MAJOR MINING RESEARCH PROGRAMS CONDUCTED BY BITUMINOUS COAL RESEARCH, INC., FOR THE UNITED STATES BUREAU OF MINES. R. D. Saltsman, Bituminous Coal Research, Inc., 350 Hochberg Road, Monroeville, Pa. 15146; Joseph Grumer, U.S. Bureau of Mines, 4800 Forbes Ave., Pittsburgh, Pa. 15213; Kelly Strebig, U.S. Bureau of Mines, Twin Cities, Minnesota 55111.

Enough rock dusting prevents a coal dust explosion from propogating by absorbing heat from the otherwise possible flame. Theoretical calculations are presented which compare adiabatic flame temperatures for coal and coal plus rock dust with adiabatic flame temperatures of gas flames at their lean limits of flammability; these calculations show that concentrations of rock dust in mine dust determined empirically to be necessary to inert coal dust are reasonable. Based on this, United States mines are equipped to maintain certain incombustible levels in the settled dust, created during the mining process, by applying rock dust to the surfaces created when the coal is removed. Monitoring and enforcing these requirements are time-consuming and expensive. Results of a data gathering and statistical study to find methods and procedures for reducing the number and quantity of samples required are described. The present respirable dust standards that American coal producers have to meet require new developments. Optimizing the conventional use of air and water to control the respirable dust will not be sufficient. One promising method involves the flushing of the cutting bits with water while the bits are cutting coal. The water flushing concept, pioneered in England for longwall machines, has been adapted to American continuous miners. The problems encountered and the results obtained to date, will be described.

APPLICATION OF ENGINEERING FUNDAMENTALS TO EVALUATION OF DUST COLLECTION DEVICES

George T. Preston

Garrett Research and Development Co., Inc.
La Verne, California 91750

One consequence of the high degree of mechanization of modern coal mining techniques is the production of a large amount of coal dust during the process of bringing the coal from seam to surface. Wide-spread use of the continuous mining machine has contributed substantially to coal production figures, but it has also added to the level of coal dust in the air the miner breathes. Dust control techniques for the area around a continuous mining machine are relatively ineffective up to the present, compared to other dust sources in the mining operation.

One of the possible approaches to this problem is to collect the dust at or near the mining face as it is generated. In late 1969 a one year contract was awarded by the U.S. Bureau of Mines to Garrett Research and Development Company to evaluate those types of commercially available dust collection equipment which might be applicable to this purpose.

In retrospect, the work done in performance of this contract provides a clear example of how engineering fundamentals can be applied toward solving problems of industrial significance. Two such problems which arose during the planning of this work were: first, what is respirable dust, and what efficiency would be required of a collection device in the underground mining environment? -- and second, how could we evaluate representative dust collection devices with only a small amount of time-consuming experimental work, and yet obtain results which would be general enough to apply to devices and operating conditions other than those specifically investigated?

Details of this work such as experimental procedures are discussed in earlier papers (1,2) and in the final contract report (3).

RESPIRABLE DUST

Mine dust control regulations currently in effect are based on allowable loadings of respirable dust in the mine atmosphere. From the legal point of view, respirable dust is that part of the total dust which is collected by certain portable sampling devices after the dust passes through sections of those devices intended to simulate the dust-collecting abilities of the nose and throat. In order to specify the performance required of a dust collector in a coal mine entry, it is necessary to do the following:

1. Define the size distribution of the "total dust," i.e., the dust generated by a continuous mining machine at a working face.

2. Compute the size distribution of respirable dust, i.e., the part of the total dust which would pass the pre-classifier ("nose and throat") of a portable sampler.

3. From an estimate of the respirable dust loading in a mine, determine the respirable dust collection efficiency required to reduce the respirable dust loading to a legal level.

Three sources of data on dust from continuous coal mining operations were used: the U.S. Bureau of Mines (4), publications of E.J. Baier (5), and mine samples taken as part of this work. With the assumption that the total dust size distribution was logarithmic-normal, and after resolving some inconsistencies in the available data, it was concluded that a conservative (i.e. small) estimate of the size distribution of the total dust near the working face is a mass mean diameter of 15 microns with a standard deviation of 3.0. The size distribution of the respirable dust fraction was then calculated by combining the total dust size distribution with collection efficiency data for the respirable dust sampler. The resulting size distribution for the respirable dust fraction based on the AEC sampler collection efficiency data is a mass mean diameter of 2.55 microns with a standard deviation of 1.7.

The loading of respirable dust in the mine atmosphere near a continuous mining machine has been estimated at 5 to 10 mg/m^3, based on the AEC personal sampler. The current maximum allowable respirable dust loading to which a miner may be exposed is 2 mg/m^3. However, this limit is based on measurement with the MRE portable sampler which, because its pre-classifier operates on a different principle, measures dust loadings differently from the AEC sampler (6,7). An approximate relationship between the dust loadings measured by the two samplers is

$$\text{MRE} = 1.63\ \text{AEC} + 0.67 \qquad 1)$$

Thus, the legal limit of 2 mg/m^3 (MRE) corresponds to an AEC-measured loading of 0.81 mg/m^3. Accepting a conservative (high) 10 mg/m^3 estimate for the loading of respirable dust from the continuous miner, the collection efficiency of a device must be

$$\frac{10 - 0.8}{10} = 92\%$$

on respirable dust in order to achieve compliance.

The problem now is, given experimental measurements on the performance characteristics of a dust collection device, how can this information be related to that collector's efficiency on respirable dust? Obviously it is impractical to attempt to use sample dust with the exact size distribution of respirable dust for performance tests on collection equipment. The overall efficiencies measured, therefore, will pertain to the size distribution of the sample dust, rather than respirable dust. The solution is to use a testing procedure which determines the collection efficiency as a function of particle size, so that the primary result of each test is a <u>penetration function</u> $P(D_p)$, the fraction of particles of diameter D_p which penetrates (is not collected by) the collector. (Efficiency and penetration are related by $E \equiv 1 - P$.)

The size distribution of a dust is defined by $f(D_p)$, where $f(D_p)dD_p$ is the mass fraction of particles having a diameter in the range D_p to $D_p + dD_p$, and

$$\int_0^\infty f(D_p)dD_p \equiv 1 \qquad 2)$$

The gross penetration for any dust through any collection device can be calculated from

$$P = \int_0^\infty P(D_p)f(D_p)dD_p \qquad 3)$$

where $P(D_p)$ is the penetration function of the collection device and $f(D_p)$ is the size distribution frequency of the dust. Thus, if the penetration function can be measured or calculated for a given collection device, it is possible to calculate the gross penetration of any dust of known size distribution.

A numerical integration computer program was written to perform the integration of Equation 3, so that the effect of any penetration function and size distribution on gross penetration could rapidly be calculated.

It was implied earlier that the size distribution and loading of respirable dust in a coal mine are not well known. This means Equation 3 is a particularly valuable tool, because it separates the effects of the dust size distribution and the penetration function of the collection device. Hence, experimental results expressed in the form of a penetration function for a given device have complete generality and are applicable to any dust whose size distribution is known.

DUST COLLECTION MECHANISMS

Several basic mechanisms can be used for particle collection, including gravity, inertia, diffusion, electrostatic attraction, fabric filtration, radiation, magnetism, and agglomeration. Most of these are either impractical or dangerous for use in the control of coal dust underground (3). Only inertia and fabric filtration appear to be technically feasible mechanisms in an underground mine, and because of space limitations, inertia is the more promising. Accordingly, this work was limited to investigation of collection devices operating by one or more inertial mechanisms.

Since the intent of the program was not simply to test several commercial dust collection devices but rather to determine what collection mechanisms could best be applied to the problem of coal mine dust, the strategy adopted was to select or develop a mathematical model for each potentially applicable mechanism and to test collection devices which used these mechanisms in order to confirm or disprove the models. With the ultimate goal of evaluating each inertial mechanism for application to coal dust, the immediate goal of the modelling and testing program was to determine for each inertial mechanism the penetration function $P(D_p)$ under various operating conditions. Once this was done, the penetration function could be applied using the technique described in the previous section.

Cyclone

In a cyclone separator, rotary motion of the entire gas stream throws dust particles to the outer wall under the influence of centrifugal force. The particles then fall through the bottom of the cyclone or are otherwise removed. The cyclone separator tested in this work was a multiple cyclone collector consisting of a bank of 46 small cylindrical cyclones in parallel. The dusty air inlet flow was parallel to the axis of the cyclone, and a tangential motion was imparted to the air by fixed vanes set at an angle to the axis. Each cyclone had a hub occupying the central part of its volume. This device was modelled by adapting equations given in Strauss (8) for more conventional cyclones. In this model it was assumed that if a particle reaches the outer wall at any time before leaving the cyclone with the gas, it is collected. No consideration was given to re-entrainment by the gas. The resulting penetration function is given by

$$P(D_p) = 1 - \frac{1 - \sqrt{1-(D_p/C)^2}}{1 - \left(D_{hub}/D_{shell}\right)^2} \qquad 4)$$

where C is a constant characteristic of the cyclone geometry and the air and dust properties. The observed dust penetration through the multiple cyclone as a function of particle size is shown in Figure 1. The curve was calculated from Equation 4. (The value of C in Equation 4 was calculated from the model, not fitted empirically.) The observed gross penetration of the test dust was 76%, and the gross penetration calculated using Equation 4 with Equation 3 was 79%.

It was concluded that within the precision of the experimental techniques used, the penetration function of Equation 4 is an adequate representation of the penetration function of the multiple cyclone which was tested. Equation 4 was then used again as the penetration function in Equation 3 to predict that the gross penetration of respirable dust through this device would be 53% at the test conditions. This is to be compared with the estimated 92% efficiency (8% gross penetration) required to achieve a legal respirable dust loading.

Curved Passages

Many dust collectors use essentially the same principle as the cyclone separator, but the gas is caused to turn by curved passages of some sort. This type of collector includes those with louvers, deflectors, or corrugated passages, and packed beds. In these devices, dust particles are spun out against solid or liquid surfaces by centrifugal force as the gas stream flows through a multitude of curved paths.

A simple packed bed scrubber was fabricated in-house to represent this collection mechanism. For a packed bed, the penetration function can be expressed (9) as

$$P(D_p) = \exp\left[-C \frac{Z}{D_t} K\right] \qquad 5)$$

MULTIPLE CYCLONE
PENETRATION AS FUNCTION OF PARTICLE DIAMETER

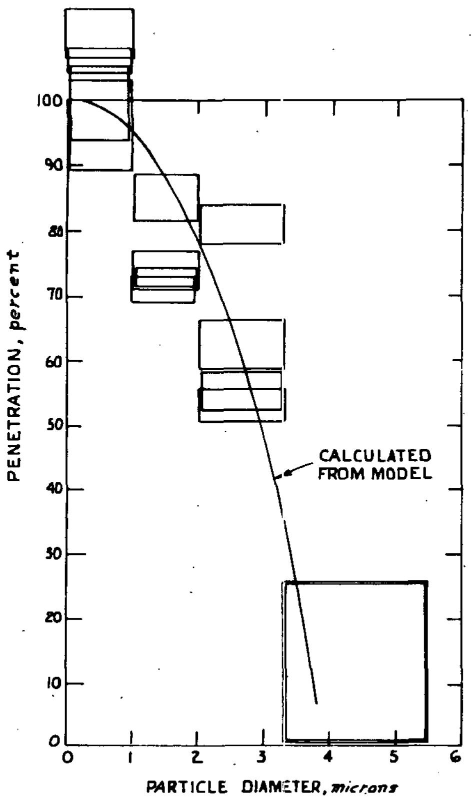


FIGURE 1

PACKED BED SCRUBBER
PENETRATION AS FUNCTION OF PARTICLE DIAMETER

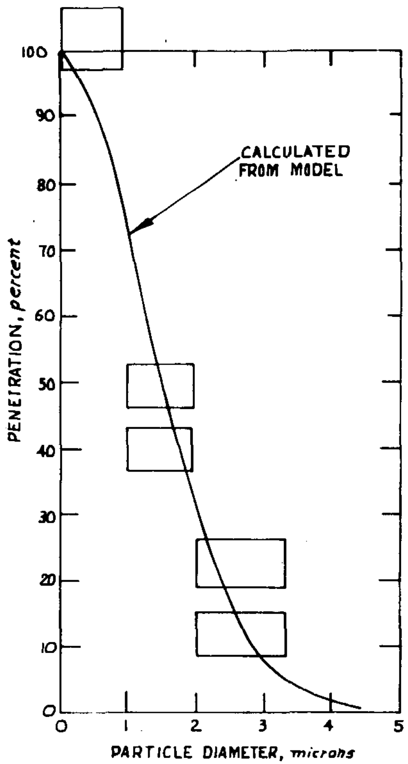


FIGURE 2

where K is the inertial impaction parameter, $U\rho D_p^2/9\mu D_t$, and C is an empirical constant (9) which depends on the packing geometry. The test results for the packed bed scrubber are shown in Figure 2. The curve was calculated from Equation 5 using a value of C = 12 for 1-1/2 inch Pall rings. The observed gross penetration of the test dust was 39%, and the gross penetration calculated using Equation 5 with Equation 3 was 41%.

Impaction Targets

Cylindrical objects such as rods, wires, and fibers are used in a large number of collection devices. As the gas stream flows around the target, the inertia of the dust particle tends to make it impact on the target instead of passing around it. The factors influencing the collection efficiency of such a device are the gas velocity relative to the impaction target, the size of the target, the gas viscosity, the number of targets the gas stream must pass, and the particle density and size. All of these effects except the number of targets are included in the inertial impaction parameter K. An empirical penetration function based on past experimental results for impaction on cylinders and on spheres is

$$P(D_p) = 1 - \left(\frac{K}{K+0.7}\right)^2 \qquad 6)$$

(Impaction on spheres is one of the mechanisms involved in a venturi scrubber, discussed below.)

A wetted screen device was tested which employed the impaction target mechanism. An accordion pleated screen constituted the impaction targets, and the screen was continuously wetted to keep the collected particles from blinding it. The screen was followed by a horizontal cyclone functioning as an entrainment separator. The test results are shown in Figure 3. The higher of the two penetration curves was calculated from Equation 6 directly. However, since the total area of the screen wires was about twice the cross sectional area of the scrubber, it was hypothesized that the scrubber might comprise two impaction stages. The penetration

WETTED SCREEN SCRUBBER
PENETRATION AS FUNCTION OF PARTICLE DIAMETER

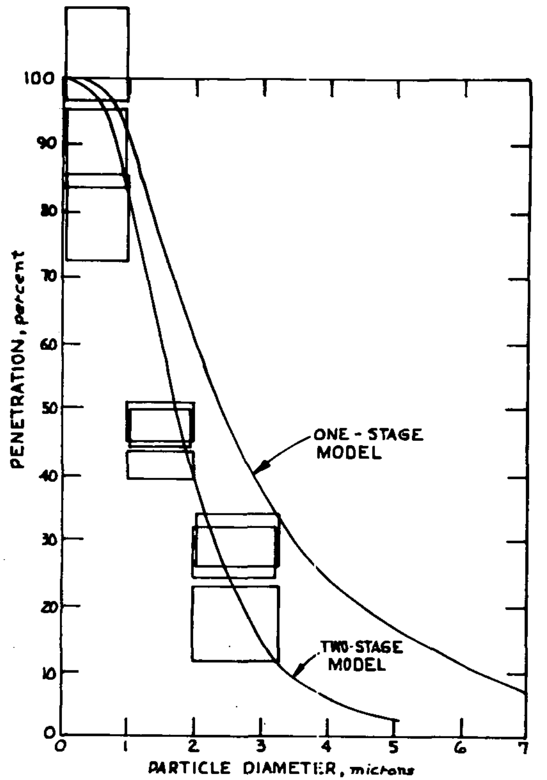


FIGURE 3

PARTICLE COLLECTION EFFICIENCY
VS IMPACTION PARAMETER
FOR IMPINGEMENT OF JETS ON
LIQUID OR SOLID SURFACES

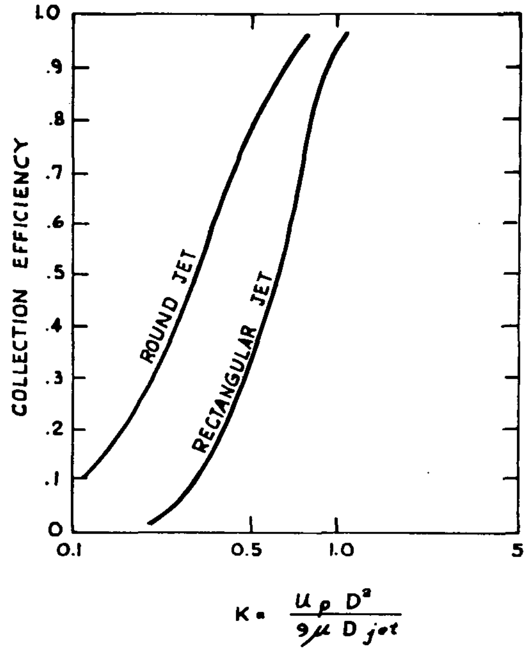


FIGURE 4

function indicated by the lower curve is just the square of the penetration calculated from Equation 6. The observed gross penetration of the test dust through the wetted screen was 54%, and the gross penetration calculated from the two-stage penetration function and Equation 3 was 56%. The same two-stage penetration function applied to the estimated size distribution for respirable dust using Equation 3 gave a gross penetration of 35%.

