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UTILIZATION OF TUNNEL LINERS IN ROOF FALL AREAS

falls of roof

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One of the most difficult provisions of state or federal mining laws to comply with is that which dictates that no person shall proceed beyond the last permanent support unless adequate temporary support is provided. Many roof fall fatalities have occurred as a result of non-compliance with the above mentioned provision. The provision is further compounded when a roof fall occurs in a coal mine.

The majority of all approved roof control plans have anywhere from ten to 12 safety precautions that relate to the installation of temporary supports and method of supporting roof in cavities where falls have occurred. The purpose of this presentation is to relate and describe a method that the Rochester & Pittsburgh Coal Company has successfully tried in the rehabilitation of a fall area where it was impossible to set proper temporary supports.

When a roof fall occurs and it is one of such a nature that it is next to impossible to secure the roof with temporary supports during cleanup, the so-called safety precautions for cleanup of roof falls, now included in the mine operator's roof control plans, is in my opinion so much window dressing. If an accident were to occur during the rehabilitation of such an area, the operator would be subject to many violations of the safety precautions and would also be subject to certain violations of 75,200 of the Federal Regulations. The seriousness of these violations of the operator's safety precautions is further compounded under the unwarrantable failure provisions contained in Section 104(c) of the Coal Mine Health and Safety Act.

Early this year, we had a roof fall occur at our Urling No. 3 Mine in Indiana County, Pennsylvania. The fall was at an intersection, was approximately 42 feet in length and funneled to a cone, the apex of which was approximately 30 feet high. The fall material was of such a nature that it was loose as the result of many mud slips. This created a problem where setting of temporary supports in the area for the purpose of bolting could not be done. The sides of the fall area needed to be scaled and this, again, was impossible to do without exposing workmen. It was one of the type of falls that occur which make it impossible to comply with safety precautions and related laws.

The initial discussions as to how we were to protect workmen during the rehabilitation of this area were conducted by operations, mine management, and engineering. They decided to attempt to use a new concept which we had never tried before, that would be using Armco steel liner plates which are 4'5" x 18" and 3' x 18". The plates would be bolted and would be 5-gauge plates. This would make a semicircular arch which would be approximately 20 feet at the base and approximately 11½ feet at the highest point. Safety Department was contacted as to how we would proceed to first include the above mentioned materials in the roof control plans and also get approval from MESA to progress with this approach on providing protection for the miners during cleanup of the fall. The proposal was made to the district manager of MESA and the state inspector, along with a description of the fall area and the problems we were facing and a request for utilization of tunnel liner to provide protection during cleanup. The district manager had the situation investigated immediately and tentatively approved our proposal, providing certain precautions would be adhered to. The precautions were reasonable, and preparations were made for the cleanup and installation of tunnel liner to begin.

As a result of the fall, water presented another problem. We were getting approximately 100-plus gallons a minute from near the top of the fall area. We were also faced with the problem of how the void in the cavity would be filled and secured to prevent further roof falls once the tunnel liner was installed. Inquiries were made as to what type of material was available that would be lightweight and would serve the purpose described above. Several materials were considered, the majority of which were finally rejected, and a decision had to be made on two remaining systems. One system was to cover the liner plate with a lightweight concrete approximately 18 inches in thickness on top of the liner plate. The other would be to clean up the fall and set the liner plate as cleanup progressed and as soon as the liner plate was installed after cleanup operations, then a bore hole would be drilled from the surface and the void completely filled with first a slag and clay mixture for stability purposes, then flyash filling the void to approximately eight to ten feet from the apex of the fall. The remaining area to be completed with concrete fill. The latter system was chosen primarily for the purpose of sealing off the water flow from the uppermost portion of the cavity.

The installation of the liner plate was started by laying eight-inch channel iron, 10-foot sections, on the mine floor parallel with the right and left ribs and then slid inby so as not to expose miners to unsupported roof. The first 18-inch width section of liner plate was completely bolted together on the mine floor and then raised into position by the workmen not exposing themselves to unsupported roof since the first ring was started at the outby edge of the roof fall. The next sections were installed by bolting two pieces of liner plate together and then installing them by bolts on the first semi-circle. This system continued throughout the cleanup operation.

Stability for the first few sections was by shoveling material between the outer edge of the liner plate and the rib. This void was filled to a height of approximately three feet and extended back approximately six to seven feet. As more liner plate was advanced, the stability of the structure increased.

As a safety precaution, four-inch flange, 14-foot diameter arch, was installed underneath the liner plate, the reason being if in the event additional roof falls occurred, it was not known whether or not the liner plate would be able to withstand a large fall before the void was filled. In addition, channel irons were installed at the base of the arch on approximately six-foot centers to prevent the arch from pushing together if a fall should occur. The only problem with this is that during the cleanup operation in which a continuous miner was used, the cats of the miner severely damaged these channels and it is questionable as to whether or not they were really needed. At one time during the operation, a small fall did occur, striking the tunnel liner; this did very little damage, it only dented the tunnel liner and did not cause any movement.

After completing the cleanup and installation of tunnel liner, a bore hole was drilled from the surface. This bore hole was 6¼-inches in diameter, cased to four inches. As soon as this was completed, filling of the void started; a slag and clay mixture was dropped. Flyash was then put in the void to within approximately eight to ten feet of the apex and the rest filled with concrete. Approximately 700 tons of flyash and clay were used. This reduced the flow of water to approximately 25 gallons per minute.

This method of providing protection for miners during the rehabilitation of difficult falls is, in our opinion, one of the safest methods that we have ever tried. The advantages are: very limited exposure to the workmen, all materials used being incombustible, the filling of the void prevents any future roof falls, eliminates necessity of testing for methane in a high cavity, and in this case reduced the flow of water by approximately 75 per cent.

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roof falls

IMPROVEMENTS IN ROOF BOLTING PROMISE GREATER SAFETY AND PRODUCTIVITY

production

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The most serious safety problems encountered in U.S. underground coal mines today are related to the roof bolting operation. Also, roof bolting is usually the operation which limits coal production of the section. The purpose of this paper is to review some of the research, developments, and new ideas related to roof bolting. Many of these ideas have originated at coal mines or with manufacturers. Most of the research on fundamental improvements has been funded by, and is a part of, the excellent and broad plans of the Bureau of Mines.

The Accident Record

MESA has published a report entitled "Analysis of Fatal Roof Fall Accidents in Coal Mines, 1972-1975."* This report details fatal accidents by occupation, by activity, and by other classifications not important to this discussion.

