SCIENTIFIC PAPERS

OVERVIEW OF RESPIRABLE DUST CONTROL FOR UNDERGROUND COAL MINES IN THE UNITED STATES

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ABSTRACT

Control of respirable dust is an important consideration in the design of the production cycle of an underground coalmine. In order to create an effective and efficient system, the mining engineer must integrate the regulatory requirements with the specific conditions that exist in a coal mine. Typical mine development is by room and pillar. Second mining is by mining rooms, extracting pillars or by retreating longwalls. Each of the mining systems can have specific constraints depending on the type of equipment used. Continuous miners and conventional mining systems (cut, shoot and load) are used for room and pillar application. Single and double drum shearers primarily are used for retreating longwall systems. This paper provides a review of the specific federal regulations affecting dust control and description of the various dust control systems commonly used to supplement those regulations for the various mining systems.

INTRODUCTION

There are over 2,000 mechanized mining sections in underground coal mines in the United States. Each of these sections must utilize a dust control system capable of maintaining their dust levels below the specified standard.

The purpose of this paper is to provide an overview of the specific federal regulations affecting dust control and a description of various respirable dust control systems currently used in underground coal mines. Utilization of these systems has been successful in controlling workers exposure to coal mine dust.

FEDERAL REGULATIONS

Current authority to establish and enforce a respirable coal mine dust standard was given to the Mine Safety and Health Administration (MSHA) of the Department of Labor through the Federal Mine Safety and Health Act of 1977. Primary responsibility of enforcing the respirable dust standard rests at the federal level as state laws generally do not specify a respirable dust standard. Specific regulations pertaining to the dust standard and dust control are contained in Title 30, Code of Federal Regulations.

Part 70—Mandatory Health Standards—Underground Coal Mines, contains the dust standards and the sampling procedures that must be followed by the coal mine operators. Part 70 establishes a respirable coal mine dust exposure standard of 2.0 milligrams per cubic meter (mg/m^3. If the dust contains more than five percent quartz, the dust standard is computed by dividing the percentage quartz into the number 10. Additionally, Part 70 establishes a dust standard for intake air of 1.0 mg/m^3. Part 70 also requires mine operators to collect and submit five dust samples from a designated occupation during each bimonthly sampling period.

Part 75—Mandatory Safety Standards—Underground Coal Mines, contains various ventilation regulations that pertain to the control of respirable coal mine dust. Part 75 contains various regulations pertaining to the design and performance of a mine's ventilation system which also have an impact on dust control. Specifically, each mechanized mining unit must be ventilated on a separate split of intake air. This prohibits series ventilation of working sections so that the return of one section cannot be used to ventilate another section.

To provide dilution, the ventilation system must deliver 9,000 cubic feet of air per minute (cfm) to the last open cross cut of a set of developing entries and to the intake entries of a retreating section. The system must also supply 3,000 cfm to each working face where coal is being cut, mined or loaded.

Unless otherwise approved by the local enforcement official, the line brattice or face ventilation device must be maintained within 10 feet of the face. For exhausting face ventilation systems, the minimum mean entry air velocity in working places where coal is being cut, mined or loaded is 60 feet per minute (fpm).

Each coal mine operator must also submit for approval a ventilation system and methane and dust control plan. The plan must show in detail the methane and dust control practices along all haulageways and travelways, at all transfer points, at underground crushers and dumps, in all active working places and in any other areas which may be required by MSHA's local enforcement official.

Prior to approval, dust samples are collected by inspection personnel to verify system performance. The dust control plan concept was developed to provide flexibility, yet ensure that appropriate measures were being taken to control respirable dust. The following discussions provide more information on specific dust control systems used for various mining systems.

DUST CONTROL ON CONTINUOUS MINER SECTIONS

Approximately two-thirds of the mining sections in the United States utilize continuous mining machines. Continuous miners are used to both develop and retreat room and pillar mining sections. Dust generated on a drum type continuous miner is controlled by two primary means, ventilation and water. The two basic types of face ventilation are exhausting and blowing. In an exhausting ventilation system, air is brought to the face at a lower velocity, captures the dust cloud and then extracts it from the face at a higher velocity. For a blowing face ventilation system the return air passes over the mining machine. This situation necessitates the use of additional controls such as machine mounted dust collectors (scrubbers) to maintain adequate dust control.

Water sprays are used in addition to ventilation to suppress and direct the dust cloud generated at the face. Typical suppression sprays are mounted on the miner as close to the cutting drum and gathering arms as possible. These systems are designed to deliver water to strategic dusty locations around the machine. Directional sprays (spray fan systems) are mounted on the body of the miner up to 10 to 15 feet from the face. These sprays are designed to use the momentum of the water to direct the dust cloud away from the machine operator. Spray fan systems are normally used in conjunction with exhaust line brattice.

Each continuous mining section utilizes one or more roof bolters to install roof support in the entries mined. Dust control on roof bolters is especially important because the drilled strata can contain high levels of quartz. The two primary methods of controlling dust generated during roof bolting operations are through proper use and maintenance of the machine dust collection system and proper ventilation of the working place.

DUST CONTROL ON CONVENTIONAL MINING SECTIONS

In a conventional mining system the coal is extracted in a series of operations each performed in proper sequence. The operations in a conventional mining system are: cutting, drilling, blasting, loading and hauling. Each operation in the cycle employs a specialized piece of equipment to perform that operation.

The cutting operation is performed with a mobile cutting machine which most nearly resembles a large chain saw on wheels. Dust from the cutting operation is controlled by the use of a "wet" cutter bar and external water sprays mounted above the cutter bar as well as proper ventilation. The wet cutter bar is made by plumbing a water pipe inside the cutter bar which terminates in a small opening at the end of the bar. The movement of the cutting chain around the bar distributes the water along the length of the cut. External water sprays should be directed towards the ingoing and outgoing bits and also toward the pile of cuttings being deposited on the mine floor.

The drilling operation employs a mobile drilling machine with a single movable drill capable of drilling to the same depth as the cutting machine. The number of holes drilled depends on the height of the coal seam, width of the face, hardness of the coal and the desired size of the coal lumps. The period of highest dust concentration is when the drill is first sumped into the coal. Once the drill has penetrated the coal, the hole itself helps contain the dust. The use of a wet auger (drill steel) is the preferable method of controlling dust on a coal drill. Water is directed through the hollow auger to the bit and is then forced out of the hole after it has mixed with the cuttings and dust. The coal cuttings and dust are thoroughly wet and come out of the hole in the form of a slurry, thus producing very little dust.

Blasting is done chiefly with permissible explosives. An explosive charge is placed in each hole and then stemmed with an inert material (either water or clay dummies). The charges are wired together and then detonated. The rapid release of energy by the explosives breaks the coal and also generates a large amount of dust. However, the dust is rapidly dissipated if the face is properly ventilated. If the blasting is done on the return air side of the other mining operation, then personnel will not be exposed to the dust generated by blasting. The next operation is the loading of the coal by either a loading machine or a scoop. Loading machines have mechanical gathering arms which pull the coal onto a chain conveyor located along the centerline of the machine. The movement of the gathering arms and chain conveyor produces dust. This dust is controlled by the face ventilation system and by external water sprays mounted on the body of the loading machine. Prior to loading, the coal pile should be thoroughly wetted. Wetting the coal pile is particularly important since subsequent loading of the coal is done with scoops that are not equipped with water spray systems.

DUST CONTROL ON LONGWALL MINING SECTIONS

In general longwall mining systems in the United States use single or double drum shearers to retreat mine a block of coal. Longwall faces range from 400 to 1,000 feet wide with total panel length often in excess of 4,000 feet. There are approximately 100 operating longwalls which produce approximately 15 percent of the underground coal mined. Normally seven people are required to operate the longwall face equipment.

When identifying and attempting to control a longwall system's dust source(s), the longwall can be divided into three primary sources of dust generation. These sources are the machinery in the headgate area, the shearer and the shields.

The dust generated in the headgate area affects personnel on the entire longwall face since it contaminates the intake air before it traverses the face. The headgate sources are the stageloader, crusher and product transfer points. The common practice employed for dust control is to enclose the stageloader and crusher on the sides and top and to install flat jet water sprays across the product inlet and outlet. To assist the water sprays in creating a tighter enclosure on the product inlet and outlet, a strip of mine conveyor belting or brattice is installed on both ends. Usually flat jet water sprays are located in the crusher and along the length of the stageloader. To control dust at transfer points, various types of water sprays are used.

The shearer's primary dust source is the cutting of the coal by the bits on the drum(s). To combat this dust source, four control methods are normally used. The four dust control methods are: internal water sprays, external water sprays, remote control and work practices.

Internal water sprays are the water sprays in/on the shearer cutting drum. The internal sprays are used to suppress the dust at the source and provide a cooling effect for the cutting bits. The number of sprays range from 25 to 45 with the orifice ranging from 1/8 to 3/16-inch. The operating water pressure measured at the spray nozzle ranges from 40 to 100 pounds per square inch (psi).

The external water sprays are the water sprays located on the shearer body or on any attached bar and/or arm. The best practice is to use these sprays to direct the dust laden air over the shearer body so that the shearer operator is maintained in a clean split of intake air not contaminated by the dust generated by the shearer. The operating water pressure measured at the spray nozzle ranges from 40 to 120 psi. To assist the external water sprays in directing the dust, passive barriers (usually made of mine conveyor belting) are sometimes attached to the shearer body, bars and/or arms.

A remote control unit(s) is a device that allows the shearer operator(s) to control the shearer from various locations. It is used to remove the shearer operator(s) from the dust being generated by the shearer. Radio control or umbilical cord are the two types of remote control units available. Radio control is more versatile but not as durable as an umbilical cord unit. Approximately 50 percent of the shearers are equipped with a remote control system.

Administratively controlled work practices are also used on longwalls to lower the dust exposure of personnel. The most common work practice employed to lower exposure is to reduce the amount of time personnel spend on the face. This is accomplished by having personnel move to the upwind side of the shearer after they have completed their primary tasks. Also changing the cutting sequence of the shearer can reduce the exposure of face personnel. A common practice employed is to cut unidirectional, cutting two-thirds of the face height in one direction and cutting the remaining one-third coming back. The shields (roof supports) are then pulled on the upwind side of the shearer. This practice keeps the shield setters out of the dust that is created by the shearer. However, the shearer operators are exposed to the dust generated by the shields. Bidirectional cutting, cutting fullface height in both directions, exposes shield setters to the dust generated by the shearer for half the mining cycle and the shearer operators to the shield dust for half a mining cycle.

The movement of the shield top creates a dust problem because the crushed and ground material on top of the shield falls. The severity of the dust problem will vary depending on the amount of this falling material. The dust problem can range from negligible to very severe. To circumvent this problem, the industry is phasing in electrohydraulic shields. The electrohydraulic shields have controls connected to a computer on the shields that allow a set of shields (1 to 15) to be electronically controlled. This allows shield setters to achieve an upwind position from this dust source.

SUMMARY

Prior to the 1969 Act respirable dust levels of 9 mg/m^3 were commonly reported. Today the industry average exposure for the designated occupation is approximately 1.0 mg/m^3. These dust levels have been mainly achieved through the application of the various dust control methods previously discussed which include:

1. A supply of uncontaminated intake air.
2. Suppression through the use of machine cutting head design and water.
3. Containment through the use of properly designed and maintained face ventilation systems, water sprays or barriers.
4. Dilution from an adequate supply of fresh air.
5. Avoidance through the use of remotely operated cutting and loading machines.
6. Administratively controlled work practices.

With continued application of these techniques, respirable dust levels can be maintained at acceptable limits.

EXTRACTION DRUMS AND AIR CURTAINS FOR INTEGRATED CONTROL OF DUST AND METHANE ON MINING MACHINES

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INTRODUCTION

Over the past 30 years British Coal has expended a considerable amount of research effort on solving the problems of environmental control at and around the production machines in coal mines. That effort has borne much fruit in the field of respirable dust control, with the levels of pneumoconiosis falling from over 10% of the workforce in 1970 to the current level of 0.9% for mineworkers of all ages. This improvement has been achieved despite a doubling of productivity at the coalface. However, the current rapid rise in output demands even more efficient dust control systems for the future.

The other major environmental hazard at the production machine is the frictional ignition of methane, caused by cutting tools striking quartzitic or pyritic strata in the presence of explosive mixtures of methane. There has been little reduction in the incidence of frictional ignitions over the last 20 years, with an average of 14 ignitions reported each year,[1] despite improvements in the ventilation of the cutting zone to dilute dangerous concentrations of methane and the more recent use of water sprays to cool the ignition source.

There is often a conflict between the requirements for good dust control and those for effective dilution of methane in the cutting zone or dispersal of methane layers in the roof of drivages. Excessive amounts of dust are often dispersed by the high air velocities blown into the cutting zone or roof area to get rid of methane. On longwall shearers the hollow-shaft ventilator does this job,[2] while in drivages where exhaust ventilation is used to control dust, machine-mounted fans can be fitted to disperse methane layers. The high air velocities these fans produce can result in roll-back (or back-up) of dust to the operator's position on the machine.

To overcome the conflicting requirements for dust and methane control, and also to provide the improvements needed to ensure that the vital productivity increases being gained by British Coal are not jeopardized by dust sanctions or increasing numbers of ignitions, two new control technologies have been developed at Headquarters Technical Department, the Extraction Drum for longwall shearers, and Air Curtains for use in exhaust ventilated drivages.