Jet Impingement

Some dust collection devices are based on impingement of a jet of the gas stream on a solid or liquid surface, a mechanism quite similar to impaction on targets. In this case, however, the space through which the gas stream must pass is much smaller, and the gas flow characteristics are thus different from the case where the flow is around relatively isolated bodies. The common examples of the impingement mechanism are sieve tray and ballast tray scrubbers. When the gas jet impinges on a surface, the inertia of the particles prevents them from following the sharp change of direction taken by the gas. Again, the collection efficiency is affected mainly by the variables included in K, the inertial impaction parameter. Past experimental data are correlated in Figure 4, which gives the penetration function indirectly by showing the efficiency as a function of K, for impingement of round and rectangular jets on surfaces.

The collection device tested to represent the jet impingement mechanism was an impingement scrubber consisting of a cylindrical tower containing water spray nozzles and baffle plates. The test results are shown in Figure 5. The penetration function curve shown was calculated by assuming the collection mechanism was two stages of impingement of round air jets on flat plates, with each stage having a penetration function obtained from Figure 4. Using this penetration function with Equation 3 and the estimated size distribution, a gross penetration of 27% was predicted for respirable dust through this device.

IMPINGEMENT SCRUBBER
PENETRATION AS FUNCTION OF PARTICLE DIAMETER

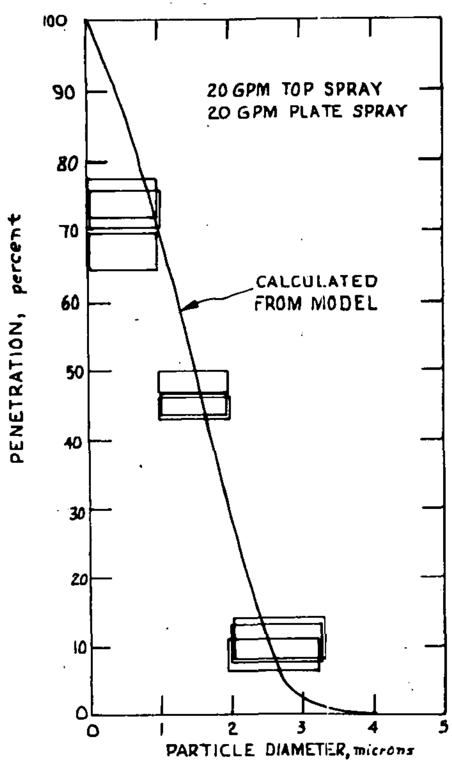


FIGURE 5

VENTURI SCRUBBER
PENETRATION AS FUNCTION OF PARTICLE DIAMETER

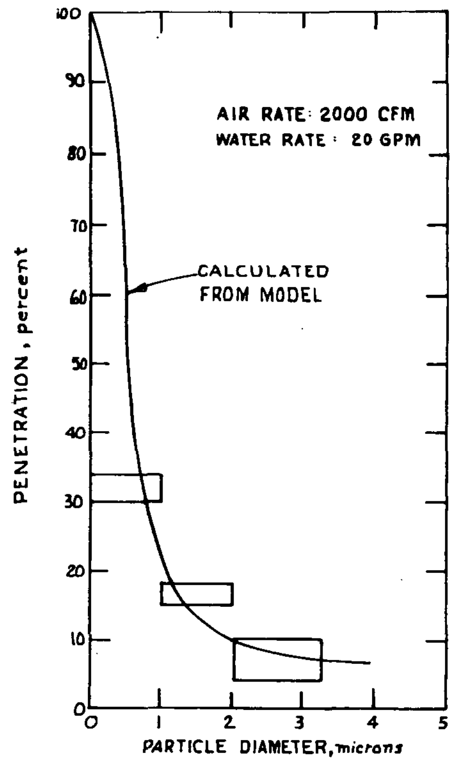


FIGURE 6

Venturi

In venturi devices, the gas stream is forced through an orifice or a narrow throat. A scrubbing liquid is introduced at or upstream of the throat, and is atomized at the throat due to the high gas velocity. The dust particles are collected by impaction on the atomized droplets. The penetration function for the venturi mechanism is given (9) as

$$P(D_p) = \exp\left[-2 \times 10^{-5}(1 - P_s)(13{,}500\ L + 1.2\ UL^{2.5}\right] \quad 7)$$

for the air-water system at room temperature and atmospheric pressure. P_s is the penetration calculated from Equation 6 for impaction on spheres. The spheres which are the impaction targets are the droplets atomized in the venturi throat. Their diameter (for calculating the inertial impaction parameter K) is estimated from the Nukiyama-Tanasawa correlation, which can be written as

$$D_t = \frac{16500}{U} + 1.45\ L^{1.5} \quad 8)$$

for air and water at room temperature and atmospheric pressure. Equation 8 is applicable when the air velocity is greater than 200 ft/sec.

In the experimental program, a simple venturi collector was tested, as well as two commercial devices operating primarily by the venturi mechanism. Partial test results for the simple venturi are shown in Figure 6. The observed gross penetration of the test dust was 19%, and the gross penetration calculated by using the penetration function given by Equations 7 and 8, in Equation 3 was 19%. The same penetration function applied to the estimated respirable dust size distribution gave a gross penetration of 6%. The test results for the two commercial venturi scrubbers also confirmed the venturi mechanism model.

DISCUSSION

A major conclusion from the contract program was that currently available knowledge of dust collection mechanisms is adequate both to explain the performance of a wide variety of dust collection devices and to provide a basis for the design of dust collection systems.

Potential dust collection mechanisms cannot fairly be evaluated only on the basis of the test results reported, because the tests on the various collectors were not always run at comparable conditions. Indeed, little in the way of useful conclusions could have been achieved if the program had been confined to testing various devices, even if the testing had been more extensive. But the purpose of the tests was to confirm the available models, and based on the application of those models, some general conclusions can be offered about the applicability of the different collection mechanisms to the coal mine dust problem.

1. A dry centrifugal collector cannot perform adequately for this application.

2. None of the "wet dynamic" mechanisms -- curved passages, impaction targets, jet impingement -- could meet current standards based on the respirable dust loading (10 mg/m^3) assumed in this work. However, if the respirable dust loading could be lowered, for instance by redesigning the air system in the mine entry, these mechanisms would be worth reconsidering, particularly in view of their relatively low energy and water requirements and potentially small size of the collection device.

3. The required collection efficiency could be achieved with a high energy scrubber using a venturi mechanism. The disadvantages of such a device are high pressure drop (i.e. high energy requirement) and a large water requirement. Also, there is at present no commercially available high energy scrubber which could fit in a coal mine entry.

NOMENCLATURE

(Any consistent units may be used, except in cases noted.)

D_{hub}	diameter of cyclone hub
D_p	particle diameter
D_{shell}	inside diameter of cyclone
D_t	target size: sphere - diameter (microns in Equation 8) cylinder - diameter round jet - diameter rectangular jet - width packing - nominal size
f	size distribution frequency
K	$\equiv \dfrac{\rho U D_p^2}{9 \mu D_t}$, inertial impaction parameter
L	liquid rate, gallons/thousand cubic feet of air
P	penetration
U	air velocity (ft/sec in Equations 7 and 8)
Z	packed height
μ	gas viscosity
ρ	particle density

REFERENCES

1. Legatski, L.K. and Clavert, S., "Scrubbers for Control of Coal Mining Dust," presented at A.I.Ch.E. meeting, Chicago, December, 1970.

2. Preston, G.T., "Dust Control in Underground Mines," Proceedings of the Conference on the Underground Mining Environment, University of Missouri at Rolla, October, 1971.

3. Calvert, S. and Legatski, L.K., "A Comprehensive State-of-the-Art Evaluation for All Types of Dust Collection Equipment That May Be Applicable in Underground Coal Mines," National Technical Information Service, U.S. Department of Commerce, Document No. PB 197 739, $3.00.

4. U.S. Bureau of Mines, private communication.

5. Baier, E.J. and Diakun, R., J.Am.Industr.Hygiene Assoc. 25, 476 (1964).

6. "Sampling and Evaluation of Respirable Coal Mine Dust," Mineral Industry Health Program Training Course 100.

7. "Proceedings of the Symposium on Respirable Coal Mine Dust," USBM Information Circular 8458, 1970.

8. Strauss, W., "Industrial Gas Cleaning," Pergamon Press, New York, 1966.

9. Calvert, S., in "Air Pollution," ed. A.C. Stern, Academic Press, New York, 1968, vol. 3, pp. 457-496.

THE U.S. BUREAU OF MINES PROGRAM TO CONTROL RESPIRABLE DUST IN COAL MINES

Welby G. Courtney

Dust Control and Life Support Group
Pittsburgh Mining and Safety Research Center
Bureau of Mines, U.S. Department of the Interior
Pittsburgh, Pa.

INTRODUCTION

Inhalation of dust has long been recognized as a health hazard leading to pulmonary diseases. In the 16th century, Agricola, the father of mining, discussed dust inhalation during mining and called it a "widow maker." In the 19th century the "black lungs" of autopsied coal miners were recorded. The 11th Edition of Encyclopedia Britannica, some 60 years ago, described fibroid changes and included a graphic photomicrograph of a cross section of a coal miner's lung.

The Federal Coal Mine Health and Safety Act of 1969 (PL 91-178, December 30, 1969) established health standards in underground coal mines for the first time in the United States. The Act specifically stated that the working conditions in each underground coal mine should be sufficiently free of respirable dust to allow each miner to work underground without incurring any disability from coal workers' pneumoconiosis (CWP) or other occupation-related disease during his working life. Based largely upon British studies, the Act specified that the maximum allowable concentration of airborne respirable dust particles (nominally less than 7.1 μm in diameter) in bituminous coal mines should be 3 mg/m^3 by June 30, 1971 and 2 mg/m^3 by December 30, 1972. The maximum permitted dust level is decreased if the silica content of the dust is greater than 5 weight percent so that the Threshold Limit Value (TLV) of 100 $\mu g/m^3$ of silica for an eight hour exposure will not be exceeded.

CWP is the most severe health and safety problem facing the coal mining industry today.

1. The number of permanent disabilities and deaths of coal miners due to CWP is 3.5 times the disabilities and deaths due to all other mine accidents (1964-1970 Pennsylvania data). (1)

2. About 30 percent of the present working miners have CWP (1972).

3. About 165,000 claims for CWP have been approved and some 100,000 additional claims are currently under review.

The cost to Government and industry for CWP compensation has been estimated to be $1 billion per year.

However, it has been estimated that coal production must double by the year 2000 in order to meet the energy requirements of the nation. The necessity for society to devise methods for mining coal at a rapid and economical rate while maintaining a healthful mining environment is apparent.

The 1969 Act directed that the Departments of the Interior and Health, Education and Welfare shall "develop new or improved means and methods of reducing the concentrations of respirable dust in the mine atmospheres of active workings of the coal mines." The Bureau of Mines was assigned the responsibility for planning and implementing a research and development (R&D) program to provide the advanced technology for reducing the amount of respirable dust in coal mines. This advanced technology would assist the mine operator to meet the stringent dust limits imposed by the Act.

The present problem of dust control during coal mining operations is considerably different from the problems normally encountered during industrial operations. For example, coal is often mined underground using continuous (mechanical) mining methods,* where coal is removed by cutting bits on a rotating wheel or chain. Dust is formed by the cutting action of the bits, and also at nearby locations by secondary handling operations. Water sprays are used to suppress the dust being formed by the cutting action. Also, air is passed over the machine to the face and then back behind a brattice cloth (usually) to "push" the dust away from nearby personnel. However, dust control during actual coal mining operations is not so clear-cut. An analogy to a typical industrial operation would be to devise ways to control a hazardous impurity in a chemical processing plant where:

1. the sources of the impurity (dust) are imperfectly known;

2. laboratory and pilot plant information is of limited usefulness (it is almost impossible to simulate the underground mining situation in the laboratory);

3. the amounts of the impurity are variable (cutting bits become broken, cutting rate varies with machine operator);

4. the flows in the plant are ill-defined and highly variable, e.g., a pipe (spray nozzle) plugs, the flow through a main pipe changes intermittently and irregularly (the distance of the brattice cloth to the face changes);

5. the chemical process varies (variety of coal seams, variation of coal structure in a seam and in a given mine, variety of mining machines);

6. imperfect operating equipment (numerous equipment malfunctions in the rugged mine environment);

7. inadequate sampling equipment (long-duration dust samplers to determine transient dust concentrations);

8. erratic sampling equipment (difficult to scientifically sample airborne dust);

9. hostile environment (a 30-inch-high seam, roof falls down).

The Bureau's program has been divided into four general categories: dust control, personal protection, instrumentation, and chemical analysis. Expenditures are summarized as follows:

	FY 70	FY 71	FY 72	FY 73 (estimated)
	($1.000)			
Dust control	1,168	2,176	1,639	1,523
Personal protection	32	90	106	-
Instrumentation	270	447	253	379
Chemical analysis	168	227	347	135
	1,638	2,940	2,345	2,037 (estimated)

*About 50 percent of the coal mined underground in the United States is obtained by continuous (mechanical) mining, 47 percent by conventional (blasting) mining, and 3 percent by longwall (mechanical) mining. This paper is limited to dust control by the first mining technique.

The program involves inhouse work at two Bureau research facilities, Twin Cities Mining Research Center and Pittsburgh Mining and Safety Research Center, and also an assortment of contracts and grants with outside organizations.

This paper briefly describes the major items in the Bureau program and their status.

DUST CONTROL

Continuous mining machines were designed to mine coal at a fast rate. They are very efficient mining machines. However, by using blunt high-speed bits, they probably are the best machines for forming dust that could be invented, except for a grinding stone.

Bureau research has shown that continuous mining forms approximately 5,000 grams of respirable dust at the face per ton of mined coal. About 2 grams of this dust becomes airborne at the face; the remainder remains adhering to the run-of-face broken coal.

An exploratory statistical study indicated that the same mining occupations in different seams often had significantly different dust exposures. For example, the continuous miner operator in the Pittsburgh and Pocahontas seams is exposed to more dust than the same man in the Kittanning seam, but curiously, the Kittanning seam gives more dust along a haulage road than the other seams. Such information suggests basic differences in the dust-forming characteristics of different coal seams, but the explanation for these differences is not presently known.

Dust control techniques at the face include:

- available technology
- machine cutting parameters
- supplementary ventilation; dust collector
- water sprays
- wetting agents
- pick flushing
- foam
- infusion

These items are described and are followed by a concluding section discussing secondary dust:

Available Technology. Considerable technology is already available, but it is often not effectively used because of inconvenience or expense. For example, while water sprays provide a valuable dust control technique, their practical usefulness underground is limited because they frequently clog in the rugged mine environment. Cleaning or replacement of a clogged nozzle is expensive to the mine operator in terms of time and cost. The development of a non-clogging spray nozzle system would reduce operator expense and inconvenience and therefore would increase the actual effectiveness of water sprays as a dust suppression technique. A contract to develop a non-clogging nozzle is expected to be awarded in late FY 73.

Machine Cutting Parameters. The British conducted pioneering work examining the effect of machine parameters on the formation of airborne dust and concluded that sharp, slow-moving, deep-cutting bits produce less dust. The problem now is to obtain quantitative information on the amount of airborne dust versus machine parameters with American coal and mining methods, along with cost and other engineering information. Improvement in cutting parameters is estimated to offer a 50-percent decrease in airborne dust.

A full-scale machine which permits varying of the machine parameters (rpm, sump and shear rate, bit geometry and array) is being constructed by Ingersoll-Rand under Bureau contract. The machine will be automatically controlled to avoid operator variables and will be used in full-scale underground tests to obtain the desired information on the effect of the machine parameters versus dust formation. Also, a research mining machine having a single full-scale cutting wheel was designed and constructed. This research machine permits the machine parameters to be varied over a wider range than the full-scale machine and will be used as a "pilot plant" to further investigate the effect of cutting parameters on the formation of respirable dust. Finally, several laboratory programs are investigating dust formation during cutting in order to better understand the fragmentation process. Such studies hopefully will lead to new bit designs and the selection of machine parameters that reduce dust production.

Supplementary Ventilation. Air flow at a mining face is normally controlled by line brattice or by extensible tubing, occasionally in conjunction with an auxiliary fan. However, supplementary ventilation techniques that involve machine-mounted fans to draw or exhaust the dusty air from the vicinity of the face appear very attractive for dust control because they reduce the recirculation of face dust back to nearby personnel and can in principle be applied to various kinds of continuous mining machines and local mining situations. A disadvantage of this local exhaust approach is that the dusty air must either be discharged into the return via a duct or be passed through a machine-mounted dust collector with the partly cleaned effluent air being discharged at the mining machine.

A current program is examining the exhaust approach with an auger-type mining machine in low coal. A machine-mounted dust collector is used, but the effluent air is ducted into the return. The unit is presently being tested underground. Initial results are encouraging, but additional testing is required.

A severe problem in coal mines, especially in low coal, is the space limitation. For example, the available high-cfm fans were too large to install on the low-coal auger machine and two small-cfm fans had to be used in parallel. This was undesirable from an engineering viewpoint but was the only alternative at the time. It has since been established that technology is indeed available to fabricate a high-cfm, small-diameter fan suitable for operation in low coal, and a prototype unit is being constructed inhouse. The availability of such a fan would expedite the use of supplemental ventilation techniques in low coal and other areas where space limitations are critical.