Roof Fall Fatalities by Occupation

Occupation	Per Cent
Roof Bolter/Helper	17.5
Continuous Miner Operator	13.5
Timberman/Jacksetter	11.9
Foreman	11.3
All Other	45.8

Roof Fall Fatalities by Activity

Activity	Per Cent
Setting Posts or Jacks	18.5
Operating Continuous Miner	11.8
Roof Bolting Operations	11.2
Operating Loading Machine	8.5
All Other	50.0

The explanation of the difference in the percentages on occupation and activity related to roof bolting is that the roof bolter and helper usually set safety posts or jacks during the course of the bolting operation, and many of the fatalities listed as having occurred during the activity of setting posts or jacks occurred to the roof bolter operator or helper.

At this time, MESA has not published comprehensive data on lost-time accidents. Nor has MESA analyzed recent data on lost-time accidents with respect to occupation, activity, and cause.

The Safety Department of Consolidation Coal Company investigates and reports on all lost-time accidents. The objective is to study the cause of each accident and to determine methods or procedures which would prevent reoccurrence. The reports are accumulated and subjected to data processing by computer.

The data generated by Consol's records consistently show that accident frequency is highest among roof bolter operators and helpers. For the year 1976, Consol's records showed for the five occupations having most accidents:

Accidents by Occupation

Occupation	Per Cent of Lost Time Accidents	Per Cent of Employees*
Roof Bolter	15.06	1.68
Laborer	13.71	1.32
Mechanic	13.26	1.28
Shuttle Car Operator	9.21	1.12
Supervisor	5.62	0.46
Other	43.15	—

*Clerical, management, technical, and staff employees excluded.

Further detailed analysis of roof bolting accidents at Consol mines during the

*Seals, E. D.: "Analysis of Fatal Roof Fall Accidents in Coal Mines." MESA, Health and Safety Analysis Center, Denver, Colorado.

years 1974, 1975, and 1976 have shown the following:

Roof Bolting Accidents by Causes

Cause	*Per Cent*
Struck by Rotating Tools	31.25
Caught by Rotating Tools	16.25
Roof Falls	18.75
Caught Between Boom & Top	7.50
Tripping and Falling	5.00
Caught Between Boom & Rib	3.75
Other	17.50

Conclusions from Accident Studies

The statistics of accident experiences and detailed studies of the circumstances of many accidents involving the roof bolting operation lead to the following conclusions:

1. There must be alternatives to the use of manually set roof jacks. The setting of a temporary roof jack requires that the man be exposed under unsupported roof for at least 15 seconds.

2. In certain states, all tests for methane must be made with a flame safety lamp, requiring the setting of many jacks to protect the man. This legal requirement must be changed or some acceptable substitute must be developed.

3. It should be possible to drill holes and set bolts without having a hand on rotating steel. It must be possible to start the hole exactly in line and the machine must be stabilized adequately so that thrusting the drill steel does not push the machine out of line. Alignment in starting the hole and holding the drill steel mechanically is especially important when drilling the smaller hole used for resin bolts. We call this condition of bolting without having the operator's hand on rotating steel "hands off bolting."

4. Self-centering hydraulic valves should be used on feed and rotation. Detented valves should not be used.

5. The panic bar on bolters should cause the oil to dump to tank in addition to de-energizing the motor. We have found enough coast in the motor after de-energizing that the panic bar affords little or no protection to a man caught by the machine.

6. Tram controls available at the operator's bolting station should move the machine at reduced speed only.

7. Canopy designs are not satisfactory in all cases. Especially in lower coal, the so-called flyswatter designs are a real hazard to the operator of a squirm type bolter if he trams the machine while beneath the canopy.

8. As might be expected, a serious risk of accident during roof bolting occurs when the regular operator or his helper are absent and must be replaced by an employee who is less skilled. Consol has developed written and detailed safe work instructions, and a man is only permitted to operate a machine for which he has been trained. But he cannot operate with the best skill if the machine is not his regular assignment. The labor contract makes management helpless to improve this situation.

Characteristically, a detailed industrial engineering performance study of a continuous miner or conventional mining section will show that the performance of the roof bolting machine limits the production of the section. Even when this is not so, relatively little improvement in the performance of other machines on the section will cause bolting to become the limiting operation.

Thus, improvements in bolting methods or machinery usually will immediately result in greater productivity. Because most of the new improvements in bolting promise safer operation or improved permanent roof support or both, the coal industry can look forward to greater productivity and improved safety.

Improvements in roof bolting are being developed and tried by manufacturers and by many mining companies. The Bureau of Mines has funded a number of contracts which if successful will contribute importantly to the technology of roof bolting or permanent roof support. With so many diverse efforts underway, it is important to set up some sort of organization of the information on and the discussion of the efforts. Accordingly, I shall try to put the various efforts in one of the following categories: improvements in the bolt; elimination of manually set roof jacks; hands-off bolting; cabs and canopies; bolter-transfer vehicles; components for automation; and miner-bolter. These categories are arbitrary, and it is obvious that many of the efforts can affect more than one of the categories.

Improvements in the Bolt

Probably the most significant improvement in bolts is the resin bolt, which has been in use long enough so it is hardly newsworthy. Information from suppliers indicates that the coal industry is continuing to increase its use of resin bolts as more mines find that in many cases it's easier to hold difficult top with resin bolts. The fact that resin bolts shorter than the mining height are holding the roof better than longer conventional bolts has great implications in the future development of automated roof bolters. This will be discussed further when automation of the bolting operation is considered.

An experimental variation of the resin bolt was tried by the Roof Control Branch, Technical Support, MESA. Their scheme involves cementing, with a small resin cartridge, a relatively short threaded sleeve to anchor a standard threaded 5/8-inch roof bolt. The sleeve has given very good anchorage of a standard bolt, but does not result in the unique properties of a full resin bolt. The idea is to try to improve anchorage without the high cost of a full resin bolt.

A new roof support invented by Dr. J. J. Scott of the University of Missouri in Rolla is known as the "split set." The split set is a long steel tube with a slot through its entire length. At first, the split tube was forced into a hole drilled somewhat smaller than the tube, but later a slug was developed to expand the upper end of the split tube. Tests using the split set have been conducted by White Pine Copper Co. and Old Ben Coal Co. While the split set has not received widespread use, the results reported from the test installations are impressive.

Elimination of Manually Set Temporary Roof Supports

A key provision of the Federal Coal Mine Health and Safety Act of 1969 requires that "No person shall proceed beyond the last permanent support unless adequate temporary support is provided . . .." Regulation 75.200-13 sets criteria for temporary supports where permanent support is required. A minimum of two temporary supports are required and the maximum space between supports and ribs or face is five feet. Frequently the roof control plan reduces this spacing, so that usually three or more roof jacks are set as temporary support prior to bolting. Because of state regulations in one case, where laws require check for methane with a flame safety lamp, commonly nine jacks must be set prior to bolting.