DUST EXTRACTION ON SHEARERS

Numerous attempts have been made in various countries to provide effective dust extraction systems on shearers, using fans and dust collectors. All failed, because of problems with blockage of ducting by coarse material, or the large size of equipment needed to supply adequate extracted airflows. However, work on small, water-powered dust capture tubes in the early 1970's[10] led to the development in the UK of effective dust extraction systems for use on shearers with cutting drums well shielded from face ventilation.[8] In these systems the non-blocking, open-ended tubes were integrated with the coal loading doors or cowls around the cutting zone. Efficient dust control on ranging-drum shearers and those with unshielded drums was not possible until the concept of the extraction drum was devised in 1981.[7]

Description of Extraction Drum

The extraction drum was developed after laboratory tests showed that the best place to extract dust was from the face side of the drum. A number of dust capture tubes are built into the drum barrel, with the tube inlets at the face side remote from face ventilation. Dusty air is drawn from the cutting zone, cleaned by the tubes, and blown out at the goaf side, from where it is turned back into the cutting zone, together with the water spray and debris, by an angled deflector plate fitted to the gearhead. Figure 1 shows a version commonly used on medium-sized drums. It has nine, 100 mm diameter, tubes which extract 1.5 m^3/s of air using 60 l/min of water, released from hollow-cone, wear-resistant spray nozzles at a pressure of 100 bar. Even though up to 70% of the air is recirculated, nearly 0.5 m^3/s of fresh air is provided to dilute methane. On smaller drums rectangular section tubes are used to minimize drum diameter, while up to twelve 100 mm tubes have been fitted to drums above 1.5 m in diameter in order to maintain air velocities across the cutting zone.

High pressure water is fed through the drum shaft to nozzles on the face-side spray ring by a dual pressure water distribution system, which also delivers up to 45 l/min of water, at approximately 7 bar pressure, to sprays on the drum to wet the cut coal before it is loaded out. It is essential to

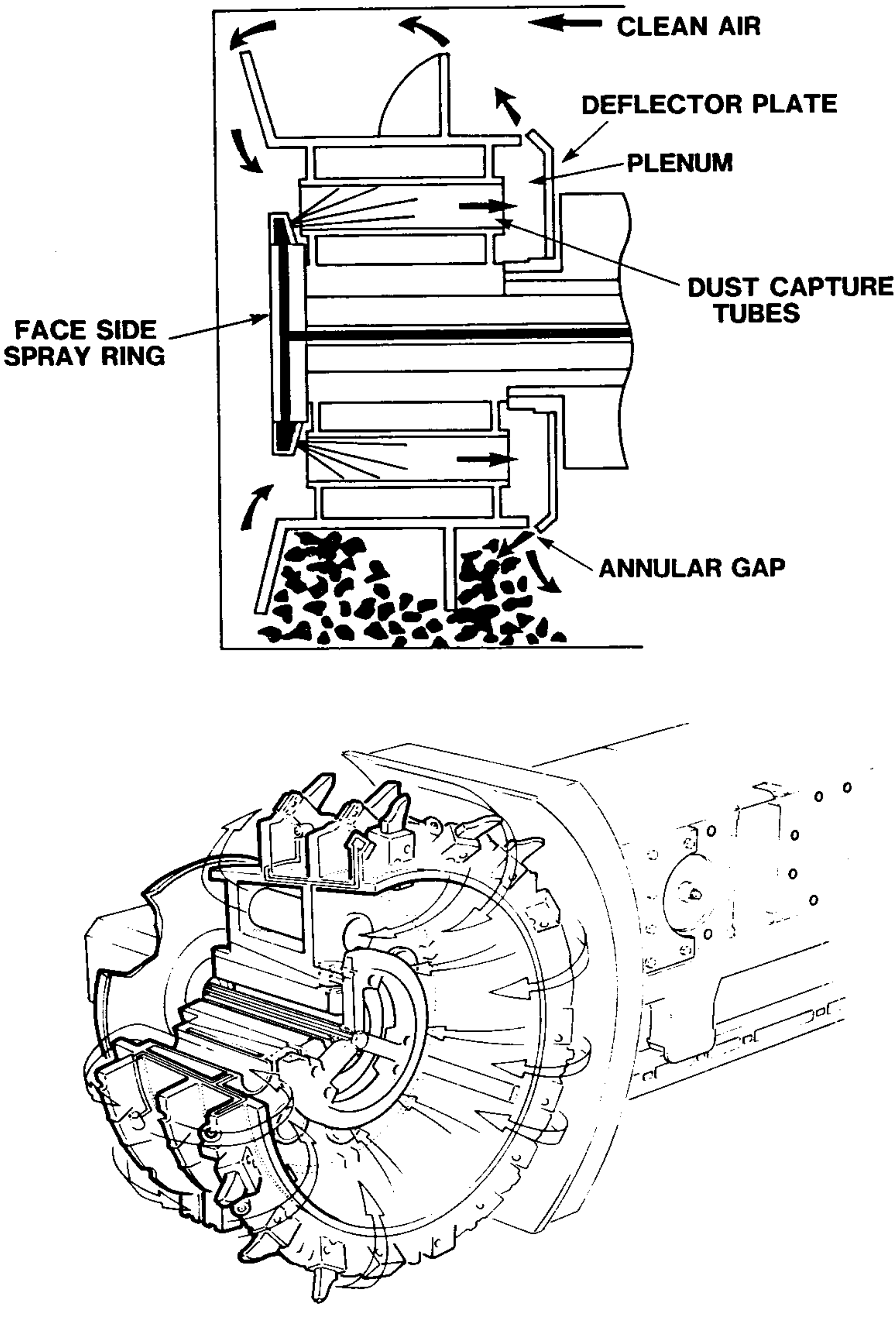

Figure 1. Schematic view and cross-section of a 9-tube extraction drum.

operate the tubes at high water pressures to provide high airflows and freedom from blockage, together with a respirable dust capture efficiency exceeding 95%.[5]

Operational problems experienced with the extraction drum have primarily resulted from inadequacies in the water supply. It is therefore essential to install water pumps with sufficient capacity, together with the correct water control and monitoring equipment. The high pressure water pump can be sited in the roadway at the end of the face, or integrated with the shearer. These pumps are expensive (for a double-drum machine, £32,000 for a roadway pump and £14,000 for a shearer-mounted pump) and represent the major cost for a system. The additional cost of fitting the two extraction drums, water distribution equipment, and deflector plates is only about £5,500.

Dust Control Efficiency

Results from underground trials on a range of shearers, see Table I, have indicated dust reductions during cutting operations of between 40% and 80% when extraction drums replaced drums incorporating the normal pick-face-flushing water spray systems. In most cases water flows were similar for each system. At one site in the UK, installation of extraction drums enabled output to be raised from 1000 to 1600 tonnes per shift without exceeding the statutory dust limits. This result shows the size of the benefits to be gained from the use of the extraction drum.

Results quoted are for dust levels in the face return air. Evidence from the USA[11] indicates somewhat less improvement at the operator's position, possibly due to the effect of the high air velocities leaving the exit annulus of the drum.

Effect on Methane Dilution

Extensive surfaces and underground trials[3] have shown that at least 30% of the air drawn in by the extraction drum is fresh air which dilutes methane in the cutting zone. Thus, for a nine-tube drum approximately 0.5 m^3/s of fresh air is provided for methane dilution, which is more than twice the airflow given by the hollow-shaft ventilator normally used at ignition risk sites. In laboratory tests on a shearer in an artificial coalface,[4] a hollow-shaft ventilator prevented frictional ignitions up to a methane emission rate of 5.5 l/s. Using the extraction drum ignitions did not occur until methane emission reached 15 l/s, which is above the emission rate on most UK coalfaces.

Measurements taken during the underground trials,[3] of methane emission rates at the shearer and methane concentrations in the cutting zone, confirmed the superiority of the extraction drum for ventilation. Consequently, British Coal now considers the extraction drum to be the best device for methane dilution, and is installing them at a number of sites primarily for ignition control. In such cases, attempts are being made to continuously monitor the extracted airflow by measuring the air pressure developed across the outlet annulus between the edge of the drum barrel and the deflector plate.[5] Alarms are activated when the airflow falls below a preset level. Systems have been fitted to a number of machines, and development is continuing to improve their reliability.

Utilization of Extraction Drums

Since 1985, when 15 drums were in use, there has been a rapid increase in numbers, with more than 85 drums in operation on some 20% of faces in the UK. In addition, drums

Table I

Reduction in Dust Produced During Cutting with Extraction Drums as Compared to Normal Water Sprays

Machine Type	Drum Diameter m	Number of Tubes	Face Air Flow m^3/s	Reduction in Dust %
Fixed Height	1.3	9	12	80
	1.5	9	15	78
	1.8	9	5	62
Single-Ended Ranging Drum	1.4	9	14	40
	1.4	9	18	72
Double-Ended Ranging Drum	1.5	10	18	53
	1.7	10	12	60
	1.8	12	13	55

have been installed both in the USA and Australia. The drums have been fitted to most types of shearer, operating on faces ranging from 1.07 m to 3.0 m in height.

HQTD have produced a comprehensive training package on the extraction drum system to aid the transfer of this technology to the collieries. This includes interactive video to cover the fault-finding and maintenance aspects.

Future developments include the use of higher water pressure to increase efficiency.

AIR CURTAINS FOR DRIVAGE MACHINES

Exhaust ventilation gives effective dust control in drivates, providing the exhaust duct entry is kept in front of the machine operator and a forward air velocity of 0.5 m/s is maintained around the machine. In practice, these requirements are often not met, and even when they are, exhaust ventilation alone cannot provide high enough air velocities to disperse methane. The air curtain system was developed to generate these velocities without dispersing dust, and also to increase dust control efficiency at sites where the ventilation criteria for preventing dust back-up were not being met.[6]

Air Curtain System

The air curtain system directs 'sheets' of fast moving air forward from the top and side of the machine body into zones of the drivage where air velocities are low and dust therefore backs up, as illustrated in Figure 2. An additional tube is usually fitted above the machine's conveyor to prevent dust from being pulled back to the operator's position by the outgoing debris. Air curtains are produced from 100 mm diameter steel tubes, fitted with cover plates from which the air is released tangentially to the tube surface through 2.5 mm deep slots running the length of the tube. The 'Coanda Effect' causes the discharged air to cling to the tube surface until directed off in the required direction by a 'splitter' bar on the tube, as shown in the tube cross-section illustrated in Figure 3.

Air is fed to the tubes at pressures of between 0.75 and 2.0 kPa by a small centrifugal fan powered from the machine's hydraulic supply at a flow of 40 l/min. The total airflow to a system depends on the length of air curtain tube used. It ranges from 0.15 m^3/s on a small boom-type machine, like the Dosco 2A, to about 0.30 m^3/s on a continuous miner, such as the BJD/Dresser Heliminer.

On some machines the exhaust duct can be installed on either side of the heading, whilst on others, aircooled motors are fitted which draw dust back beneath the exhaust duct. For such cases, tubes are sited on both sides of the machine, and the air pressures in the tubes are balanced to provide the correct flow of air around the front of the drivage towards the duct inlet.

At present systems are available for ten different boom-type machines, and three continuous miners, with equipment for a further two of the latter soon to follow. Figure 4 shows a typical system, fitted on a Dosco LH1300 machine. Prices range from £4,500 to £7,500 for a complete system, dependent upon the number and complexity of the air curtain tubes.

Airflow monitoring systems are currently under development to ensure that adequate airflows are provided for methane dispersal whenever the machine starts to cut. A new technique is at present also under development as an addition to the air curtains, to give integrated ventilation of the cutting zone for continuous miners. Air for this system would be taken from the same fan as the air curtain, and it is hoped that use of both systems will provide effective ventilation of the cutting zone and the roof, whilst maintaining effective dust control.

Dust Control Benefits

Underground trials[6] have shown that the air curtains significantly reduce dust back-up, see Figure 5. Over the range of forward airspeeds and duct entry positions used, the proportion of dust from cutting that reached the operator was reduced by at least 70% when the air curtains were

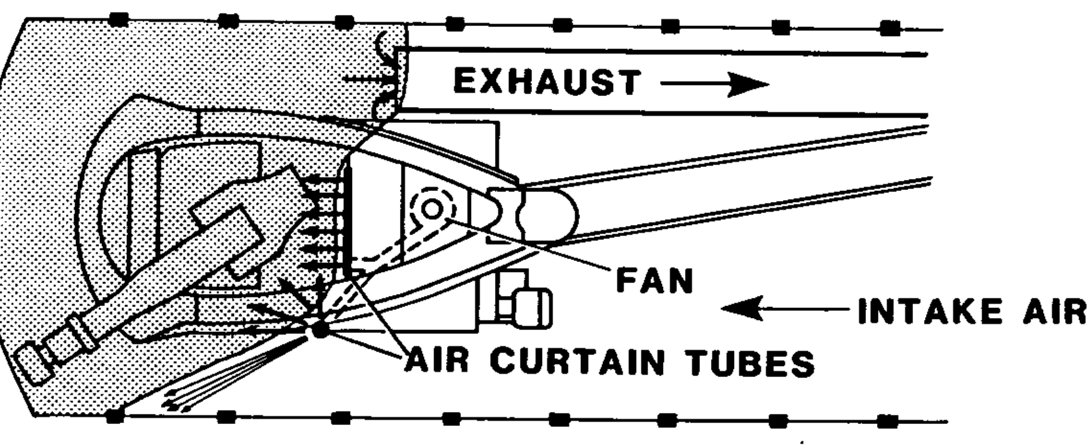

Figure 2. Plan view of drivage showing air curtains containing dust cloud.

switched on. The visual improvements when using air curtains are dramatic on most types of machines, and operators are loath to work with them turned off. Time lost in waiting for dust to clear is reduced, with consequent improvement in production. As a consequence, to date more than 80 systems have been installed.