Passing the exhaust dusty air through a machine-mounted dust collector has mushroomed in popularity during the past year. This approach has the distinct advantage of avoiding ducting from the machine to the return. However, the dust collector must be very efficient because any effluent dust may bathe the machine operator and nearby personnel and could even increase their dust exposure. Available dust collectors tested in 1970, comprising a large assortment, were all found unsatisfactory because of a low collection efficiency for respirable size dust or because of bulk or safety problems. The Bureau has fabricated an above-ground facility to evaluate the collection efficiencies of new full-scale collectors as they become available. For example, the collection efficiency of a typical scrubber designed for mounting on the boom of a continuous miner ranged from 80 percent for 1 micron dust to 99 percent for 5 micron dust. While surprisingly high, these efficiencies are still too low to scrub anticipated incoming dust levels to a 2 mg/m^3 level.

A venturi wet-collection approach appears to be the most attractive mechanical approach for achieving high collection efficiency. A Bureau-designed research-type venturi collector will be used to investigate collection efficiency versus power input, water flowrate, and other engineering parameters. Results will provide guidance for the design of dust collectors for specific situations and can assist a mine operator to determine whether a mechanical collector will bring him into compliance. In the interim, the Bureau is fabricating a new, simpler, low-cost, venturi collector that is especially designed for use in coal mines.

An alternative and new approach for a respirable dust collector is to use plastic surfaces such as polystyrene and polyethylene. Such materials usually have "islands" of electrical charge that rapidly collect airborne coal dust if the dust has an electrical charge. Exploratory inhouse underground tests have shown that mine dust often has an electrical charge and that an appreciable collection can be obtained merely by passing the dusty air through a plastic tube. In principle, such collection should be especially effective for smaller particles because of their higher mobility. In view of the difficulty of collecting very small particles, the feasibility of a plastic-type dust collector for coal mine use is being explored.

In general, the air flow pattern in the vicinity of the mining machine and at the face is crucially important in affecting the transport of face dust back to the machine operator and nearby personnel in exhaust ventilation. However, local flow patterns are largely unknown. Since the machines usually occupy a large fraction of the cross section of an entry, their presence would be expected to significantly influence flow patterns. Recent inhouse work confirmed this expectation and also found that the motion of the cutting wheels influences the local flow pattern.

A 1/10-scale laboratory model of a low-coal entry including the auger-type mining machine is being used in an inhouse study of local flow patterns and the effect of these patterns on the transport of respirable dust. This work supports the low-coal field contract. Results to date indicate that drawing about half of the incoming ventilating air through the machine leads to a drastic reduction of the dust levels at typical personnel locations. Modeling appears very attractive as an inexpensive technique for screening proposed auxiliary ventilation techniques before underground testing is initiated. Underground measurements are currently being made to verify the modeling concept.

Water Sprays. One of the main dust control techniques presently in use is water sprays. Sprays are reported to reduce the respirable dust level 20 to 60 percent, although 30 percent is a typical number. The type and placement of spray nozzles is currently selected in an arbitrary manner because guidelines are not available; a typical approach seems to be to merely add more nozzles in the hope of reducing more dust. Techniques seemingly could be devised which would make a more effective use of the sprays, e.g., either greater dust suppression with the existing water flow or sufficient suppression with a smaller water flow rate.

British laboratory studies indicated that the capture of airborne respirable-size particles with water drops is dependent upon the size, concentration, and velocity of the drops, although optimum spray parameters to be used to achieve maximum dust suppression for underground spray systems were not determined.

A Bureau program to determine these optimum parameters for a spray system was undertaken. A theoretical model for the capture of airborne dust was developed and verified in the laboratory. Capture efficiencies of up to 75 percent were obtained. The theory can be used to select an optimum spray nozzle which gives the maximum collection efficiency of airborne dust at a specific spray-nozzle location in a mine for the water flowrate, line pressure, and geometry at that location. In practice, of course, the water spray drops can also impact and moisten the surface of coal and thereby suppress the formation of airborne dust by interfering with the dust-forming cutting process or by enhancing the adhesion of newly-formed particles. The development of a theoretical impaction model is being studied inhouse. Combination of the impaction and airborne models will then be attempted.

In the interim, the usefulness of the above airborne theory for improved dust suppression at the front end of a continuous mining machine was tested underground. In one test series, dust suppression was about equal with all spray nozzles, although the "good" sprays used about one-third less water than other sprays. In another less extensive series, the good sprays also gave one-third less dust. The use of one-third less water is a major accomplishment because many mines already have excessive moisture. Additional underground testing is required to obtain definitive data regarding dust suppression.

Another Bureau program indicated that steam and water spray were about equal in effectiveness for suppressing the formation of airborne dust or for collecting airborne dust. The use of steam underground would involve difficult logistics and is not recommended.

Wetting Agents. The usefulness of wetting agents for increasing the effectiveness of water sprays as a dust control technique is controversial. Some workers state that they are worthless, others state they are helpful, and there is little published data to support either statement. Clarification is warranted in view of the expense and inconvenience associated with using such agents. The Bureau had a contractor measure the wetting behavior of 16 wetting agents on coal from six different seams. All agents essentially wetted all the coals. At present, another contractor is measuring the drop size and velocity of the sprays from several nozzles with several wetting agents. Underground tests will be conducted to establish the usefulness of wetting agents as a dust suppression technique.

Bit Flushing. British studies indicated that a 75 percent reduction in dust is obtainable by directly flushing the cutting bits with water. However, internal plugging of the orifices and seal leakage have prevented this technique from being widely used in the United States. Although approximately 40 such "wet-head" machines are underground, only two are reported to be using the wet-head mode. A contractor has designed an improved seal with one type of wet-head ripper machine and is presently conducting underground tests.

Foam. The use of foam for dust suppression is based on the concept of using a high-expansion foam to "blanket" the cutting site and thus physically prevent any dust from becoming airborne. Several studies to establish the merit of foam have been attempted over the past 10 years, but results were inconclusive because of insufficient underground testing. The Bureau therefore initiated a new, more detailed effort in order to obtain definitive results on the usefulness of foam. Brief underground tests in FY 72 established that the foam broke rapidly, that there were no slip hazards, and that the foam was well received by mine personnel. However, the underground testing regarding dust suppression was inconclusive, and the program is being continued to obtain sufficient data in order to be statistically significant.

Infusion. Water infusion has been useful in Europe for reducing dust formation during subsequent mining. German mining regulations require water infusion wherever possible as a dust control technique, but implementation seemingly is left to the discretion of individual mines. About 15 percent of the collieries in England are using infusion to control dust but are having problems due to the low permeability of English coal beds.

Infusion has received only limited attention in the United States owing to engineering-type equipment problems. The Bureau initiated an infusion program with the dual objectives of controlling methane and reducing the formation of the dust. To date, significant methane control has been achieved and dust seemingly is reduced by about 50 percent. However, considerable additional field work is required to obtain definitive results.

Secondary Dust Generation. Points of secondary dust generation include the gathering arms on the continuous mining machine or loader, dumping of material into the shuttle car, operation of the shuttle car along the roadway, belt operation, belt transfer points, etc.

Inhouse Bureau work has shown that enough respirable-size coal dust adheres to 20 pounds of ordinary run-of-face broken coal to contaminate approximately 1 million cu ft of air up to the 2 mg/m^3 level if it should become airborne. The potential danger of secondary hauling as a dust source is obvious. The magnitude of the forces involved in physical adhesion of coal particles to massive substrates was measured and found to approximately agree with expected adhesion forces.

Laboratory work indicates that only about 10 percent of the dust adhering to the run-of-face coal is dislodged and becomes airborne during a typical drop operation. Dust generation can be reduced only somewhat by changing the belt parameters, e.g., the dust would be reduced by about 30 percent by decreasing the drop height or slowing the belt by a factor of 2. A 70-percent reduction of dust should be obtainable by passing the broken coal down an inclined chute instead of a vertical drop or by using water sprays along the belt somewhat upstream of the drop point. These laboratory conclusions have not yet been tested in a full-scale operation.

At an underground belt transfer point, water sprays or a low-expansion foam injected directly into the falling coal reduced the formation of airborne dust by about 50 percent. Additional application of water sprays onto the underside of the belt reduced airborne dust by 60 percent, while foam on the underside of the belt reduced dust by 90 percent.

Laboratory research indicates that the formation of new dust due to secondary breakage during dropping is insignificant compared to the dislodgment of adhering dust.

PERSONAL PROTECTION

While the Bureau does not consider personal protective devices such as face masks as a primary approach for reducing the miner's exposure to dust, such personal protection can be visualized as an interim measure and also as a "last alternative" for certain dusty operations in case remedial measures are unsuccessful. The present filter-type face mask is undesirable because of (1) a high-pressure drop when clean and an excessive pressure drop during use due to plugging, (2) an imperfect match to the facial contour, (3) poor day-to-day refit in the field, (4) irritation due to dust at the mask-face juncture, and (5) interference with voice communication, spitting, etc.

A personal device employing the air curtain concept is presently being developed under Bureau contract. Dusty air from the environment is filtered and an air curtain of dust-free air is passed from the hat brim down over the miner's face, thereby shielding him from the dusty environment. Such a unit requires considerable power and therefore is limited to machine operators but hopefully will be of interest to industry in general.

INSTRUMENTATION

Improved dust samplers for monitoring the eight-hour exposure of miners and also for research purposes are required. The state-of-the-art in dust sampling is aptly summarized in an International Labor Organization report (1967), which concludes that "No evaluation or comparison of dust content has any significance... unless the type of equipment, the method of sampling, and the nature of the dust are precisely known."

The dust hazard in United States mines currently is assessed gravimetrically with a personal sampler continually worn by the miner during his working shift. The sampler uses a battery-powered pump to draw 2 l/min of dusty air through a cyclone, which collects the nonrespirable dust, and then through a membrane filter, which collects the respirable fraction. The filter is weighed in the laboratory to determine the total mg/m^3 of dust exposure; 2 mg of weighed dust approximately corresponds to 2 mg/m^3 exposure for eight hours.

Although this system functions, the weight and size of the personal sampler is burdensome to the miner, the sampler is subject to mischief, the entire approach is expensive, and the accuracy of the system has been questioned.

Bureau research has indicated that about half of the 1 to 2 µm underground airborne particles are agglomerated to large nonrespirable airborne particles. This means, unfortunately, that a size classifier must be used with any sampler, i.e., the sampler merely cannot collect all the dust and the respirable fraction measured in the laboratory, for the respirable dust concentration then would be overestimated.

Respirable dust nominally involves 7.1 µm-diameter or smaller particles and is specifically defined by the Act in terms of the size classifier of the British MRE dust sampler. However, medical authorities are seriously considering revising the TLV standard for silica to emphasize the smaller size particles. Also, some medical workers feel that dust particles somewhat larger than the respirable fraction may lead to some respiratory diseases, while others feel that the submicron particles are especially dangerous to human health. Furthermore, medical authorities are becoming increasingly concerned that short exposures to high dust levels may be more hazardous than the eight-hour time-averaged value of the dust exposure, which is the present basis of the dust standard.

Research purposes require a short-duration, fast-response field sampler to test the effectiveness of a dust control technique and to assist the identification of dust sources. The present midget impinger unit requires about 10 minutes to obtain sufficient dust for analysis by the Coulter counter. This technique is reasonably satisfactory for screening, comparison-type purposes. However, a 10-minute sample time is excessive for monitoring many mining operations owing to the short duration of the operation.

Light scattering is attractive because response time is rapid and the electrical output signal can be readily measured and could be telemetered or used in control circuitry. However, light scattering is related to the area of the particles in a dust cloud, and results must be converted to mass concentration (mg/m^3). With laboratory dust, this conversion is reasonably reproducible. With mine dust, the size distribution varies, and variable amounts of noncoal materials that have different densities and indexes of refraction are often present. Such variations will lead to a variable (and unknown) conversion factor. Despite the potential uncertainty in the conversion factor, a prototype unit was fabricated by a contractor. The unit has a fast response time (5 seconds) and already has been valuable in underground work as a "screening" tool.

CHEMICAL ANALYSIS

The 1969 Act specifies a maximum exposure to respirable coal mine dust of 2 mg/m^3 but also stipulates that the permitted level is reduced below the 2 mg/m^3 standard to the value given by the expression 10/(percent quartz) to match the TLV of 100 $\mu g/m^3$ quartz for an eight-hour exposure. All the silica in a coal-mine dust sample is assumed to be quartz.

At present, the Bureau analyzes the quartz content in mine dust by removing the dust from about 10 personal respirable dust samples, combining the dusts, ashing the combined dust, making a KBr pellet of the ashed dust, and measuring the amount of silica in the pellet with infrared (IR) techniques. This approach is time consuming and expensive and, by combining the silica content of the several samples, does not give the eight-hour exposure of an individual.

A technique for analyzing the silica content of a single field filter sample was recently developed by the Bureau wherein the dust was removed and redeposited onto a new filter and then analyzed by IR and X-ray techniques. Results obtained by both techniques agreed, giving support to the values obtained by both techniques. However, the true accuracy of any of the mentioned methods is exceedingly difficult to determine. Also, the effect of particle size, the occurrence of other SiO_2 polymorphs, and perhaps surface effects must be examined.

Direct measurement of the silica in the mine dust as collected on the field filter sample would greatly expedite analysis and is being explored by the Bureau and a contractor using IR techniques and by another contractor using a new soft X-ray approach.

Recent work in West Germany indicates that silica dusts from different coal mines have significantly different toxicities despite similar particle sizes and concentrations. Considerable clarification of the hazards associated with silica by medical authorities appears necessary.

Medical researchers have become increasingly concerned that small quantities of inorganic and other materials in the coal mine dust may add to the health hazard. The Bureau initiated several programs to develop laboratory analytical methods to measure the concentrations of assorted materials in anticipation that standards may be established. In addition, techniques were developed for measuring the surface areas and density of different size fractions in the respirable dust, in anticipation that such information would be of medical value. However, medical authorities have been hesitant to promulgate new standards in these areas, and only one program examining carcinogenic organic compounds in respirable dust has been continued.

CONCLUSION

The final objective of the present Bureau program is to provide advanced technology to control airborne respirable dust in coal mines by 1975 and preferably earlier. An assortment of approaches is being explored, for it is unlikely that a single technique will be equally suitable for the diverse mining operations. The present respirable dust problem is almost unique in industry because controlled experiments are difficult to perform, dust samplers are imperfect, and the environment is exceedingly hostile.

REFERENCES

1. P. Dessauer, et al, Annals of the New York Academy of Sciences, v. 200, 1972, p. 220.

INVESTIGATION OF MATERIALS FOR USE AS A MINE SEALANT/COATING, J. R. Carlson and W. P. Banks, Continental Oil Company, P. O. Box 1267, Ponca City, Oklahoma 74601.

The deterioration of rib-roof surfaces in coal mine entries which must be maintained for many years is a serious problem. A coating of sprayed urethane foam is known to effectively control deterioration in many cases, but at the same time poses a fire hazard. This paper presents an overview of recent work done with Consolidation Coal Company to find an effective, non-hazardous coating. Performance criteria are summarized for various mine conditions and are related to desired basic material properties. Results of laboratory and field tests on several types of materials using varied approaches to the problem are discussed. Comparisons are made on the safety, performance and economics of these materials.

THE EXTINCTION LIMITS OF AN ESTABLISHED FLAME

Spencer C. Sorenson*, Lester D. Savage*, and Roger A. Strehlow†

*Department of Mechanical and Industrial Engineering and †Aeronautical and Astronautical Engineering Department, University of Illinois at Urbana-Champaign, Urbana, Illinois 61801.

INTRODUCTION

Many coal mine explosions originate when the methane-air mixture at the working face is ignited. The air motion resulting from the initial combustion causes the coal dust layer on the walls to become stirred into the air ahead of the propagating flame. The explosion process then becomes one of a propagating coal dust methane flame. Therefore, any device which is designed to extinguish the flame by adding some sort of suppressant to the unburned air must be effective with coal dust methane flames of varying proportions. It is, therefore, essential to have a knowledge of the effectiveness of various suppressants and the mechanism by which they work.

In this project, we are developing a technique to study both the effectiveness and suppression mechanism for various solid and gaseous suppressants in methane coal dust air flames. In this paper, we are reporting some initial results dealing with the effect of gaseous suppressants on a gaseous methane air flame.

Traditionally, the flammability of such fuel-air suppressant mixtures has been determined by observing the upward or downward propagation of a flame over a fixed distance in a relatively large diameter tube after ignition at the open end of the tube (1). There is, however, some question as to the applicability of this type of data to the case where a flame propagates from a fully flammable region into a region that contains a suppressant.

This paper describes a new technique for determining flammability limits using a large steady flow burner in which the suppressant mixture is placed in contact with a flame propagating through a mixture devoid of suppressant in an attempt to more realistically model the mine situation.

THE BURNER

In order to investigate the problem of direct extinguishment, a special steady flow burner was constructed. Provisions have been made for the use of coal dust as a fuel and solids as suppressants although they were not used to obtain the results discussed in this report. The basic objective of the burner design was to obtain two relatively large area streams, one of which will support a steady oblique flame sheet such that the flame can be made to propagate from this fully flammable region into another region containing suppressant. This was accomplished by dividing the flow areas at the burner head in the manner shown in Figure 1.