Also, allowed tolerances in spacing of roof bolts are variously interpreted by inspectors. In some cases, spacing called for on the roof bolt plan may be exceeded by as much as six inches, while in other cases the inspector may not allow any deviation which results in a spacing between bolts which exceeds the spacing of the roof control plan. The result of this emphasis on spacing between bolts is that often the bolter operator, using a measuring stick, will mark the bolt locations in advance of setting roof jacks. In this way, he will avoid locating roof jacks at points where they will interfere with the subsequent installation of bolts.

Thus, normal practice variously exposes the roof bolter operator or helper to functions carried out under unsupported roof. If the roof is marked before setting jacks, the man is exposed; and setting jacks exposes the man until the jack is adequately tightened. The record of fatalities and lost-time accidents from falls of roof which have occurred during setting temporary roof support underscores the need for methods of providing mechanical temporary roof supports (TRS) which are controlled from some point back on the machine and under previously set bolts.

A particularly good example of a manufacturer's response to this problem is the model DDM bolter built by J. H. Fletcher & Co. This is a large, dual-boom bolter using mast type drills and is usually equipped with a massive TRS which effectively sets two cross bars between the bottom and the roof. With a telescoping capability of each boom of nine feet and a swing of six feet, the machine can bolt a full 20-foot cut with just one machine move. It affords excellent protection for the operators during bolting. There is an adequate cab at the rear of the machine to protect the operator while tramming between working places. The Fletcher Co. has other versions of the TRS and other models of bolters adaptable to a wide variety of operating conditions, but the DDM is outstanding for use in reasonably high seams.

The Lee-Norse Co. has developed a TRS package called the "safety arm," which is integrated into their standard boom bolter mechanism. The safety arm thrusts from the same stabilizer pad used by the bolter boom, and its top moves in the same straight line as the drill pot. The top of the safety arm is a ring, available in a wide range of sizes, to allow the placing of cab boards of different lengths. In the preferred version, a hydraulically operated centralizer is attached to the ring. The centralizer is a few inches below the roof when the ring is set against the top and the centralizer jaws are in line with the line of movement of the drill pot. In most versions, the safety arm carries a safety canopy which projects over the operator when he is in the position of the bolting controls. The safety arm adequately meets MESA requirements for a canopy. The tram controls and the controls for positioning the boom and setting the safety arm are located five feet back on the machine, at a point where the operator would be under previously set bolts. Most of the Lee-Norse bolters are dual boom units mounted on a large, rugged chassis.

MESA's Technical Support Roof Control Branch, working with Valley Camp Coal Co., has developed a pair of temporary roof supports which are mounted at the front of a small, single-boom squirm type bolter. The temporary roof supports have hydraulically positioned base pads which act as stabilizer pads. The upper portions of the TRS carry hinged channel section bars which are parallel to the bolting boom. The stabilizer pads are pushed downward to accept the front weight of the machine and the channel sections are raised to provide sturdy support of the roof each side of the boom. Tram controls and controls for the TRS are set back on the chassis, about five feet away from the drill pot. This scheme was put together by Valley Camp and is a significant development of a TRS which can be retrofitted to single-boom squirm bolters—the machine which is in highest population in the coal industry.

An especially rugged TRS has been developed by Lee Engineering Division of Consolidation Coal Co. This TRS resembles a longwall chock with two double acting jacks and large wide roof pads which are resiliently mounted at the top of the TRS. There is a substantial base which contains hydraulic valves and other components. The TRS is carried by lugs added to the boom of a small squirm type bolter which engage saddle-like lugs on the TRS. With controls mounted back on the bolting machine, the boom of the bolter is raised until the roof pads contact the roof. Then the cylinders are extended to set the base on the bottom, the hydraulic pressure is raised to set the TRS between top and bottom, and the boom is lowered, separating the bolter from the TRS. The bolter is then able to operate freely to set a row of bolts just outby the TRS. The connection between the bolting machine and the TRS is provided by a pair of hydraulic hoses, like an umbilical cord. The valving on the TRS copies longwall practice, with pressure-operated checks to prevent collapse of the TRS if a hose is ruptured. This detachable TRS is also aimed at providing a safe, remotely set TRS which can be retrofitted to the much used small-boom type squirm bolter.

Doubtless, other schemes will be developed both for new roof bolters and for the large population of bolters in use. Little has been done in low coal, and even in high coal, experience is limited.

Hands-Off Drilling

The analysis of lost-time accidents by the Safety Department of Consolidation Coal Co. showed that almost half of the accidents involving bolter operators and helpers might be prevented by methods which allowed "hands-off" bolting. Hands-off bolting means that the operator's hand is not needed to stabilize the drill steel while it is rotating, either in drilling or in removing the drill steel from the hole. Also, since containment of the wrench is important to safe operation during tightening of the bolt, mechanical centralizing or holding of the wrench is needed to achieve full hands-off operation.

While mast type drills have had centralizers available for a long time, the most flexible and most popular boom type drill has not had any centralizer system until recently. The Lee-Norse safety arm, mentioned previously, can be fitted with a centralizer which is located near the top when the arm is set against the roof. Then, with the centralizer properly located, the hole will be started accurately and alignment is assured. Since one of the main causes of drill steel breakage is misalignment resulting from starting the hole out of line, the centralizer at the roof will reduce accidents resulting from drill steel failure. It will also eliminate the need for manual guidance. However, a top centralizer will not prevent bowing of the drill steel caused by excessive thrust.

Accurate alignment of the hole with a boom bolter can be obtained by using a short starter steel fitted with an alignment disc or collar. However, use of the short starter steel requires an extra motion up and down with the boom, and it is difficult to get the operator to use the starter steel conscientiously.

Another scheme which is working well is a deep hex chuck fitted to a standard drill pot. The hex chuck is about five inches deep and works with commercially available hex drill steel. If the mechanism which gives straight line motion and leveling of the drill pot on a boom type bolter is in reasonably good condition, the deep chuck will cause the hole to be started in line and the hole will be drilled straight. The friction of the deep chuck, while rotating the drill steel, is enough to grip the steel to pull it down out of the hole even when sticking occurs. The tightening wrench uses hex steel also, so it is aligned and retained. The deep chuck appears to be a practical and simple way to obtain hands-off operation. The penalty of this system is that the hex chucks made so far add to the height of the drill pot and can reduce the vertical travel of the drill pot.

The Galis Division of FMC has built a prototype single-boom bolter with a centralizer which can be adjustably located near the upper half of the stroke. The design of this unit looks very good, but so far the machine has not been put to work underground.

Other ideas being worked on include several designs of centralizers capable of being retrofitted to single boom bolters and various types of chucks which will retain the drill steel or wrench. Until these designs are proved by actual use, the details are not important.

Cabs and Canopies

Fortunately, the design and acceptance of cabs and canopies on all equipment is improving. Experience is being gained and lives are being saved. But the problems of adapting cabs and canopies to underground equipment is getting more difficult, as the industry is being required to provide them in lower seams.

