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Methane Control in United States Coal Mines—1972



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Methane Control in United States Coal Mines—1972

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METHANE CONTROL IN UNITED STATES COAL MINES-1972

by

M. G. Zabetakis,¹ Maurice Deul,² and M. L. Skow³

ABSTRACT

This Bureau of Mines report presents a brief summary of the explosion trends in United States coal mines for the past 160 years. These trends show that in spite of the overall improvements in mine safety, the number of fatalities from ignitions of methane has actually increased in recent years. Suggestions are made of procedures that can be utilized to reverse this trend.

INTRODUCTION

The Bureau of Mines has been interested in mine safety since its inception in 1910. In this connection, a series of studies has been undertaken in recent years in an attempt to remove methane from underground mines in a controlled fashion thereby reducing or eliminating the hazards associated with the presence of this gas. This report presents a summary of the work conducted in 1972.

BACKGROUND

Methane has created a hazard in United States coal mines almost as long as underground mining has existed. While coal was first obtained from the outcrops in Virginia near the turn of the 18th century, it was subsequently removed from shafts as the seam was followed underground. The first explosion from an accumulation of methane reportedly occurred near Richmond about 1810 (9).⁴ Other explosions occurred periodically until mining was discontinued, as open flames were used both for illumination and to test for the presence of methane during this period. With the development of the flame safety lamp in 1815, methane could be detected without the hazards associated with open flames. However, even when this gas was found in an underground area, little could be done to clear it readily. During this period, a miner could use his jacket to dilute the methane by creating a current of air (9, 26)

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⁴Underlined numbers in parentheses refer to items in the bibliography preceding the appendix.

or he could eliminate the gas by igniting it with an open flame; needless to say, the latter procedure was extremely hazardous. Unfortunately, forced ventilation was not utilized extensively at that time even in the older European mines.

As mining rates and depths increased, the number of ignitions, and consequently the number of explosions per year, increased markedly. The first recorded major explosion (one in which five or more persons were killed) occurred in the Black Heath Mine near Richmond, Va., in 1839. Eleven more explosions occurred in the next 36 years with a resultant loss of 280 lives. During this same period (1839-75), coal production rose from 2.07 million to 52.8 million tons per year (4). In the next 35 years there were 180 major explosions, and the rate of occurrence rose to 17 per year in 1909 and in 1910. The number of fatalities from major explosions rose from 14 per year in 1876 to 903 per year in 1907, while coal production rose to 480 million tons in

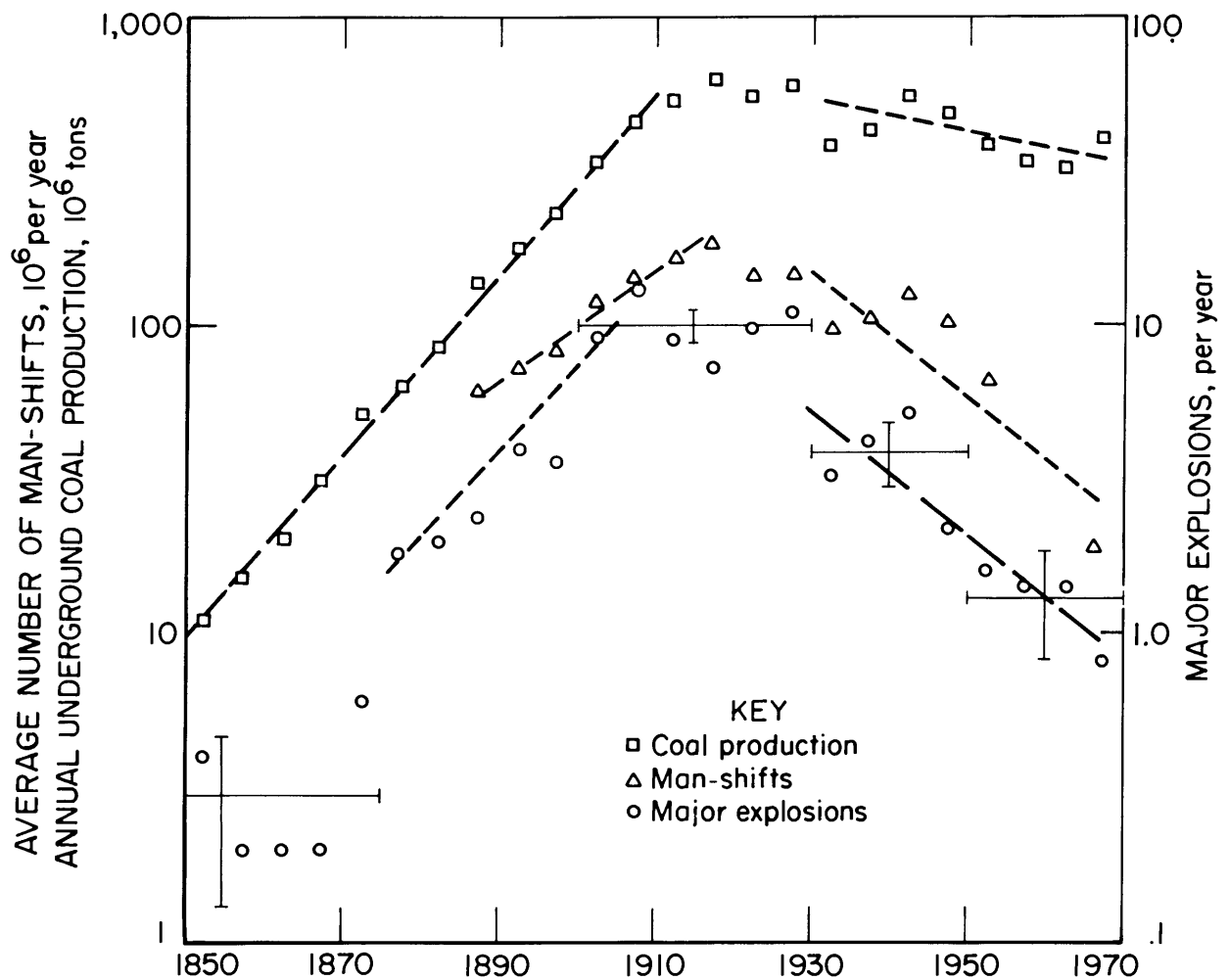


FIGURE 1. - Underground coal production, man-shifts, and major coal mine explosions in United States mines between 1850 and 1970 (5-year averages).

1907. The trend was unmistakable: both coal production and the number of explosions were increasing rapidly. In essence, the increase in coal production at greater depths resulted in an increase in the methane emission rate; this in turn resulted in an increase in the number of accidental ignitions and gas and dust explosions.

The steps taken by the various States and the Federal Government to prevent mine explosions are discussed in detail by Humphrey (9). These included the elimination of a number of ignition sources (open flames, black powder, and high explosives), improved ventilation, generalized rock dusting, the use of sprays at the face, education of underground workers, and improved inspection procedures. A detailed analysis of the trends observed to date, and the inferences to be drawn from these trends in view of the present demand for coal are considered in the following sections.