The two inner rectangular regions of this burner are fed by flows that contain only a fuel-air mixture for the larger annular shaped rectangular region and fuel-air suppressant mixtures for the central rectangular region. The outer of these two regions therefore provides a typical non-suppressed premixed laminar flame as an ignition flame while the inner region contains a flow in which a suppressant may be added to the mixture to test for flammability limit behavior. The flow rates and composition of each region may be varied and measured independently using rotameters. The nitrogen flows along the two short edges of the burner prevent flame attachment at the end of the burner and facilitate end-on observation of the flame (i.e., observation along the major axis of the burner). The air channels along the longer outer edges of the burner shield the outer edges of the flame from external disturbances and can be used to stabilize large diffusion flames in the annular rectangular flow regime. Flow diffusers have been placed in all of the flow streams below the burner head in order to obtain a uniform flow velocity in each outlet region of the burner. The final exit plane of the burner is filled with over 2000 closely packed 1/8 inch stainless steel tubes which are long enough to develop and stabilize a fully laminar flow at the burner head. These tubes also serve to quench the flame and prevent flash back into the body of the burner at low flow velocities. In addition, as shown in Figure 1, the burner head contains a number of parallel stainless steel shim stock spacers which help stabilize both the flow and the flame.

THE TECHNIQUE

The flame shapes that are observed in this burner are actually quite complex and, in general, depend upon the stoichiometry of both the surrounding ignition stream and the central suppressant stream as well as their velocities.

It has been found that in the ignition stream a single large tent flame, open at both ends, may be easily stabilized on the rich side even though on the lean side it is easier to stabilize a flame which attaches to most of the shim spacers and thereby producing a shorter height multiple tent flame which is also open at the ends. In all these cases, the inner test stream, which may contain suppressant, is contacted on four sides by a hot product stream issuing from the ignition region.

In the experiments on flammability, we are interested in how this central stream behaves as its composition passes through the flammability limit of the mixture. Preliminary observations showed that for suppressant and ignition stream equivalence ratios which were lean-rich or rich-lean and for a flammable mixture in the suppressant stream the burner always exhibited a central tent flame in the suppressant stream which was anchored at the rectangular stream divider edges. However, for a rich-rich or lean-lean interface, the flame did not attach at the interface divider edge but instead propagated across the interface to produce a flame which, in general, situated itself at a different oblique angle than that which existed in the ignition stream. These two behaviors are illustrated for a section of the flame in Figure 2.

It was observed that if one viewed the flame along the major axis of the burner as one altered the suppressant stream composition from flammable to inflammable, the included angle of the central tent flame at first rapidly approached an angle which was near zero and then remained at or near that small angle with further changes in composition. Interestingly enough, this type of distinct limit behavior was observed for either of the interface geometries described in Figure 2.

Based on the above observations, the following procedure for determining flammability limits was developed. A premixed laminar flame was established in the outer (non-suppressed) stream, while the mixture in the inner (suppressant) stream was set well beyond the flammability limit in order to prevent flashback. The flows were adjusted so that the inner stream velocity always remained close to that of the outer stream.

During an experimental run the composition of the inner (suppressant) stream was changed systematically so that its composition varied in steps from a completely non-flammable to a completely flammable mixture. This was done by either changing the percentage of fuel (methane) or the percentage of suppressant depending upon the region of the flammability curve that was to be investigated in that particular run. Observations of the flame angle were made through a transparent plexiglass window located approximately three feet from the burner head. For each of the flow settings (i.e., for each of the compositions of the suppressant stream) the operator placed his eye in line with each flame tent edge and drew lines parallel to each side of the flame sheet as it existed just inside of the suppressant stream. A transparent plastic sheet was mounted on the plexiglass window for this purpose. In this way an accurate measurement of the included flame angle was obtained for that particular set of rotameter settings. This includedangle between the two flame sheets was divided by two to obtain the oblique flame angle α. Since this operation was performed for a number of points during a systematic change in composition, the technique in effect involves titrating the flame for an end point corresponding to the flammable limit of that particular mixture. Figure 3 illustrates the experimentally obtained relationship between the included flame half angle, α, and the fuel concentration in the suppressant stream for three different suppressant percentages. In this case the percent methane was the titration variable. Observe the decrease of α towards zero with a distinct change in the slope of the alpha-fuel composition curve when the angle becomes close to zero. In some cases, the stream lines were slightly divergent or convergent so that the maximum inflection in the curve (end point) was observed for values of alpha slightly less than or slightly greater than zero. In actual practice, the flammable limit composition was chosen as the point at which the flammable and inflammable branches of the alpha, percent composition curves intersected as indicated by the smooth extrapolation of these curves. It should be noted that it was still possible to observe an "apparent flame sheet" in our burner even though the suppressant stream composition was well outside the flammability limit. This can be attributed to reactions occurring as the fuel in the suppressant stream encounters hot combustion gases from the ignition stream due to diffusional processes. Thus, some observable reactions were occurring although under these conditions they were not of sufficient magnitude to support a propagating flame in the suppressant stream.

ACCURACY AND REPRODUCIBILITY OF THE TECHNIQUE

In order to check on accuracy and reproducibility, a few titration runs were repeated at different suppressant flow velocities and on different days. A comparison of the α versus CH_4 percent for these cases is shown in Figure 4.

Both of these curves illustrate that the reproducibility of the data is ± 1/2 percent CH_4 and that the location of the inflection point is relatively insensitive to the flow velocity. However, sensitivity of the technique is determined by the flow velocity to some extent because at high flow velocity and at high suppressant concentrations, the maximum value of α becomes very small, as is shown in Figure 3. Therefore, the experiments were always performed with the lowest possible suppressant flow velocity for the particular desired titration.

We feel that the equivalence ratio of the external ignition stream may possibly have an effect on the end point, i.e., on the measured flammability limit. However to date all the data has been taken with a lean surrounding ignition stream. This is the reason why the rich end of the α curves in Figure 4 all have inflammable branches whose value of α is greater than zero degrees. Under these conditions the rich branch of the flammability curve is always measured from an attached flame and flow divergence does not occur easily. The lean branch, on the other hand, is always oriented as shown in Figure 2b and under these conditions central stream divergence occurs relatively easily. Thus, some of the lean end points occur for relatively large negative values of α.

RESULTS

The effects of four suppressants (Ar, N_2, CO_2, and Halon 1301) on the flammability limits of a methane-air flame are shown in Figure 5. The flammability limits determined by this technique were generally found to be wider than those determined by standard vessel propagation techniques. The lean limits obtained tended to be about 1 percent CH_4 leaner than vessel propagation limits and conversely, the rich limits were about 1 percent CH_4 richer than vessel propagation limits for a given percentage of suppressant. The only exception noted was the case of Halon 1301 in which the rich flammability limits were approximately 2 percent CH_4 richer than those from vessel propagation limits (2).

It was observed that the rich limits for N_2 and AR were indistinguishable and that the lean limits were very close, although the N_2 lean limits were 10 percent higher than the values of Ar lean limits. The results of other investigators (1) show a substantial difference between N_2 and Ar limits. The reason for the differences in the results is not clear at this time. One of the features of the new technique which may have a bearing on the comparison with other techniques is the lack of a heat sink for the suppressed gases. Thus, the thermal conductivity of the gases may have a stronger influence in one system than in the other. In addition, since the new technique involves the use of a forced-flow system, the physical properties of the gases could also have varying degrees of influence in the different techniques.

CONCLUSIONS

A technique has been developed for the measurement of the extinction limits of well-established flames by gaseous suppressants. The results of this technique are similar to the results of conventional techniques, although the flammability limits are wider for this technique than those obtained for conventional techniques.

ACKNOWLEDGMENT

This work was supported by the Bureau of Mines, U.S. Department of Interior under Contract H0-122-122, Dr. David S. Burgess, Technical Monitor. The authors would also like to express their sincere appreciation to Richard A. Moreau, James H. Rush, and Gary M. Vance for their contributions to this work.

REFERENCES

1. Coward, H.F. and Jones, G.W. "Limits of Flammability of Gases and Vapors" Bulletin 503, U.S. Bureau of Mines, 1952.

2. Kuchta, J.M. and Burgess, D.S. "Effectiveness of Halogenated Agents Against Gaseous Explosions and Propellant Fires" Proceedings of Symposium on an Appraisal of Halogenated Fire Extinguishing Agents, National Academy of Sciences, p. 257, 1972.

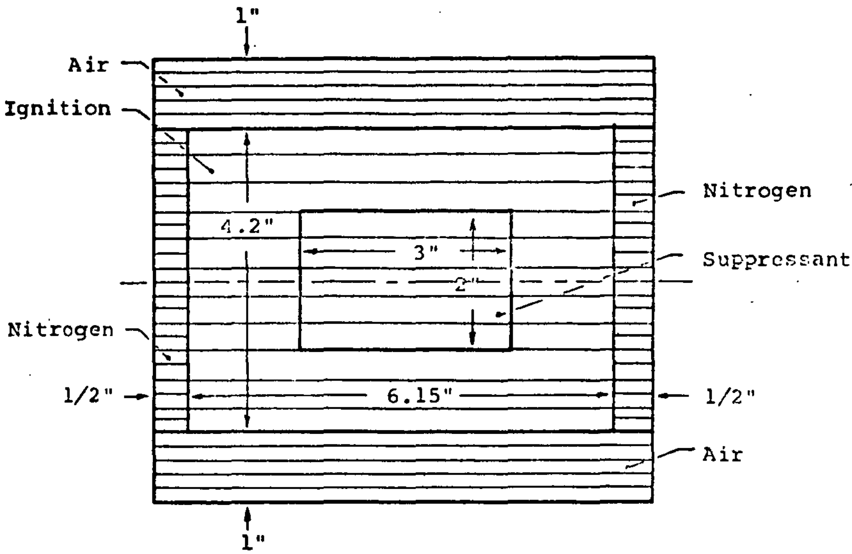
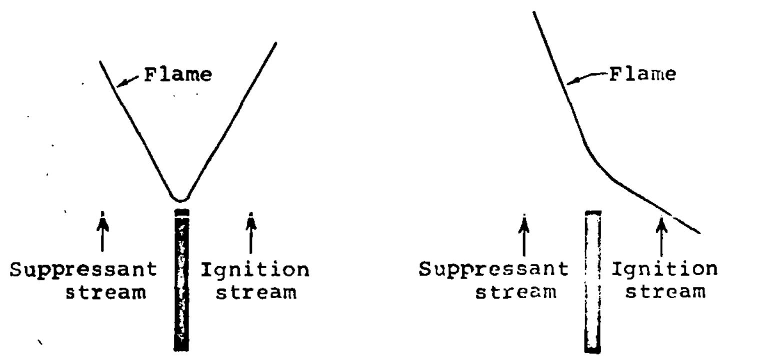


Figure 1. The burner head showing the different flow regions and the shim stock sheets (thin lines) used as flow straighteners and flame holders.



(a) Rich-lean or lean-rich interface (b) Rich-rich or lean-lean interface

Figure 2. Flame geometries as determined by relative equivalence ratios at the interface.

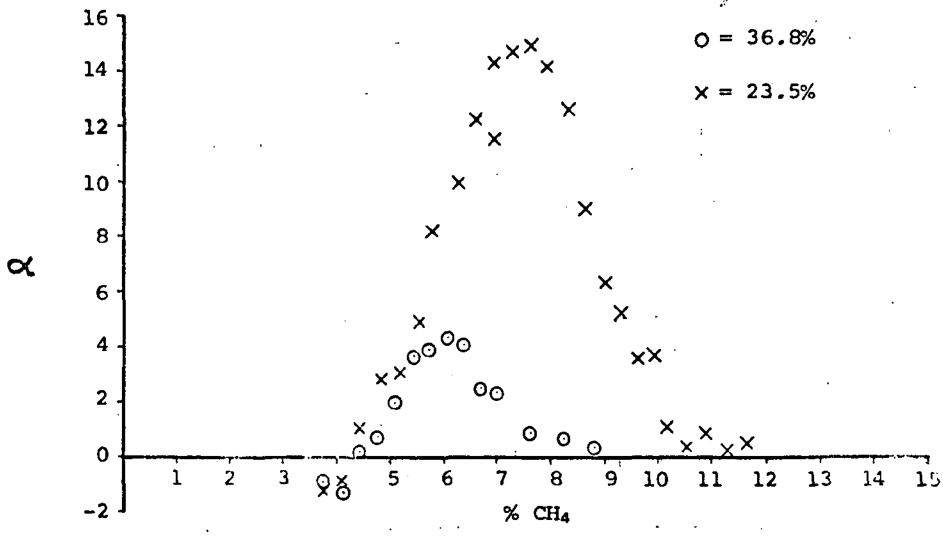


Figure 3. Effect of fuel concentration on flame angle.

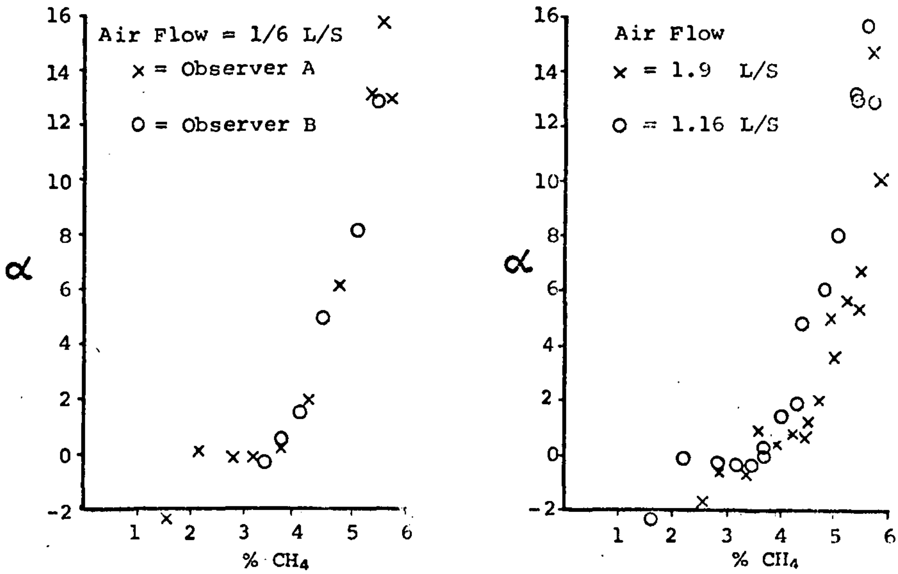


Figure 4. Effect of observer and flow velocity on end point.

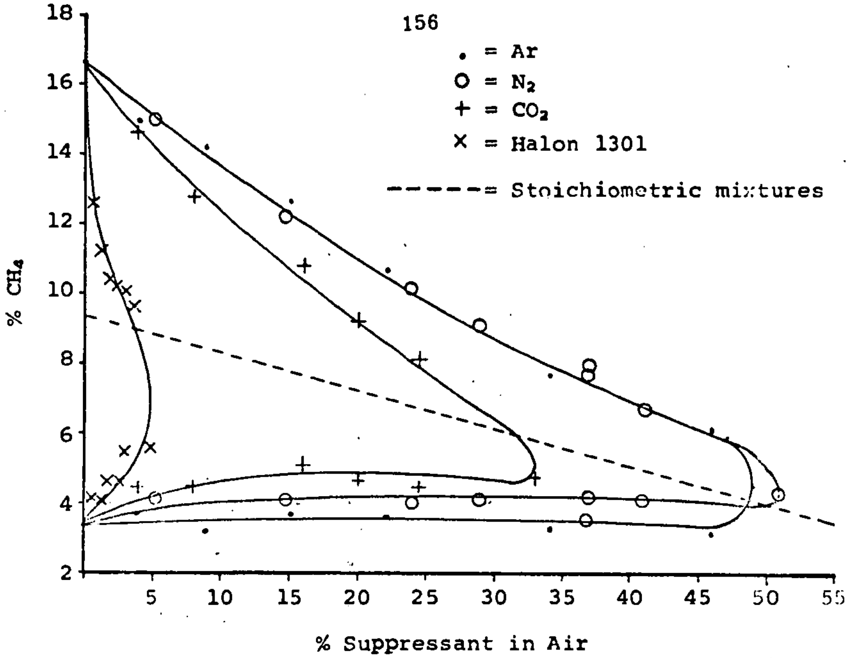


Figure 5. Flammability limits for methane-air suppressant mixtures.

THE USE OF GEOLOGICAL INFORMATION TO DESCRIBE COAL MINE ROOF CONDITIONS.

L. G. Benedict and R. R. Thompson, Homer Research Laboratories, Bethlehem Steel Corporation, Bethlehem, PA 18016

Since bad mine-roof conditions directly affect mine production and safety, it is important to be able to detect or identify the location of areas where such conditions are likely to be encountered in undeveloped portions of coal reserves. From our studies of the geology associated with the coal seams in our Cambria Division mines in central Pennsylvania as well as from discussions with our coal mine operators, we worked out a model based on geological variables that will make it possible to identify areas of potentially bad mine-roof conditions prior to mining. The geological factors in this model are: the intensity, direction and extent of surface fractures as determined from aerial photographs; the nature and thickness of the strata immediately above the coal seam; the thickness of the overburden; and the presence and extent of ancient stream-channel deposits. Taken together, these geological factors in a conventional mine-development plan should make it possible to assign a relative ranking of the roof conditions that are likely to be found as mining progresses over the extent of the coal reserve. Of course, quantification of this ranking will require a thorough knowledge of the independent role of each of the geological variables in mine-roof conditions, and this will be accomplished as mining advances over a larger portion of the area under study. Once the required information is obtained, the model will not only enable us to pinpoint the location of bad mine roof in coal reserves but should ultimately provide the guidelines for working out mine-development plans that will result in optimum productivity and maximum safety throughout the life of the coal reserve.