The front or operating position canopy on a small squirm type bolter has not been a complete success. The squirm type bolter has a short wheel base, and springing or equalization is not possible. As this machine is tramming over uneven bottom, the overhanging ends of the machine pitch up or down severely. Since the canopy is at the front end of the machine, it must be lowered to give reasonable clearance for moving. If the operator is beneath the lowered canopy and trams the machine over an uneven spot, the canopy may come down on him. In low coal, where clearance for a man is reduced, this type of canopy has received the name of "flyswatter" for obvious reasons.

Regulations also require installation of a cab or canopy at the rear of the bolter to protect the operator while tramming the machine from place to place. This presents no unusual problems on a large chassis or new design, but when a cab is added to a small squirm bolter, it can hurt the maneuverability of the machine.

Some thought is being given to the idea of providing an extra-strong full cab at the operator's location which permits the operator to perform all the necessary functions while in a strong cab. This appears to be a very useful idea for eliminating setting of roof supports prior to bolting and for giving maximum protection for the operator. So far, interpretation of the regulations has prevented any trial of this idea, so nothing is known about the problems of operating a bolter from a full cab.

motor

Bolter Transfer Vehicles

The objectives of this system are to eliminate exposure of personnel to unsupported roof and reduce the risks and lost production caused by place changing. The system uses a continuous miner which is equipped with remote control. The miner operator controls the miner from the miner is discharged into a is always under bolted roof. The coal from the miner is discahrged into a hopper built at the front of the bolter transfer vehicle (BTV). The overlap of the hopper by the rear conveyor of the miner allows the miner to mine up to five feet while the BTV is stationary. The BTV has two boom-mounted, mast-type roof bolting drills which can set bolts as close as two feet from each side of the center line of the machine. The BTV is equipped with strong roof supports which give good temporary support of the roof in front of the bolters. The BTV has a conventional loading machine conveyor from the hopper to the standard swinging tail section. The BTV has potential only in those mines where the freshly mined roof is self-supporting from the distance from the face to the bolters, generally about 35 feet.

So far, two versions of the bolter transfer vehicle have been built. Bethlehem Steel Company's Research Division commissioned J. H. Fletcher Co. to build a BTV on rubber tires. The machine has a heavy steel cross bar inby the bolters which is set against the roof with hydraulic jacks. Also, hydraulic roof jacks on each side of the hopper are in front of the cross bar. Each bolter carries a small canopy to protect the bolter operator. The bolters are mounted between the wheels, and the hopper is in front of the front wheels.

The other version was built by Consolidation Coal Company's Lee Engineering Division. The chassis of this machine was taken from a Lee-Norse miner and widened to accept a 30-inch conveyor. The bolters are just in front of the crawlers, and the receiving hopper is in front of the bolters. The original arrangement of roof supports used two rows of jacks with cross bars in front of the bolters and two rows, having canopies thereon, behind the bolters. It was found that the inby jacks were too vulnerable to damage, so they were replaced with supports projecting ahead from the next jacks. The setting pressure on all of these jacks is adjustable and very low. Yield pressure on each jack is separately adjustable, and is set so that the jack would yield under a heavy load of 25,000 pounds. The heavy miner chassis on crawlers allowed the machine to accept loads of this magnitude without damage.

None of the machines built so far are in operation at this time. The original BTV built for Bethlehem was intended for use in southwestern Pennsylvania, as was Consol's machine. The Pennsylvania mine inspectors blocked the use of both machines for reasons which were unrelated to the machine. A Fletcher BTV was tried in Illinois by Old Ben Coal Co., but they had problems in their narrow entries. It is hoped that better conditions and improved attitudes will allow these machines to perform as well as they should.

Bureau of Mines-Sponsored Research

At this point, the discussion turns to the comprehensive research programs initiated and sponsored by the Bureau of Mines. The objective of the Bureau program is to bring an increasing level of automation to the bolting operation, starting with simple functions to eliminate operations, installing remote control which will move the operator to a safe location, then moving to the development of automated bolter packages or modules, and finally the marriage of bolter packages to the continuous miner. This will permit bolts to be set as close to the face as is reasonable and with a minimum delay to the operation of the miner. The problems of achieving remote or automated bolting are comparatively easy on paper when the length of the bolt being used is 1-1/2 to two feet shorter than the mining height. It is much more difficult to drill holes and to insert bolts which are close to or longer than the mining height. Fortunately, the development and the increasing acceptance of resin bolting, which generally uses shorter bolts, are improving the eventual utility of the simpler systems which place bolts shorter than mining height.

Components for Automation

The first attempts at some level of remote control or automation were the miner-bolter and the two remote control single-mast bolters made as part of the ISMS contract. The miner-bolter was designed for use in high coal and had four remotely controlled bolter modules which were expected to place four bolts simultaneously. The miner stopped mining during the bolting cycle. Unfortunately, the bolter modules never developed the high level of reliability needed to make this miner-bolter a success. The remote control, single-mast bolter which accompanied the miner-bolter was intended for auxiliary use only, and so was not in the cycle. Full experience with this machine was never developed, but it seemed to have a good capability. A second remote control mast bolter was designed for lower coal and was intended to be used in the cycle. Unfortunately, when the small experimental mine was developed, it was found that bolts longer than the mining height were needed, so this machine also never went on cycle.

A new contract with FMC in which Peabody Coal Co. is contributing technically and financially is now underway. This contract will develop a second generation remote control bolter with two mast-type bolter packages. The limited experience with the remote control single-mast bolters of the ISMS contract showed that the bolter system was too slow. The only way to speed up this machine is to have at least two bolter packages so that two bolts can be set simultaneously. The bolter masts will rotate to a horizontal position and the bolts and drill steels will also be stored horizontally to give more clearance for tramming. This machine is scheduled to go underground by the end of 1977.

The Bureau of Mines has funded three miner-bolter concepts. Wisely, the Bureau and its contractors have decided to prove out the bolter modules before combining multiple bolter modules with the remainder of the machine. The proposed machines will be described later in this paper, but it is appropriate to discuss the bolter packages at this point, because these packages install bolts shorter than the mining height. They go beyond the FMC remote control bolters in that they insert resin cartridges into the hole before inserting the bolt. One bolter

cyclic miners
bolting

module under development by Joy Manufacturing Co. uses a conventional mast-type bolter with a circular turntable that holds five bolts and one drill steel. A separate mechanical feeder moves into position to push a resin cartridge into the hole. The bolt must be one foot shorter than the mining height. The Joy bolter module is under test now, and the plans are to mount four modules on a crawler chassis.

Another contract with Jeffrey Mining Machinery Division of Dresser Industries, Inc. is developing a module which also uses a mast-type bolter mechanism consisting of a carousel holding seven bolts, seven drills, and seven tubes containing the needed resin cartridges. The module employs air mist dust control during drilling and uses the same air source to blow the resin cartridges into the holes. Bolts must be at least one foot shorter than the mining height.