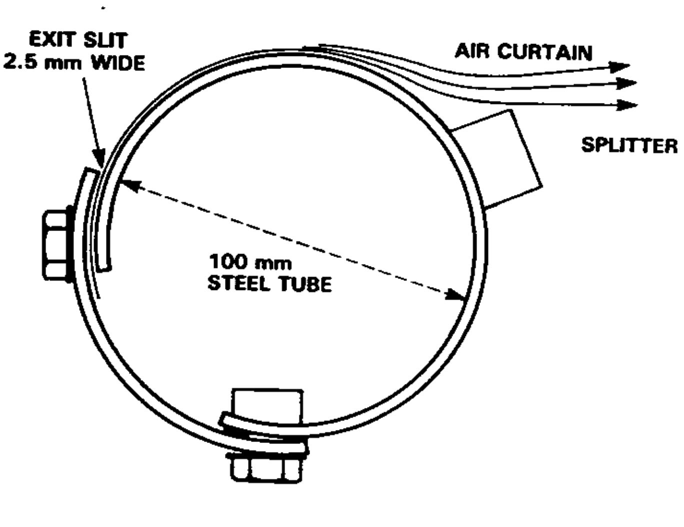

Figure 3. Cross-sectional view of air curtain tube.

Effectiveness of Methane Layer Dispersal

Full-scale laboratory tests were carried out to ascertain the air velocity profiles around Dosco 2A and LH1300 machines in an arched section drivage.[9] These tests showed that the air curtains directed air into the roof area at velocities well above the 1 m/s required for the dispersal of methane layers. In addition, they were just as effective as a machine-mounted blower fan for removing the 'dead zones' present at the front of the drivage when exhaust ventilation was used alone. Underground evaluation confirmed these results. It is now British Coal policy to fit air curtains to all drivage machines.

CONCLUSIONS

The extraction drum and air curtain systems both provide effective control of dust and methane on longwall coalfaces and in exhaust ventilated drivages respectively. Each system can be easily integrated with mining machines without detriment to operational performance, and offer solutions to the problems of environmental control on high performance coalfaces and in rapidly advancing drivages.

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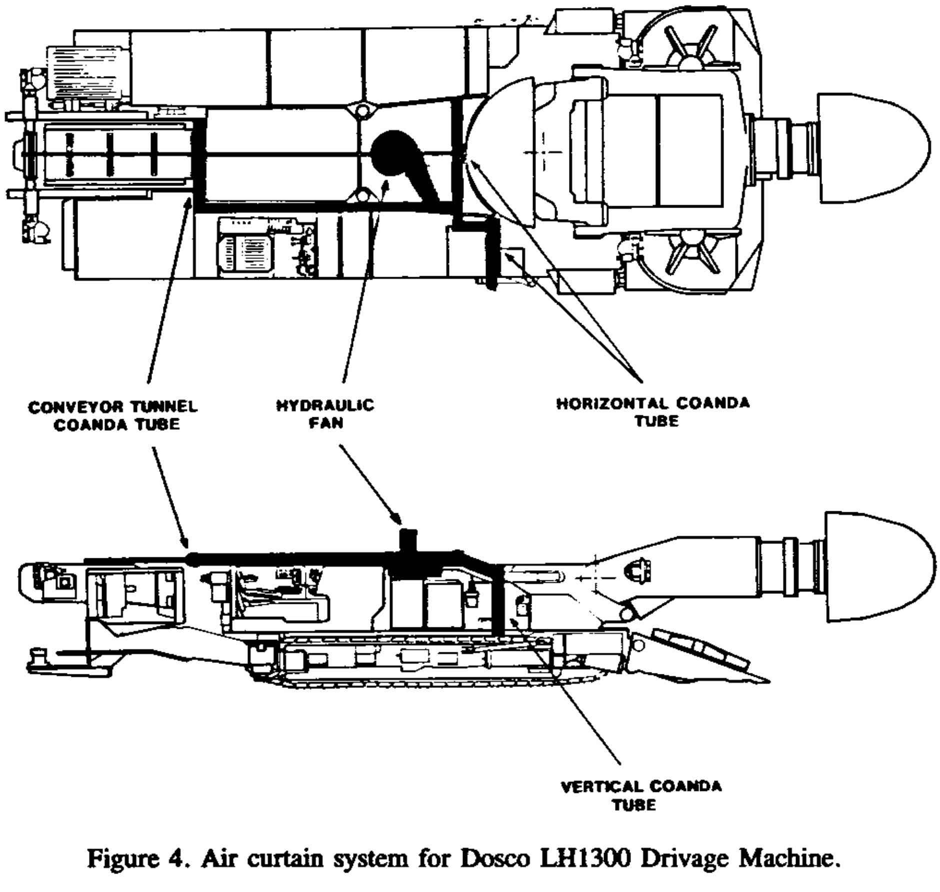

Figure 4. Air curtain system for Dosco LH1300 Drivage Machine.

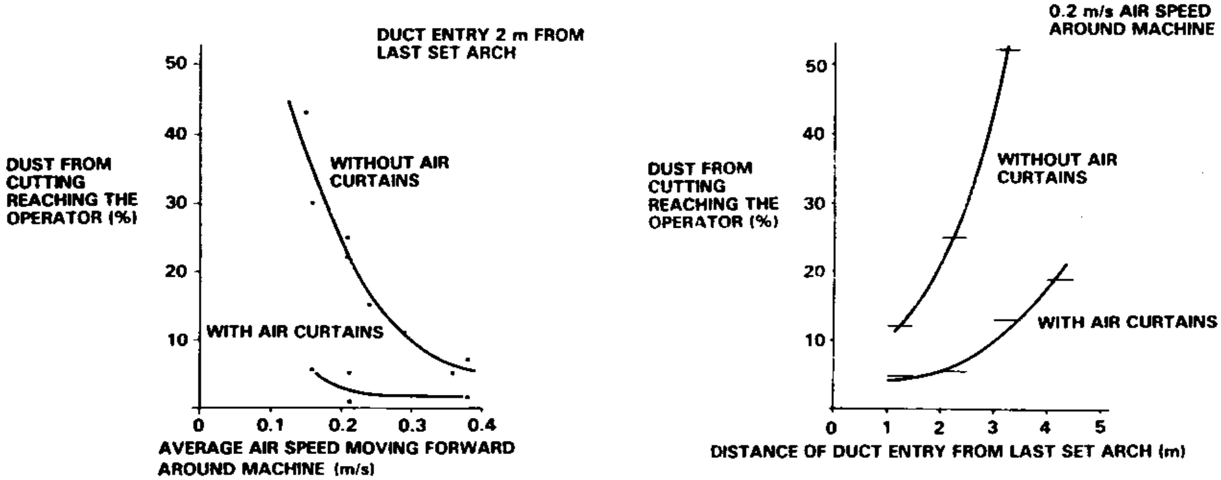

Figure 5. Underground results with air curtains on Dosco MK2A Machine.

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INCREASING COAL OUTPUT WILL REQUIRE BETTER DUST CONTROL

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BACKGROUND

In 1969, the Federal Coal Mine Health and Safety Act (FCMHSA) was passed for the purpose of reducing the incidence of Coal Workers' Pneumoconiosis (CWP), or black lung, a chronic lung disease caused by coal dust inhalation. The FCMHSA limited the average exposure of coal miners over an eight hour working shift to 3.0 mg/m^3 (milligrams of respirable dust per cubic meter of air); this maximum dust level was reduced to 2.0 mg/m^3 in late 1972, effective in 1973. Additionally, in order to reduce the incidence of silicosis, a lung disease caused by the inhalation of silica dust, the FCMHSA requires that the Mine Safety and Health Administration (MSHA) enforce a more stringent standard if dust samples contain silica in excess of 5.0 percent. (Dust standard = 10/(percent SiO_2 in sample); the standard is less than 2.0 mg/m^3 if the silica content of the sample exceeds 5.0 percent.)

The annual costs of the black lung program, which include compensation payments to retired miners or their survivors and the program costs of the Departments of Labor and Health & Human Services, have leveled off in the \$1.6–1.7 billion range since 1979. The cumulative cost of the program from 1970 through 1985 is estimated at \$18.4 billion.[2,5] In constant 1970 dollars using the Consumer Price Index (CPI) to adjust for inflation, however, the cumulative cost of the program was \$10.0 billion, and annual costs have declined every year since 1979, from \$834 million to \$585 million in 1985.

Due to the time lag between initial exposure of miners to respirable coal dust and the filing of black lung claims, sometimes as long as 25–30 years, it is likely that future compensation payments will decline, if compliance with the standard is maintained, as miners who worked in dustier conditions prior to passage of the FCMHSA leave the compensation rolls. Based on a British study predicting the incidence and progression of CWP over a ten year period as a function of mean dust concentration and assuming compliance with the 2.0 mg/m^3 dust standard, Attfield forecasted the future incidence of CWP Category 1, a less debilitating form of the disease, to be about 9 percent of the underground work force and the incidence of CWP Category 2/Progressive Massive Fibrosis, a disabling form of the disease, at 1–2 percent.[1,10]

Throughout the remainder of this analysis, it is accepted as given that there is a direct relationship between lower dust levels and reduced worker morbidity and mortality. Therefore, this paper evaluates the relationship between dust control and mine worker health indirectly through its impact on mine dust levels rather than directly on incidence of dust related disease.

UNDERGROUND COAL MINING METHODS

The three major underground mining methods employed by the domestic coal industry are conventional, continuous, and longwall mining. Since conventional mining currently accounts for only 11.7 percent of underground coal production and is predicted to decline to 4.2 percent by 1995 it will not be further considered in this analysis.[3,8,11,17]

Longwall mining is more productive than continuous mining and generates more coal dust.[12,13] The silica dust problem, however, is currently almost entirely restricted to continuous mining due to the cutting pattern used in this mining method.

DUST LEVELS AND COMPLIANCE

Due to improvements in dust control technology, average dust levels of continuous and longwall mining sections are currently at or below the required dust levels (Figure 1). These data are average values, implying that not all mines operate in compliance with the dust standard. This is evident when the standard deviations of these average data are examined (Table I). Furthermore, compliance data indicate that the problem is far from having been solved—through May 1987, 70 percent of longwall sections were in compliance and only 59 percent of continuous mining sections could comply with more stringent dust standards due to the presence of silica in excess of 5 percent (Figure 2). As an example of the remaining problem, several U.S. longwall mining sections having the highest output per shift recorded an average dust exposure value of 3.8 mg/m^3, more than two standard deviations above the longwall average.[15]

The costs to the underground coal mining industry of the decline in the average dust level fall into two categories: (1) direct costs, and (2) opportunity (i.e., lost production) costs. In fiscal year 1986, for example, mine operators submitted 83,985 samples at a cost of \$10.3 million.[14] The General

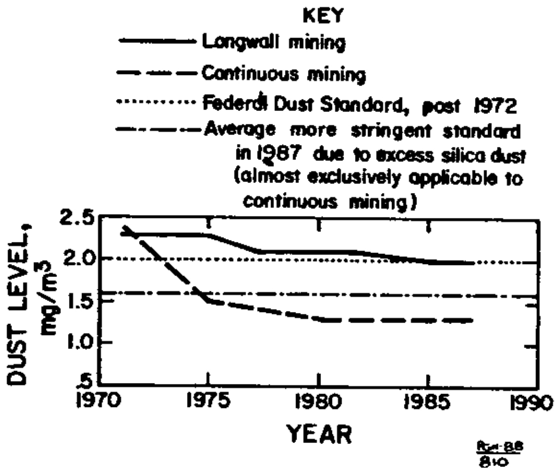

Figure 1. Average dust levels of operator samples from selected underground mining methods.

Table I

Dust Levels, by Underground Mining Method (mg/m^3)

Year	Continuous Mining		Longwall Mining	
	Ave.	Std. Dev.	Ave.	Std. Dev.
1975	1.5	0.62	2.3	1.40
1980	1.3	0.53	2.1	0.71
1985	1.3	0.42	2.0	0.52
1987	1.3	0.48	2.0	0.87

Source: (16); Bureau of Mines records

accounting Office cited a National Coal Association claim in 1977 that 15–20 percent of the total payroll in large underground coal mines is paid to employees involved with MSHA-related tasks; it is uncertain whether this figure is still accurate.[9]

The opportunity costs associated with lowering dust levels include: (1) the present value of production lost due to reductions in production rates to generate less dust per eight hour shift and thereby maintain compliance, and (2) the present value of production lost as a result of closure of mines unable to meet the standard. Longwall operators employ unidirectional cutting methods instead of bidirectional cutting solely to comply with dust regulations, resulting in an estimated production loss of 12 percent per working face. (Estimated based on personal communications with Consolidation Coal, Old Ben Coal, Jim Walters Resources, and Island Creek Coal Corp.) In 1985 this translated into a loss in potential revenues of approximately \$200 million. (Revenue Loss = {[(350.8 million tons mined underground in 1985) × (14.7 pct longwall mining underground)/(100−12 pct)]—[(350.8 million tons) × (14.7 pct)]} × {\$28.18 per ton average underground coal price in 1985} = \$198.2 million.)