MINE EXPLOSION TRENDS

The major mine explosions frequency-of-occurrence data included in Humphrey's compilation (9) are summarized in figure 1, along with data obtained from more recent Bureau of Mines reports for the period 1955-70. While the general trends noted above are evident (that is, a low initial rate before 1875, followed by an exponential rise to about 1910, and then a gradual decline), a detailed analysis of these data indicates considerable variability in the rate at which major explosions have occurred over the years. However, if the total number of major explosions against time (fig. 2) is plotted, a figure is obtained that can be used to correlate these data by using a series of straight lines. These lines define certain periods during which the rate of occurrence is fairly constant (table 1). Obviously, there are factors such as methane liberation rate, coal production rate, dustiness, ignition sources, and behavior of workmen that affect the frequency with which major explosions occur. Presumably, during the four periods noted here (1836-75; 1901-30; 1931-50; 1951-70), these factors interacted in such a way as to yield a fairly

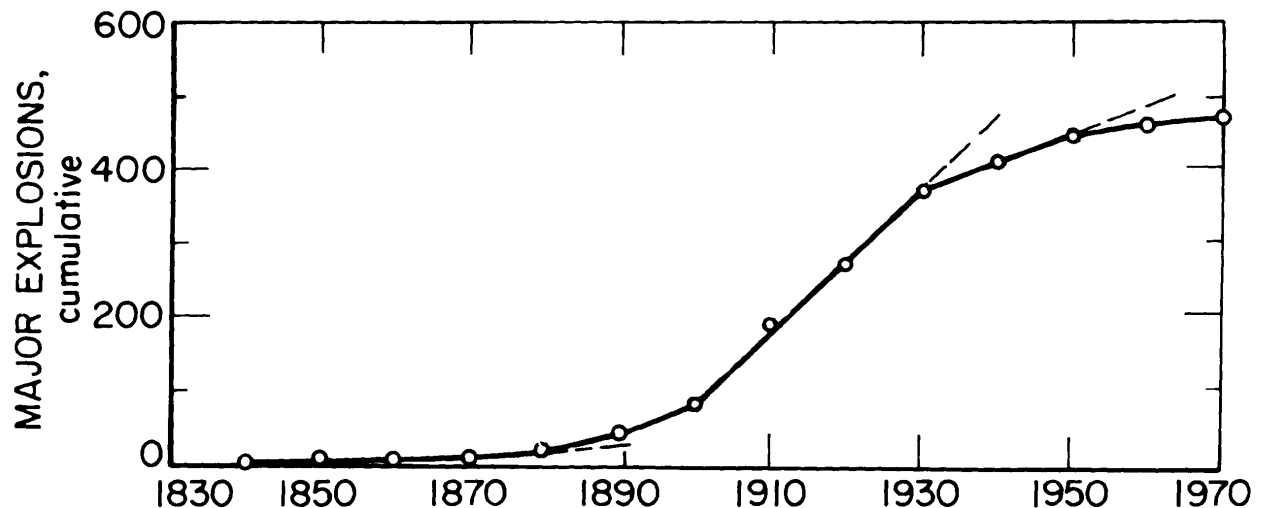


FIGURE 2. - Cumulative major coal mine explosions (1840-1970).

constant rate over each specified interval. This idea can be evaluated statistically by determining the nature of the distribution of these explosions with time, the frequency of occurrence, and scatter of the data (see appendix).

TABLE 1. - Major mine explosion rates during selected periods from 1836-1970

| Period: | <u>Major mine explosion rate (number/year)</u> |
|--------------|--|
| 1836-75..... | 0.30±0.17 |
| 1901-30..... | 9.93±1.17 |
| 1931-50..... | 3.70± .86 |
| 1951-70..... | 1.30± .53 |

During the 30-year period from about 1875 to 1905, major mine explosions occurred at an annual rate, y_1 , given by the regression equation:

$$\ln y_1 = -1.336 + 0.2950X_1, \quad (1)$$

where X_1 is the coded time (year - 1847.5)/5. This regression equation accounts for 91.8 percent of the scatter ($r = 0.958$) in the data about the average value for this period. During the period 1885 to 1920, the number of man-shifts worked per year (in millions) is given by the expression:

$$\ln y_2 = 3.9048 + 0.1952X_2, \quad (2)$$

where X_2 is the coded time (year - 1882.5)/5. This equation accounts for 97 percent of the scatter ($r = 0.985$). An even better fit is obtained for the coal production rate during the period 1850 to 1910 (fig. 1). Here we have:

$$\ln y_3 = 2.0793 + 0.3416X_3, \quad (3)$$

where $X_3 = X_1$. This equation accounts for 99.6 percent of the scatter ($r = 0.998$) in the data about the average value.

Alternatively

$$y_1 = 0.263 \exp (0.2950X_1) \text{ major explosions per year,} \quad (4)$$

$$y_2 = 49.6 \exp (0.1952X_2) \text{ million man-shifts per year,} \quad (5)$$

$$y_3 = 8.0 \exp (0.3416X_3) \text{ million tons per year.} \quad (6)$$

The exponential rate of increase in major explosions before 1905 almost equals the rate of increase in coal production, but each exceeds the rate of increase in underground exposure (man-shifts; fig. 1). Both underground exposure and coal production increased until about 1920 before dropping, while the major explosions rate dropped around 1910, tended to a constant value until about 1930 and then dropped abruptly. Subsequent changes in the major explosions

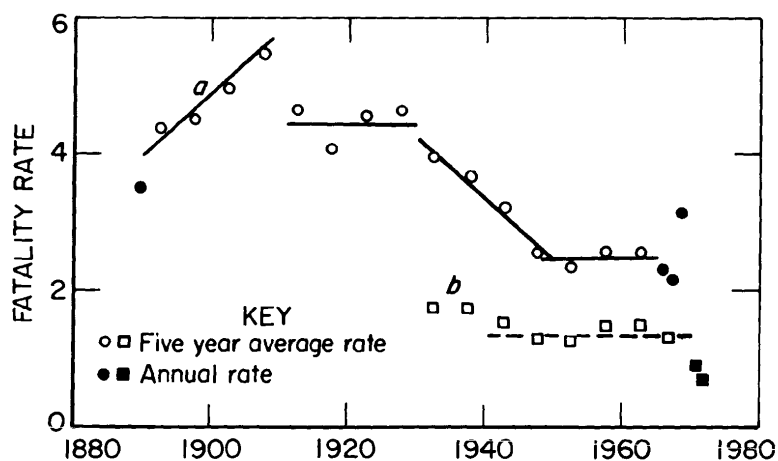


FIGURE 3. - United States coal mine fatality rate: *a*, per thousand 300-day workers and *b*, per million man-hours exposure.

coal mine fatality rate per thousand 300-day workers over the period 1890-1968 (21) shows an increase in this rate until about 1910 when the Bureau of Mines was formed, an abrupt drop to a relatively constant level until about 1930, then a gradual decrease during the next 15 years (as employee turnover decreased and the proportion of experienced personnel increased (1)). Subsequently the annual rates have fluctuated about the 5-year average. Unfortunately, the actual underground exposure times are not known with any degree of certainty during this extended period. Reliable data, available only for the period 1930-72,⁵ are included in curve *b*, figure 3. The fairly constant 5-year average rate during the period 1941-70 (about 1.36 fatalities per million man-hours underground exposure) is noteworthy despite the fact that the annual rates ranged during this period from a low of 0.98 in 1969 to a high of 1.80 in 1968 (standard deviation = 0.20). The annual rate subsequently dropped to a low of 0.91 per million man-hours underground exposure in 1971 and more recently (1972) to a new low of 0.71. By comparison, the fatality rates associated with coal mining operations in general were 0.72 and 0.56 per million man-hours total exposure in 1971 and 1972, respectively.

Approximately one in eight of the more than 90,000 fatalities that have occurred in the Nation's coal mines during the period 1906-68 were caused by gas and dust explosions. Despite improvements in other aspects of mine safety in recent years (figs. 1 and 3), the number of fatalities from ignitions of methane and from major explosions has actually increased slightly in recent years, but the fraction of the industry total ascribed to these causes has increased markedly (table 2). Thus, if present trends were to continue, the number of fatalities from ignitions of methane could be expected to increase still further when production of coal from underground workings increases. Fortunately, this trend can be reversed by the implementation of coal degasification techniques developed here and abroad in recent years. These

rate of occurrence appear to parallel the underground exposure (man-shifts), while coal production has remained at a relatively high level since the mines have mechanized.

DISCUSSION

While only the major mine explosions data (gas and dust) are considered in the preceding section, these have followed the same trends exhibited by other mine accident statistics based on worker exposure. For example, a plot (curve *a*, fig. 3) of the United States

⁵ Compiled by Office of Assistant Director--Technical Support, Washington, D.C.

techniques can be best described in terms of the basic factors that govern the behavior of methane in coal.

TABLE 2. - Fatalities due to ignitions and explosions in underground workings of coal mines

| Year | Fatalities | | Percent of total |
|--------------|----------------|--------------------------|------------------|
| | Industry total | Ignitions and explosions | |
| 1956-60..... | 1,654 | 130 | 7.86 |
| 1961-65..... | 1,178 | 139 | 11.8 |
| 1966-70..... | 991 | 152 | 15.3 |

Basic Factors

Methane is a component part of coal that is formed during the coalification process (23, 27, 33); according to Mott (20), approximately 3,000 cubic feet of methane is produced in forming 1 ton of medium rank bituminous coal (along with copious quantities of carbon dioxide and water). In most United States mines, only a fraction of this methane has been retained in the coal since the balance is lost to the atmosphere.