ENGINEERING ASPECTS OF COAL MINE VENTILATION SYSTEMS

R. V. Ramani and A. Owili-Eger

Department of Mineral Engineering, The Pennsylvania State University

INTRODUCTION

The single most critical factor affecting the health and safety of workers engaged in the coal-winning process is the mine environment; which is broadly defined as the space in which man works when underground and includes the physical and chemical conditions of the surrounding enclosure and the nearby mining equipment. A fundamental objective of a mine ventilation system is to supply this environment with an adequate and sufficient quantity of uncontaminated fresh air. It is the largest single logistics application in health and safety for underground coal mining operations. It is sometimes necessary to course intake air through several miles of underground airways in order to achieve this purpose. Leakages will be severe and yet an adequate supply of fresh air must be delivered to the last open cross-cut outby of the active face. This process is identified as quantity control. Equally important, if not more crucial, is the quality control process dealing with the control of respirable contaminants liberated during mining operations.

The Federal Coal Mine Health and Safety Act of 1969, Sec. 303(b), stipulates that the primary ventilation system must deliver at least 9,000 cfm of uncontaminated fresh air to the last open cross-cut. Auxiliary ventilation systems are required to supply 3,000 cfm to the coal face. Sec. 202(b), (2), of the 1969 Act states that each mine operator shall continuously maintain an average concentration of respirable dust in the mine environment during each shift at or below 2.0 mg/m^3. Respirable dust can be defined as solid coal particles in the minus 10 micron range which become airborne and do not settle easily.

With respect to gaseous concentration, the 1969 Act states that no working section of the mine shall contain more than 0.5 percent carbon dioxide and no harmful quantities of other noxious or poisonous gases such as the oxides of nitrogen. The concentration of methane should at all times be maintained below one percent. A split of air returning from any working section shall contain no more than 1.5 percent of methane and air that has passed by an opening of any abandoned area shall not be used to ventilate any working place in the coal mine if such air contains more than 0.25 percent methane. Methane concentration in the returns from the bleeder entries should not exceed 2.0 percent, and no air that has passed through an opening which is inaccessible for examination shall be used to ventilate any active areas. While the air quantity requirements alone can easily be met, the governing condition in most mines is the dilution requirements for quality control. A sufficient quantity of air is required to render harmless and carry away noxious and respirable pollutants.

Good ventilation is essential for efficient mine operation. On one hand, available ventilation facilities place certain limitations on production level. On the other hand, ventilation possibilities and requirements cannot be defined other than in relation to a production plan. Thus, ventilation and production planning are interdependent. Experience in the last decade has shown that the development of larger and more powerful machines for higher production rates calls for increased sophistication in ventilation planning. Over the last twenty years, ventilation standards have risen steadily and stringent regulations have been enacted. It is also evident that in the near future, ventilation requirements will become increasingly more demanding. The trend is a consequence of change in both mining conditions and equipment. These changing conditions, such as higher production from fewer mines, extraction of thinner seams, increasing depth of workings and additional installed horsepower, influence the planning and design of ventilation systems. It is, therefore, becoming increasingly important that ventilation requirements should be adequately assessed at the planning stage.

COMPUTERS IN MINE VENTILATION STUDIES

The role which computers can play in evaluating ventilation parameters and processes is tremendous. During the past decade, the role of the digital computer has expanded in an unprecedented manner from purely commercial applications to problems involving the design, maintenance, and control of technical systems. Major development of computer applications for mine ventilation has been in the field of ventilation planning. When an enormous amount of data has been collected, a computer can be used advantageously to relieve the ventilation engineer from the tedium of routine repetitive calculations. In addition, digital data processing has both short and long term significance. Long range data processing involves such observations which are routine and are being continually recorded. An accumulation of such data could be tested for statistically significant trends (6). This information could then be used in planning the ventilation of new sections or adjacent mining operations. Short term data processing involves spot testing and checking of such parameters as fan performance, optimum roadway sizes, air horsepower losses, and statistical examination of information surveys.

Development of computerized methods of calculations is only one aspect of the use of digital machines by research personnel. An example of ventilation research which is completely computer dependent is the theoretical investigations to study the patterns of methane flows and rates into the mine openings from the roof, sides, and floor. Mathematical models have been developed which enable the computer to simulate gas flow rates while varying such parameters as the position and emissivity of the gas sources, boundary pressures, and the permeabilities of the intervening strata.

The application of energy and mass transport phenomena, or more appropriately, physico-chemical principles, to quantify mining engineering parameters for quality and quantity control of the mine environment has taken an unprecedented outlook since the advent of digital computers. These machines enable complex problems in ventilation to be modeled and solved numerically. It is, therefore, not surprising that solution of problems of temperature and humidity, fluid flow dynamics, and toxic emission dilution in the mine environment are being attempted through mathematical models and computers. The parameters obtained from such numerical estimation can then be utilized in engineering and process control.

Today, more than ever, much attention is being focused on the development of a system of remote monitoring and control of environmental parameters. Significant progress is being reported in remote sensing and monitoring of environment. Some work has already been done in the United Kingdom on automatic monitoring of a methane drainage system. Such automatic controls can be extended to other environmental parameters such as heat and humidity, temperature and dust levels. As studies are perfected in the determination of suitable parameters for control, their range of operation and choice of monitoring sites, these advances can eventually be used to develop on-line computer control systems of the complete mine environment.

VENTILATION NETWORK ANALYSIS

The theory of network analysis has long played an important role in many branches of engineering sciences. Transportation and other distribution problems have been solved by applying tools of network analysis. Literature review reveals that the application of this theory to mine ventilation planning and network analysis is a recent development. There has, however, been a growing awareness in recent years that certain concepts of network theory can be successfully applied in many other fields as well. It is, therefore, not surprising that in the last ten years, these tools have found increasing application in mine ventilation network analysis.

The major advance in ventilation network analysis in the last decade has been, therefore, the development of computer programs capable of developing and solving the systems of equations defined by the junction and pressure laws. Ever since Hardy-Cross iterative technique (4) was adapted and modified by Scott and Hinsley (13) for the solution of ventilation network problems, several programs have been developed to solve for mine ventilation parameters. A brief description of most of the quantity flow mine ventilation programs is presented by Geiger (5).

METHANE GENERATOR MODELS

The quantity of methane emitted into the mine atmosphere and the movement of gas through solid coal and the adjoining country rock are dependent on gas emissivity, boundary conditions, and the initial gas distribution pressures and the combination of natural and mining factors. Functional relationships between these factors are not yet known. Consequently, the development of rigorous mathematical equations to simulate methane flow into the mine air is difficult, and to date, no model has been reported which is capable of such a function. Several researchers in many parts of the world have attempted to quantitatively describe the pattern of gas emission in mines as functions of seam characteristics and the confining gas pressure (1,2,9), none with complete success.

Thus far, empirical formulas available for the calculation of gas released from underground sources appear amenable to analytical computation. These estimations are at best only crudely approximate. The numerical methods available have not been made practical enough for use in the industry. Therefore, a relationship amenable to numerical analysis and practical enough for application in the industry would provide the flexibility lacking in analytical approaches. The mathematical model developed and presented by Owili-Eger (10) is yet another attempt to correlate as many flow governing parameters in a single equation. This model solves gas flow-rates into mine workings from coal seams and intervening layers of rocks and is designed to handle only two dimensional steady state flow systems. It represents a modified gas diffusion system for flow through porous media. Mathematical considerations and derivation of the flow equation are presented in the reference.

Various investigators have shown that temperature also affects the rate of gas flow. However, at temperatures usually encountered in mines, methane has a very low rate of diffusion. As long as shallow deposits are being extracted, the effect of temperature on the rate of gas flow will remain insignificant. At great depth where rock temperature will be high, temperature will affect flowrate of gas and has to be considered. The chemistry of multi-component multi-phase systems has also been considered in a recent report on methane flow modelling (12). It has been pointed out that the presence of water affects the migration characteristics of methane in coal seams even though the two fluids are said to be chemically unreactive.

HEAT AND HUMIDITY CONTROL

Temperature and humidity control is important in the face area, or more generally, any active section of the mine where men are exposed to the environment for as long as a shift. Factors that affect the temperature of the ventilating air in such areas are the presence of men and machines, the temperature of the incoming air, chemical oxidation of coal, evaporation of water which may extract part of the latent heat from the air and the heat transfer from the surrounding strata. This last factor is very significant especially in deep mines where wall rock temperature is much higher than that of the incoming ventilation stream. In the U.S. coal mines today, rise in temperature and humidity as a result of heat transfer from the wall rock is not significant because coal seams being mined presently are very shallow - less than 1500 feet. However, these coal reserves are being depleted at a very fast rate. It is only logical to say that operation at great depth in the near future is inevitable. In deep metal mines, heat transfer from the enclosing strata can be significant and, consequently, techniques for predicting air temperatures have been developed. The first requirement in mathematically formulating heat transfer problems is knowledge of the thermal properties of the strata surrounding the roadway or model area. Jones (7), Jordan (8), and Paulin (11) have presented details of mathematical considerations which can be taken to estimate such parameters if the thermal history of the region is unknown. Extensive work in this area is reported in South Africa and in Japan, Amano and Shigeno (3) and Vance and Kathage (15).

DUST IN COAL MINES

Coal dust is defined as any solid coal particles smaller than 100 microns (1 micron = 10^{-4}cm) which become airborne when disseminated. During the coal winning process, dust is inevitably generated by the mining machines. Transfer points and transport systems, and high air velocities, contribute to the dust problem in the mine roadways. Inadequate suppression of coal dust can lead to explosion hazards and the coal workers' pneumoconiosis. Pneumoconiosis is supposed to be caused by the inhalation of respirable coal dust (-10 micron) for extended periods of time.

In addition to the human suffering, cost of compensating pneumoconiosis victims is high. For example, in the fiscal year 1972, the Department of HEW provided $384 million for this cause which marked an increase of $142 million over the 1971 budget (14). This cost is expected to rise even more steeply in the next few years as more and more coal workers become eligible for compensation benefits.

In the light of economics of remedial actions due to dust hazards, the prediction of dust content in air is very necessary from health and safety points of view. The amount of coal dust deposited along the mine roadways is of paramount importance because it serves as the assessment of the dust explosion hazard, and it can be used to calculate the amount of incombustible dust required for rock-dusting. Mechanisms of dust deposition and quantity estimations have been discussed at length (5). Two processes have been employed to control dust levels at the working face. These are 1) water sprays, and 2) face ventilation to reduce the concentration of respirable coal dust. Although water sprays reduce dust load handled by ventilation systems, the current spray techniques are not effective in the control of respirable dust (5). Currently, face ventilation is the most commonly used procedure.

A dust control program should, therefore, be able to deal effectively with both the fine and ultra-fine dust sizes. Work in the area of dust characterization, particularly with respect to the size-consist, chemical properties is reported (14). Under the Coal Mine Health and Safety Act of 1969, research on dust control has accelerated.

HAZARDS FROM DIESEL EQUIPMENT

The principal hazards of using diesels underground have been identified as:

1) The transportation and storage of a highly flammable and volatile fuel with resultant hazards of fire and/or explosions.
2) Unhealthy conditions caused by the discharge of toxic substances from the engine exhaust.
3) Ignition of flammable atmospheres by hot surfaces of the engine, such as exhaust manifold or by burning particles of carbon from the exhaust gas.

It has been long established from engineering tests that diesel exhaust quantities are related to displacement and speed characteristics of the engine, the design of the engine, and the fuel-air ratio needed to produce useful power. It has also been recognized that diesels under proper control produce only minor amounts of toxic and noxious fumes although there is an irreducible limit for the exhaust gases. The contaminants that are released into the mine atmosphere must be diluted immediately to minimize local concentrations. Furthermore, since multiple units usually will be operating in the same air stream, there is a cumulative effect on contaminant concentrations. The rate of contamination, therefore, is related to the volume and velocity of the ventilating air, frequency and duration of engine operations, and engine load and location. Ventilation, in addition, affects the rate at which the contamination moves through the workings. Because of Threshold Limit Values (TLVs) and excursion rates allowed on some of the contaminants, the calculations for ventilation requirements must take into account the time variant characteristics of the contaminant concentrations and average them over the entire operating period of the mine. The objective of a research program at Penn State is the development of a digital simulation model to study the effect of using diesels underground on mine ventilation systems (16).

SUMMARY

In its most elementary form, coal mining is materials handling and consists of removing in-situ coal from multiple mine origins to final destinations. There are certain special environmental considerations peculiar to underground mining. These include the problems of toxic, noxious, and explosive atmospheres caused by gases and dust. A fundamental objective of a mine ventilation system is to supply the mine environment with an adequate quantity of uncontaminated fresh air to deal with the control of respirable and explosive contaminants and to provide for the health and safety of the workmen. This paper has briefly reviewed some recent research programs presently underway in the mine ventilation area.

It is difficult to quote an average cubic feet per minute ventilation figure for coal mines. The weighted figure is unknown and really has no significance. However, in a modern coal mine, for each ton of coal produced each day, on an average, 4-6 tons of air is circulated. Similarly, cost figures are very difficult to obtain from the industry because of the concern for releasing proprietary information. Also, accounting principles vary from one mine to another to make comparisons less meaningful. However, as a typical figure for a group of mines, operating costs are 15¢ per ton and capital costs 25¢ per ton, accounting for about 6% of the total costs. However, the indirect costs of mine ventilation, though difficult to ascertain, can be quite high. Severe disruption of work and losses in production cannot be ruled out. In any case, the end results of inadequate control in mine ventilation can be sudden and catastrophic e.g., explosions, ignitions, fires, suffocations, etc.

Much research - theoretical, empirical, laboratory, and field studies - has been done and is being done toward the identification and quantification of the hazards posed by inadequate ventilation. Model studies of gas and heat flow problems provide the necessary input parameters to sophisticated quantity and quality control models. Research efforts are also aimed at the determination of suitable parameters for process control, their range of operations, the development of remote sensing and monitoring equipment, choice of monitoring sites, etc. In the future, it is not difficult to foresee a highly sensitive monitoring system coupled to computers that are programmed for automatic corrective action.

REFERENCES

1. Airey, E. M., 1968, "Gas Emission from Broken Coal - An Experimental and Theoretical Investigation", Int. J. Rock Mech. Min. Sci., Vol. 5.

2. ____________, January 1969, "Diffusion of Firedamp in Mine Airways", The Mng. Engr., No. 100, Vol. 128.

3. Amano, K., and Shigeno, S., 1969, "An Underground Ventilation Network Analysis and Estimation of Temperature of Air Current", A Decade of Digital Computing in the Mineral Industry, edited by A. Weiss.

4. Cross, H., 1936, "Analysis of Flows in Networks of Conduits or Conductors", Bull. Engrg. Exp. Sta., Univ. Ill., No. 286.

5. Geiger, G. E., September 1, 1972, "Development of an Air Quality Simulator for Coal Mines", U.S.B.M. Project No. Go101478 (MIN-35).

6. Jollife, G. V., February 1970, "The Emission of Methane from Rapidly Advancing Coalfaces", The Mng. Engr., Vol. 129, No. 113.

7. Jones, C., June 1964, "Air Temperature Along a Main Intake Roadway", Coll. Guard., Vol. 208.

8. Jordan, D. W., 1965, "The Numerical Solution of Underground Heat Transfer Problems - I. Methods Relating to Dry Roadways", Int. J. Rock Mech. Min. Sci., Vol. 2, pp. 247-270, Pergamon Press Ltd., London.

9. Lidin, G. D., Airuni, A. T., Klebanov, F. S., and Matrienko, N. G., 1964, Control of Methane in Coal Mines (Translated from Russian), Israel Program for Scientific Translation, Jerusalem.

10. Owili-Eger, A. S. C., March 1973, "Simulation of Quantity and Quality Control in Mine Ventilation", M.S. Thesis, The Pennsylvania State University, University Park, Pennsylvania.

11. Paulin, A., 1970, "Thermal Conductivity of Porous Materials", Mining and Metallurgy Quarterly, No. 1, NOLIT Publishing House, Belgrade, Yugoslavia.

12. Price, H. S., and Abdalla, A. A., 1972, "Mathematical Model Simulating Flow of Methane and Water in Coal Mine Systems", U.S.B.M. Project Contract No. S0110752, PMSRC, Int. Comp. Appl., Houston, Texas.

13. Scott, D. R., and Hinsley, F. B., December 1951, "Ventilation Network Theory", Coll. Engrg. Parts 1-4, February, April, June, and December, 1951.

14. Thakur, P. C., September 1971, "Mass Distribution, Percent Yield, Non-Settling Size and Aerodynamic Shape Factor of Respirable Coal Dust Particles", M.S. Thesis, The Pennsylvania State University, University Park, Pennsylvania.

15. Vance, W. E., and Kathage, B. A., March 1967, "A Model Experiment to Determine Ventilation Cooling Occurring Around Underground Roadways", J. Mine Vent. Soc. of S. Africa, Vol. 20, No. 3.

16. Stefanko, R., and Ramani, R. V., January 1973, "The Effect of Diesel Equipment on Mine Ventilation Systems", Paper presented at the U.S. Bureau of Mines' Diesel Symposium, Pittsburgh, Pennsylvania.