EFFECT OF COAL OUTPUT ON DUST LEVELS

A fundamental fact of coal mining is that as coal is mined at a faster rate, more dust is generated. Coal producers must balance increased production per eight hour shift against the reduction of average dust levels per eight hour shift.[4] This has become more difficult in recent years since: (1) the use of longwall mining, a more productive yet dustier mining method than continuous mining, has increased from only 3.6 percent of underground coal production in 1975 to 20.8 percent in 1987 (Figure 3), and (2) longwall mining technology has advanced dramatically. The average production of longwall sections per shift was approximately 850 short tons in 1978 and has increased to 1,968 short tons in early 1987, an increase of 132 percent.[14]

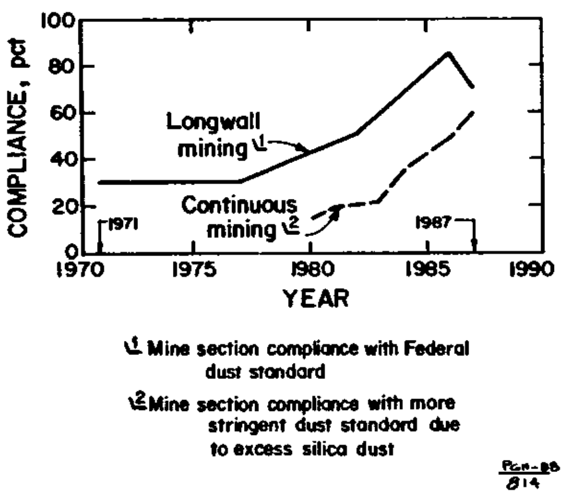

Figure 2. Compliance of selected underground mining methods with dust standards.

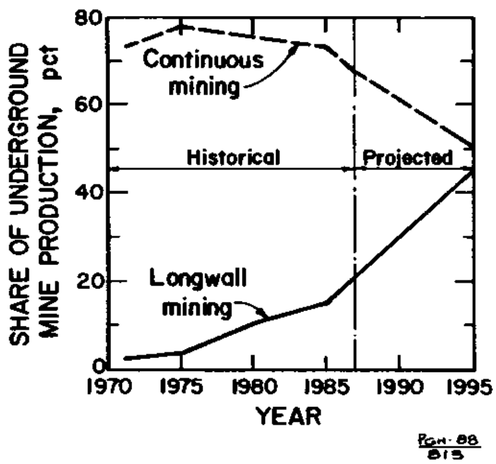

Figure 3. Production by underground mining type.

Due to the direct positive relationship between output and dust generated by longwall mining and its growing share of underground coal production, plots of dust levels against time (Figure 1) are extremely misleading. It is evident that for a given amount of dust control technology, dust levels will rise as coal output per eight hour shift rises. Average dust levels have decreased through time despite the fact that coal output per hour has increased considerably, but not as much as they would have, given the dust control technology implemented, if output per hour had remained constant. In Figure 4, the observed path of dust reduction is indicated by the round markers. Had output per shift remained at "output level 1," dust would have been reduced even further, as indicated by the square markers.

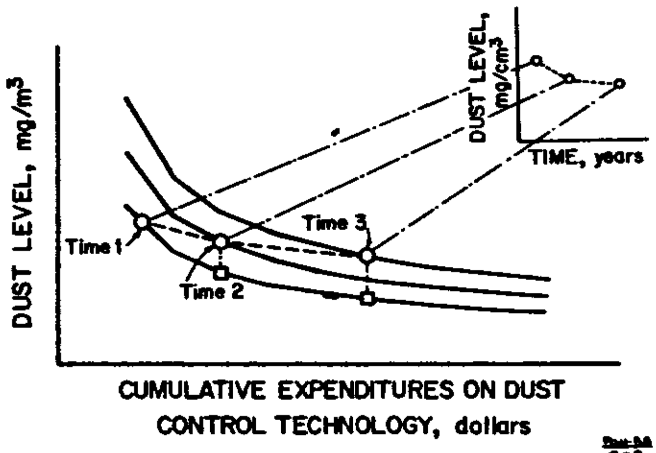

Figure 4. Effect of shifting output level on dust versus expenditures on control technology.

Dust levels of longwall and continuous mining sections adjusted for output per hour are presented in Figure 5. These adjustments were made as follows: output per hour data for the years 1970, 1978, and 1986 were indexed to 1986 levels and these ratios were used to adjust the raw dust data. The adjusted curves, then, show the dust level assuming output per hour had been held constant at the 1986 level, ceteris paribus. The adjusted average dust level in longwall sections declined from 7.29 mg/m^3 in 1970 to 5.50 mg/m^3 in 1978 to 2.0 mg/m^3 in 1986. Raw data indicate a decrease from 2.3 mg/m^3 to 2.1 mg/m^3 to 2.0 mg/m^3 in these years, respectively. Thus, these curves indicate that, particularly in longwall sections, average dust levels have been lowered more drastically since 1970 than is apparent from the raw data.

The 1986 average dust level was then adjusted to the year 1995 given forecasted output per hour of the two mining methods. Output per hour data for 1986 were indexed to forecasted 1995 levels and these ratios were used to adjust the 1986 dust data. (Output per hour is forecasted to increase by 28 percent for longwall mining and by 25 percent for continuous mining by 1995.) Under this scenario, if output per hour were allowed to increase to the forecasted values, by 1995 dust levels would exceed the current dust standards by 28 percent in longwall sections and by 2 percent in continuous mining sections (Figure 5).

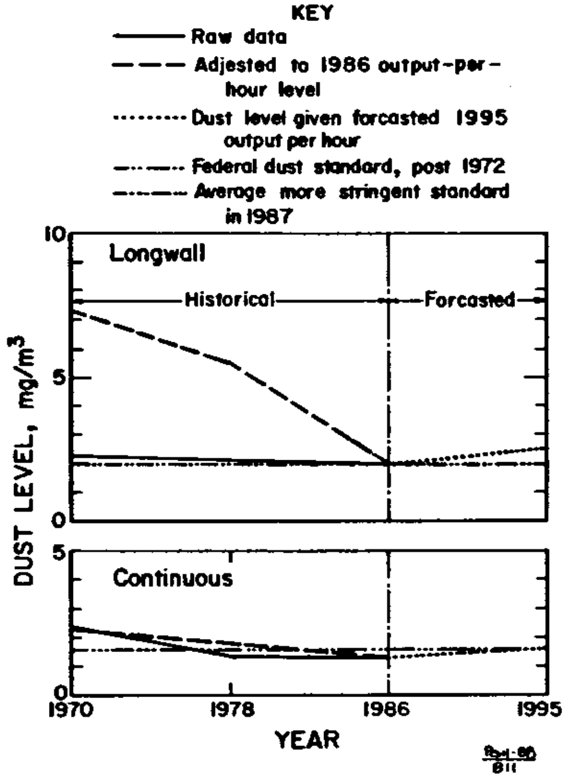

Figure 5. Dust levels of longwall and continuous mining sections adjusted for output per hour.

Unless new dust control technology is developed which enables compliance to be reached at these higher production rates, it is likely that output per hour will be significantly constrained in the future due to required compliance with the dust standard. Indeed, because the average dust level of longwall mining sections is already at the 2.0 mg/m^3 standard, future increases in output per hour are already constrained, on average.

Barring the introduction of new dust control technology, the lost 28 percent increase in longwall mining output per hour forecasted for 1995 translates into a loss in potential revenues in 1995 of $584 million from currently existing longwall sections. (Coal production from longwall mining is expected to total 74 million tons in 1987 (based on calculations from data in 3, 8, 11)). Revenue Loss = {[1.28 × (74 million tons)] − [74 million tons]} × {$28.18 per ton average underground coal price in 1985} = $583.9 million.) This estimate is a maximum figure because even if no new dust control technology is developed by 1995, it is expected that more of the existing technology will be implemented by the industry before 1995.

COMPETITIVENESS

The United States is a major coal exporting nation; exports totalled 85.5 million short tons in 1986, 50 percent going to Europe and 17 percent to Canada.[6] There are numerous indications, however, that the U.S. is losing market share to foreign competitors despite the transition to more efficient underground mining technology. Coal exports have dropped significantly from the 1981 high of 112.5 million tons. The Energy Information Administration reported that the U.S. share of the European market declined from 42 percent in 1981 to 31 percent in 1985; Australia and South Africa appear to have gained market share at the expense of the U.S.[7]

The reason for this loss in competitiveness is apparent from a comparison of the price of delivered coal to Europe (Figure 6)—the U.S. price is by far the highest of the major coal exporting nations to this market. The U.S. has been losing market share even though European coal imports have been rising. And European coal imports have been forecasted to increase from 139 million tons in 1985 to 174 million tons in 1995. Thus, unless the U.S. is able to improve its competitiveness, a continued loss of market share in Europe can be expected.

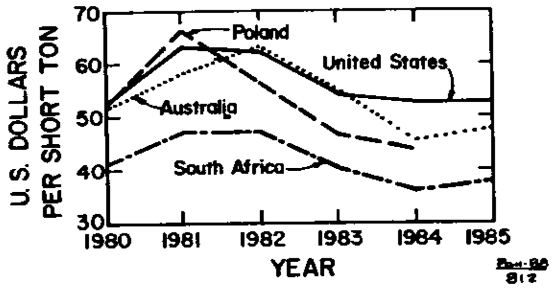

Figure 6. C.I.F prices of non-EEC coal delivered to Europe.

CONCLUSION

To reduce unit costs and thereby ameliorate its competitive position in world markets, the domestic coal industry must continue to increase output while holding the line on production costs. Output from longwall mining sections is forecasted to increase to 45.0 percent of underground coal production, from 20.8 percent currently as the industry attempts to achieve this goal.

The silica dust problem, presently uncommon in longwall sections, is anticipated to become more prevalent as a consequence of increased longwall production because continuous mining machines are used to develop coal panels for extraction by longwall methods. In addition, due to geological conditions—mining of thinner and more heavily faulted and fractured coal seams—the amount of silica dust in airborne respirable dust is expected to increase.

In light of the industry trend toward longwall mining, advancement of dust control technology is necessary to enable associated increases in production while maintaining compliance with the mandated standard. If no new control technology is made available, the dust standard will act as a binding constraint on future output per hour. This is especially pertinent to longwall mining where the average dust level is already 2.0 mg/m^3.

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ON THE TRANSPORT OF AIRBORNE DUST IN MINE AIRWAYS

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ABSTRACT

One of the primary means of control of health hazards from respirable contaminants in mine atmospheres is through design and operation of mines to meet mine health and safety regulations and recommended practices. A U.S. National Academy of Sciences study concluded that for significant progress in coal mine dust control, research should be directed more toward obtaining fundamental understanding of the origin, transport and characteristics of respirable coal mine dust. Theoretical and experimental studies on transport of dust in mine airways, particularly coordinated efforts to validate theory with practice, are scarce. Some empirical models, developed on the basis of experimental data, are available but these models cannot be applied to new conditions. The purpose of this paper is to present the results of theoretical and experimental studies on the transport and deposition of dust in mine airways. This study is a part of an ongoing research project in the Generic Mineral Technology Center on Respirable Dust.

In the paper, the assumptions of the modeling phase of the project and the development of a convection-diffusion equation for dust transport in mine airways are outlined. The important aspect of the modeling effort is the capture of the deposition phenomenon. The experiments performed under controlled conditions in a typical mine airway, as well as under normal mine operating conditions, are discussed. The comparison of the model predictions with experimental results are made to identify critical areas of agreements and deviations. The implications of the findings and areas for further research and development are presented.

LIST OF SYMBOLS

- b class of size distribution
- c concentration at center of duct
- d particle diameter
- D_p Brownian diffusivity
- K_{ij} collision frequency function
- L length of airway under consideration
- n_k number of particles in size class k
- N deposition rate
- r_i radius of particle in i^{th} class
- R radius of dust
- Sc_i molecular Schmidt number
- Sc_t turbulent Schmidt number
- u_* friction velocity
- v_t terminal velocity
- V deposition velocity of particles
- y distance from surface of deposition
- ε eddy diffusivity
- σ dimensionless particle

INTRODUCTION

The objective of this study was to aid in the control of dust in underground mines through an improved understanding of the behavior of dust clouds in mine airways. The results of the study presented in this paper span three phases. Phase I involved the development of a mathematical model; Phase II related to experimental studies in underground mine airways; and Phase III dealt with comparative analyses of the mathematical model predictions with experimental data. A summary of the three phases is presented in this paper.

MATHEMATICAL MODEL

The dispersion and deposition of dust in underground mine airways was modeled as a convective-diffusion problem. To achieve this, the constituents of the model were identified, relationships developed, and assumptions made that closely approximate the physical conditions in a mine airway. A brief description of the major components are presented in this section.

PARTICLE DEPOSITION

The three major mechanisms of deposition in turbulent airflow in mine airways are Brownian diffusion, convective diffusion, and sedimentation. Deposition due to other mechanisms such as electrostatic and thermal force, and inertial impaction were considered not significant compared to the mechanisms considered.

The equation for the turbulent diffusion of particles to the sides of the airways may be written as (Friedlander, 1977):

(equation 1)

$$N = (D_p + \varepsilon) \frac{dc}{dy}$$

while the flux towards the floor and roof are (Sehmel, 1973): (equation 2)

$$N = (D_p + \varepsilon) \frac{dc}{dy} \pm v_t c$$

The value of the eddy diffusivity ε varies within the boundary layer. Therefore, different values of ε have to be used when integrating the flux equation from the deposition surface to the core of the airflow.

