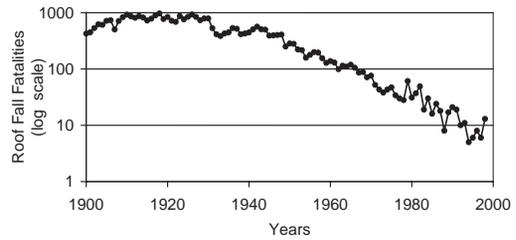
**Figure 2. Roof fall fatalities in underground coal mines**

loaded the coal or prepared the face for the next shot (Figure 4) (Paul et al., 1930). These temporary supports required extra time and effort, and their use was often at the discretion of the miner.

Under these circumstances, it is understandable that much of the blame for roof fall accidents was placed on the inexperience and carelessness of the miners themselves (WVU Press, no date). In 1912, the USBM asked:

How can you, the miner, escape harm from roof falls?" The answer was: "Be careful...do not take the risk of loading a car before putting up a prop...set extra posts, even though they are in your way (Rice, 1912).

Over time, however, safety professionals began to recognize that "a condition responsible for many fatalities from falls of roof is the absence of any policy on the part of management with respect to systematic methods of roof inspection and support" (Paul, 1927). Encouraging mine managers to prepare, promulgate, and enforce a *systematic timbering*



**Figure 3. U.S. roof fall fatality rate for underground coal mines (fatalities per million employee hours)**



**Figure 4. Hand loading with a safety post**

*plan* became a key element in Bureau's roof control efforts.

The USBM also exhorted miners to "comply with systematic methods of timbering, where such systems have been adopted, and exercise judgment in placing additional posts for your own protection." But as long as the typical mine foreman was responsible for about 80 miners, and seldom spent more than 5 minutes with each one during a shift, enforcing timber plans would present a challenge.

### THE IMPACT OF MECHANIZED MINING ON GROUND CONTROL

Between 1930 and 1948, the portion of underground coal that was loaded by machine rose from less than 10 pct to nearly  $\frac{2}{3}$  (Dix, 1988). In many ways, machine mining actually made roof support more difficult. Loading machines required a prop-free front in which to work. The machine operator was usually protected by posts and crossbars, but the helper had to venture into the unsupported face zone (Figure 5) (Coal Age, 1945). A detailed and widely reported Bureau of Mines study conducted in 1951 concluded that "mechanical operations are,



**Figure 5. Machine loading with timber support**

to a considerable degree, more dangerous from the standpoint of roof falls” than hand loading, “notwithstanding that much closer supervision is maintained in such operations.” The study also found that 74% of roof fall fatalities occurred within 25 ft of the working face, and that 3 out of 4 of these took place by the last permanent support (between the last support and the face) (Forbes et al., 1951). The same study also concluded that:

Regardless of roof conditions, minimum standards of roof support suited to the conditions and mining system of each mine should be adopted and followed.... The judgment of the person should never be substituted for the minimum support required in the systematic roof support plan (Forbes et al., 1951).

The difficulties posed by traditional timber supports increased as the early track-mounted loading machines were replaced by crawler-mounted ones. When rubber-tired shuttle cars that carried coal away from the face were introduced in the late 1930s, timbering became a critical bottleneck in the mechanical mining process. Little wonder, then, the Bureau’s Edward Thomas wrote in 1948 that:

The more progressive mining companies are continually searching for improved types of roof support that will give maximum protection and at the same time offer minimum interference with the preparation and loading of coal (Thomas et al., 1948).

### THE INTRODUCTION OF ROOF BOLTING

Even as Thomas wrote those words, the roof bolt was emerging as the leading candidate “temporary legless support” in machine mining (Thomas et al., 1948). As a support, roof bolts are superior to timbers because

“timbers offer support after the strata they are supporting have failed; whereas roof bolts reinforce the roof rock, which contributes to its own support.”