The amount of methane retained by a particular coal is determined in part by the hydrostatic pressure at the depth below surface at which the coal is found. Figure 4 gives the shut-in pressures measured at the base of a number of vertical wells drilled recently into coal from the surface. In general, these pressures are less than the hydrostatic head:

$$P_{hyd} = 0.435h, \quad (7)$$

where h is the well depth. Similar results have been obtained from measurements made underground (8). For example, a plot of pressure against horizontal distance from the face shows a rather steep pressure gradient near the face, but this gradient decreases rapidly with depth of penetration and finally the pressure approaches the hydrostatic value (fig. 5).

The volume of gas adsorbed by a coal at a fixed temperature is given by the Langmuir isotherm:

$$V_{ad} = \frac{V_a bP}{1+bP}, \quad (8)$$

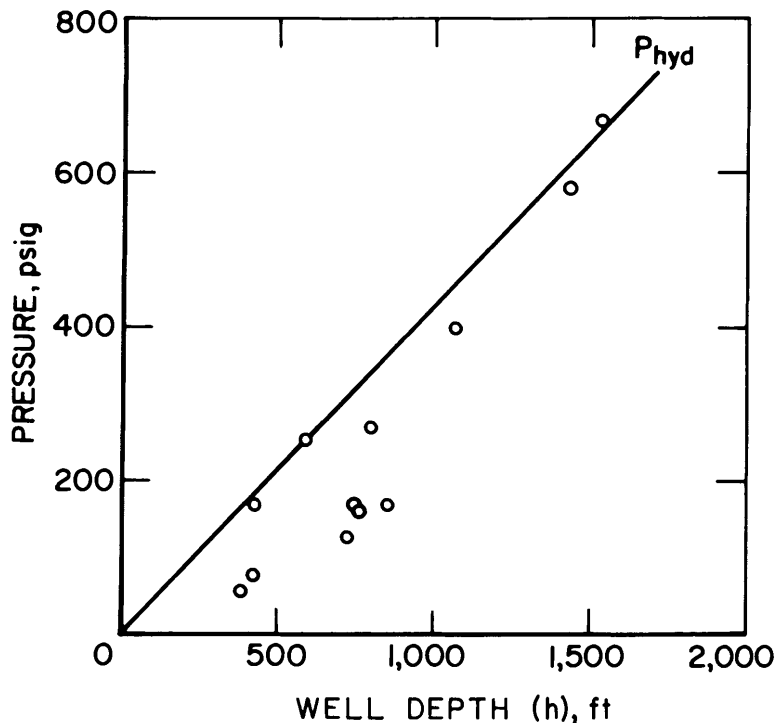


FIGURE 4. - Shut-in pressure versus well depth (h).

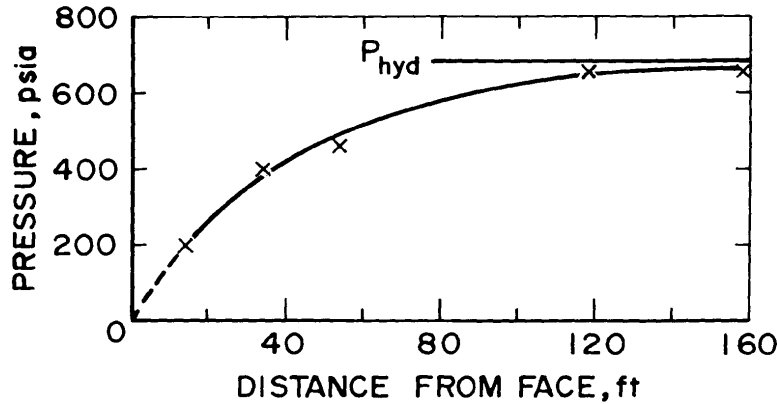


FIGURE 5. - Pressure versus distance from the face.

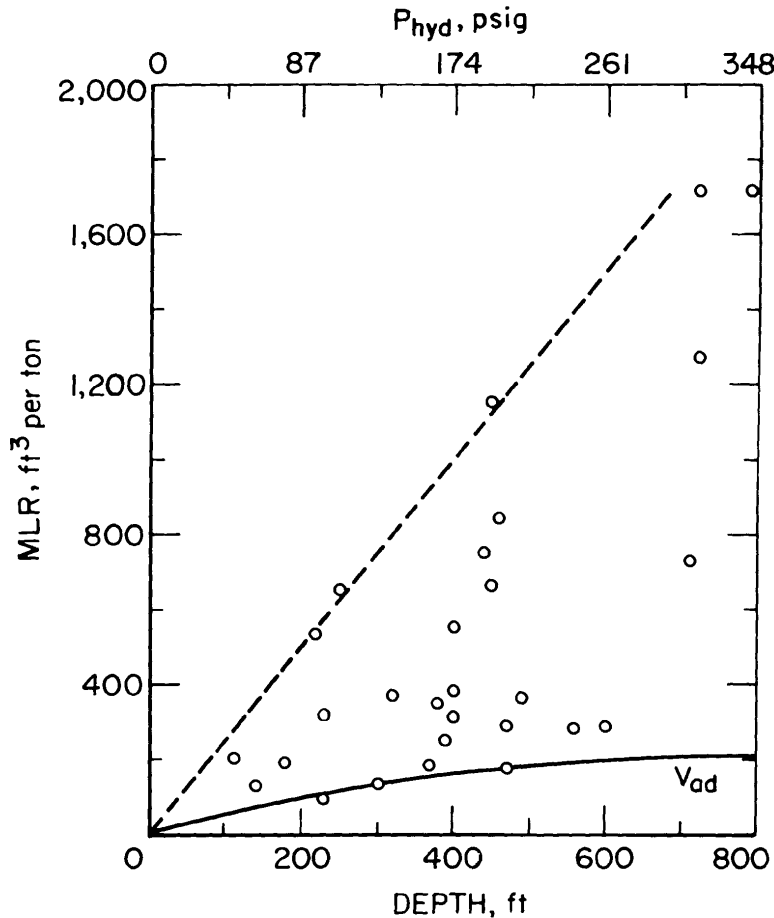


FIGURE 6. - Methane liberation rates of various mines in the Pittsburgh coalbed; V_{ad} for Pittsburgh coal in its normal water-saturated state.

where V_m is the saturation (monolayer) volume,

b is a constant,

and P is the equilibrium pressure in psia.

For example, V_{ad} for Pittsburgh coal in its normal water-saturated state is given approximately by the expression (11):

$$V_{ad} = \frac{1.53P}{1+0.0039P} \text{ ft}^3/\text{ton}, \quad (9)$$

where P is in psia. Taking P to be the hydrostatic pressure, this can be written as:

$$V_{ad} = \frac{0.666h}{1+0.0017h} \text{ ft}^3/\text{ton}, \quad (10)$$

where h is the coalbed depth in feet. A plot of V_{ad} against h is given in figure 6, along with data on gross methane liberation rates per ton of coal mined (MLR)⁶ in a number of commercial mines in the Pittsburgh coalbed (10). With few exceptions, more methane is emitted than that contained (adsorbed) in the mined coal under saturation conditions, and deviations from V_{ad} appear to increase with increase in mine depth. Thus while the rate of change of adsorbed gas with depth (dV/dh) is approximately 0.67 cubic feet per

⁶This is actually a fictitious figure but is now commonly used as a measure of gassiness during periods of active mining.