An empirical relation is used for describing deposition due to turbulent diffusion in the inertial range, given by (Wood, 1981): (equation 3)

$$V^+ = \frac{N}{cu_*} = 0.13 \quad \text{for} \quad 17 < \sigma < 265$$

and: (equation 4)

$$V^+_{Skyrme} = \frac{2.6}{\sigma} \quad (1 + \frac{50}{\sigma}) \quad \text{for} \quad \sigma < 265$$

The total deposition due to all mechanisms is given by: (equation 5)

$$\frac{N}{u_*} = \int_0^{r_1} n(r)\, dr\, V^+_{diff} + \int_{r_1}^{r_2} 0.13\, n(r)\, dr + \int_{r_2}^{r_3} V^+_{Skyrme}\, n(r)\, dr$$

The deposition due to gravity is a function of the terminal velocity and can be written as: (equation 6)

$$V_{gravity} = v_t.$$

COAGULATION

Coagulation of airborne particles was represented in the mathematical model by a modified rate equation (Chung, 1981) and is given as: (equation 7)

$$\frac{dn_k(t)}{dt} = \frac{1}{2} \sum_{\substack{i=1 \\ i+j=k}}^{k-1} K_{ij} n_i n_j + n_k \sum_{i=1}^{b} K_{ik} n_i - n_k \sum_{i=b+1}^{\infty} K_{ik} n_i$$

where the first term represents the gain in particles in size class k due to the collision of particles of size i and j. The second term represents the loss of particles from size class k due to collision of class k particles with other particles. The last term represents those k class collisions occurring with particles of class less than b, the resultant size being less than the upper boundary of size class k. K_{ij} is the collision frequency term that takes into account the motion of the particles with the air, relative motion due to the air and relative motion due to sedimentation. The formula proposed by Saffman and Turner (1956) was used in the model.

The governing equation is a convective-diffusion equation. A one-dimensional equation was adopted and is represented by the relation: (equation 8)

$$\frac{\partial c}{\partial t} = E_x \frac{\partial^2 c}{\partial x^2} - u \frac{\partial c}{\partial x} + \text{sources} - \text{sinks}$$

The equation was solved for a range of particle sizes obtained by discretizing the particle size distribution of the source dust. The behavior of the total dust cloud is a weighted average of the contribution from the various sizes. The initial condition to solve the equation is of the form: (equation 9)

$$c(x, t) = 0 \text{ for } t = 0, 0 < x < L$$

where L is the length of the region of interest. The boundary condition was developed by assuming that the concentration of the dust becomes asymptotic at the end of the region of interest. It is represented as: (equation 10)

$$\frac{dc}{dx} = 0$$

The source term (S(t)) was developed as a step function and is given as: (equation 11)

$$S(t) = \sum_{i=1}^{n} A_i (\delta(t - t_i)$$

when n = the number of operating modes, A_i is the amount of dust released in mode i, and δ is the dirac delta function. The model was solved numerically, using an implicit scheme (Bandopadhyay, 1982) and programmed in WATFIV.

EXPERIMENTAL STUDIES

To obtain a better understanding of the spatial and temporal behavior of dust clouds in underground mines and data to compare with the predictions of the mathematical model, a set of mine experiments were conducted. They were performed in the Lake Lynn Laboratory of the U.S. Bureau of Mines. The laboratory was formerly a limestone mine. Six experiments were conducted. The salient parameters are listed in Table I. The experiments provided data on ambient concentration, floor deposition, particle size distribution, and cross-sectional variation of dust at various stations along the length of the airway. In addition, two experiments were performed in the return airway of a longwall section.

Table I

Salient Data on Controlled Experiments

	Dust Type	Velocity, m/s
Experiment 1	Semianthracite	0.838
Experiment 2	Bituminous	0.838
Experiment 3	Semianthracite	1.855
Experiment 4	Bituminous	1.855
Experiment 5	Semianthracite	1.525
Experiment 6	Bituminous	1.525

The sampling plan for airborne concentration and deposition is shown in Figure 1. Centerline and cross-sectional airborne dust samples were collected as shown in the figure. Twelve samples were collected at each of the three cross-sectional sampling stations. The sampling systems were designed for isokinetic sampling, using specially shaped sharp-edged nozzles. Corrections as suggested by Belyaev and Levin (1974) were applied to those data for which isokinetic sampling conditions were not achieved.

Floor samples were collected at about 13 stations, 100 feet apart, along the airway. Samples were collected along the center and across the width of the airway. Flat deposition plates covered with preweighed, lightweight, "sharkskin" filter papers were used to collect the dust.

The dust was dispersed by a fluidized bed-type trickle duster through a four-port system of tubes. Each port was located at the center of the four quadrants of the airway cross-section. Semi-anthracite and bituminous dust, with top size of 25 μm and median size in the 4.96 to 7 μm range were used as source dusts.

COMPARISON OF MODEL OUTPUT WITH EXPERIMENTAL DATA

The experimental data were compared with the output from the mathematical model for similar physical conditions. Comparisons were made for ambient concentration, deposition, particle deposition rates, dispersion coefficient and cross-sectional concentration of the dust. The inputs to the model were based on the physical conditions prevailing during the experiments. These included the airway, source dust characteristics, and airflow conditions. For reasons of brevity, comparisons for only a select set of experiments are presented.

The results of comparison of model output and experimental data for experiments 1 and 6 are presented in Figures 2 and 3 for ambient concentration, and in Figures 4 and 5 for deposition.

The comparison of predicted and actual concentrations for experiment 1 (Figure 2) shows that predicted concentration falls rapidly with distance from the source tending to an asymptote towards the end of the region of interest. The experimental data also shows a rapid decrease in concentration from the source, in fact, more than that predicted by the model. However, part of this decrease may be due to agglomeration induced increase in deposition rate. The two data sets closely follow each other after 120 m from the source. The respirable dust data show that while the predicted and experimental data are generally in agreement, the experimental data show a more consistent deposition along the airway.

The concentration data for experiment 6 (Figure 3) shows a closer match between the predicted and experimental data up to 180 m, after which the experimental data tends to assume a less steeper decline in concentration. This pattern is also true in the case of respirable dust data for the experiment. The experiment was conducted at 1.55 m/s. It appears that some of the differences between the concentration data sets may be due to the greater sensitivity and scope for errors in concentration data measurement. The deviation between predicted and experimental data at the first two stations near the source may possibly be due to inadequate dispersion of the source dust.

The deposition data for experiment 1 is presented in Figure 4. The data shows good agreement between the predicted and experimental data. The agreement is especially close between 60 and 400 m. The deposition data for experiment 6 (Figure 5) also show good correlation between the two data sets between 100 and 420 m.

In addition to comparison of the predicted and experimental ambient concentration and deposition, comparisons were also made between the deposition rate per unit concentration, per unit time, and the dispersion coefficient of the dust cloud. The comparisons could be made for floor deposition only, as the amount of dust deposited on the sides and roof could not be collected with an acceptable degree of accuracy. Very little dust, compared to floor deposition, could be collected on the sides and roof.

The theoretical deposition rate was assumed to be dependent on only the physical parameters relating to the particle and flow properties. The volume concentration of the dust was assumed to be low enough to be considered a 'dilute' flow. Therefore, the particles were assumed not to affect the fluid flow properties and the theoretical deposition rate was considered to be independent of concentration or location of the dust cloud in the mine airway. However, the experimental data showed that deposition rate decreased with distance from the dust source, becoming fairly constant towards the end of the airway (Figure 6).

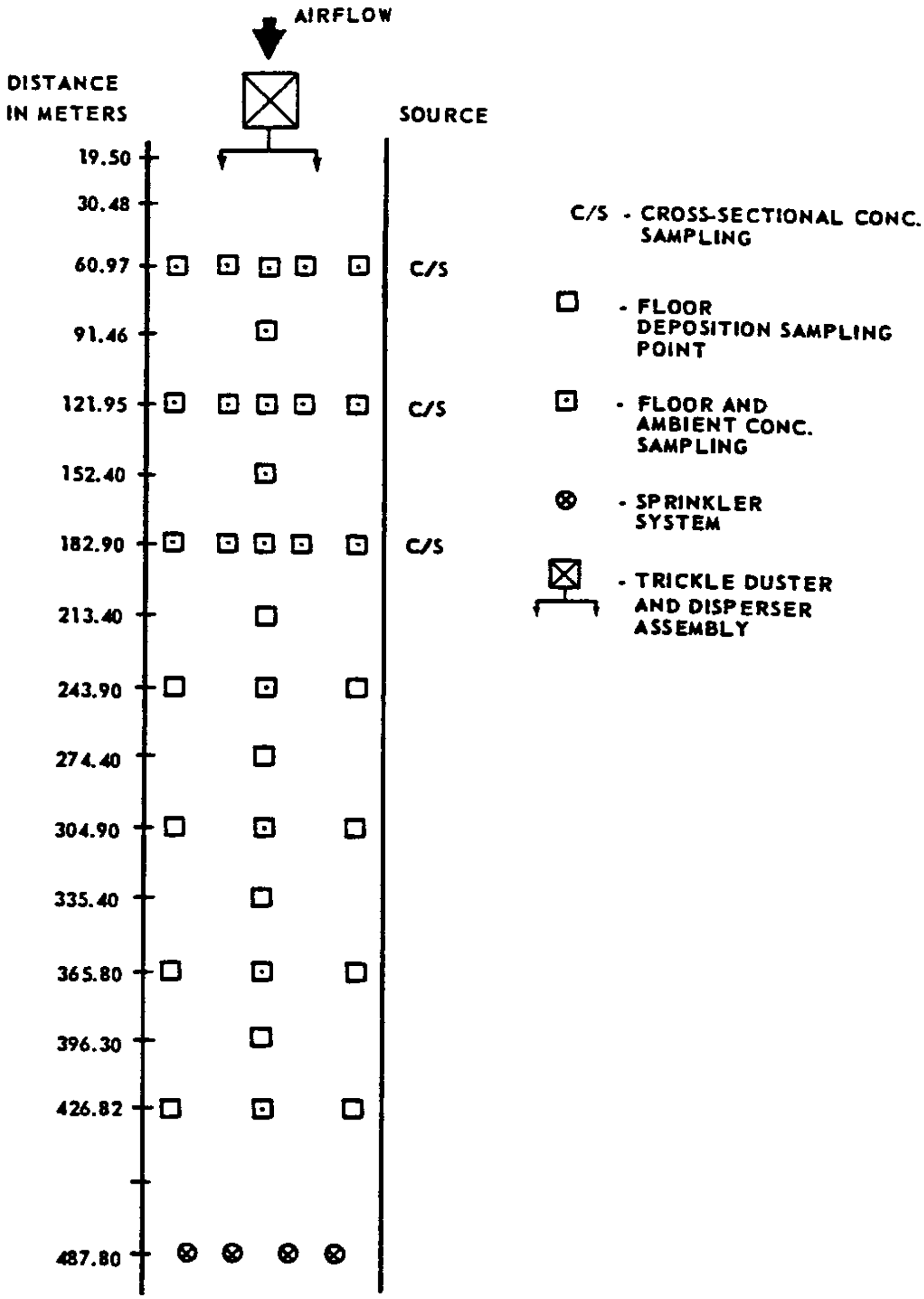

Figure 1. Ambient concentration and deposition sampling plan (Controlled Experiment 6).

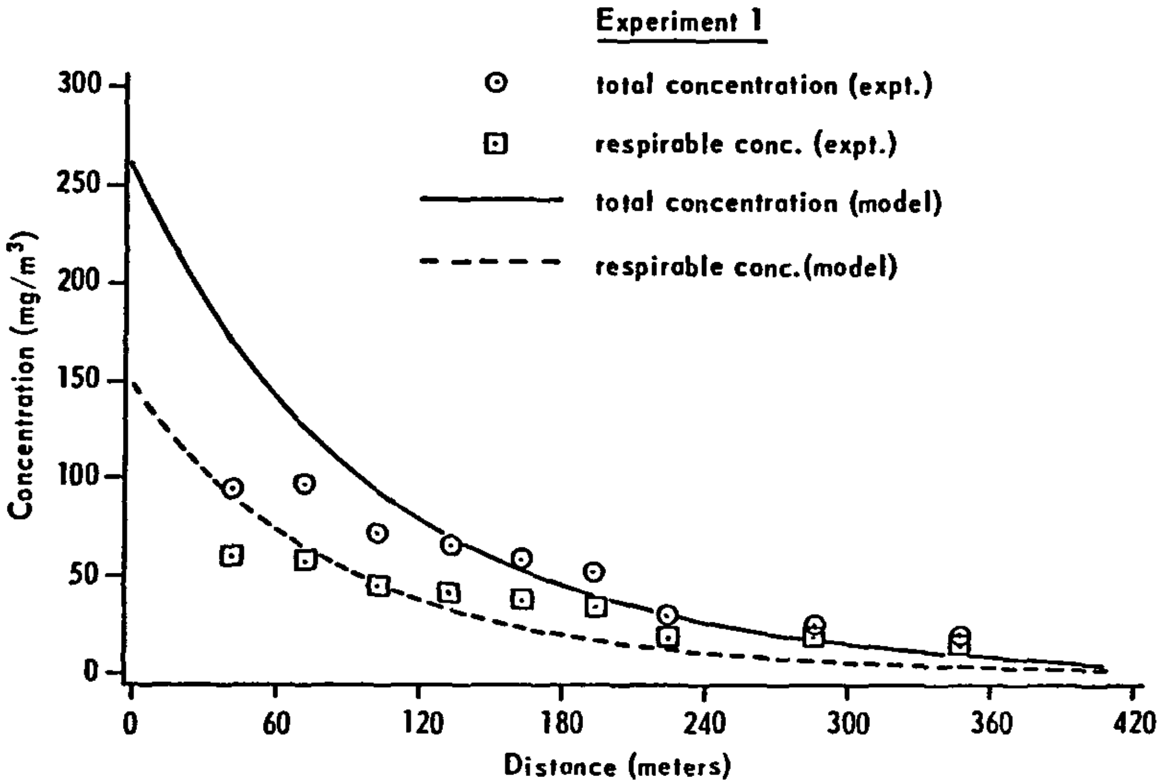

Figure 2. Comparison of model predicted concentration with experimental data (Controlled Experiment 1).