The St. Joe Lead Company was the first major mining company to make extensive use of roof bolts, beginning in the 1920s. Early in 1947, C. C. Conway, Chief Engineer for the Consolidation Coal Company in St. Louis, visited one of the St. Joe mines near Bonne Terre, Missouri, and was impressed with the roof bolts he saw there (Jamison, 2001). He determined to try them at Consol’s Mine No. 7 near Staunton IL. The first roof bolts were installed in Mine No. 7 using hand-held stopper drills (Figure 6). The anchors were expansion shells “similar to those used to support trolley wire,” though slot-and-wedge type anchors like the ones “ordinarily used in the metal mines” were also employed.

The USBM was apparently involved in the roof bolt trials at Mine No. 7 almost from the beginning. Subsequently, the Bureau initiated a major effort to promote the use of roof bolts nationwide. They touted both the safety and efficiency advantages of the new supports (Thomas, 1951). Safety features included:

- A systematic support “within inches” of the working face
- Can’t be dislodged by blasting or equipment;
- Improved ventilation (because of less air resistance)
- Reduced accumulation of explosive coal dust (because places can be cleaned more thoroughly) (Thomas et al., 1949)

Economic advantages included:

- A reduction in the time required to load a place by 15–50%
- Potential for widening rooms



**Figure 6. Installing roof bolts with a hand-held stopper drill**



**Figure 7. An early roof bolt machine**

- Faster haulage
- Reduced labor cost for roof support and material handling

Since the Bureau was without regulatory powers, it had since 1910 “mastered the art of prodding operators into implementing new technologies that resulted from its scientific investigations (Aldrich, 1995). The Bureau’s roof bolting campaign had to overcome numerous barriers to the new technology. The first was the cost and availability of the required equipment. The development of carbide alloy insert bits was essential because it made it possible to drill holes cheaply in hard rock (Thomas, 1956). Hand-held stopper drills were already available at most mines, but they now required a mobile source of compressed air. Many of the first mines to install roof bolts built their own cars to carry the drills, compressor, bolts, and other supplies from face to face (Figure 7). The bolts themselves were not readily available, and sometimes had to be fabricated in the mine’s own shop. Finally, miners were used to the reassuring presence of heavy timbers, and roof bolting seemed to be “reverse in principal to the old methods,” because it “appears at first glance to approximate holding oneself up by one’s bootstraps.”

Between 1949 and 1955, numerous case histories of the successful application of roof bolting from all over the coalfields were reported in the mining press. For example, a major Bureau study in five southern West Virginia mines found that they “produced over two million tons of coal without a fatal accident and only 4 lost-time accidents, as compared with two fatal and 71 lost time accidents over a similar period when conventional timbering methods were used. Production increases ranged from 0.86 to 10.7 tons per man shift.” That these results were achieved during the dangerous process of pillar recovery made them even more impressive. In northern WV, the success of roof bolting in pillar work

allowed 7–10% more coal to be recovered (Flowers, 1953).

Such figures led many safety professionals to concur with Joe Bierer of the West Virginia Department of Mines, that:

Herein lies a wonderful opportunity for the coal industry to bring about an epochal advance in safety for the mineworker, a humanitarian accomplishment to compare with the great social advances of recent years. No such immediately effective and readily confirmed benefit has derived from any other measure ever conceived, or devised, for safety in coal mines (Bierer, 1952).

Unfortunately, even as roof bolting continued its dramatic expansion, overall fatality rates stubbornly refused to go down. The total number of roof fall fatalities declined, but three out five mining jobs also disappeared between 1948 and 1960. Each of the remaining miners actually had a greater chance of being killed in a roof fall than his counterpart in 1948.

The Bureau advanced two explanations for the frustrating lack of progress. The first was that when miners go beyond the last support, they are unprotected, regardless of what type of support is used. A 1954 Bureau study had found that more than 50% of roof fall fatalities occurred in the unsupported space between the last row of supports and the face (Sall, 1955).