The third contract is with Ingersoll-Rand Research. The I-R module uses rotary or rotary impact drilling with a mast-type mount. A rotary magazine holds five bolts with the necessary resin cartridges. Testing of the rotary impact drill head has achieved drilling rates of as much as 11 fpm into hard Indiana limestone. The module uses a short starter steel and then shifts the final steel into place. This long steel must be at least one foot shorter than mining height. The early rotary impact drill head was excessively noisy, but the newest model is expected to operate at less than 100 dBA. Obviously, if these automated bolter modules work successfully, they might be placed on a new generation of bolting machines to provide safer and more productive bolting.

The Bureau has funded a whole series of contracts to develop components to install roof bolts longer than the mining height. A key component, developed by Bendix, is the bolt bender-inserter. This machine grasps the bolt, bends it 90 degrees around a powered pulley, then straightens the bolt and runs it up into the hole. In this way, a long bolt can be placed horizontally in the bender and the machine places the bolt in the vertical hole.

The Bureau has funded five contracts for the difficult problem of drilling the long holes. Three basic schemes have been developed. One arrangement uses a jointed drill steel so that the hole is drilled in a conventional manner and when the drill head has traveled to the limit of its stroke, the drill steel is retained in the hole and an extension drill steel is added. This is similar to drilling a long hole manually, except that the handling of the extension lengths of drill steel is accomplished by the machine. The second scheme uses a special flexible shaft, designed for high torque-transmitting capability. The third method uses a segmented drill steel made of short precision segments which lock together into a straight rod under torque. Without torque or thrust or both, the segments unlock so that the pieces are flexibly jointed or free to swivel.

A contract issued to Ingersoll-Rand Research has been successfully completed with an underground demonstration in which over 4000 bolts were installed using jointed drill steel. In the test, a straight rotary drill was used, but the drill steel is rigid and strong enough to permit the use of rotary impact, if very hard material had to be drilled. The test bolter used drill steel in 18-inch lengths. A 1-3/8-inch hole was drilled, and standard expansion shell bolts were placed. One comment made on the drill is that it is slow because of the number of extensions used and the time needed to handle the extensions. It is hoped that this drill package can have continued testing to improve the unit and to further assess its utility.

Two contracts have been awarded to develop versions of a flexible shaft drill. The first was given to Foster-Miller Associates. The F-M flexible shaft consists of counterwound steel coils attached to a top tubular starter section which holds the drill bit. The inner steel coil is a closely wound spring of round steel wire. The outer coil is wound from rectangular steel and, under torque, the coil winds up so that the flat sides lock up, creating a rigid straight member. The drill head is set at the roof, frictionally grips the outer coil, and gives torque and thrust to the flexible shaft. The gripping is accomplished by hydraulic pressure alternately applied to reinforced plastic tubes, allowing one tube to turn and thrust while the other tube is repositioning to take over when the thrusting motion by the first tube has reached the end of its stroke. It is claimed that the drill head can develop 1200 in/lbs. of torque, 7000 lbs. of thrust, and a drill speed up to 500 rpm. The unit was underground for testing but saw limited use because of seal and other problems. Envirotech built a different flexible shaft drill which so far has not gone beyond laboratory testing. The flexible shaft is made of five wire-wound layers, three to transmit drilling torque, and two layers are counter. The flexible shaft is driven from the free end, and the shaft is contained by a guide mechanism outside of the hole. Torque of 500 in/lbs. at the bit is reported, which limits the applicability of this drill to soft material.

Two contracts use the segmented flexible drill scheme. Washington State University is beginning to test a robust design driven by friction pads. The projected design should develop 3000 in/lbs. of torque with 6000 lbs. of thrust. Until lab testing is followed by successful in-mine testing, the utility of the Washington State design is still unknown. On the other hand, the segmented flexible drill designed by Bendix Corp. has installed over 400 bolts in a mine in eastern Ohio. The segmented flexible drill performed very well, while the bolt bender and bolt tightener required relatively little help from the operator. Torque capability of the flexible drill is in the order of 7000 in/lbs., with 5000 lbs. of thrust. A smaller segmented flexible drill will be available for one-inch holes and will have a torque capability of around 400 in/lbs. The Bendix engineers found certain areas where engineering improvements could be made, and mine personnel suggested other improvements. Most of the improvements are under way now.

Another important contract awarded to Bendix has been funded to combine the flexible roof drill with a resin cartridge inserter, the bolt bender-inserter, and a bolt tightener. The contract was funded omitting the flexible roof drill, as it is expected that the best system developed by the five contractors working on flexible roof drill contracts will be used. This unit is intended to drill one-inch holes, insert resin cartridges, feed a plate into position, insert a 3/4-inch rebar bolt, and rotate and hold the bolt until the resin sets. With the operator in a cab and back under bolted roof, this machine should give safe bolting in lower coal.

Three contracts have been awarded to develop a drill to drill one-inch or smaller holes. The contract awarded to Bendix did not go beyond Phase I since Bendix was able to show how its segmented flexible drill could be modified to drill one-inch holes. Bendix's own studies have developed and improved drill head design and modified segmented flexible drills to drill one- or 1-3/8-inch holes interchangeably with the same drill head.

The other two contracts to drill holes one-inch in diameter or less use high pressure water jets combined with a drag bit. Colorado School of Mines has designed a laboratory unit and fabrication is under way. This unit is intended to operate with high pressure (20,000 to 40,000 psi) and low volume (0.5 to 1.5 gpm) water. The drag bit provides feedback so that when the drag bit encounters resistance, rotation speed is reduced.

The laboratory test unit of Flow Research, Inc. uses a lower pressure (less than 20,000 psi) and a greater water volume (up to 10 gpm). It will use a drag bit with feedback, also. These water jet drills are expected to be able to drill holes as small as 1/2-inch. Tests run so far seem to show that penetration of the water jets is very effective in sandstone, and the drag bits will clean off any projections left by the water jets. There are slight variations in hole diameter when different materials are penetrated, and these are expected to improve the anchorage of the resin. The reason for reducing hole diameter is to permit the use of 5/8-inch bolts and less resin. It would appear that the minimum useful hole size is about 3/4-inch, and that the technique will be restricted to drilling holes shorter than mining height.

Miner Bolter

While the Bureau lists the contract with Jeffrey Mining Machinery Division as the development of a miner-bolter system, the machine being developed can be described more accurately as a bolter transfer vehicle (BTV) equipped with four automated bolter modules. The bolter packages were described previously. The BTV is mounted on crawlers and provides cabs on each side at the rear of the machine so that the BTV operator and helper can initiate and supervise operation of the bolters in front of them. The four bolter modules are in front of the crawlers and behind a small receiving hopper. The miner to be used with remote control in front of the BTV will be a Jeffrey 120 Heliminer. Testing and improvement of the bolter modules is under way now. The BTV will be completed and assembled with tested bolters by January 1978. Testing of the completed BTV will be started then, and the entire system will go underground by mid-1978.