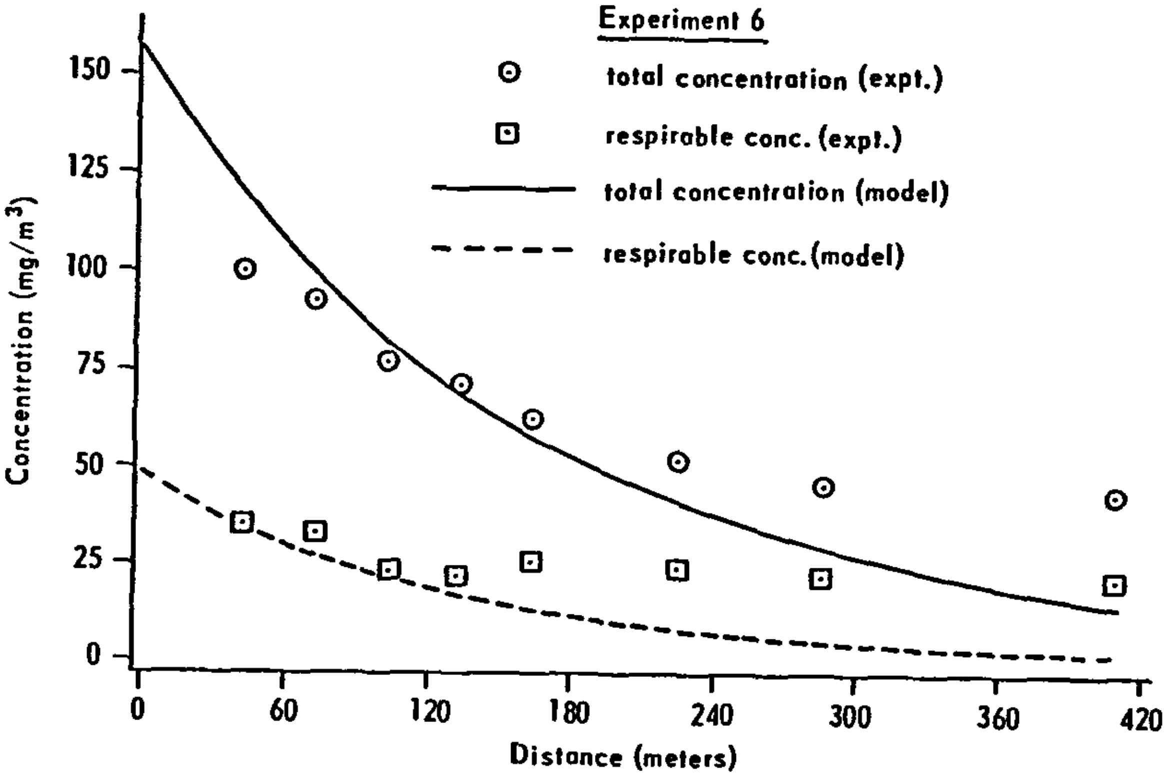

Figure 3. Comparison of model predicted concentration with experimental data (Controlled Experiment 6).

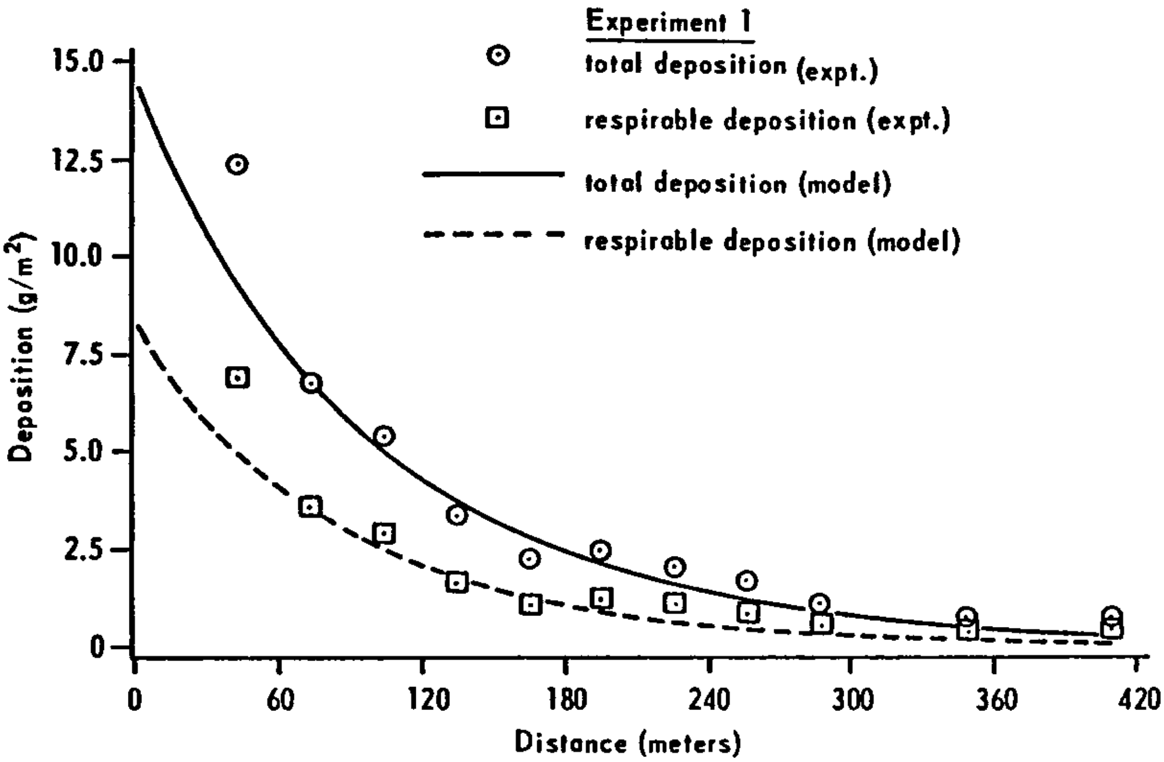

Figure 4. Comparison of model predicted floor deposition with experimental data (Controlled Experiment 1).

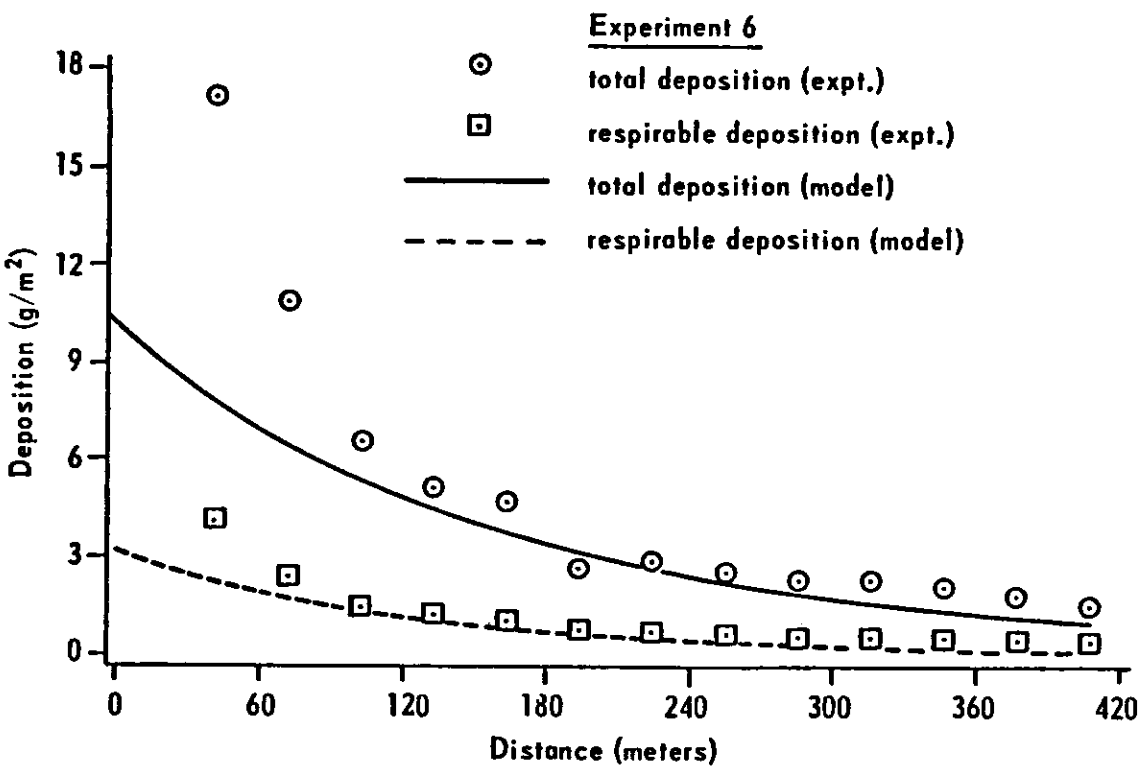

Figure 5. Comparison of model predicted floor deposition with experimental data (Controlled Experiment 6).

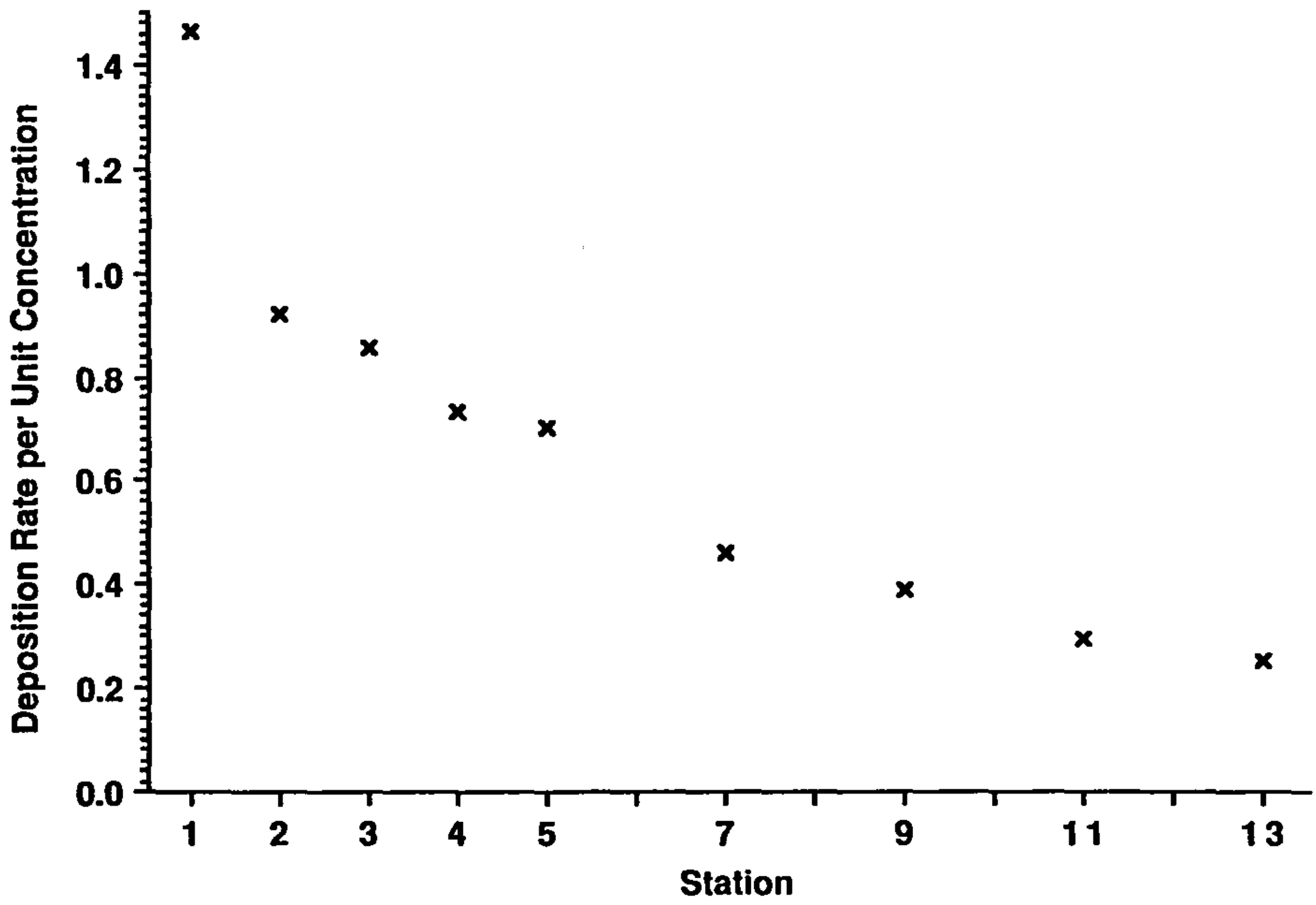

Figure 6. Normalized deposition rates of dust along mine airways (Experiment 5, size 3.73 microns).