The second explanation for the lack of progress was that “many mines are now bolting where the method is marginal in the sense that perfect anchorage cannot be obtained.” Leon Kelly, a Bureau engineer in Vincennes IN, described this process in remarks he made to the 1950 Annual Meeting of the IL Mining Institute. He cited examples from three mines in his own experience in which the level of support had been reduced, and concluded:

In each case, when bolts were first used at the mine, everyone was more or less afraid of them and the pattern that was adopted was followed religiously. As time went on, and none of the bolts fell out, they were taken for granted and it was assumed that bolts would hold up the roof as long as there were bolts in the roof. Some operators are beginning to tell me that we are all overbolting, and naturally when they feel that way they will reduce the either the number of bolts or the length of the

bolts they use.... If failures are accepted as a calculated risk, it is only a matter of time until a serious accident occurs (Kelly, 1951).

Looking back, it seems clear that simply replacing timbers with bolts would not be sufficient to substantially reduce roof fall rates. The success of any support system in a particular application depends not just on the *type* of support, but also on the density of the pattern, the capacity of each unit, when they are installed, the quality of the installation, and many other factors including the span, the rock quality, and the ground stress.

Simply stated, roof bolts can only prevent roof falls if enough of them are installed. That costs time and money. The mine operator's natural tendency was to adjust the expenditure to achieve an *acceptable* level of roof fall risk. Unless the mining culture was changed to reduce the acceptable level of risk, competitive pressures would mean that the new technology would be adapted to obtain the same results as before.

Moreover, it seems clear that roof bolting actually introduced a vicious new hazard, silica dust. The Bureau publicized research that showed dry drilling, with either pneumatic stoper or electric rotary drills, could result in silica dust concentrations up to *200 times* the recommended level of 5 million particles per cubic ft of air. Such concentrations were a serious menace to not just the drill operators, but also anyone working downwind in the return air. But in 1957, the Bureau reported that of the 424 mines using roof bolts, just 8% employed water to allay dust, 35% employed dry dust collectors, and nearly half employed no means of dust control other than respirators (Figure 8). There is little doubt that the prevalence of occupational lung disease among the generation of miners who worked in the dusty,



**Figure 8. Dry drilling using respirators for dust protection**

mechanizing mines of the postwar period was higher than among most others in the past. Unfortunately, it seems that silica dust from roof bolting must have contributed to this terrible human toll.

## THE 1969 ACT

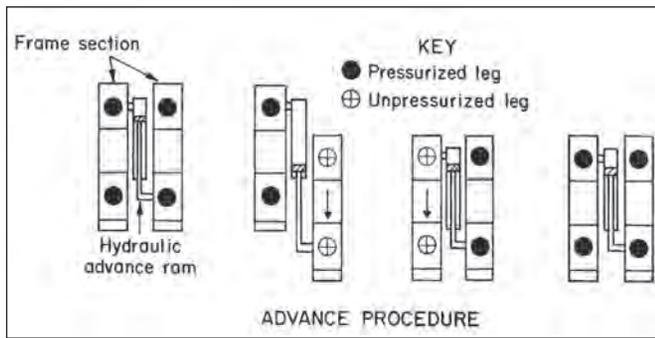
On Nov. 20, 1968, a massive gas and dust explosion destroyed the Farmington Mine and killed 78 miners. The disaster led directly to the Federal Coal Mine Health and Safety Act of 1969, which was signed into law by Richard Nixon. Under the Act, Federal inspectors were given much expanded enforcement powers, and a detailed set of health and safety standards was made mandatory for all mines (Lewis-Beck and Alford, 1980). The key ground control provision, one that Bureau ground control specialists had been advocating for two generations, was that each mine was required to develop a Roof Control Plan which included systematic support throughout the mine. Strict guidelines regarding bolt spacing, bolt length, entry width, and other ground control parameters were also included. Working under unsupported roof without safety posts was banned.

The results were quick and dramatic. In the six years following 1968, roof fall fatality rates plummeted by two-thirds, and they stayed approximately constant at that new level for the next 15 years. The improvement might have been due to regulatory enforcement, or to changes in safety standards implemented by the operators themselves, or to mandated safety technologies like protective canopies and Automated Temporary Roof Support systems. But there can be little doubt that the reduction was not due primarily to new technology, but rather to widespread application of technologies that were already available.