Respiratory Protection and Respirable Dust in Underground Coal Mines

H. E. Harris, W. C. DeSieghardt

Eastern Associated Coal Corp.
Research Center, Everett, Massachusetts 02149

I. INTRODUCTION

During the past several years, and especially as a result of the enactment of the Federal Coal Mine Health and Safety Act of 1969, much attention has been focused on respirable coal dust and various means of preventing the inhalation of such dust, including the use of dust respirators.

The use of respirators in coal mines is certainly not new and, in fact, almost 40 years ago the Bureau of Mines first established performance requirements under Schedule 21(1).* However, there was little information available about the usage of respirators in the field and, importantly, there was no information on how effective are dust respirators under actual working conditions. Consequently, the National Institute for Occupational Safety and Health sponsored a research project with Eastern Associated Coal Corp., with the Harvard School of Public Health acting as a subcontractor. The three major objectives of this project were:

a. To determine, by means of a field survey, the current status of respirator usage with regard to duration and frequency of use, types, and maintenance levels.

b. To determine protection factors provided by respirators worn by working miners.

c. To make recommendations on ways to improve existing units, or on research needed to develop new types of respiratory protective devices for coal miners.

II. FIELD SURVEY

A field survey (2), which was carried out in 1970 and 1971, involved visits to 47 mines and interviews with 511 supervisory and underground mining personnel; personnel interviewed included representation of all of the major job classifications found in underground mining operations.

Results from this survey showed not only was there rather widespread possession and usage of dust respirators (a small percentage of which, incidentally, were not Bureau of Mines approved) but the working miners expressed strong sentiments for the need for use of respirators, Table I.

---

*Underlined numbers in parentheses refer to references at the end of this paper.

TABLE I. Need for Use of Respirators in Coal Mines

	Percent of Underground Work Force*
Generally Needed	42
Used Whenever Dust is Present	45
Used Only When Necessary	4
Needed, but are Hard to Wear	8
Prevent Dust to Make Usage Unnecessary	1

*428 people in various job classifications, plus 17 Section Foremen

It was also found that virtually all coal miners use respirators on an intermittent basis, i.e., putting the respirator on and taking it off a varying number of times during a work shift. Based on intermittent use, a significant number of miners found the presently available, approved respirators to be only marginally acceptable or unacceptable, Table II.

TABLE II. Respirator Acceptability Based on Intermittent Use

	Percent of Underground Work Force*
Completed	2
Generally	64
Marginally	24
Unacceptable	10

*See Note on Table I.

Major complaints about current dust respirators in use could be placed in two categories, namely, breathing difficulties and physical discomfort, Table III, and, consequently, the miners want respirators that are more comfortable and provide easier breathing, Table IV.

TABLE III. Problems Associated with Respirator Use

		Percent of Underground Work Force*
Cause Breathing Difficulties		37
Physical Discomfort		55
Generally Cumbersome and Uncomfortable	13	
Cause Perspiration	9	
Interfere with Tobacco Chewing	9	
Troublesome Head Harness	7	
Respirator Too Large	6	
Facepiece Troublesome	5	
Dust Inside Mask	5	
Improper Fit	1	
Interference with Work		9
Restricts Vision or Interferes with Wearing Glasses	5	
Exhalation Valve Troublesome	2	
Interferes with Communications	1	
Difficult to Carry	1	

*See note on Table I.

TABLE IV. Improvements in Respirators Desired by Mining Personnel

Improvements	Percent of Underground Work Force* A	B**
Easier Breathing	19	28
Comfortable Facepiece	12	18
Smaller Unit	11	16
Comfortable Head Harness	11	16
Lighter Unit	6	9
Better Filter	5	7
Better Valves	2	4
Easier to Carry	1	2
Educate Men to Use Them	3	-
Cannot Be Improved	2	-
Do Not Know	28	-
	100	100

* See Note on Table I.

** Percentage Recomputed from Part A by eliminating last three items in Part A.

Further information on results of the field survey have been reported elsewhere.(2)

III. PROTECTION FACTORS

1. General

As mentioned previously, the field survey revealed that virtually all underground miners wear respirators only on an intermittent basis. This, coupled with the fact that the accumulated exposure of miners to respirable coal dust is considered to be of importance with respect to the incidence of coal workers pneumoconiosis, indicated two protection factors should be determined. One protection factor, entitled "Effective Protection Factor (EPF), represents the amount of protection obtained by working coal miners over the entire work shift when the respirators are used intermittently and worn according to the miner's training and work habits. Therefore, EPF was determined, in the field, by sampling separately, but concurrently, the ambient air and the air inside the respirator facepiece; over the entire working shift the concentration of respirable dust was determined for each sample. EPF was calculated as follows:

$$EPF = \frac{DC_A}{DC_M} \qquad 1)$$

where:

EPF = Effective Protection Factor

DC_A = Dust Concentration in the mine air

DC_M = Dust Concentration in the air in the respirator mask.

Since sampling was done over the entire working shift both DC_A and DC_M are time weighted average concentrations of respirable dust.

While EPF represents the protection provided to the working coal miner, it does not tell how much protection is provided by the half-mask respirator when the respirator is actually worn. Consequently, True Protection Factor (TPF), which is defined as the amount of protection the user receives when he is actually wearing the respirator and in accordance with the manufacturer's instructions, was determined by sampling separately but concurrently the ambient air and air inside the facepiece only when the respirator was worn; respirable dust concentrations were determined for both samples. TPF was calculated as follows:

$$TPF = \frac{DC_S}{DC_R} \qquad 2)$$

where:

TPF = True Protection Factor

DC_S = Dust Concentration in the Mine Air in the Vicinity of Miner wearing the respirator

DC_R = Dust Concentration in the air inside the respirator facepiece.

2. Equipment and Procedure:

   a. Equipment

For determining EPF's, mine air sampling was done with conventional personal mass respirable sampling equipment (3) in use throughout the coal industry. Air inside the facepiece of the respirator was sampled using the same mass respirable sampling equipment, Figure 1, with the cyclone mounted on the respirator and connected to a sampling port inside the respirator (4). Also, located inside the facepiece, Figure 2, was a thermistor which is a part of a time-of-wearing device (4) that was used to determine the amount of time the respirator was actually worn.

In the case of TPF, sampling of the mine air and the air in the mask was done using two GCA RDM-101 Respirable Dust Monitors (5) both equipped with the same 10mm AEC Cyclone as used with the personal samplers. Figure 3 shows the sampling equipment in actual use underground.

FIGURE 1 - TEST SUBJECT WEARING SAMPLING EQUIPMENT

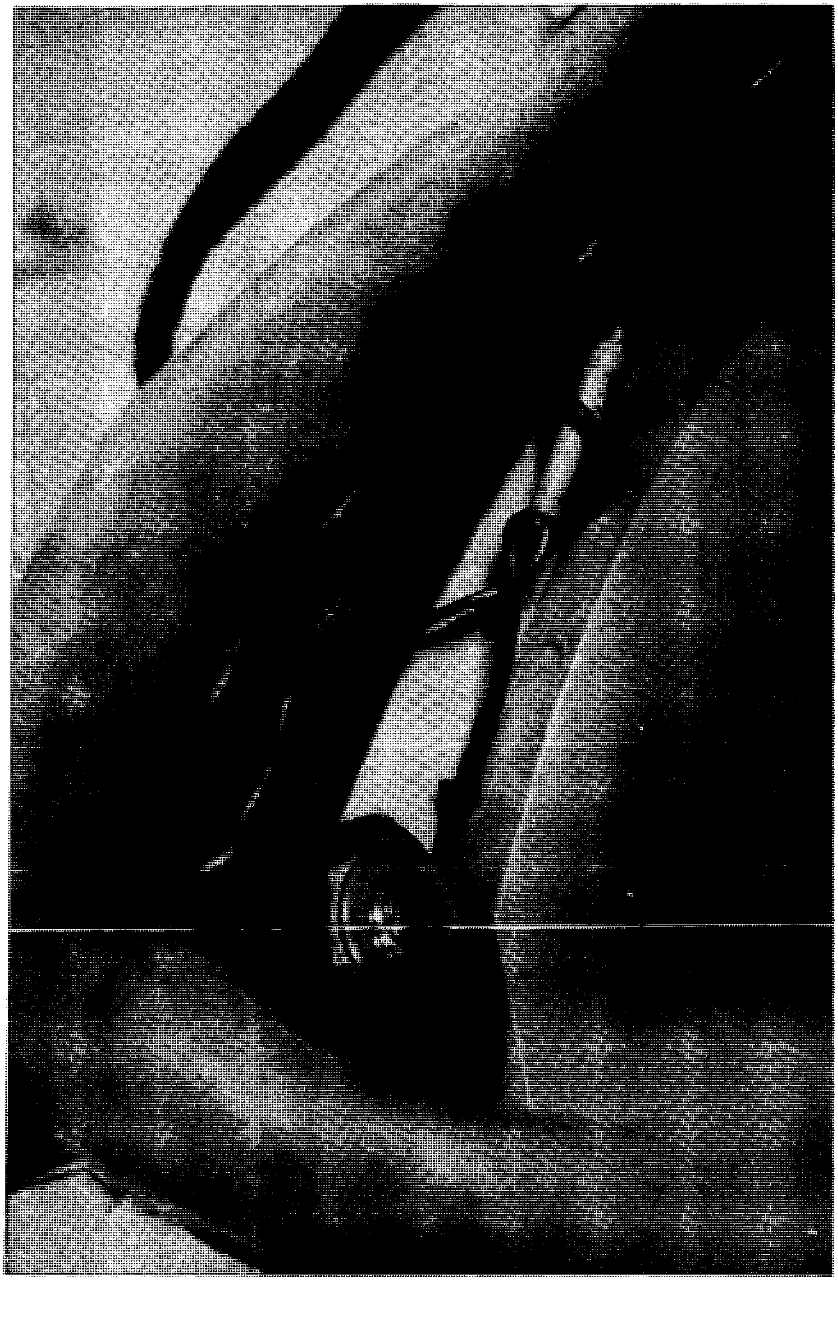


FIGURE 2 - INSIDE OF RESPIRATOR FACEPIECE SHOWING THERMISTOR

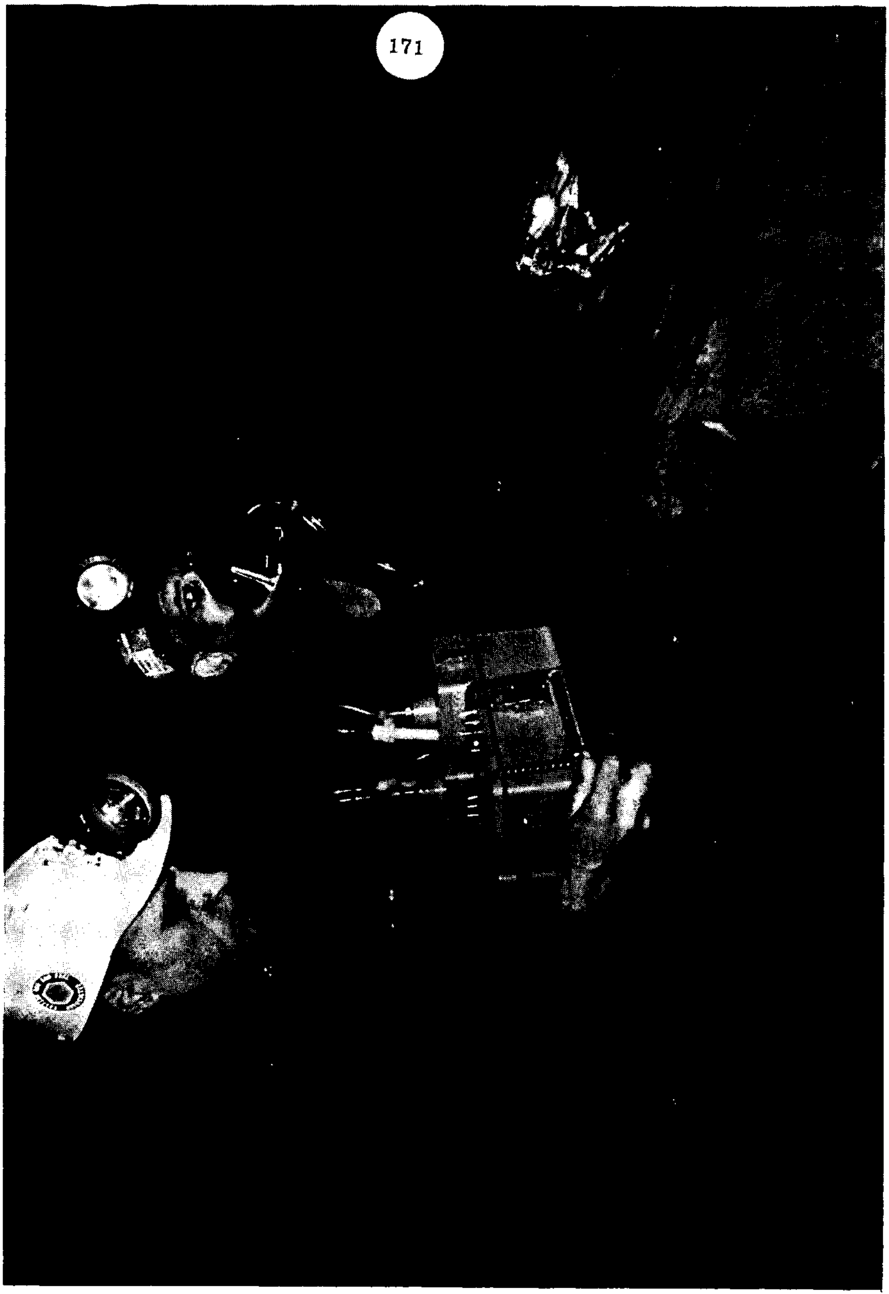


FIGURE 3 - TRUE PROTECTION FACTORS SAMPLING EQUIPMENT

b. Procedures

For EPF's, air and in mask sampling was done from the time each test subject miner started work until work ceased at the end of the shift, except for the lunch period. As shown in Table V, testing was done in five different mines and involved 208 man shifts and 13 different job classifications, mostly those at the working face. Five different models of respirators were used.

TABLE V. Scope of Testing - Effective Protection Factor

No. of Mines	5
Days of Testing	26
Man Shifts of Testing	208
Test Subjects (by job classification)	Number
Continuous Mining Machine Operator	5
Continuous Mining Machine Helper	1
Loading Machine Operator	6
Roof Bolter	3
Shuttle Car Operator	7
Bratticeman	2
Cutting Machine Operator	2
Coal Driller	2
Longwall Machine Headgate Operator	1
Longwall Machine Tail Operator	1
Longwall Machine Jack Machine Operator	2
Safety Technician	4
Rock Duster	8
Research Investigator	1
Total	45

For the TPF, 8 different face miners and one research engineer were used as test subjects. These people, which included 6 different job classifications of face miners, represented 8 different facial sizes as classified by the system set forth by Hyatt, et al (6); a diagram of this system is shown in Figure 4. Each of the test subjects wore 5 different respirator models over a 3 day period. During the period each respirator was worn, four sampling runs, each of four minutes duration, were made in which the mine air in test subjects' breathing zones and the air inside the respirator facepiece were sampled concurrently.

3. Results

While all of the data have been obtained, the analyses of the data had not been completed at the time this manuscript was prepared; consequently this should be considered in the nature of a progress report.

The distribution of EPF's for all the test subjects who were face miners is shown in Figure 5 and, similarly, the distribution for TPF's is shown in Figure 6. Some interesting differences can be observed.

Figure 4

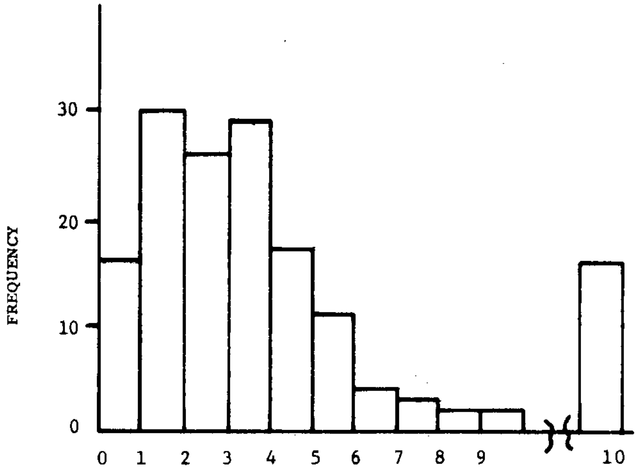
Facial Size Classification Diagram

(Job Classification of Test Subjects put in appropriate block according to facial measurements)

Face Width, mm

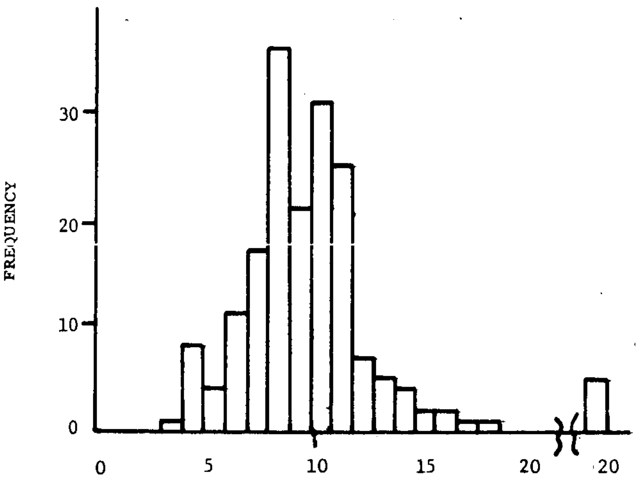
FACE LENGTH, mm	129 - 139	140 - 145	146 - 155
136 126-	A Roof Bolter	B Cutting Machine Operator	C Continuous Miner Operator
125 116-	D Bratticeman	E Timberman Loading Machine Operator	F Roof Bolter
115 105-	G Research Engineer	H Loading Machine Operator	I None

DISTRIBUTION OF PROTECTION FACTORS



EFFECTIVE PROTECTION FACTOR

FIGURE 5



TOTAL PROTECTION FACTOR

FIGURE 6

For the EPF's, values ranged from less than one to as high as 40 and above; however, most of the values were in the one to four range. Of the 151 values obtained, 16, were less than 1.0. While it may seem surprising that on occasion a respirator user is, either getting no protection at all or is possibly inhaling more respirable dust than is present in the ambient air, field observations indicate such is the case. For example, it is quite possible that respirable dust collected on the miner's clothes could be brushed off or knocked loose and be collected in the mask, which was worn hanging loose on the wearer's chest, thereby creating the higher dust concentrations found in the mask.