Joy Manufacturing Co. is working on one of the true miner-bolters. The bolter modules were described previously, and four modules will be assembled in pairs; each pair straddles a crawler. Initial testing will use a continuous miner frame from which the cutter boom, clean-up head, and conveyor have been omitted. After the four bolter modules have been thoroughly proved, the mining functions of the machine will be added and in-mine testing of the full system is expected to start in late 1977. In operation, the bolter modules, equipped with heavy 50-ton roof support jacks, will be anchored between roof and bottom by the heavy jacks. Then the miner is sumped and mines two feet, moved by hydraulic jacks which thrust from the bolter modules. The machine mines another two feet with the sump again provided by thrusting from the bolter modules. In this way, the miner mines four feet while the four bolter modules each are setting a bolt. When the four-foot mining cycle is completed, the roof jacks are lowered and the modules are moved ahead four feet to catch up with the machine, completing the cycle. The Joy miner-bolter will place four bolts in line and, at the beginning of the cycle, the holes will be started about 13-1/2 feet from the face. At the end of the cycle, the bolts will be installed and will be about 17-1/2 feet from the face. The machine will cut a 16-foot entry and the bolts will be on four-foot centers. At the back of the fixed frame of the machine there will be cabs for the miner operator and helper on each side of the machine. The back bumper which protects these cabs is about 28 feet 10 inches from the face, or about four feet further than today's standard hard head miner.

The Ingersoll-Rand miner-bolter is intended to meet virtually the same mining specifications as the Joy miner-bolter; namely, 16-foot entry width, 5'0" to 9'9" mining height (Joy 6'0" to 8'3"0, resin bolts on four-foot centers with maximum bolt length one foot less than mining height. The I-R bolter module was described previously. The I-R bolter module is 26 inches wide, too wide to be mounted in pairs with the crawlers between each pair. Instead, two bolters are to be mounted on the outside of the crawlers and toward the front of the crawlers. These bolters would set the two outside bolts 12 feet apart. The bolters which set the inside bolts would be located just behind the crawlers, eight feet back of the outside bolters. All of the bolter modules would set themselves by pushing jacks down to the bottom and up against the top. The bolter modules would be attached to the miner by loose slides, which would allow the miner to mine a little over four feet while the bolters are setting bolts. After setting bolts and mining four feet, the jacks on the bolters would be retracted, the bolters are to slide toward the front of the miner to their starting location, and the cycle would be repeated. All available space between the crawlers and a reasonable location for the rear bumper would be needed for the rear bolter modules and the slides. Therefore, operators' cabs *could not be located on the miner*, and the miner-bolter would be operated by remote control. One I-R bolter module is being built at this time. It will be mounted on a crawler chassis. The remainder of the contract has not been funded, and funding depends upon successful operation of the bolter.

Summary

In a survey such as this, it is impossible to cover every development, project, or contract in adequate depth or clarity. I hope, however that the first part of this paper has convinced you that we have *serious safety problems in the roof bolting operation* and that we can do something about them, even with the equipment which we are now using. When more people become convinced that *improvements in roof bolting* are possible, we will see a flood of new ideas. Some will hit the mark and may be much better than any I have mentioned.

In my opinion, the program and contracts of the Bureau of Mines related to roof bolting have the greatest potential for improving safety and productivity. There are *two shortcomings* in the Bureau's program as I see it. First, funding has not matched the need or the potential. I cannot believe that most of the potentially good ideas are being researched. Second, the importance of long-term underground trial of equipment and methods is not fully appreciated. Coal miners can be innovative and constructive. There is no piece of equipment important in mining today that was not modified and improved by the people who operate and maintain it. The prototypes developed under the Bureau's program need more of the same practical input.

The goal of this research is a family of modules, of limited height and width, able to set bolts of the required length and type in a wide variety of roofs. The *physical dimensions of the modules* are critical to their utility and applicability. Each succeeding generation should be better and more useful, but progress will be slow, expensive, and difficult. The Bureau *should be commended* for its vision in getting this program started.

THE STATUS OF DIESEL POWERED EQUIPMENT FOR UNDERGROUND MINING

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The National Institute for Occupational Safety and Health (NIOSH) was created by Congress primarily to perform research and provide technical services and consultation. We operate under four acts: PHS Act, CMH & S Act, OSHA Act and TSCA. Part of our function under the CMH & S Act is to make recommendations to MESA, the Mining Enforcement and Safety Administration, for health standards in the coal mining industry. Accordingly, the administrator of MESA asked us whether diesel engines should be prohibited in underground coal mines because of adverse health effects.

In preparing our reply, we already knew that there were toxic substances, including carcinogens, in diesel exhaust; however, we could find no adequate studies to determine whether or not these toxic exhaust components, at concentrations and in combinations found in mines and in combination with coal mine dust, would result in adverse health effects to miners after a working lifetime of exposure. Also, we were not aware of any data which supported the claim that diesel engines provided an increase in safety compared to electrical systems. Consequently, we did not feel that NIOSH could make a recommendation one way or the other, but we did resolve to conduct studies and accumulate data on this subject as rapidly as possible.

NIOSH has several ongoing research projects related to this subject. In a joint effort with MESA, we are studying the effects of diesel exhaust and silica on non-coal miners working underground. NIOSH has conducted medical examinations on over 5000 miners in 19 mines, while MESA has collected relevant environmental data. The combined results from this work should be available in about a year. coal

A retrospective mortality study of 12,600 non-coal miners is nearing completion. A component of this study is an attempt to determine standard mortality ratios for cohorts with varying degrees of exposure to diesel exhaust. Hopefully, this work will be ready for publication in about nine months.

We have also collected medical and some environmental data from five dieselized coal mines. The field work was completed this summer, and we are presently analyzing the data. More detailed environmental investigations will also be conducted in four of these mines during this coming year. The results should be available at the end of this fiscal year. It should be noted that most of these mines have only been using diesels for about six years. Because of this limitation and the small population of exposed miners available for epidemiologic study, the results of this work may be inconclusive.

NIOSH is also planning an experimental animal study of exposure to diesel exhaust and coal dusts to assess any adverse health effects. This project will take several hundreds of thousands of dollars and over three years to complete. The situation, then, is that available data appear to be inadequate for our purpose, and several years may be required to complete applicable studies.