## ROOF SUPPORT AND LONGWALL MINING

Modern longwall mining began in 1952 with a Bureau-sponsored trial at the Stotesbury mine of Eastern Gas and Fuel Associates near Beckley, WV (Haley and Quenon, 1954; Mason, 1976). This marked the beginning of mechanized extraction of a relatively large coal panel using a German coal planar. Roof control for the retreating longwall face was provided by wood cribs and mechanical 40-ton-capacity friction props with I-beam caps. Three panels were successfully mined between 1952 and 1958 producing an average of 530 tons per shift (Haley and Quenon, 1954).

The primary constraint to increased production from longwall operations during this era was roof support advance. Mechanical friction props and



**Figure 9. Self-advancing frame-type roof support**

wood cribs were the predominant support system, and they had to be manually removed from roof contact, and in the case of the wood cribs, reconstructed as the face advanced. Support capacity during this era typically ranged from 25 to 75 tons per jack supplemented by 100 ton capacity wood cribs. The jacks were often damaged during the panel extraction and tended to be unstable in heavy strata conditions, which further reduced their effectiveness.

The next innovation was the introduction of powered roof supports, which were hydraulically pressurized and advanced without manual labor. Eastern Associated Coal Company installed the first hydraulic self-setting and self-advancing roof support system in 1960 in a 52-in Pocahontas coal seam at their Keystone mine in southern West Virginia. These initial self-advancing roof supports were frame-type constructions where two hydraulic jacks were connected by a beam to form a frame construction as shown in Figure 9 (Chironis, 1977a). The units were operated in sets of two with two frames linked together by a central shifting ram. As one frame remained set between the roof and floor, the other frame was lowered and advanced, thus creating a self-advancing support system. Longwall installations using self-advancing frame supports were successful in strata conditions that provided easy caving of the roof behind the supports. However, there were also very serious failures when mines attempted to install these supports under more competent roof strata, such as massive sandstone and limestone geologies.

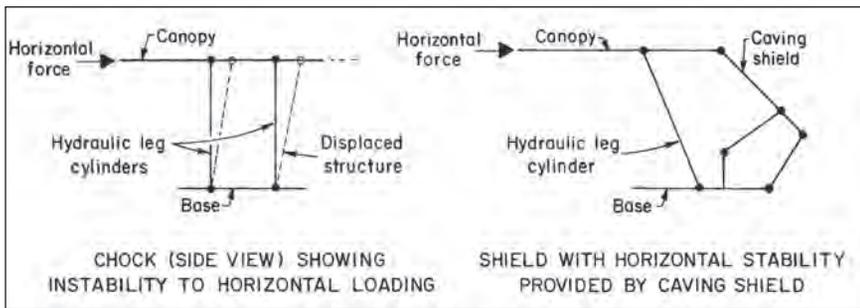
The first high-capacity support system was installed in Island Creek's Beatrice mine in 1966. This system featured 560-ton units that provided excellent control of the competent sandstone roof, and operated for a period of ten years without failure while producing a respectable 1,000 tons per shift of raw coal (Kuti, 1972). Nationwide, support capacity

increased by a factor of 2 to 4 during the next few years, with four-leg supports providing resistances as high as 700 tons.