Unlike the EPF's, the TPF's showed a reasonably normal distribution and with little difference between mean and median values.

As shown in Figure 7, during the EPF test work the time the respirators were actually worn during the work period by the test subjects varied from a low of about 10 percent of the time to almost 90 percent; the mean average was about 46 percent of the time. It might be expected that a relationship should exist between the length of time the respirator is actually worn and the level, or effectiveness, of the protection obtained; in other words, the longer the respirator is worn, the better the protection (higher EPF) obtained. However, so far we have found no relationship to exist between the time the respirator was worn and the protection obtained. This suggests there are probably other factors that obviate the effect of time of wearing.

Test results for the five different models of respirators tested (for EPF) is shown in Table VI. It should be noted that the data shown include some very high values and values, as mentioned previously, where the EPF is less than 1.0. Both can, of course, influence the mean average and, consequently, the median value is also shown. Although we have not completed a statistical analyses of the data, it does appear there are differences among the respirators tested with respirators B and E being less effective than the others.

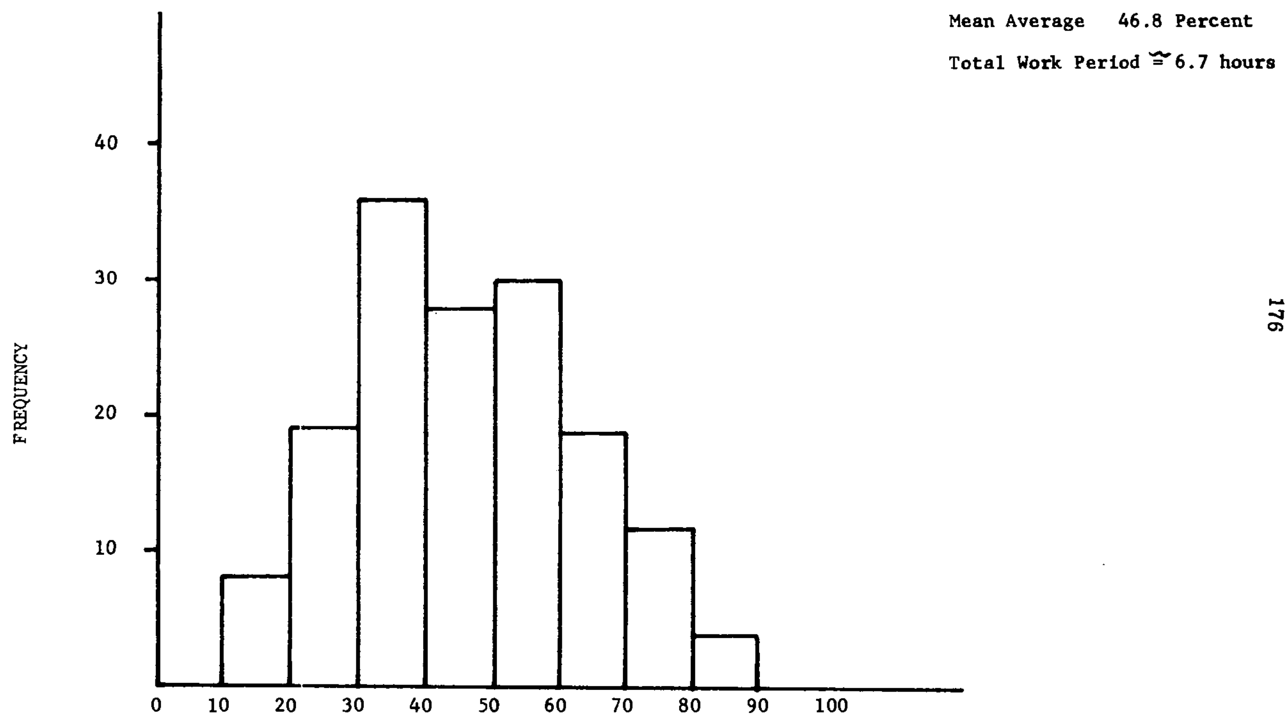
TABLE VI. Comparison of Different Respirator Types Tested

Respirator	No. of Test Subjects	Effective Protection Factor (EPF)	
Model		Mean Avg.	Median
A	8	4.9	3.4
B	11	5.0	2.6
C	11	6.8	3.7
D	6*	8.5	3.9
E	4	3.2	2.0

* Actually 3 different test subjects.

The TPF's obtained for the same five respirators is given in Table VII. While, in each case, the TPF obtained was substantially higher than the EPF, the TPF is still somewhat less than that theoretically possible based on filter efficiencies.

PORTION OF WORK PERIOD THAT RESPIRATORS WERE WORN



PERCENT OF TIME WORN

FIGURE 7

TABLE VII. Comparison of Different Respirator Models

Respirator	No. of Test Subjects	True Protection Factor (TPF) Mean Avg.	Median Avg.
A	9	11.1	10.6
B	9	9.2	8.9
C	9	11.4	10.8
D	9	8.7	8.4
E	9	8.7	8.5

(Penetration of filter media by particles 0.8 to 1.0 micron in size is usually less than 4 percent.) We feel this loss in respirator effectiveness is primarily caused by the lack of an ideal fit or seal between the facepiece and the subject's face. Not only is a proper face seal disturbed by facial and body movement, but undoubtedly the facial size and shape affects the seal obtained. This is indicated by the range of values obtained for the different test subjects, Table VIII, each of whom wore all the different respirators.

TABLE VIII. Range of TPF's

Test Subject	TPF* High	Low
1	12.3	6.6
2	13.5	5.8
3	14.2	4.0
4	11.4	8.2
5	15.4	7.2
6	19.5	8.3
7	10.2	6.6
8	11.9	9.6
9	10.0	5.9

* Average of four values obtained for one respirator worn.

It is probably this difficulty of easily achieving and maintaining a good fit and face seal that accounts for much of the difference between EPF and TPF. In this connection and in the case of EPF the fit problem becomes more difficult because miners find it impractical to wear a two-strap respirator head harness in the prescribed manner. Therefore, the miners wear the two straps in a single-strap configuration below the ears. The development of a more appropriate head harness is an area wherein research is much needed. In addition, there is need for better materials of construction and better designs which will provide both a more comfortable respirator and better face seal.

V. CONCLUSIONS

1. Half-mask dust respirators are in general use in underground coal mines and working miners feel there is a definite need for respiratory protective devices.

2. Most miners feel presently available approved respirators are acceptable for intermittent use but over a third of the miners feel the current units are unacceptable or marginally acceptable.

3. Discomfort to wear and breathing resistance are cited by miners as the major disadvantages of present day half-mask respirators.

4. As used in the field, presently available respirators provide the working miners a reasonable level of protection against the inhalation of respirable dust. However, the level of protection obtained is significantly lower than possible under ideal conditions. Difficulty in maintaining the proper seal between facepiece and face is one of the major reasons for reduced protection levels under actual working conditions.

5. There is a need for more comfortable respirators with reduced resistance to breathing. Likewise, there is a need for better materials of construction and better designs so that a good fit between facepiece and face can be secured and maintained with half-mask type respirators.

6. In the development of improved respiratory protective devices for coal miners a system approach should be used.

VI. FURTHER OBSERVATIONS

It is evident that improved respiratory protective devices for underground coal miners are needed. In the development and design of such devices, a systems approach should be used because of the need to integrate protection requirements of different kinds with the constraints of the work requirements and work environment. In the case of the coal miner, there is, as a minimum, a need, all or part of the time, for the following, each of which can be considered a system:

a. head protection
b. illumination
c. eye protection
d. noise protection
e. others, e.g., carrying of special equipment or tools

At the same time, the miner needs to have the maximum amount of mobility and the work environment often imposes severe constraints in terms of space and size and weight limitations. Consequently, it will be necessary, for example, when developing and designing improved respiratory equipment to take into account the miner's need for such things as head, eye, and noise protection and illumination, and to integrate these systems.

An example of the lack of systems approach are the present difficulties associated with respirators with the two-strap head harness and the miner's use of the hard hat (7). Clearly, such difficulties must be eliminated in newer and improved designs.

REFERENCES

1. Schutz, R. H., "The Dust Respirator Problem", Proceedings of the Symposium on Respirable Coal Mine Dust, Washington, D. C. November 3-4, 1969.

2. DeSieghardt, W. C., H. E. Harris, W. A. Burgess, and P. C. Reist, "Respirator Usage in Underground Bituminous Coal Mines", Mining Congress Journal, pp 43-48, August 1971.

3. IC 8503, "Sampling and Evaluating Respirable Coal Mine Dust," U. S. Bureau of Mines, February 1971.

4. Harris, H. E., W. C. DeSieghardt, W. A. Burgess and P. C. Reist, "Respirator Usage and Effectiveness in Bituminous Coal Mining Operations." American Industrial Hygiene Conference, San Francisco, Calif. May 18, 1972.

5. Lilienfeld, P. and J. Dulchinos, "Portable Instantaneous Mass Monitor for Coal Mine Dust,"AIHA Journal, Vol. 33, No. 3, pp 136-145 (1972)

6. Hyatt, C. C., A. L. Hack, T. O. Moore, and C. P. Richards, "LASL Respirator Test Panel - Representative of U. S. Male Facial Sizes.

7. Harris, H. E., W. A. Burgess, and W. C. DeSieghardt, "Effectiveness of Dust Respirators in Underground Coal Mines", AIME Annual Meeting, Feb. 25 - Mar. 2, 1973, Chicago, Ill.

EQUIPMENT MODIFICATION FOR INCREASED SAFETY

Donald G. Rogich

Chief, Division of Mine Systems Engineering
U. S. Department of the Interior
Bureau of Mines

INTRODUCTION

This paper is a progress report on the work performed to date under the Federal Bureau of Mines "Inherently Safe Mining Systems" contract.

The concept of initiating a large action-oriented contract, directed towards obtaining coal-mining systems with greatly improved safety characteristics, was conceived by the Bureau in January 1971.

The concept of working demonstrable systems, rather than just a conceptual study, was felt to be necessary for two significant reasons. In the development of any new system, the translation from concept to working hardware is usually a costly and time-consuming process that acts as a deterrent to implementation. Second, many decisions in the mining community to change equipment or systems are based on first-hand observation of the new system, in operational practice, an attitude which is certainly understandable in view of the high capital costs of equipment.

Based on a competitive procurement process, the Inherently Safe Mining Systems contract was awarded to the FMC Corp., San Jose, Calif., on June 18, 1971. The major goal of this contract is to develop and demonstrate safer mining equipment and procedures that can be developed using existing state-of-the-art technology.

This program is aimed at demonstrating improved safety of operations in the area from the working face to the first point of coal transfer in underground bituminous coal mines with seam heights of 4 to 8 feet, with emphasis placed on the prevention of fatal and nonfatal accidents by avoiding, warning, or protecting against the physical hazards of machinery and against falls of roof and rib. Long-term health problems and catastrophes (for example, explosions) are not being directly addressed, although procedures and equipment developed for this project will be evaluated to verify that they do not aggravate these problems.

The program is being conducted in three phases. Phase I, which has been completed, involved data accumulation and analysis leading to problem definition and the development of concepts to alleviate these problems. We are currently well into Phase II of the program, in which all the basic equipment elements are to be procured and modified and new equipment fabrications completed. Subsequent portions of this paper present the status of these major equipment developments. Phase III, the 1-year operational demonstration, is scheduled to begin in August of this year for the conventional section and later for the continuous section.

PROBLEM DEFINITION

A brief review of the magnitude of the safety problem associated with those activities within the scope of this program is of interest. This work in problem definition was performed during the first phase of the program and has been reported in detail in a report entitled "Accident Analysis by Functional Classification for Bituminous Coal Mines in 4-Foot to 8-Foot Seam Heights." This report was completed on

September 11, 1972, and has been placed with the National Technical Information Service.

Data on fatal and nonfatal accidents in underground bituminous coal mines operating in 4- to 8-foot seams was gathered to evaluate the problem and to assist designers in developing designs for equipment and procedures, which will reduce the incidence of these accidents. The data was structured in a form that would identify --

The types of accidents that are occurring

The causes of these accidents

The aspects of the job or equipment that contribute to these accidents

The nature and duration of the hazards to which workers are exposed.

The fatal accident data base consisted of 283 accidents resulting in 302 fatalities.

Nonfatal accident data was gathered in the field from company records on file at various mining company offices. The specific accident information sources have been masked to preserve their identities. Information on 65 conventional mining sections and 155 continuous-mining sections is contained in the data base. This sample represents 10,907,757 total annual man-hours in 4- to 8-foot seams, which is about 14% of the industry wide total annual labor hours in 4- to 8-foot seams.

Table 1 is an overall summary of the frequency and severity of nonfatal accidents, the incidence of fatal accidents, and the industry-wide expectation of lost man-days per year by system function.

Worthy of notice is the total of 220,500 estimated man-days lost each year as a result of compensable and noncompensable accidents. This is equivalent to a labor force of approximately 890 men, enough men to mine coal in 75 to 90 underground mining sections.

Table 2 provides information on the principle causes of injury for each of the major system functions.

Based on the results of accident studies and unit operation analysis, criteria for concepts to improve the safety of the conventional and continuous mining systems were developed. These criteria were made available to design personnel through accident study conclusions, operation analysis study conclusions, and recommended design approaches.

CONVENTIONAL MINING SYSTEM CONCEPTS

Work is progressing on the modification of equipment for use in the conventional mining section demonstration. The specific modifications planned or underway are described in the following sections. The major common features in these modifications are all-around protective suspended cabs and improved and standardized controls. The cab suspension feature, incorporated in a variety of unique ways on the section equipment, has two advantages: first, in the event of a major fall the cab can deflect in a controlled manner to enable the machine frame to pick

Table 1 INDUSTRY IMPACT — ACCIDENT SURVEY SUMMARY

Underground Bituminous Coal Mines -Inby The Face
Seam Height 4 - 8 Feet

	NON-FATAL								FATAL
	Compensible				Non-Compensible				
SYSTEM FUNCTIONS	One Accident for Every X Man-Years	Estimated Accidents per Year	Severity: Man-Days per Accident	Estimated Man-Days Lost per Year	One Accident for Every X Man-Years	Estimated Accidents per Year	Severity: Man-Days per Accident	Estimated Man-Days Lost Per Year	Five Year Total Accidents
4.0.0 Support Roof Roof Bolter	5.7	1114	53.7	60000	3.8	1621	1.6	2600	50
52.0.0 Load Coal Loader	6.1	293	70.4	21000	2.6	673	1.5	1000	47
1.0.0 Extract Coal Continous Miner	16.5	447	56.1	25000	10.9	671	1.7	1100	43
3.0.0 Move & Load Coal Shuttle Car	25.8	320	51.0	16000	11.7	799	1.8	1400	38
9.0.0 Maintenance All Maintenance	10.6	468	48.8	23000	3.1	1560	1.4	2200	20
51.1.0 Cut Face Cutter	12.9	158	115.3	18000	7.7	252	1.7	400	14
			Sub-Total	163000			Sub-Total	8700	212
All Other Functions (At Face)	-	952	48.4	46000	-	1842	1.5	2800	71
	Totals	3752	-	209000	Totals	7418	-	11500	283

Table 2 ACCIDENT MATRIX

Summary of Analysis
By System Function
And Source Of Injury

Note: Percentages will not add to 100 because only major sources are listed.