The dispersion coefficient relationship used in the model was developed by Skubunov (1974) and is given by: (equation 12)

$$E_x = 15.8 \; UDSc_i^{-0.6} \; Sc_t \sqrt{\lambda/\lambda_r}$$

The values obtained by this relation was compared with the experimental data. The experimental dispersion coefficient was calculated using the procedure outlined by Klebanov and Martynyuk (1974). The results are presented in Table I. The comparison of the calculated and experimental data show that both are in the same order of magnitude. The experimental values vary from 11.79 to 45.06 m^2/s while the model assumed value was 61.46 m^2/s.

Cross-sectional concentration data were also collected during the experiments. The data showed that the average concentration across the cross-section is 75% of the concentration at the center of the airway, with all the points in the cross-section given equal weights. The concentration decreases from the roof to the floor, with the top third of the airway having a concentration 72% of that in the lower third, while the concentration in the middle third being 89% of that in the lower third of the airway. Complete details of the theoretical and experimental study are presented in Bhaskar (1987).

SUMMARY

A mathematical model describing the behavior of dust clouds in mine atmospheres was developed with special reference to the condition prevailing in a mine. The model was programmed for the computer and outputs ambient concentration and deposition data as a function of time and location. The output includes both the total and respirable size ranges. In addition to mathematical modeling, experimental studies were performed in mine airways for two types of dust at three velocities. The experimental data were compared with the output of the mathematical model for similar conditions.

The results show that there are areas of agreement and deviation between the two data sets. The comparison highlighted areas, such as deposition rate, reentrainment and diffusion coefficient, where additional studies have to be performed. Studies in these areas have been initiated and are continuing.

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DUST CONTROL ON LONGWALL SHEARERS USING WATER-JET-ASSISTED CUTTING

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INTRODUCTION

Since 1977 the number of U.S. longwall mining sections using double-ended ranging-arm shearers has more than doubled. Improved productivity is a primary reason for using the longwall mining method. Average U.S. longwall production is 700 to 1,200 tons/shift compared with 300 to 400 tons/shift for room and pillar mining. However, in some cases, production on longwall sections must be limited because the levels of airborne respirable dust exceed the mandatory standard.

The best way to suppress dust generated by the shearer is to add water to the coal at a location near the cutting bit. The most effective way to accomplish this is to supply water through the rotating drum and distribute it to nozzles located in the bit block. All longwall shearers operating in the United States are equipped with this type of water spray system for dust control. Typically the water pressure measured at the nozzle is 100 to 200 psi. Increasing the water pressure delivered through the drum-mounted sprays will usually decrease dust levels.

Water-jet-assisted cutting uses moderately high pressure, 2,000 to 10,000 psi (13.5 to 67.5 MPa), solid streams of water, called water jets, that are directed to strike near the cutting bit tip. The Bureau of Mines and others have evaluated the potential advantages of using high-pressure streams of water for water-jet-assisted cutting. Water-jet-assisted cutting was used with a roadheader. Energy supplied by the water jets enabled the roadheader to cut hard rock that could not be cut when operating dry.[1] Results of an earlier laboratory test program showed that airborne dust formed during cutting could be reduced by using water-jet-assisted cutting.[2] The objective of this research program was to determine what effect use of water-jet-assisted cutting has on respirable dust levels generated during cutting with a longwall shearer.

Testing was conducted on the surface at a simulated longwall face and on an operating underground longwall section. The initial study took place at the Bureau of Mines' surface test facility in Pittsburgh, PA. Operating parameters could be controlled more precisely at the surface site than underground. A 60-ft-long (18.5 m) by 6-ft-high (2 m) coalcrete block, composed of coal, fly ash, and concrete, was used to simulate a longwall face. Because coalcrete has a higher silica content, it is more abrasive than coal; however, when using conventional drag bits, its cutting properties are similar. Overall the coalcrete face was homogeneous.

The shearer used to cut the coalcrete was a Joy 1-LS1* double-drum machine (Figure 1). For each test the shearer cut from right to left. Only the left hand, or leading drum, was supplied with high-pressure water and used for cutting during the tests. The right-hand drum was positioned so that it traveled within the cut made by the left-hand drum. A longwall face conveyor, located adjacent to the coalcrete block, provided continuous removal of the cut material, as well as functioning as a support along which the shearer moved. The diameter of the cutting drum (bit tip to bit tip) was 54 in. (137 cm), and the drum width was 28 in. (71 cm). During the tests, web width (thickness of the cut) varied from 25 to 29 in. (63.5 to 73.5 cm). The machine tram rate was maintained at approximately 5 ft/min (1.5 m/min). Drum rotation speed was 46 r/min with a bit tip speed of 650 ft/min (200 m/min). Thirty-two radial attack bits were mounted on the drum.

The site for the underground work was a longwall section in the Auguste Victoria Mine, which is located in Marl, West Germany. The face was 7.54 ft (2.3 m) thick, 919 ft (280 m) long, and mined on retreat. During the tests, one single-drum and one double-drum shearer were operated on the face. Figure 2 shows the relative locations of the shearers on the longwall face. The single drum machine, an Eickhoff model EW-200/170-L shearer, was supplied with high-pressure water (Figure 3). This shearer operated within 164 ft (50 m) of the longwall tailgate. While making dust measurements, the shearer cut only in the upper part of the face. Shearer tram rate and web width were maintained as constant as possible.

Underground testing at low pressure was conducted using the cutting drum that was originally supplied with the shearer. This drum was not designed for use with high-pressure water and a new drum had to be designed and built for the water-jet-assisted cutting tests. Table I compares features of the original and new drums. Included with the high-pressure

*Reference to specific products does not imply endorsement by the Bureau of Mines.

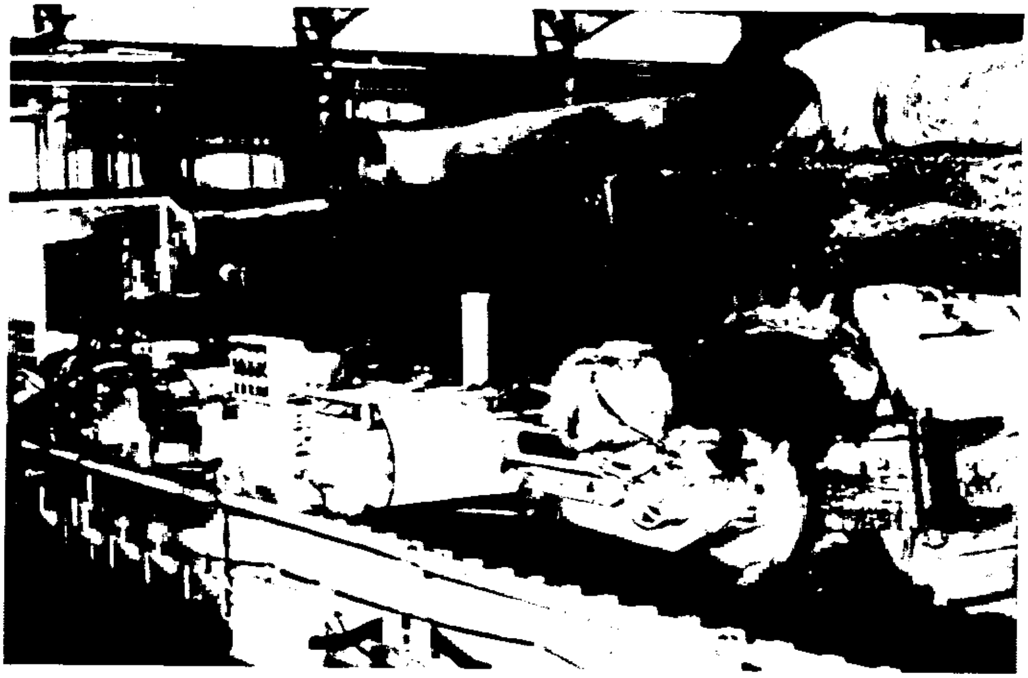

Figure 1. Shearer use for surface testing.

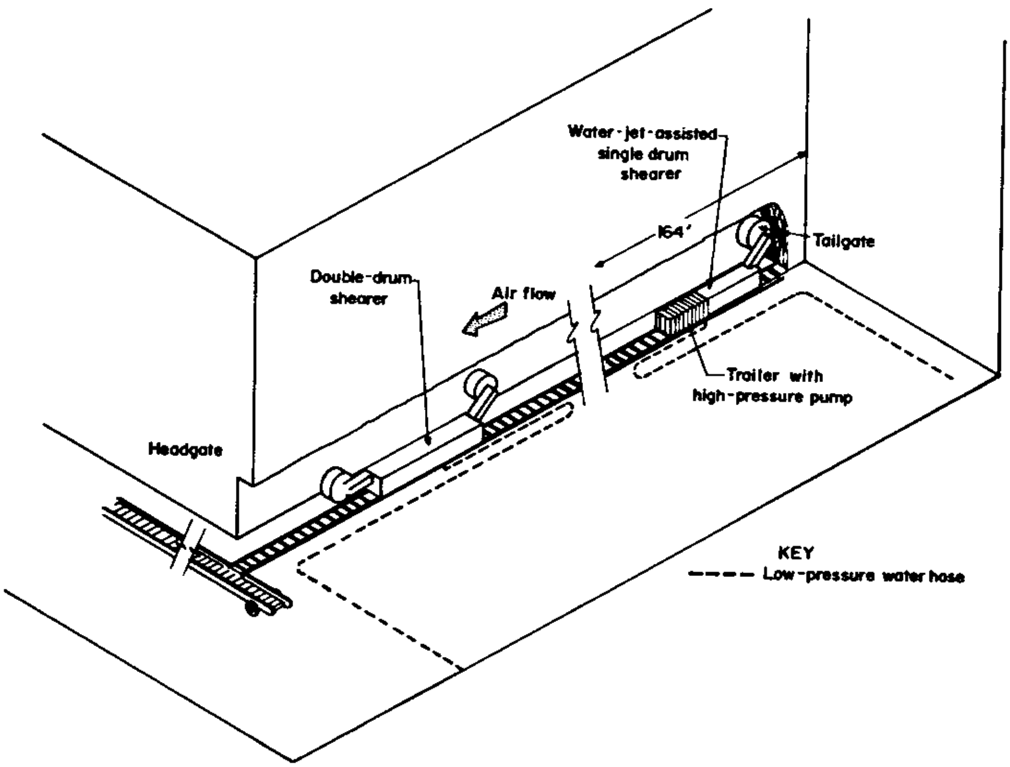

Figure 2. Underground test area.

Table I
Comparison of Cutting Drums Used During Underground Tests

	High-Pressure Drum	Low-Pressure Drum
Diameter(in)............	67	63
Web depth(in)...........	33.4	33.5
R/min...................	23.6	48
Bits (No./type).........	51/conical	55/radial
Bit tip speed (ft/min)..	413	791
Spray nozzles (No./type)	50/Sapphire	41/conical
Flow rate(gal/min)......	10 to 21	10

Figure 3. Shearer operating underground.

drum was a newly designed ranging arm with double planetary gearing that provided a drum rotational speed of 23.6 r/min. The slower rotational speed allowed a more efficient distribution of fluid energy, i.e., more energy could be supplied per length of cut. However, another consequence of slower rotation speed was a deeper depth of cut. The bit lacing was modified to provide more efficient cutting and loading at deeper cutting depths.

SURFACE WATER DELIVERY SYSTEM

A 200-hp (112-kW) Aqua-Dyne triplex pump was used to supply the desired water pressures to the shearer. The pump was placed adjacent to the coalcrete face and water was transported to the shearer through a 2-in (5.1 cm) flexible hose. Water pressure during the low-pressure tests was maintained at 190 psi (1 mPa). During each water-jet-assisted cutting test the pressure was maintained constant. High pressures between 1,000 and 6,000 psi (7 to 40 MPa) were used. Water entered the cutting drum through a high-pressure Aqua-Dyne rotary seal, located in the drum hub. Six hoses were attached to the rotary seal. Each one of the six hoses carried water to a sector of the cutting drum which contained approximately 1/6 of the water jet nozzles. A water jet nozzle was located in front of each of the 32 cutting bits on the left cutting drum (Figure 4). All water nozzles in the drum operated continuously during the surface tests.

Each nozzle used for these tests had a 13 degree Leach and Walker configuration (Figure 4). To maintain approximately the same flow rate during the high- and low-pressure tests, 0.024 in. (0.6 mm) and .07 in. (1.78 mm) orifices, respectively, were used. Nozzle flow rates for each test pressure are given in Table II. Each nozzle delivered a solid stream of water to a location about 0.1 in. (3 mm) in front of the bit tip. Distance from the nozzle to the bit tip averaged about

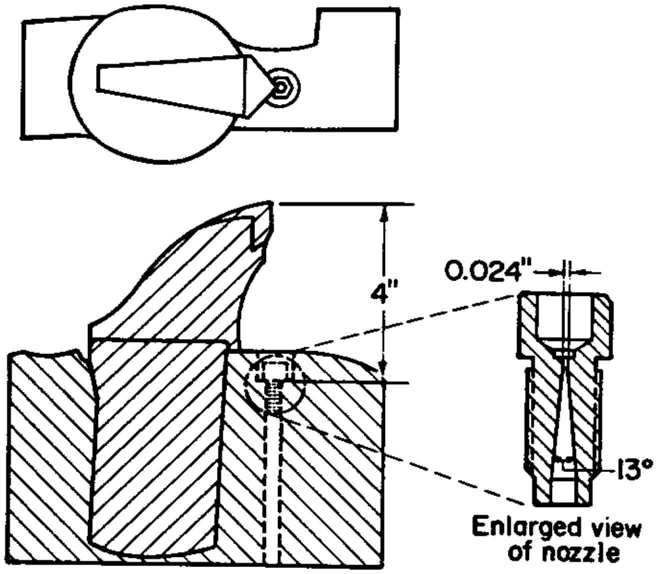

Figure 4. Bit block and nozzle configuration for surface testing.