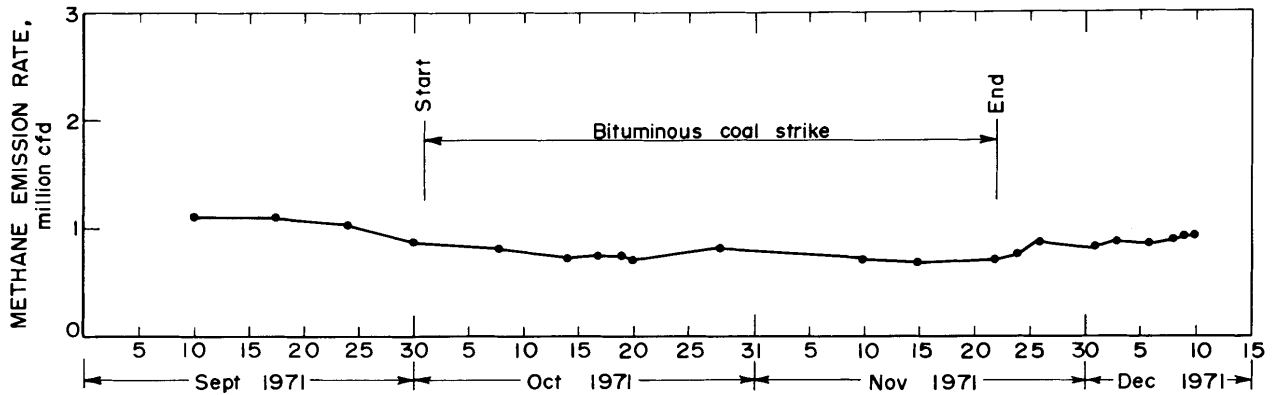


FIGURE 7. - Effect of an idle period on the methane emission rate of an operating mine in Pennsylvania.

ton of coal per foot depth near the surface, the rate of change in the methane liberation rate ($dMLR/dh$) is actually 2.5 cubic feet per ton per foot depth for the more gassy mines. This gives a ratio $MLR:V_{ad}$ of 3.75 for shallow mines; and an even larger ratio for the deeper and more gassy mines. Such behavior results from the fact that a mine is a dynamic system; the gas emitted comes not only from the extracted coal but in practice as much as 70 to 80 percent of the methane enters the mine atmosphere from the exposed ribs and adjacent strata. A striking example is presented in figure 7 which shows the effect of a 7-1/2-week strike-induced idle period on the methane emission in an operating mine (10) where the methane emission rate (MER) of about 1 million cubic feet per day before the strike dropped only to about 0.7 million cubic feet per day during the strike. Assuming that the decrease represents the adsorbed gas normally released during mining, then an $MLR:V_{ad}$ ratio of approximately 3.3 results; this means that nearly 70 percent of the gas enters the mine workings from the ribs and adjacent strata. Admittedly, some of the gas liberated during the strike would have entered the mine as desorbed gas from the mined coal so that this ratio (3.3) may be a little high.

Assuming that a fixed proportion p of the total methane enters a mine through the exposed ribs and adjacent strata, then the methane liberation rate can be predicted by determining the gas content of the coal from a core sample. For example, if p is 0.75 then:

$$MLR = \frac{1}{1-p} V_{ad} = 4 V_{ad} \text{ ft}^3/\text{ton}, \quad (11)$$

In a recent study of cores obtained from a number of rather gassy mines in the Pittsburgh, Pocahontas, Illinois No. 5, and Hartshorne coalbeds, Kissell (14) actually found average $MLR:V_{ad}$ ratios above 6.

Methane Control Techniques

The control techniques considered to date for use in coal mines can be divided into three categories:

1. Controlled dilution with air (ventilation).
2. Blocking or diverting gas flow in the coalbed by means of adequate seals.
3. Removal of pure or diluted methane through boreholes.

The first and most widely used technique involves the use of ventilation air to reduce the concentration of methane to a safe level. It is the only one used universally in underground mines and is a requirement of Section 303 of the Federal Coal Mine Health and Safety Act of 1969 (32). Title 30, U.S. Code of Federal Regulations, Subchapter O, Part 75, Subpart D--Ventilation (31) contains the statutory provisions and mandatory standards related to the ventilation of mines. Other methane control techniques can be considered under the provisions of Section 301(b) of the Federal Coal Mine Health and Safety Act and Section 75.316 of the mandatory safety standards in Title 30, U.S. Code of Federal Regulations. While a few mines have employed other control procedures (5-6, 12, 17-18, 24-25, 28), most mine operators have been rather reluctant to attempt anything that might change their mining plans, this despite the fact that many of the new procedures will decrease ventilation air requirements while increasing productivity by eliminating much of the methane from the active face areas (33). The general tendency to date has been to use additional quantities of ventilation air as methane emission increases. Thus, the large mines in the Pittsburgh coalbed circulate between 4 and 15 tons of air per ton of coal extracted. In most cases this is adequate to keep the methane concentration at the exhaust below 0.5 percent. However, an examination of recent Federal mine inspection reports shows that some mines exhaust air containing from 0.7 to 1.0 percent methane at the fan outlets. Although on the average this meets the statutory requirements, the distribution of ventilation air through a mine is such that there are probably areas of such a mine where the methane concentration is well over 1 percent. This is particularly true in newly developing mines or in mines which are driving development headings into virgin coal. Under these circumstances ventilation alone is usually not adequate, and development is curtailed or stopped. Furthermore, as the methane emission rate depends on both the coal production rate and depth (10), it would be expected that even more air will be needed as mining rates and depths increase unless some of the methane is removed before mining proceeds.

The techniques considered to date in our attempts to prevent the entry of methane into the ventilating airstream are summarized in table 3. Briefly, there are four regions of interest--the working space, the gob, the coal seam, and the adjacent strata; the latter regions are considered because they can and do influence the atmosphere in the working space. Furthermore, as those regions represent the seat of the problem, they have been the focus of all the initial methane control procedures considered by the Bureau in recent months. In particular, attempts are being made to remove the methane before it enters the working space. To date, vertical and horizontal boreholes, water seals, isolation techniques, improved bleeders, tracer gases, and various combinations to determine the best method of gas control in any particular circumstance have been used. For example, in recent months, we have found that as

much as 25 percent of the available methane can be removed from a longwall gob in an operating mine in the Pocahontas No. 3 coalbed by the use of a single partially slotted vertical pipe (19); approximately 180 million cubic feet of gas was removed in the first 15 months. A summary of total methane flow before and after borehole intersection on November 9, 1970, is given in figure 8. At least three coal mining firms are experimenting with such pipes in gob areas.

TABLE 3. - Methane control techniques considered for underground use

| Region | Methane control technique | | |
|--------------------|-----------------------------------|--------------------------------------|---|
| | Diluting with air | Sealing | Removal of CH ₄ |
| 1. Working space.. | Ventilate; isolate and ventilate. | Water infusion; grout; plastic seal. | Chemical process; physical process; biological process. |
| 2. Gob..... | Ventilate; bleeders | Gob..... | Surface boreholes. |
| 3. Coal..... | - | Coal..... | Vertical surface boreholes; horizontal and sloped underground boreholes; multipurpose borehole. |
| 4. Adjacent strata | - | Strata..... | Boreholes. |

In a series of experiments in the Pittsburgh coalbed, it was found that the bulk of the gas entering the face area in a very gassy mine can be diverted by infusing the face with water (3); a summary of the methane flow at two locations before and after infusion of an active face in this mine is given in figure 9. In a companion study, the method of moving averages was used to show the effects of oil and gas wells on the methane flow into the same mine (34). Figure 10 shows the methane flow rate and the rate of change of this rate with distance along an entry in the vicinity of two abandoned wells.