System Function (Job)	Location	Analysis Factors	TOTAL	Roof Falls	Face & Rib Falls	Falling and Flying Objects	Person Falling	Handling Material	Hand Tools	Stepping on Object	Striking or Bumping	Electricity	Machinery	Bursts - (Pressure)
1.0.0 EXTRACT COAL (Continuous Miner)	Worker at Controls	% of Accidents % Man-Days Lost Average Severity	52.1 55.4 24.8	17.1 26.4 35.9	6.2 2.9 10.9	7.5 4.2 13.1					4.1 4.7 26.8		14.4 13.2 21.4	
	Worker in Support	% of Accidents % Man-Days Lost Average Severity	48.0 44.6 21.7	3.4 6.1 41.4	4.1 8.1 45.8	4.1 13.4 75.7		11.6 6.2 12.4	2.1 0.2 1.7				15.1 9.0 13.9	
51.1.0 CUT FACE (Cutter)	Worker at Controls	% of Accidents % Man-Days Lost Average Severity	73.2 71.6 43.3	23.2 13.8 26.4	14.3 15.0 46.5	10.7 0.4 1.5					7.1 7.2 44.5		16.1 35.2 97.1	
	Worker in Support	% of Accidents % Man-Days Lost Average Severity	26.8 28.4 47.0	1.6 22.2 one acc.			10.7 2.9 12.0	5.4 2.9 24.3					8.9 0.4 2.0	
51.2.0 DRILL FACE (Drill)	Worker at Controls	% of Accidents % Man-Days Lost Average Severity	75.0 86.5 19.0	14.3 14.3 19.0	12.5 13.7 20.9	8.[illegible] 25.[illegible] 55.0							35.7 26.8 14.3	
	Worker in Support	% of Accidents % Man-Days Lost Average Severity	25.0 13.5 10.3			5.4 0.7 2.3	7.1 4.5 12.0						5.4 5.2 18.3	
52.0.0 LOAD COAL (Loader)	Worker at Controls	% of Accidents % Man-Days Lost Average Severity	89.0 77.6 19.4	16.9 23.0 30.4	8.1 2.2 6.0	35.3 10.4 6.5	3.7 2.3 14.2				2.9 0.6 4.3		18.4 37.3 45.3	1.5 1.8 26.5
	Worker in Support	% of Accidents % Man-Days Lost Average Severity	11.0 22.5 15.4					3.7 15.2 92.[illegible]					3.7 2.4 14.6	
3.0.0 MOVE & LOAD COAL (Shuttle Car)	Worker at Controls	% of Accidents % Man-Days Lost Average Severity	81.6 69.7 14.5	15.3 18.2 20.4	3.5 3.6 17.7	15.[illegible] 7.[illegible] 9.7					26.5 27.1 17.5		19.4 11.1 9.8	
	Worker in Support	% of Accidents % Man-Days Lost Average Severity	18.4 39.0 28.[illegible]				6.5 19.0 50.2	2.4 10.5 76.5				4.1 0.9 3.0		
4.0.0 SUPPORT ROOF (Roof Bolt)	Worker at Controls	% of Accidents % Man-Days Lost Average Severity	92.9 75.6 23.0	39.8 43.8 24.5	2.6 4.6 40.7	6.[illegible] 4.3 15.[illegible]	3.4 2.0 13.1	2.4 1.1 10.8	1.6 0.2 3.0	1.[illegible] 0.3 5.4			32.2 36.3 25.2	
	Worker in Support	% of Accidents % Man-Days Lost Average Severity	7.1 4.3 13.4	2.4 2.0 18.8									2.4 1.7 15.7	

Table 2 ACCIDENT MATRIX

Summary of Analysis
By System Function
And Source Of Injury

Note: Percentages will not add to 100 because only major sources are listed.

System Function (Job)	Location	Analysis Factors	TOTAL	Roof Falls	Face & Rib Falls	Falling and Flying Objects	Person Falling	Handling Material	Hand Tools	Stepping on Object	Striking or Bumping	Electricity	Machinery	Bursts - (Pressure)
51.3.0 SHOOT FACE (Explosive)	All	% of Accidents	100	15.8	21.1	21.1	10.5	10.5					10.5	
		% Man-Days Lost	100	2.3	59.0	14.2	3.0	16.1					5.2	
		Average Severity	38.5	5.7	107.4	26.0	11.0	59.0					19.0	
51.3.6 SHOOT FACE (Airdox)	All	% of Accidents	100	19.5		14.6	12.2						46.3	
		% Man-Days Lost	100	15.8		23.7	6.9						47.8	
		Average Severity	8.2	6.6		16.0	4.6						8.4	
4.90.0 SUPPORT ROOF (Posts, Timbers, etc.)	All	% of Accidents	100	9.8		13.7		51.0	15.7				3.9	
		% Man-Days Lost	100	5.9		6.2		39.0	5.4				40.4	
		Average Severity	7.3	4.4		3.3		5.6	2.5				75.5	
5.0.0 SAFETY	All	% of Accidents	100	75.9					10.3					
		% Man-Days Lost	100	95.0					1.6					
		Average Severity	38.2	47.9					6.0					
6.0.0 VENTILATION	All	% of Accidents	100		11.5			23.1			19.2		23.	
		% Man-Days Lost	100		19.0			11.9			57.2		5.	
		Average Severity	23.1		19.3			6.0			35.0		3.	
7.0.0 FIRE CONTROL (Rock Dust)	All	% of Accidents	100	14.3				11.9					4.8	
		% Man-Days Lost	100	43.6				8.3					7.6	
		Average Severity	11.9	58.4				13.4					30.5	
8.0.0 MATERIAL SUPPLY	All	% of Accidents	100			2.8	8.3	77.8			2.8			
		% Man-Days Lost	100			0.3	10.2	79.5			0.8			
		Average Severity	11.2			1.0	13.7	11.4			3.0			
9.0.0 MAINTENANCE	All	% of Accidents	100	6.5		7.6	5.0	4.3	23.7			7.6	40.3	
		% Man-Days Lost	100	19.5		1.8	11.6	2.6	10.1			5.5	27.0	
		Average Severity	12.1	36.4		3.0	28.4	7.4	5.5			8.9	8.3	
10.0.0 SUPERVISION	All	% of Accidents	100	20.0		17.9	17.8	6.7					17.8	
		% Man-Days Lost	100	50.3		38.7	4.5	0.8					4.7	
		Average Severity	29.6	74.3		64.3	7.5	3.7					7.9	
	All	% of Accidents												
		% Man-Days Lost												
		Average Severity												
	All	% of Accidents												
		% Man-Days Lost												
		Average Severity												
	All	% of Accidents												
		% Man-Days Lost												
		Average Severity												

up a portion of the load; and secondly, the elevation can be adjusted to obtain the maximum visibility allowable under existing seam height and roof conditions.

Controls on all section equipment have been examined from a human engineering control function and consistency viewpoint and similar motions and locations have been established to the maximum extent possible. This was considered to be of importance in view of the learning experience required when an operator is new to a piece of equipment or is called upon to operate temporarily a different equipment element. It has been shown that this learning experience is often attendent with a higher accident rate.

All section equipment with the exception of the DC shuttle cars will employ 950-volt power. This permits the utilization of shielded cable with safety grounds without any increase in cable size or wei ht.

SHUTTLE CAR

Two National Mine Service Lokar's have been procured and are being identically modified for this program. To a large degree, the modifications being performed identify what can be accomplished when sufficient space is available on the basic machine. The basic modification is a human engineered swing-around suspended cab. This cab will elevate or depress under power from a nominal 6-inch to a maximum of 16-inch ground clearance. In the event of a fall, the cab will deflect to the floor in a controlled fashion.

The operator is not required to leave the cab when the direction of shuttle car travel is reversed. This is accomplished by the release of an underseat latch, powered rotation of the cab about an axis near the operator's center of gravity, and latch reengagement when rotation is complete. The controls travel with the operator and are automatically reversed when the new position is reached. Steering by means of an automotive-type wheel is always consistent with the direction of travel. Tram and brake controls are also consistent with automotive practice and rotate with the cab.

Cab side clearance in the 90° rotated position is 31 inches beyond the outer edge of the existing machine structure. In the fore and aft position, cab side extension is 4 inches. This was required in the interest of additional operator comfort.

FACE DRILL

The face drill procured for this program was a Long Airdox model TDF-24C. Control modifications will consist of a reduction in the number of control valves by means of combined joystick controls. The drill frame swing and elevate controls will be combined as will the drill arm swing and elevate functions thus reducing the number of levers in the operators compartment. Additionally, automotive wheel-type steering will be utilized, the tram and brake controls will remain unchanged. An operators cab similar to that employed for the shuttle cars will be used with a 10-inch nominal adjustable height.

UNDERCUTTER

The cutting machine employed in this system will be a Joy model 15RU-6B. Joystick boom and bar controls and wheel steering similar to the face drill will be used. Since insufficient space for a suitable cab existed on the standard machine, it was modified by relocation of the hydraulic tram motor in order to provide leg room for the operator.

The machine will be equipped with forced air to the end of the cutter bar for slot ventilation and water spray nozzles for dust control.

LOADING MACHINE

The loading machine will be a Joy model 14BU10. Some of the special features of this machine are an oblique mounted operator's cab with limited reel cable storage and simplified cab mounted controls. The existing standard loading machine does not provide sufficient space for installation of a cab of adequate dimensions. The availability of this space to provide a comfortable cab is felt to be important since this is the one location where a worker can be protected, and everything possible should be done to make it desirable and practical for the operator to remain in the cab.

The cab, as configured, will elevate and depress about a horizontal pivot at the operator's feet. The additional side clearance was required because of oblique mounting of the cab to facilitate the operators view of the tail boom during the loading operation.

Cable reel storage for approximately 100 feet of cable will be provided in the rear of the operator's cab. The reel, which will be controlled by a cab-mounted, foot-operated valve, should ease the cable handling requirements.

AUTOMATED TEMPORARY ROOF SUPPORT BOLTER

A piece of equipment that will be built in two versions, one for use in the conventional section and one in the continuous section, is an automated roof bolter incorporating temporary roof support. The basic machine chassis will be a Galis model 3510. The key features of this machine will be an automated magazine supplied bolter that can be remotely operated by the worker from the protective cab position to bolt a 20-foot-wide place without local repositioning of the machine.

In the remote-control operational mode, the capability will exist to drill holes and install bolts with a length of from 5 to 6 feet, depending on the machine version. The machine can also be utilized with the operator working at the head end under the remotely actuated temporary support canopy. In this mode, the capability will exist to drill holes and install bolts of whatever length required.

For tramming, the bolts and drill steel are removed from the magazine and the protective canopy is then collapsed to a maximum height of approximately 3 feet.

CONTINUOUS-MINING SYSTEM CONCEPTS

The scheduled start date for the underground demonstration of the continuous-mining system is about 4 months later than for the conventional section. The equipment development is, therefore, not as far advanced as in that discussed for the conventional section.

The principle thrust for this equipment is to develop the capability to install a full bolting pattern automatically from the continuous-mining machine in order to obtain permanent support installation as rapidly as possible and allow the convenient utilization of continuous-belt haulage systems. The benefits of success in this development are apparent, but the problems encountered are also quite significant. The major obstacle is, of course, the space constraints. Existing continuous-mining machines are large and encumbered with considerable necessary machinery components, and the dimensions of the working space are, of course, limited. The on-board space available to accommodate equipment to drill and bolt a full three- or four-bolt pattern without restricting equipment maneuverability or effecting complete machine redesign is, therefore, extremely limited.

What has been developed through the preliminary design phase is illustrated by the sketch in figure 1. The basic machine to be employed is a Joy 12CM1-15B full-face miner capable of driving a 15-1/2-foot entry. Four on-board bolters are shown which would permit the installation of a four bolt pattern with a maximum center-to-center spacing of 4-1/2 feet. The bolting machine would employ a single drilling bolting

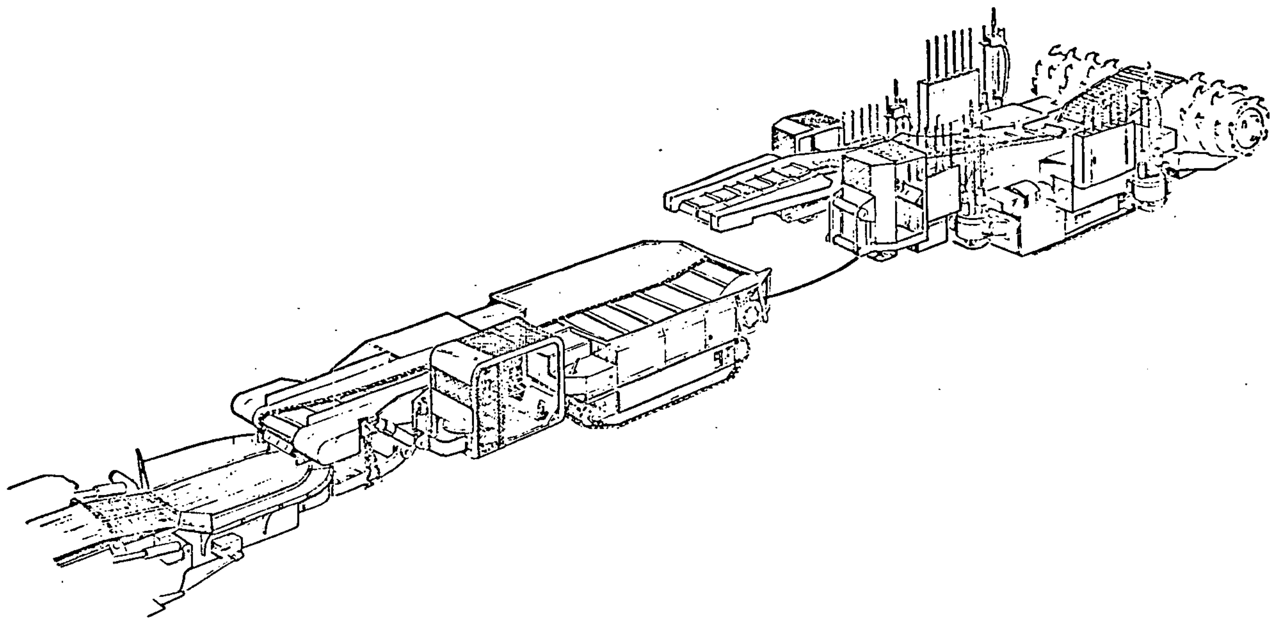


Figure 1
CONTINUOUS MINING SYSTEM

head with magazine storage and should be able to install a bolt approximately 2 feet less than the seam height.

In order to accommodate the rear bolters, the operators compartment has been removed, the conveyor extended, and the tailboom hinge pin moved forward. Two cabs have been added to the tailboom, one for the miner operator and one for the roof bolter. With the hinge pin relocation, the maneuverability of the machine is essentially the same as with the unmodified version.

Operation of the drilling bolting cycle is completely automated, and all four bolters are controlled from the operator's cab. All bolters are jack stabilized, and bolting should be possible during the shearing portion of the cycle. However, no slide-through capability is incorporated, and no drilling or bolting can be done during sumping. It is presently planned that the front bolters will have an eight-bolt magazine and the rear bolters, a four-bolt magazine. Reloading, which is a simple insertion process, would be required after about a 20-foot advance of the face. Bolting, in turn, is felt to be possible, although this will require some repositioning of the machine to accommodate bolt-spacing constraints. Some pick up bolting will be required when the machine is pulled out of a place since the forward bolts can be placed no closer than 12 feet from the face. This pick up bolting will be accomplished by means of a TRS bolter described previously.

The extensible belt haulage system to be used will be the Lee Norse system with one tram car and several drive storage units. Each drive storage unit will accommodate a maximum length of 150 feet of belt, which can be extended or retrieved at a rate of 140 feet per minute. System capacity is on the order of 12 tons per minute. Modifications to the extensible belt system include the addition of protective cabs or canopies to all units and revisions to the steering and tram controls.

ANCILLARY SYSTEMS

Given the time duration and extent of the demonstration phase of this program, it was considered desirable to integrate to the maximum extent all research and development results that could be incorporated in this phase. The additional items to be incorporated are those developed from the efforts of mining research, which is conducted both in-house and under contract programs other than the "Inherently Safe Mining Systems" contract. The following indicates some of the equipment and/or systems that will be incorporated in the areas of roof control, communications, illumination, dust control, and monitoring systems. The incorporation of these elements will include both demonstration, in those cases where the technology has already been proven, and test and evaluation where concepts are still under development.

The items under consideration for incorporation are as follows:

Roof Monitoring

1. Horizontal Roof Strain Indicator (HORSI) system
2. 3-in-1 roof deformation and failure warning device
3. U-type bolt tension indicator

Roof Monitoring (continued)

4. Micro seismic roof-fall warning system

5. Helix pressure gage

6. Hand held infrared scanner for detection of loose material

7. Mobile shield -- during experimental phases of system development in SRCM, Bruceton, and at ISMS mine, if applicable.

Monitoring, Communications, Lighting

1. Automatic monitoring of methane, air velocity, temperature, and CO

2. Advanced carrier phone systems and portable pocket paging equipment for section personnel

3. New lighting systems including wide-angle incandescent, mercury vapor, and, as development permits, polarized and/or flourescent for face area illumination and electroluminescent for machine identification. Hardware includes machine-mounted, portable, and personal units.

Dust Control and Monitoring

1. Application of foam in both coal cutting and handling

2. Air curtain respiratory device for personnel protection

3. Wet-drilling for automated roof drills

4. Continuous miner equipped with integral respirable dust scrubber and water sprays at cutter bits

5. Portable dust meters

6. Portable rock dust/coal dust analyzer, development permitting.

IN-MINE DEMONSTRATION

The precise demonstration start dates and location are currently under review and cannot be specifically identified at this time. As planned, the conventional section will be demonstrated in a seam of about 4 feet in height and the continuous section, in about 8 foot coal. Since both of these sections are to be operated for a years duration each, and numerous visitors are anticipated it is important that easy access to the sections be available both in a geographic and in-mine sense. Another important consideration is, of course, the cost of obtaining and operating these demonstration facilities over this time-frame desired.

A number of locations in West Virginia, Virginia, Illinois, and Kentucky have been examined and are currently being evaluated and it is anticipated that this question will be resolved shortly.