Although NIOSH has not made a recommendation to MESA, we have expressed concern which we feel is well justified. Unfortunately, it has been interpreted by some that NIOSH has already arrived at a final recommendation. NIOSH is an institute dedicated to the principles of preventive medicine. Protecting the health and safety of workers is our only objective. Therefore, we are cautious in our advice, but we intend to address this issue openly and to make conclusions and recommendations based upon the best available scientific data and not conjecture.

While NIOSH has made recommendations for the maximum occupational exposures on a pure substance basis to some of the constituents of diesel engine emissions, we are concerned about the possible adverse health effects from long-term exposure to combinations of diesel exhaust components and coal mine dust. Without further scientific evidence we cannot state with any confidence that adherence to individual standards will provide adequate protection when mixed exposures occur. The occupational health community has recognized this problem for many years.

Another concern is the risk of mutagenesis and carcinogenesis from exposure to these mixtures of substances. Very little conclusive information is available on the long-term carcinogenic potential from mixed exposures. Known carcinogens, polynuclear organics, are, of course, present in diesel emissions and in coal mine dust generated from other sources. We also know that respirable coal mine dust and diesel-generated particulate may act as condensation nuclei for the polynuclear aromatic hydrocarbons and, thereby, in combination, alter the health effect. Again, we have no conclusive evidence to support or to reject this hypothesis. However, the fact that other irritant gases do absorb to respirable size particulate, with a consequent potentiation of their effect, is the basis of our concern. The issue facing us is obviously complex, and the consequences of a wrong decision made in the absence of substantive and confirmatory data are possibly grave.

NIOSH recently sponsored an international workshop in Morgantown, WV, for the purpose of gathering, in one place, persons with state-of-the-art knowledge in health, safety, and production research related to the use of diesel engines in underground coal mines. It was hoped that, from the results of this workshop, priority research could be identified and initiated to fill important gaps in our knowledge in this area.

The workshop in Morgantown consisted of plenary sessions, where the experience of the representatives from the participating countries were presented along with the views and positions of NIOSH, MESA, and the Bureau of Mines, and work sessions where four groups attacked different aspects of the problem. One work group addressed the aspects of emissions and engine attached control technology, one discussed environmental characterization and industrial hygiene aspects, another the health effects, and the fourth, safety and productivity.

Task groups

As anticipated, the work groups were unable to provide substantive data at this time; however, they did identify many areas where research efforts are required. For example, the Emissions Workgroup identified the need for development of specific emissions factors, duty cycles, engine design factors, and an in-depth assessment of control technologies applied to diesel emissions. The environmental characterization group identified sampling and analytical method development as a significant research gap. With respect to the common gases such as carbon monoxide, carbon dioxide, oxides of nitrogen, and sulfur dioxide, they felt that working concentration levels have already been determined. However, it was decided that extensive sampling and analytical development is required to characterize and differentiate diesel particulate, aldehydes, organic acids, and polynuclear aromatic hydrocarbons. Whatever data exist in these areas can at best be described as qualitative.

The health effects group examined the possibility of using data from foreign countries. Some data from the U.K. and Canada may be useful; however, due to the short exposure period and small population of exposed miners, results from U.S. data are probably years away. It was recommended that epidemiologic studies of cohorts who have been exposed to diesel for over 20 years be conducted to assess potential cancer and chronic respiratory disease risk. Some suggested cohorts are railroad roundhouse workers and diesel maintenance shop workers. They also recommended performing in vitro and in vivo toxicity determinations, allowing indirect assessment of dose response effects, and ergonomic studies to assess vibration, noise, heat, and safety aspects of diesel-powered versus electrically-powered mining equipment.

It was concluded that some safety and productivity data are available, but not in an acceptable format for comparative analysis. Definitive studies in these areas continues to be a high research priority. It is anticipated that the proceedings from the workshop will be available within the next several weeks. The proceedings will be available from our Appalachian Laboratory for Occupational Safety and Health in Morgantown, WV.

The complex issues which were addressed by the experts at the NIOSH workshop are typical of the problems we now face in the field of occupational safety and health. We can no longer base our health criteria on the assumption that toxic agents will act independently of one another, but rather we must examine carefully the entire exposure environment, with all its complex interactions, in order to determine toxic potential. The American Conference of Governmental Industrial Hygienists has recognized this for many years. Let me quote from the TLV pamphlet: "When a given operation or process characteristically emits a number of harmful dusts, fumes, vapors, or gases, it will frequently be only feasible to attempt to evaluate the hazard by measurement of a single substance. In such cases, the threshold limit used for this substance should be reduced by a suitable factor, the magnitude of which will depend on the number, toxicity, and relative quantity of the other contaminants ordinarily present."

It should be noted that diesel exhaust is mentioned by ACGIH as an example of a process associated with two or more harmful atmospheric contaminants. Two aspects of diesel exhaust serve as good examples of the problem. Carbon monoxide and nitric oxide are frequently considered independent of each other in terms of relationship to their environmental standards, but the effect of each is to diminish the amount of hemoglobin available for oxygen transport; therefore, their contributions to the standard should be calculated as additive effects.

A second aspect is the problem of condensed hydrocarbons on ~~particulate~~ particles. It is inappropriate to consider the health effect to result solely from the base substance forming the particles. It might be appropriate to use the measurement of particulate as an alternative to the more complex measurement of hydrocarbons, but the concentration limit to which such measurements are compared must necessarily be reduced.

We also face the problem of attempting epidemiological studies where only a limited exposure history exists. Should we continue to allow these exposures to occur, we might be injuring the health of many of our nation's miners; however, at this time, we have no evidence to suggest that this is the case. Our concern is based on the fact that carcinogenic substances, such as vinyl chloride, asbestos, and hexavalent chromium have long latency periods, and it may be that our miners have not been exposed long enough for us to observe these adverse health effects. We, however, realize we cannot wait until we are 100 per cent certain before making a decision on this issue. The possible consequences to the health of our miners and the burden of uncertainty we would place on industry demand a speedy resolution of this issue. It is for this reason that NIOSH will be conducting extensive diesel research over the next few years.

The Bureau of Mines recently completed two preliminary industrial engineering studies of tethered (electrical) shuttle car systems and load-haul-dump (diesel) systems in coal mines. They showed that diesel mines out-produced tethered shuttle car mines by a factor of two. With the President calling for a drastic increase in coal production to meet our nation's energy needs, this determination is clearly of national significance.

It should also be noted that further recommendations regarding the use of diesel equipment in underground coal mines will most probably impact on the metal and non-metal mines, where diesels are used extensively, with thousands of jobs and valuable mineral resources in the balance.

These challenges are of the type that all of us, as health and safety professionals, will be facing in the coming years. We must face them objectively and resist emotional appeals. But we must never forget that our primary responsibility is to the health and safety of the American worker.