More recently, in a natural drainage experiment in the Pittsburgh coalbed, it was found that long horizontal holes that pass near old wells frequently produce much more gas than those that penetrate the same coalbed in areas free of wells. A schematic of the arrangement used in these studies is given in figure 11 (7). Eight 3-inch-diameter horizontal holes were drilled into the coal from the base of a cased vertical borehole. One hole was used to measure the gas pressure in the coalbed, and the other holes were used to drain water and gas (fig. 12). The latter holes ranged in depth from 500 to 850 feet; initial gas flow rates for the individual bleeder holes are given in figure 13. If we assume that holes 7, 8, and 1 are shielded from the wells by the other holes (particularly 2 and 6), then the initial gas (primarily methane) flow rate (GFR) in the absence of the wells would be given by the equation:

$$\text{GFR} = (0.18 - \cos 2\theta) \text{ cfm/ft}^2, \quad (12)$$

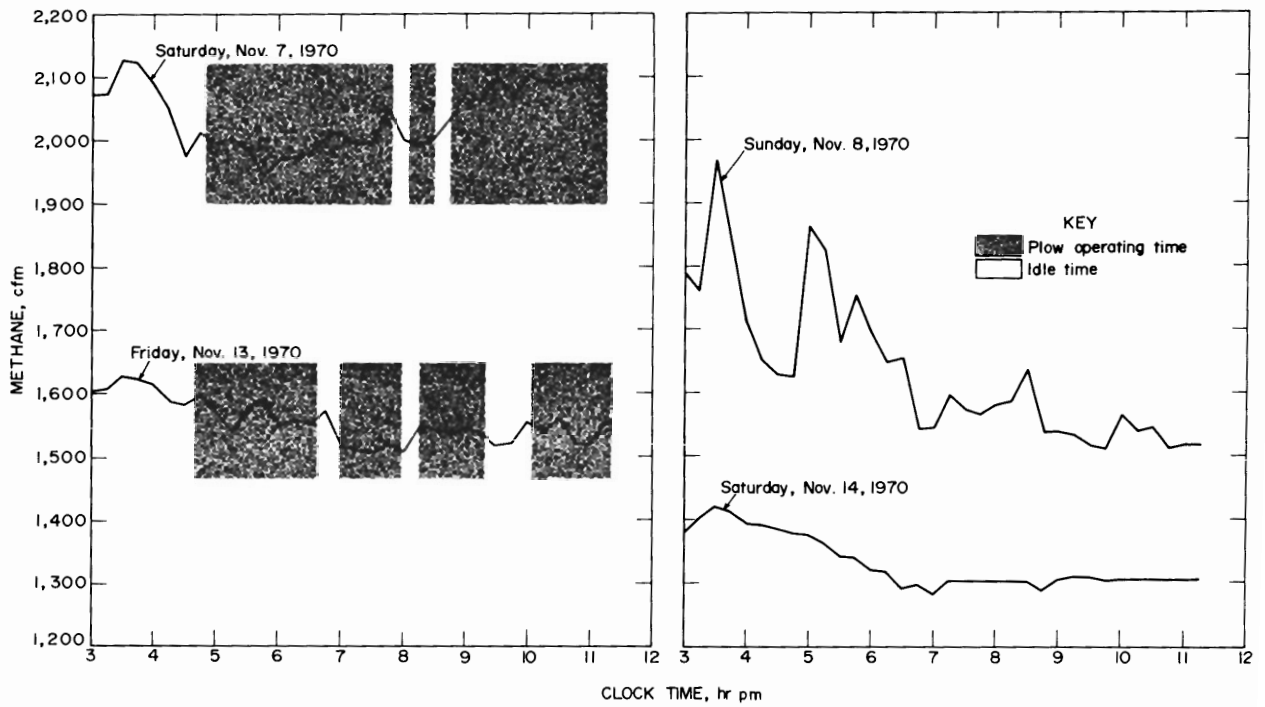


FIGURE 8. - Fluctuations in methane flow before and after borehole intersection during working and idle periods.

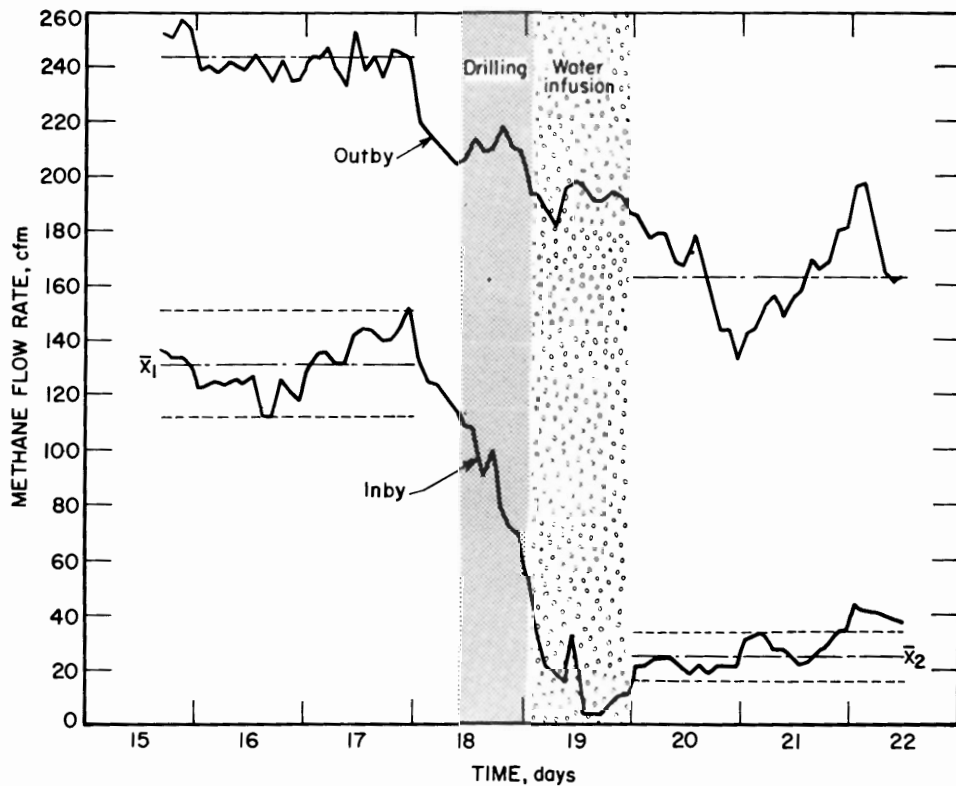


FIGURE 9. - Total methane flow rates from the study area.

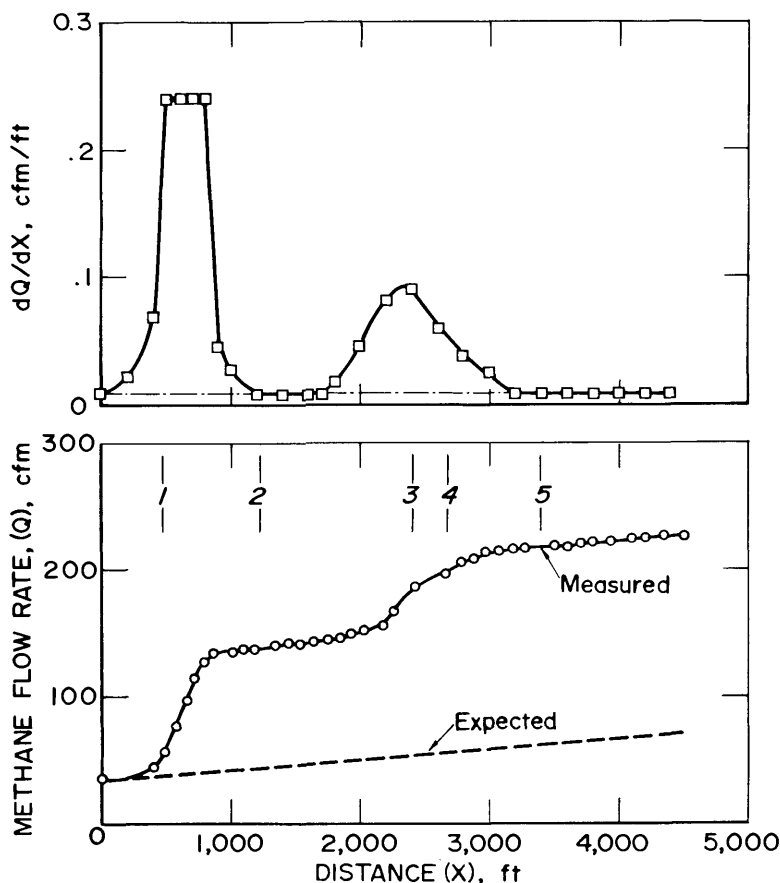


FIGURE 10. - Methane flow rates near abandoned wells.

where θ is the angle (displacement) measured clockwise from the face cleat located between holes 2 and 4. A plot of equation 12 is included in figure 13 which strikingly illustrates the effects of directional permeability on the flow; in the absence of wells, the maximum flow is from holes drilled along the butt cleats (B) and perpendicular to the face cleats (F). However, since boreholes that pass near wells may have flow rates above the predicted values, it is not surprising to find that the initial flow rates from holes 2 and 4 were approximately twice as high as those to be expected from equation 12. Furthermore, comparing the flows from holes 4 and 7 (two diametrically opposed holes), it can be seen that the flow from hole 4 which passes near a gas well was about 2.5 times higher than that from hole 7 which does not pass near a

well. This supports earlier findings of the Bureau on the effects of oil and gas wells on emission of methane in coal mines (34).