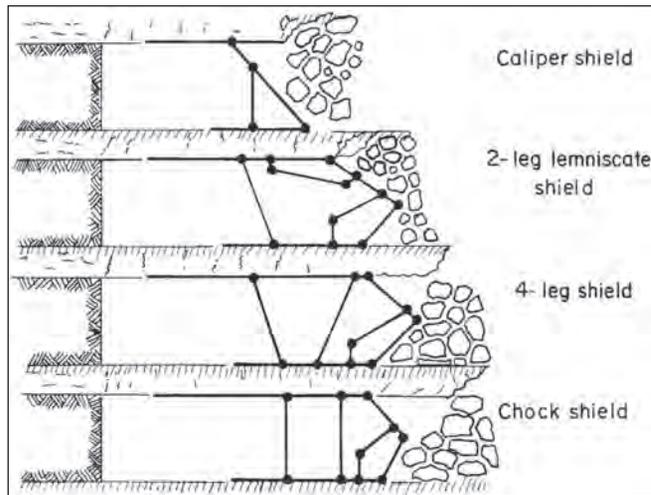
An improvement in support design was also made with development of the chock support. The chock support can be thought of as a mobile crib. It essentially tied two frame supports together with a rigid canopy and semi-rigid base. This design improved structural stability and increased roof contact area over the previous frame-type systems. The increased roof cover helped reduce accidents caused by material falling from the immediate roof, especially during support advance. However, several failures were still reported due to the inadequacy of the chock support systems to operate in competent strata that caved in large pieces and exerted rotational moments or horizontal displacements to the support structure.

It was not until the introduction of the shield support in 1975, with its improved lateral stability and more roof cover (Figure 10), that the last major hurdle to longwall utilization was overcome. The first shield-supported face installed in the United States was by Consolidation Coal Company (Consol) in 1975 at their Shoemaker mine near Moundsville, West Virginia. From the very beginning, encouraging results were obtained. The face produced 750,000 tons in its first year of operation (Chironis, 1977b). The outstanding successes of these initial shield installations created a feverish quest for utilization of this technology. Shield utilization rose dramatically from their introduction in 1975 to 1982, when 83 pct of the longwall faces were shield supported. By 1985, just 10 years after introduction, nearly 90 pct of the operating longwall faces were supported by shields.

The basic shield design itself underwent several developmental changes during this era (Figure 11). The most significant design improvement was the



**Figure 10. Comparison of chock and shield support constructions and stability**



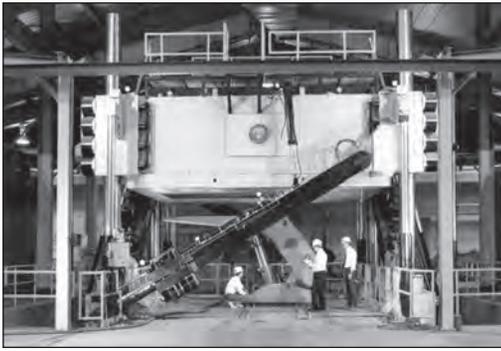
**Figure 11. Modifications of basic shield design**

incorporation of the lemniscate linkage into the caving shield assembly. Previous designs had the caving shield connected directly to the base, which caused the tip of the canopy to transcribe an arc as the shield was lowered and raised. The lemniscate system provided vertical travel of the canopy throughout its operating range, which minimized the unsupported span in front of the shield.

The USBM was extensively involved in promoting the utilization of longwall mining during the 1970s. Bureau sponsored research totaled approximately 67 million dollars, which would be the equivalent of about \$254 million today (BLS inflation calculator, <http://data.bls.gov/cgi-bin/cpicalc.pl>). Included in these efforts were: early demonstrations of the shield support system at York Canyon and Old Ben; demonstrations of thin seam longwall mining, single entry gateroad design, multi-lift extraction of a

thick seam, steeply pitching seam extraction, and an advancing longwall trial.

In 1979, the Bureau under contract to MTS Corporation built a sophisticated load frame at a cost of \$6.8 million (\$20.2 million in 2009) for conducting research on longwall support systems. The Mine Roof Simulator, as it is called (Figure 12), has the capability to provide controlled loading of up to 1,500 tons vertically and 800 tons horizontally over a 20-ft by 20-ft platen area to full scale support structures. This facility was instrumental in helping to identify design deficiencies in the early generation shield supports and to develop testing protocols taking advantage of the active loading capabilities of the simulator. Much of this effort centered around achieving proper load transfer through the caving shield lemniscate assembly, particularly for horizontal shield loading (Barczak and Schwemmer, 1988a).