Table II
Flow Rate versus Water Pressure for Surface Tests

Pressure, psi	Flow rate gal/min
High-pressure:[1]	
6,000........	1.26
5,000........	1.15
4,000........	1.03
3,000........	.90
2,000........	.75
1,000........	.54
Low-pressure:[2]	
190..........	.90

[1]0.024-in orifice [2]0.071-in orifice

4 in (10 cm). The water lines in the cutting drum were flushed frequently, and the water passed through 10 micron filters to reduce the possibility of nozzle blockage.

UNDERGROUND WATER DELIVERY SYSTEM

Normal head pressure provided water to the shearer at 340 psi (2.4 MPa). Forty-one conical spray nozzles mounted in the cutting drum were used for dust control. Total flow rate for this normal operating pressure was approximately 10 gal/min (38 l/min).

To provide the high-pressure water needed for water-jet-assisted cutting, a five-piston pump was mounted on a trailer that was pulled by the shearer. The maximum capacity of the pump was 34 gal/min at 10,000 psi. Fifty of the 51 bit blocks were equipped with jet nozzles (Figure 5). Blockage of the 0.6 sapphire nozzle orifices was reduced by installing a 10 micrometer filter in the water line.

The drum built for the high-pressure tests, was divided into 10 sectors. Water was directed to each sector through manifolds and high-pressure hoses (Figure 6). A phasing system was designed to feed the water to five of the ten sectors at a time. The average angle of the arc of rotation that was supplied with water was 195 degrees (see Figure 7). Using this phasing system reduced the water required by about 50 pct.

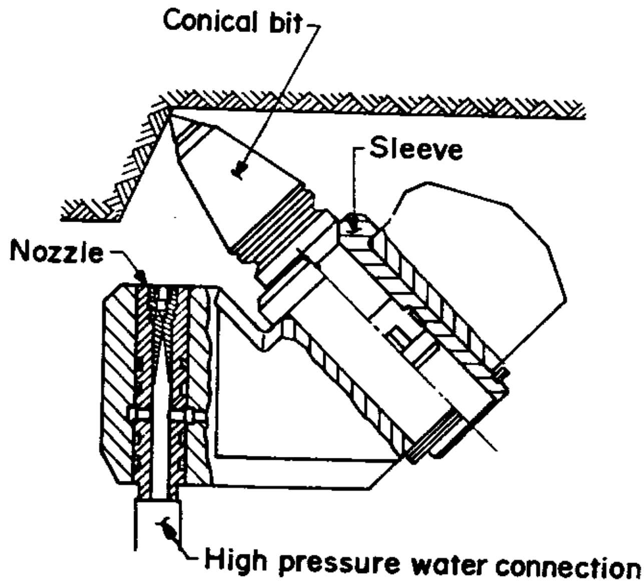

Figure 5. Bit block and nozzle configuration for underground testing.

TEST PROCEDURE

Cuts made in the coalcrete block were 5 to 40 feet (1.5 to 12.3 m) in length. Water pressure was monitored during each test cut to assure the water pressure did not vary.

Dust levels were measured at two locations near the shearer.
1. About 6 ft (1.8 m) from the cutting drum at approximately the same height as the top of the cut.
2. About 24 in. (0.6 m) from the bottom of the lead drum.

Real-time aerosol dust monitors (GCA RAM 1's) and strip chart recorders were used to track the levels of airborne respirable dust. Dorr-Oliver 10 mm-nylon cyclones were used to separate the respirable dust from the larger particulates.

Underground, one location upwind, and another downwind of the shearer were sampled. As much as possible, during underground testing, no other work that produced dust, such

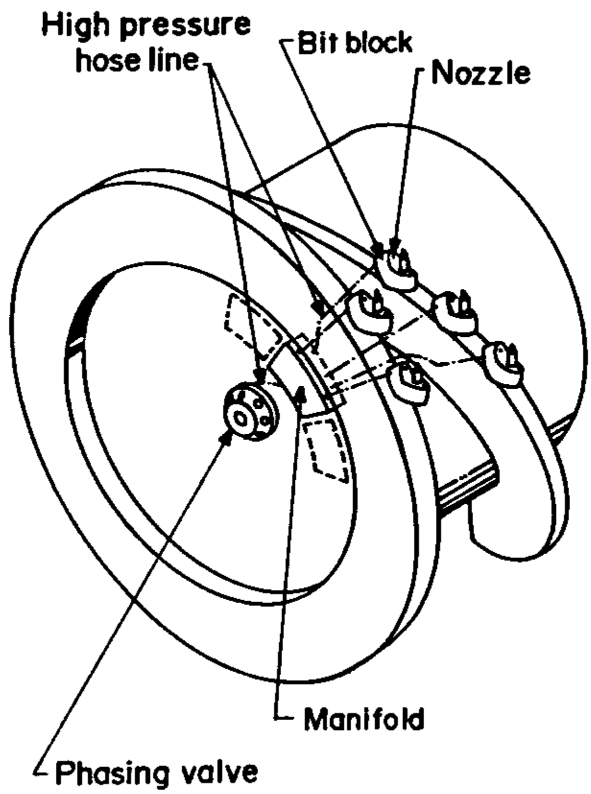

Figure 6. High-pressure water supply to cutting drum.

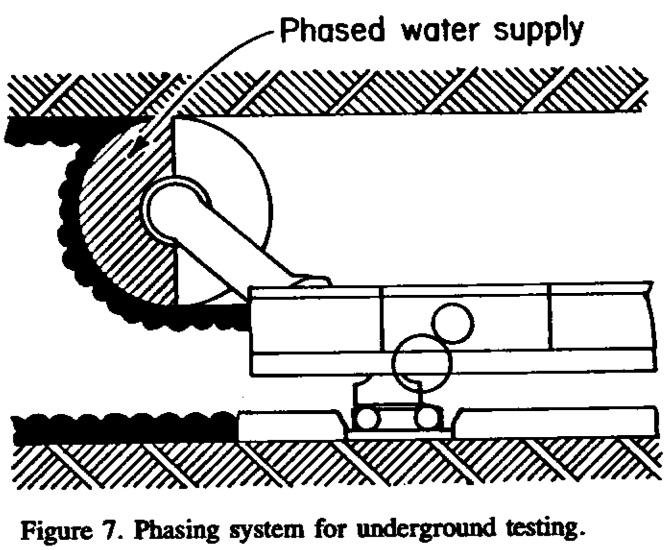

Figure 7. Phasing system for underground testing.

as moving the roof supports, was carried out upwind of the shearer. The dust generated by the second shearer, which operated on the headgate side of the test shearer, did not influence the dust readings, because airflow was from tailgate to headgate.

RESULTS

For the surface tests, the average dust levels measured while using high-pressure water (1,000 to 6,000 psi) were compared with dust levels generated while operating at 190 psi (1 MPa). The percentage dust reductions achieved by using the higher water pressures are shown in Table III. At a water pressure of 3,000 psi (20 MPa), the dust levels were 79.2 pct less than when operating at 190 psi (1 MPa). Raising the pressure further from 3,000 to 6,000 psi (20 to 40 MPa) resulted in only small additional dust reductions.

Table III

Comparison of Dust Reduction During High- and Low-Pressure Operation

Pressure, psi	Dust reduction, pct
High-pressure:[1]	
6,000........	80.4
5,000........	84.8
4,000........	80.4
3,000........	79.2
2,000........	63.9
1,000........	4.2
Low-pressure:[2]	
190..........	0

[1] 0.024-in orifice [2] 0.071-in orifice

The underground respirable dust results are shown in Figure 8. At a water pressure of 1,800 psi (12 MPa) and a water flow rate of 10 gal/m (38 l/min), average dust levels were reduced almost 80 pct compared to dust levels measured while operating at 340 psi (2 MPa) and 10 gal/min (38 l/min). Maintaining the water pressure at 1,800 (12 MPa) and increasing the flow rate to 21 gal/min (80 l/min), by increasing the nozzle orifice size, resulted in no further reduction in dust. Additional reductions in dust level due to increasing the pressure to 7,200 psi (50 MPa), with a flow rate of 21 gpm, (80 l/min) were not significant.

DISCUSSION

Dust Levels

Use of water during longwall mining reduces the levels of airborne dust by:

1. Capturing airborne dust particles.
2. Wetting the dust particles before they can become airborne.

The surface study results showed that increasing the water pressure from 190 to 1,000 psi (1 to 7 MPa) did not significantly reduce dust levels. Dust levels decreased rapidly as the water pressure was raised from 1,000 to 3,000 psi (7 to 20 MPa). Any further decrease in dust level, as the water pressure was raised from 3,000 to 6,000 psi (20 to 40 MPa), was small.

Raising the water pressure underground from 340 to 1800 psi (2 to 12 MPa) reduced airborne dust levels 70 to 80 pct. There was no significant additional reduction in dust level when the pressure was raised from 1800 to 7,200 psi (12 to 50 MPa). The fact that there is a maximum pressure above

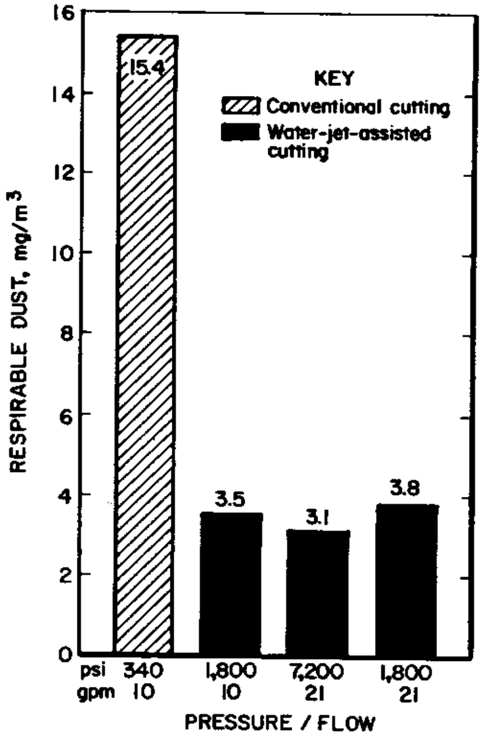

Figure 8. Underground respirable dust results.

which no further dust reductions take place further confirms the results of the surface longwall shearer study and the work performed by other researchers with roadheaders.[3]

The operation of the shearer during surface cutting of the coalcrete was similar to the operation of a shearer on an underground longwall section. However, the airflow patterns on an underground longwall face, which have a significant effect on the distribution of the airborne dust near the shearer, could not be simulated during surface testing. Also, the amount of dust generated by cutting coalcrete and coal would not be the same, due to physical differences between the two materials. Therefore, the dust levels measured during surface testing cannot be directly related to the amount of dust generated underground. However, the underground study results verify that the relative reductions in dust resulting from use of the high-pressure sprays are typical of what can be achieved underground.

Mining conditions during underground testing were representative of a typical longwall operation although the amount of dust generated was extraordinarily high. This may have been due to cutting in a faulted zone. The same reductions in respirable dust obtained underground cannot be expected for all faces. Use of high-pressure water directed through drum mounted jet nozzles would be effective for dust suppression on all longwall faces.

Interpretation of the underground dust data is complicated by the fact that during the high-pressure tests, a different cutting drum was used and the drum r/min was reduced. Cutting depth was increased because the tram rate was kept constant. Reduced drum r/min and increased cutting depth has been shown to reduce airborne dust levels.[4] It is not possible to determine how much each factor, reduced r/min, deeper cutting, or water-jet assist, contributed to the reduction in dust levels. For optimum dust control, it is recommended that high-pressure water be used with reduced drum speed and deeper depth of cut.

Supplying high-pressure water for water-jet-assisted cutting requires a large amount of fluid energy. The quantity of energy can be reduced if water is supplied only to that part of the cutting drum where the bits are in contact with the rock. Although a phasing system was used for the underground study, a suitable system wasn't available for the surface study. To more accurately reflect the amount of energy directed to the bits that were cutting during the surface tests, the total fluid energy supplied was divided by two. Using these calculations, at 190 psi (1 MPa) operating pressure, the fluid energy accounted for less than 2 pct of the total energy used during cutting. At 6,000 psi (40 MPa), almost 33 pct of the total energy supplied during cutting was provided by the water jets. During underground testing a similar proportion of the total energy was supplied by the water jets.

CONCLUSIONS

The results of the surface and underground studies showed that use of water-jet-assisted cutting significantly reduces airborne dust generated by a longwall shearer. Optimum dust suppression was achieved using pressures between 1,000 and 3,000 psi (7 to 20 MPa). These reductions in respirable dust were obtained without increases in water flow rate. Underground a phasing system, used to direct water to only those bits that were cutting, reduced water flow rate by 50 percent. The second underground trial called for under this research project will be conducted on a longwall face in the United States. During this test a double ranging arm shearer will be equipped with a high-pressure water supply system.

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