After the initial flow rates were measured, the seven bleeder holes were connected to a common manifold. The total methane emission from these holes fell for approximately 50 days and then began to increase (fig. 14). Interestingly, the emission rate fell only by a factor of 2 while the gas pressure in the coalbed fell by a factor of 20. However, during this period, water was removed continuously from the coal (the average initial water flow rate was 6.2 gpm per hole; in 50 days this figure fell to 0.5 gpm per hole), and the gas permeability of the coal increased. As a first approximation, the initial gas emission rates (GER) were found to be proportional to the inverse of the square root of the elapsed time, t (30):

$$\text{GER} = \frac{970,000 \text{ cfd}}{\sqrt{1 + t/10}}; 0 \leq t \leq 40 \text{ days.} \quad (13)$$

This simple equation yields values within about 5 percent of the smoothed (curve) data given in figure 14.

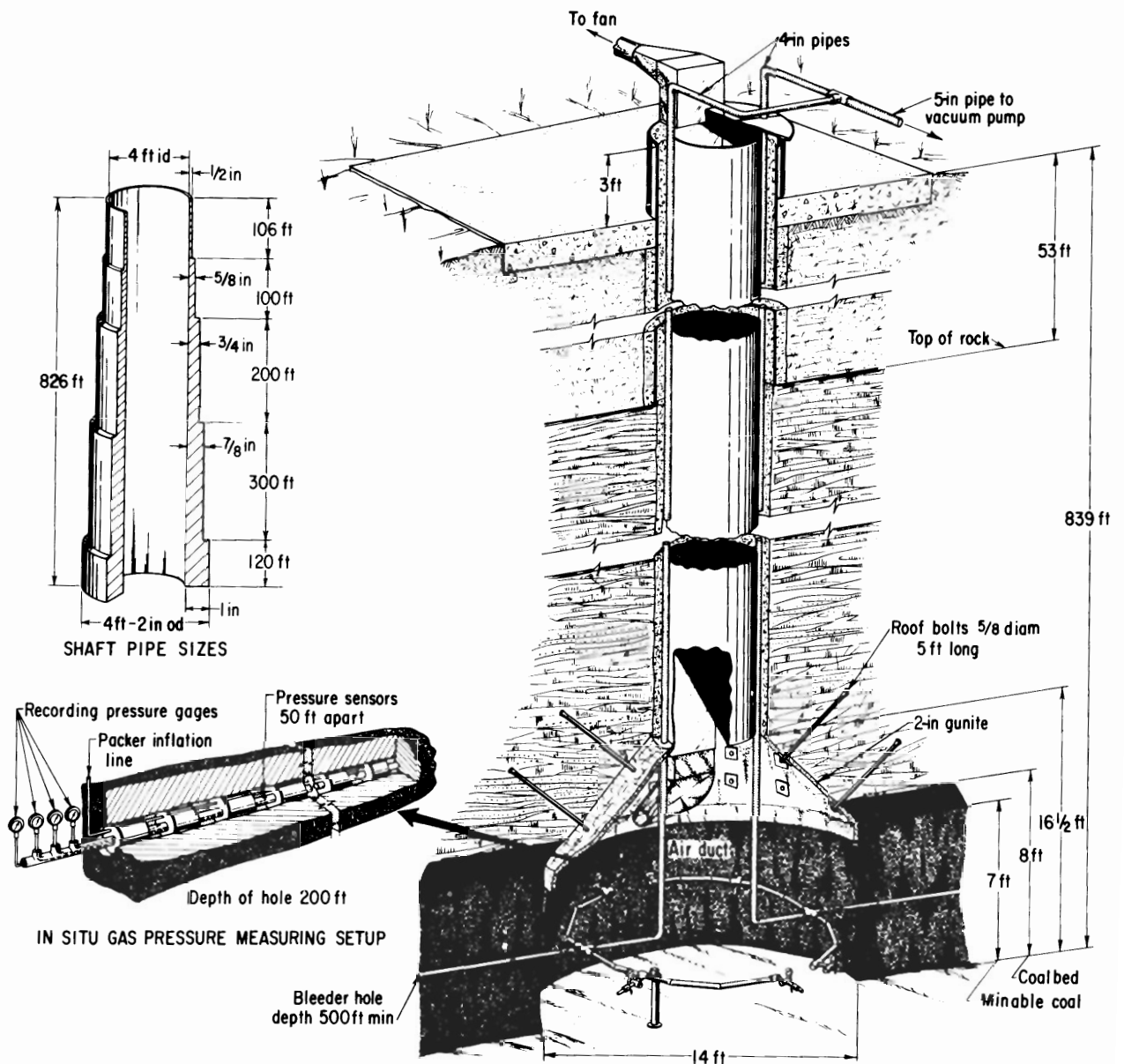


FIGURE 11. - Perspective view of multipurpose borehole.

In addition, the Bureau is presently evaluating the effects of clay veins and faults on methane flow. The results of these studies and of others on the drainage of gas from isolated panels and vertical and horizontal holes should be available soon. In a few years the results of a series of recently started studies should be available on the flow of gas from gob areas, the design of effective bleeders, and the improvement of ventilation at the face. However, even preliminary results show that technology is now available to permit a more realistic approach to methane control in most of the Nation's coal mines. Toward this end, a program was initiated late in 1972 to demonstrate that a complete mine can be effectively degasified with a concomitant improvement in safety and production. The initial results of this work also should be available within a few years.