**Figure 12. Mine Roof Simulator unique active loading frame used for full-scale support testing**

As the shield continued to grow in capacity and the leg cylinders became increasing larger to accommodate the increase in capacity, two-leg shield designs became the standard in the industry. The advantages of the two-leg shield over the four-leg design is more efficient loading, less maintenance, and active horizontal loading whereby the inclination of the leg cylinders can impart through the canopy a confining force toward the longwall face (Barczak and Garson, 1986). This acts to restore the loss of confinement of the roof strata in the face area created by the formation of the gob as the immediate roof caves. The disadvantage of the two leg design is increased toe pressure that can create problems in soft floor conditions. This problem is mitigated by increased contact area and specialized base lifting devices when necessary.

In addition to the increase in capacity, the shields have also increased in size. They have increased in length to accommodate one-web-back operations and larger face conveyors and deeper shearer webs, making these units over 20 feet in length by the end of the 20th century. The standard canopy width by the year 2000 was 1.75 m, replacing the 1.5 m designs that were prevalent during the 1990s. The first 2.0-meter wide shield was installed in Foundation Coal's Cumberland mine in 2003. This trend of increasing canopy width to 2.0 m is likely to continue. The 2.0-m width may represent an upper limit with current shield construction materials since wider shields weigh more, thereby requiring more effort to transport during face moves.

The question of is bigger better or how much capacity is needed remains a debated issue as the largest shield capacity is now 1,750 tons. A consequence of the increase in capacity by increasing the leg cylinder diameter has been a proportional increase in shield stiffness (Barczak and Schwemmer, 1988b).

This results in increased loading per unit of convergence, so convergence which is beyond the direct control of the shield causes an increase in shield loading in proportion to the increase in shield stiffness.

Less debatable has been the desire to maintain consistent and high setting forces. Modern electro-hydraulic control technologies are capable of maintaining a set-to-yield ratio in the 0.6 to 0.8 range, provided there is sufficient flow capacity from the pumping system and the support hydraulics are well maintained. Unfortunately, the increase in shearer speed, automated shield advancement with multiple shields moving and simultaneous conveyor advancement, and wider longwall faces have placed increasing demands on the hydraulic power distribution system and consistent setting pressure remain a concern at many current longwall operations.

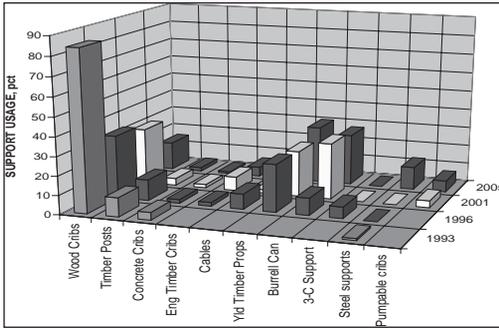
## **DESIGN OF LONGWALL GATE ENTRY SYSTEMS**

On December 19, 1984, 28 miners entered the longwall section at the Wilberg Mine near Huntington, UT. They hoped to break the world's record for one-day production from a longwall. Instead, a fire broke out at an air compressor located near the mouth of the section, and deadly smoke filled the headgate entries. Only one miner escaped.

During the subsequent investigation many safety experts insisted that miners died, not because there was a fire, but because they had no escape route. The miners had been forced to try to escape through the headgate because the tailgate from the Fifth Right longwall had been blocked by roof falls (Cocke, no date). As a result, new regulations were introduced that required longwall mines to maintain safe travel ways on the tailgate side of longwalls (30CFR75, 2000).

At the time, there was very little theory available to assist mine planners in sizing longwall pillars. Traditional pillar design methods were not appropriate because the long and narrow geometry of typical gate entry layouts makes a classic squeeze highly unlikely. In addition, the abutment loads applied to longwall pillars are significantly greater and more complex than the tributary area loading assumed by traditional pillar design methods. Effective longwall pillar design requires some way to estimate the abutment loads during all phases of longwall mining, from development all the way into the tailgate.