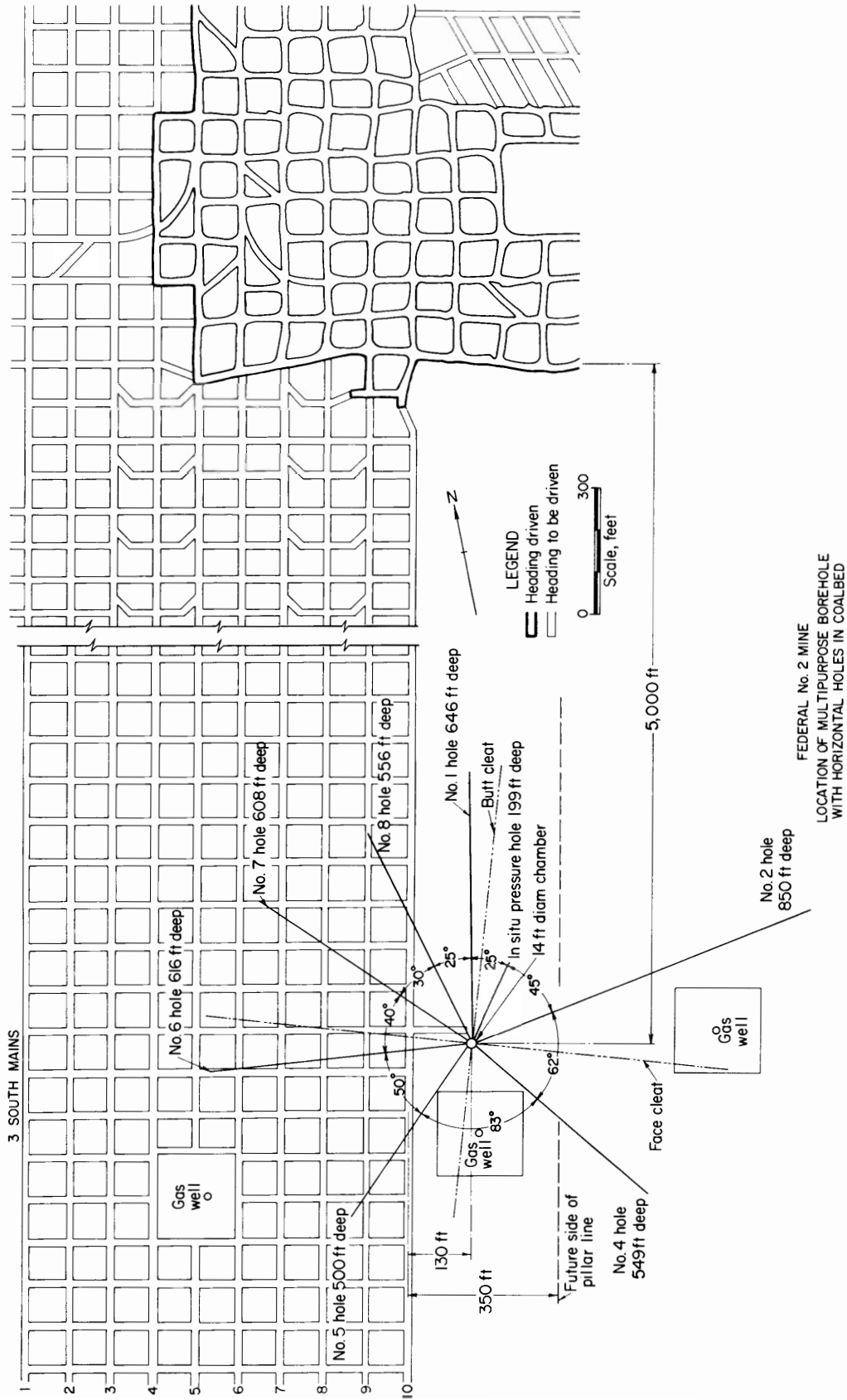


FIGURE 12. - Location of bleeder holes and gas wells.

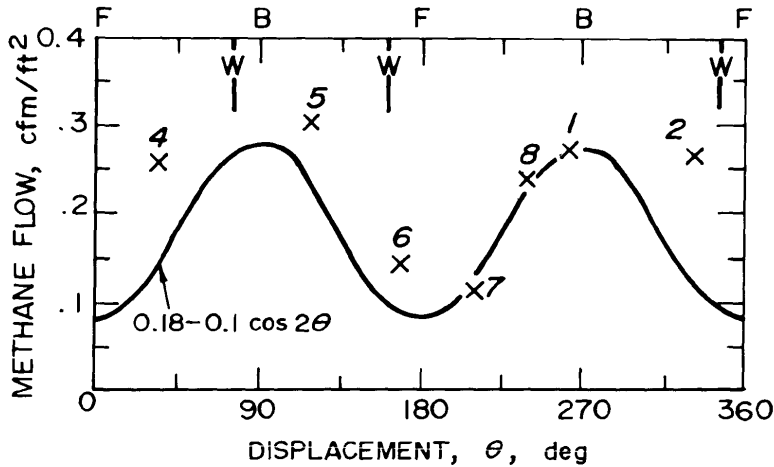


FIGURE 13. - Initial gas flow rates from horizontal holes drilled in virgin coal.

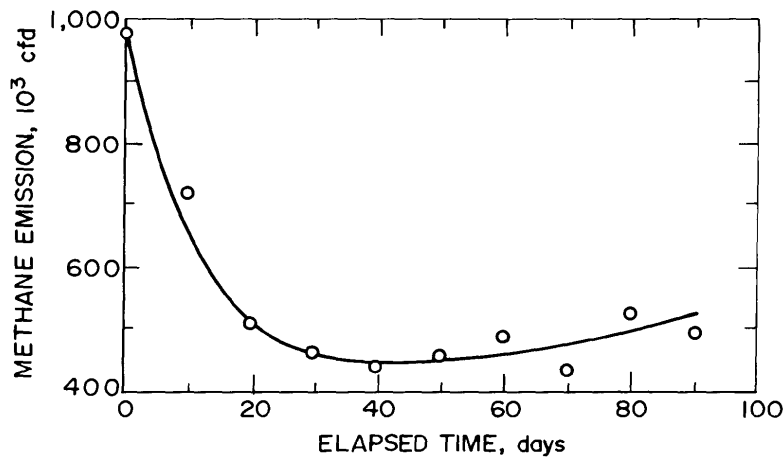


FIGURE 14. - Total flow (gas emission) from seven horizontal holes drilled in virgin coal.

Recommended Procedures

From the work conducted to date, it is strongly recommended that adequate methane control procedures be incorporated in all phases of underground mining, beginning with initial exploration and continuing until mining is completed. The problems that will be encountered in mining can be forecast by first measuring gas reservoir pressures and gas flow rates when boreholes are drilled during exploration and by determining the gas content of the coal (V_{ad}) using the direct method of Bertard (2). Briefly, this involves the taking of a coal core from a vertical hole and measuring the volume of gas that is desorbed at atmospheric pressure. Alternatively, V_{ad} can be calculated from the Langmuir isotherm (equation 8) if the pressure and temperature of the gas in the coalbed are measured. The approximate methane liberation rate (MLR) can then be obtained from equation 11.

Where V_{ad} and P are not known, an idea of the expected MLR can be obtained by combining equations such as those given here for the Pittsburgh coalbed. For example, from equations 10 and 11 we can obtain an approximate measure of the methane liberation rate for the Pittsburgh coalbed. This is plotted in figure 15 as $4 V_{ad}$ along with the MLR prediction curve derived earlier from the average methane emission rate data obtained in a number of mines in this coalbed. While there appears to be a fairly good correlation, it must be recognized that these procedures are based on averages and that the presence of gas and oil wells in the area of an active mine may tend to increase the averages, while barriers to free flow such as clay veins, faults, and nearby abandoned workings may decrease them.

Once V_{ad} or MLR is obtained, some thought must be given to the proper choice of control procedures. As a rule-of-thumb, we propose that where the

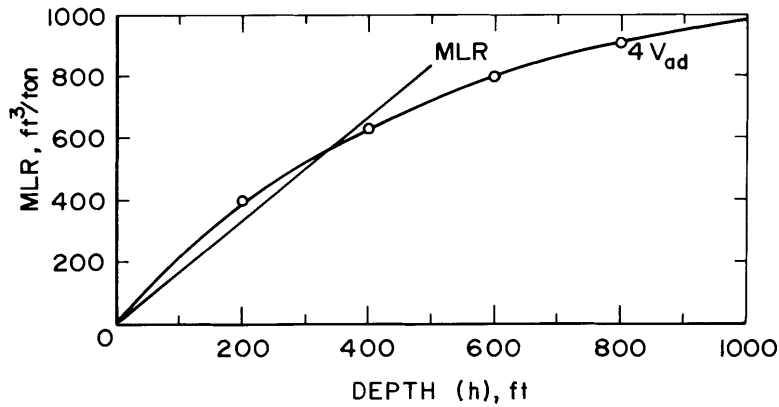


FIGURE 15. - Methane liberation rates and adsorbed gas volumes for mines in the Pittsburgh coal-bed versus depth (h).

projected methane liberation rate (MLR) approaches 400 cfm per ton of coal mined, ventilation alone will be inadequate and that direct methane control procedures should be utilized. These include the use of vertical and horizontal holes to dewater and degasify an area before the onset of mining; the use of horizontal holes to infuse an active face with water and thus divert the flow of methane; the use of long horizontal holes to dewater and degasify a section during mining;

improvement of bleeder entries; the use of vertical holes and slotted pipes to drain a gob area; the outlining of isolated panels to degasify the panels prior to mining; hydrofracing a block of coal to increase its permeability; and rapid dilution of methane with air as it enters the mine. Detailed procedures can be found in a number of references included in the bibliography.

CONCLUSIONS

While significant advances have been made in reducing the hazards associated with the mining of coal, the ignition and explosion hazards associated with the presence of methane and other combustibles have actually increased. From the data obtained in recent years, it can be expected that additional problems will arise as mining rates and depths are increased. Accordingly, special emphasis must be given to the use of established methane control procedures in all phases of underground mining and to the refinement of these techniques by a larger number of mining companies through application and innovation.