Most importantly, however, the stability of the tailgate is not purely a pillar design problem. Studies conducted as early as the 1960s had concluded that "whether or not the stress [from an extracted longwall panel] will influence a roadway depends more



**Figure 13. Distribution of tailgate support since 1993 showing change in support practices from conventional wood cribs and posts into a variety of alternative support technologies**

on the strength of the rocks which surround the roadway itself than on the width of the intervening pillar” (Carr and Wilson, 1982).

The USBM developed the Analysis of Longwall Pillar Stability (ALPS) method to answer these needs. First, design estimates of abutment loads were obtained through an extensive series of underground stress measurements. The greater challenge was how to integrate pillar design into a broader design method that focused directly on tailgate performance.

The solution was to employ a sophisticated empirical technique. The Bureau conducted studies at more than half of all the longwalls in operation in the U.S., and documented historical gate entry performance at all of them. A new rock mass classification system, the Coal Mine Roof Rating (CMRR), provided a quantitative measure of the structural competence of the roof (Molinda and Mark, 1994). Multivariate statistical analysis provided simple, quantitative guidelines for sizing longwall pillars, and showed that when the roof is strong, smaller pillars can safely be used.

The Bureau conducted an extensive program to disseminate the ALPS technology to the industry. ALPS was implemented as a user-friendly computer software package, which was widely distributed free of charge. A number of hands-on computer training sessions were held throughout the coalfields. Since the mid-1990s, ALPS has been considered the industry standard for longwall gate entry design.

In addition to this pillar design effort, an extensive development in standing support to provide increased stability in the tailgate entries occurred in the early 1990s. Timber supports in the form of wood cribs and posts were all that was used prior to mid-1980, when trials of concrete supports were

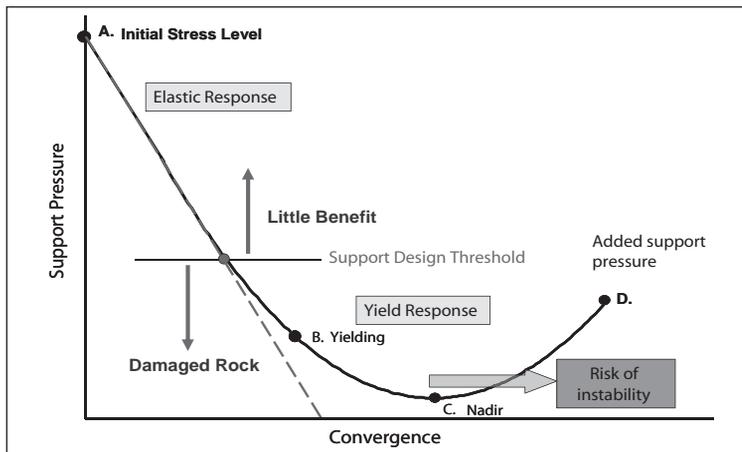
common. Then, the 1990s saw a revolution in new support technologies, providing a wide range of support capabilities. Figure 13 (figure shows a distribution of several types of tailgate supports since this time.

The soft response of the wood cribs that allowed too much roof movement to occur under abutment loading led a search to develop stiffer and higher capacity supports for longwall tailgate applications. However, a valuable lesson was learned from concrete supports that had a compressive strength and material modulus an order of magnitude higher than the conventional wood cribs. Any standing support cannot control the behavior of the overburden. Plain and simple, you cannot prevent all of the closure of a mine opening with any practical and economical support system, and if the support cannot survive this uncontrollable convergence, it will fail prematurely and not provide adequate roof support. These super high capacity and stiff concrete cribs were crushed in several longwall tailgates (Figure 14) (Barczak, 2006).