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APPENDIX

Assuming that a major mine explosion is a rare and random event unaffected by the occurrence of any other similar event, then the probability that k such explosions will occur in a given period (for example, 1 year) is given by the Poisson distribution function:

$$\Pr(k) = \frac{\mu^k e^{-\mu}}{k!}, \quad (\text{A-1})$$

where μ is the average number of such explosions that occur in a unit interval of time. For example, during the period 1836-75, the average number of major explosions (\bar{x}) was found to be 12/40 or 0.3 per year. If μ is estimated by \bar{x} then the expected frequency f_e with which 0, 1, 2 ... explosions should occur per year during the 40-year period can be computed. These expected frequencies are listed in table A-1 along with the observed frequency f_o . Similar data are given for the other periods of interest (1901-30; 1931-50; 1951-70). It should be noted that the unit interval represents the time during which observations are made in each case. This interval was taken as 1 year during the period 1836-75 so that f_e and f_o are the number of years in which 0, 1, or 2 or more explosions occurred; however, when the interval is 1 month, f_e and f_o are the number of months in which 0, 1, 2 ... explosions occurred. There appears to be a good correlation between f_o and f_e in most cases. This can be checked by computing χ^2 . For example, for the period 1836-75, we have:

$$\chi^2 = \sum_{i=0}^2 \frac{(|f_{o_i} - f_{e_i}| - 0.5)^2}{f_{e_i}} = 0.0407, \quad (\text{A-2})$$

where the second subscript (i) refers to the i th values in the sum (that is, f_{o_1} and f_{e_1} refer to the first observed and expected frequencies, respectively, and so forth). As the calculated χ^2 is less than the tabulated value ($\chi^2_{0.05,1} = 3.841$), there is no reason to reject the hypothesis that major mine explosions occurred in a random fashion during the period 1836-75 with an annual probability of occurrence of 0.3. Furthermore, as this is a Poisson distribution, the variance (σ^2) also should be approximately 0.3. Using this value, approximate 95-percent confidence limits for μ were computed by use of the expression $\bar{x} \pm 2\sqrt{\bar{x}/n}$; these limits are included in table 1 of the main text (for example, 0.3 ± 0.17). Similar data are given for the other periods considered, and the averages and measures of scatter are included in figure 1 of the main text. From these it can be seen that the averages differ markedly from neighboring values at the 0.05 significance level in each case.

TABLE A-1. - Observed and expected occurrence of major coal mine explosions in the United States during the period 1836-1970

| Period (interval) | μ | Explosions, k, | Pr(k) | f_e | f_o |
|------------------------|-------|-----------------|--------------------|-------|---------------------|
| | | (unit interval) | | | |
| 1836-75 (1 year)..... | 0.30 | 0 | 0.7408 | 29.6 | 29 |
| | | 1 | .2222 | 8.9 | 10 |
| | | 2 or more | .0369 | 1.5 | 1 |
| | | | | | $\chi^2 = 0.0407^1$ |
| 1901-30 (1 month)..... | .8275 | 0 | .4371 | 157.4 | 163 |
| | | 1 | .3617 | 130.2 | 131 |
| | | 2 | .1497 | 53.9 | 42 |
| | | 3 or more | .0515 | 18.5 | 24 |
| | | | $\chi^2 = 4.465^2$ | | |
| 1931-50 (1 month)..... | .3242 | 0 | .7231 | 173.5 | 180 |
| | | 1 | .2344 | 56.3 | 49 |
| | | 2 | .0380 | 9.1 | 8 |
| | | 3 or more | .0045 | 1.1 | 3 |
| | | | $\chi^2 = 4.605^2$ | | |
| 1951-70 (1 month)..... | .1083 | 0 | .8974 | 215.4 | 216 |
| | | 1 | .0972 | 23.3 | 22 |
| | | 2 or more | .0054 | 1.3 | 2 |
| | | | | | $\chi^2 = .0583^1$ |

¹ $\chi^2_{.05, 1} = 3.841$.

² $\chi^2_{.05, 2} = 5.991$.

Alternatively, we can consider the time intervals between major explosions rather than their rate of occurrence (16). During any period in which the explosion rate is constant, the probability density function of time interval between major explosions should be given by the expression:

$$f(t) = E \exp(-Et), \quad (\text{A-3})$$

where E is the expectation of a major explosion in a given time interval (for example, 1 month). For example, if the 74 major explosions that occurred in United States coal mines during the period 1931-50 are considered (9, pp. 40-41, 168), a list of 73 time intervals that elapsed between explosions can be generated (table A-2). From these time intervals a histogram can be constructed (fig. A-1). Then, taking E to be $1/\bar{t}$, we can write:

$$f(t) = 0.36 \exp(-0.36 t), \quad (\text{A-4})$$

where t is the time in months (actually, 30-day intervals). This curve is included in figure A-1. As expected, the exponential equation follows the trend of the histogram fairly well. Again, the χ^2 distribution can be used to evaluate the goodness of fit. This same (χ^2) distribution can be used to establish a confidence interval for E or $1/E$. For example (16):

$$\frac{2 \cdot n\bar{t}}{\chi_{1-\alpha}^2} \geq \frac{1}{E} \geq \frac{2 \cdot n\bar{t}}{\chi_{\alpha}^2}, \quad (\text{A-5})$$

so that in this case, with 90 percent probability, $1/E$ lies between 2.31 and 3.40 ($\bar{t} = 2.771$ months). By computing this interval for two successive periods, we can determine if a statistically significant change has taken place in the probability of having a major explosion in a given time interval.

TABLE A-2. - Time intervals, t (in days), between successive major explosions in United States coal mines during the period 1931-50¹

| No. | t | No. | t | No. | t | No. | t | No. | t | No. | t | No. | t | No. | t |
|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|
| 1 | - | 11 | 2 | 21 | 112 | 30 | 36 | 39 | 120 | 48 | 6 | 57 | 43 | 66 | 272 |
| 2 | 3 | 12 | 15 | 22 | 17 | 31 | 42 | 40 | 13 | 49 | 52 | 58 | 139 | 67 | 89 |
| 3 | 22 | 13 | 260 | 23 | 109 | 32 | 180 | 41 | 26 | 50 | 144 | 59 | 126 | 68 | 16 |
| 4 | 121 | 14 | 329 | 24 | 92 | 33 | 66 | 42 | 10 | 51 | 89 | 60 | 173 | 69 | 20 |
| 5 | 158 | 15 | 169 | 25 | 11 | 34 | 120 | 43 | 109 | 52 | 67 | 61 | 56 | 70 | 85 |
| 6 | 55 | 16 | 177 | 26 | 78 | 35 | 43 | 44 | 61 | 53 | 6 | 62 | 56 | 71 | 140 |
| 7 | 21 | 17 | 187 | 27 | 30 | 36 | 94 | 45 | 30 | 54 | 109 | 63 | 231 | 72 | 59 |
| 8 | 40 | 18 | 216 | 28 | 70 | 37 | 18 | 46 | 104 | 55 | 19 | 64 | 20 | 73 | 170 |
| 9 | 107 | 19 | 9 | 29 | 5 | 38 | 36 | 47 | 1 | 56 | 8 | 65 | 93 | 74 | 3 |
| 10 | 177 | 20 | 78 | | | | | | | | | | | | |

¹Average interval (\bar{t}): 83.15 days (2.771 months) between explosions.

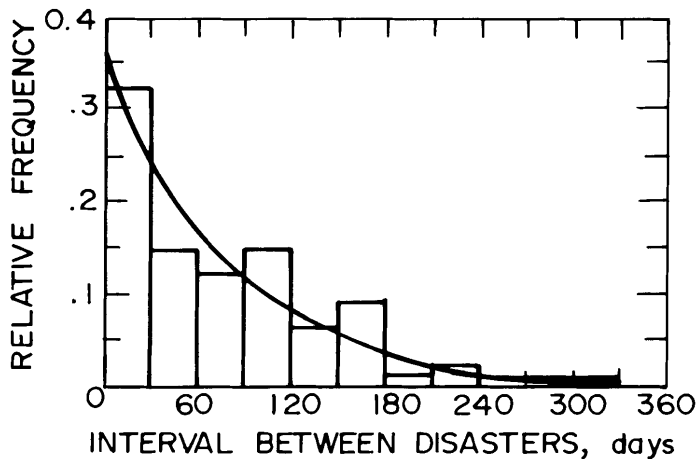


FIGURE A-1. - Histogram of time intervals between successive major explosion disasters in United States coal mines for the period 1931-50.