This lesson has transpired into a goal to utilize the ground reaction concept for support design (Figure 15). The basic concept is to match the performance characteristics to the ground response, whereby the convergence developed in the longwall tailgate is proportional to the amount or capacity of support used. Numerical models are used to develop



**Figure 14. Failure of high capacity concrete crib in longwall tailgate**



**Figure 15. Ground reaction curve for support design**

the ground response curves which are calibrated with in mine measurements of support loading and convergence. The onset of strain-soften rock response, indicating that the rock has been loaded beyond its elastic range and is now yielding causing damage to the rock mass, defines the support threshold. Providing more support than this provides little benefit since the support cannot impact the elastic ground response and will produce a negligible reduction in tailgate convergence. Conversely, too little support will result in greater convergence and increased risk of instability leading to roof fall.

The performance characteristics of the various roof support products are determined from full-scale testing in the NIOSH Mine Roof Simulator through a rigorous testing protocol that determines the limitations of the support system. This data is incorporated in the NIOSH Support Technology Optimization Program (STOP) which facilitates selecting the most effective support for a particular ground condition (Barczak, 2006 and 2008).

NIOSH subsequently employed the same basic empirical approach to develop solutions for several other complex ground control problems. The software packages, particularly the Analysis of Retreat Mining Pillar Stability (ARMPS) and Analysis of Multiple Seam Stability (AMSS), have been widely adopted by the mining community. Each is based on extensive data bases of real world case histories, gathered from in-mine studies. Because they are so firmly linked to reality, they have met the mining community's needs for reliable design techniques that can be used and understood by the non-specialist.

Unfortunately, the 2007 Crandall Canyon mine disaster provided an unwelcome reminder of

the importance of pillar design. The MSHA report on the disaster pointed to the "flawed" mine design as the root cause, because "the stress level encountered exceeded the strength of a pillar or group of pillars near the pillar line." The report documented how the two pillar design software packages used to develop the mine plan, ARMPS and LaModel, had been employed improperly. Following the disaster, MSHA distributed a series of memorandums and other documents that strongly encouraged mine planners to use ARMPS in the pillar design process. The memorandums state that any design that does not meet the NIOSH criteria should be considered "complex and/or non-typical," and be subject to more extensive MSHA review.

## RECENT ACCOMPLISHMENTS AND FUTURE CHALLENGES

*Fatalities inby supports:* Fifty years ago, miners working inby supports were the victim in half of all underground roof fall fatalities. The practice of going inby supports was banned by the 1969 Act, but through the 1990s such accidents still accounted for nearly half of all roof fall fatalities. During the past six years, however, there have been just two inby incidents. The progress is associated with the industry-wide educational campaign under the slogan "Inby is Out!" Another factor contributing to the reduction in inby fatalities is the development of the concept of the "Red Zone." In many roof control plans, miners must now stay outby the second row of bolts, particularly when making extended cuts or turning a crosscut.

*Pillar recovery:* Progress has also been made in pillar recovery. Historically, retreat mining has been less safe than other underground mining techniques. Although the intentional caving that occurs is an unavoidable part of the retreat mining process, premature caving can cause roof falls that put miners at risk. A NIOSH study published in 2003 found that retreat mining elevated a miner's risk of being killed in a roof fall by a factor of three.

The same study also found that many of the fatalities had occurred where the mine was actually following its Roof Control Plan. As a result, NIOSH, together with MSHA, advocated that retreat mines increase the roof support they employ during retreat mining by leaving final stumps, using Mobile Roof Supports, and installing longer and stronger roof bolts. The widespread adoption of such safer retreat mining techniques seems to have made a difference. During the past four years there has been just one fatal roof fall, compared to an average of two per year during the previous decade.

*Rock and rib falls:* More than 400 miners continue to be injured each year by rock falling from between supports, and 100 more are injured by rib falls. Together, these two categories also account for a large percentage of recent ground fall fatalities. Available technologies such as roof screen, rib bolting, and inside control roof bolters could dramatically reduce injury and fatality rates if they were used more widely. Further advances in these areas will likely be the next big advance in ground control safety.

### Disclaimer

The findings and conclusions in this paper have not been formally disseminated by the National Institute for Occupational Safety and Health and should not be construed to represent any agency determination or policy.

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