COAL SEAM DEGASIFICATION PRIOR TO MINING

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Methane gas had been associated with coal mining for many years. In 1973 it was reported that during the preceding 66 years one-eighth of the U.S. coal mine-related deaths could be attributed to ignitions of methane and coal dust.[1] Unfortunately, these tragedies associated with mine fires and explosions have generated many myths regarding the safe mining of coal.

Methane is present to some degree in the majority of coal seams. The increasing demand for coal will necessitate the mining of deeper, more gassy coal seams at increasingly higher rates of production. It can therefore be concluded that future coal mines will generally have to deal with increasingly larger methane inflows into their workings.

It might logically be asked: Where does the gas come from? Methane is generated during the coalification process during which vegetable matter decomposes to form coal. A hardwood could be considered to contain 65 per cent cellulose, 30 per cent lignin, and five per cent wax and resins.[2] A softwood would contain more cellulose and less lignin. Gaseous methane, hydrogen, and carbon dioxide are products of the coalification process. The process continues from peat through lignite and bituminous coals, finally terminating at anthracite. As would be anticipated, the higher the coal ranking, the larger the gas volume generated.

The quantity of gas entrapped within a coal seam is generally much less than that generated during the decomposition process. Much of the generated gas escaped from the swampy depositional environment prior to the time an impermeable cover such as mud was laid down over it. The gas generated after this time of sealing could logically be expected to remain within the seam, provided no other avenue of escape existed (such as fractures extending to an outcrop).

The process of gas generation continues along with coalification. The process is a very slow one and, indeed, is continuing today.

The mechanism by which gas is contained within a coal seam differs considerably from typical gas reservoirs and is frequently misunderstood. The quantity of gas contained within a coal seam is partially dictated by pressure, but more so by the characteristics of the particular seam. The gas quantity present can be many times greater than that of a porous sandstone gas reservoir at the same pressure. Once the porosity, pressure, and liquid saturations are known, the determination of the gas volume contained within a porous media is a straightforward calculation based upon known pressure-volume relationships. However, when these relationships are used to predict the gas content of coal seams, the calculations yield gas volumes many times too small. For example, an extensively worked northern West Virginia coal seam is known to possess more than 20 times the gas volume calculated by commonly used pressure-volume relationships.

What then is the phenomena that allows coal to possess these large gas volumes?

Several authors have studied the gas emission histories from coal and found time-dependent exponential relationships. One might observe that this behavior is similar to that for a porous media. The time required for the gas emission from the coal sample to approach completion is, however, several orders of magnitude greater than that for a comparable sandstone. The emission of gas from coal samples is very temperature sensitive. The quantity of gas coal is capable of giving up can increase significantly for a few degrees increase in temperature.

It has been concluded from gas emission studies of various ranks and sizes of coal that the gas is not only trapped within the interstitial voids in accordance with pressure-volume relationships, but also that the gas molecules share a weak chemical bonding to the coal surface. This surface phenomenon has been extensively studied for many gas-solids combinations. The mechanism of weak bonding between gas molecules and coal surfaces has been termed sorption. A few investigators have found empirical relationships that describe these observations to within a reasonable degree of accurary. For the purposes of this discussion it is only important to recognize that coal can contain much larger gas volumes than sandstones at similar conditions and that this behavior is a result of a weak chemical bonding of gas molecules to the coal surface.

The degree to which a gas molecule can migrate through a coal seam can also have a great bearing on the gassiness of a coal mine. The gas mobility can differ significantly for different coal seams.

A coal seam might be considered to be comprised of an assortment of coal particles surrounded by fractures. Many coal seams have been found to contain two well defined, mutually perpendicular, vertical fracture systems. Frequently, one of the fracture orientations is found to be considerably more permeable than its perpendicular counterpart. As would be expected, mine entries perpendicular to the more permeable fracture systems generally encounter larger gas inflows. The larger, more permeable cleat system has generally been referred to as the face cleat, while its less permeable counterpart is often termed the butt cleat. It is important to recognize that two distinctly different regimes are in effect within a coal seam and that both respond to a pressure gradient. It is a straightforward conclusion that gas has an easy access to the mine, and that it can flow significant distances to the mine workings. Similarly, it is likely that entries driven perpendicular to the more permeable face cleats will encounter larger gas inflows. Typically these methane inflows are diluted with ventilation air brought, at great expense, from the outside. The air requirements necessary to dilute the methane inflows into a gassy mine can be quite significant. Air flows of 60,000 cfm are not uncommon during the development mining of a gassy coal seam. Even with these large volumes of ventilation air, significant gas quantities flow in through the ribs and faces and is often quite audible.

An approach to reducing the gas inflows is to create a low pressure zone between the mine and the larger pressure of the virgin coal seam. An often tried solution has been to drill vertical holes from the surface in advance of mining. This approach has generally met with only limited success. Holes of this type have generally produced only token amounts of gas because of their limited exposure to the coal seam. Attempts to create additional surface area around these vertical holes through hydraulic fracturing have generally resulted in improved gas production but, unfortunately, still insufficient to warrant their large capital expense.

Another approach to coal seam degasification in advance of mining has been to drill horizontally ahead of mining. This technique permits the gas to be "intercepted" before it reaches the mine. This degasification technique is not a new one. There are a number of references in the literature regarding the use of horizontal drilling for coal seam degasification ahead of mining. Interest in this approach appears to have waned during the last 15 years, probably because of the difficulty encountered in keeping

the drill bit in the coal seam for more than one hundred meters. The gas is then "captured" by the horizontal holes and is pipelined through the mine to a vertical vent where it exits to the atmosphere.

Continental Oil is currently conducting a research effort in this area. The project has had two primary objectives; namely, to develop a drilling technique that will permit the drilling of horizontal holes within a coal seam and to degasify an active development mining section. Both of these objectives have been attained.

The horizontal in-seam holes drilled ahead of mining have significantly reduced the methane concentrations in the ventilation air and thereby provided a safer working environment. Many experienced mining personnel familiar with the gassiness of the area are quite certain that the development could not have advanced without the methane reductions brought about by these horizontal degasification holes. More than 1300 meters of an in-seam horizontal drilling have been completed during the past year.

The pipelining of methane through a coal mine is a potentially dangerous endeavor, and the Mining Enforcement and Safety Administration has made recommendations regarding the underground pipelining of methane.

The possibility of a roof fall breaking the pipeline has been one of the major concerns. Various solutions have been offered. The one presently in use utilizes a small diameter, frangible, plastic tubing along the entire length of the pipeline. The plastic line is pressurized with an inert gas; it terminates at a spring loaded, self-closing valve installed on each well head. Unless pressurized, these self-closing valves will shut the horizontal degas holes in. Should a roof fall occur, the plastic line would fail much more readily than the gas pipeline and the valves will automatically close. The plastic type was selected because of its low impact strength. In actuality, no roof falls have occurred along the pipeline route since its installation.

A check valve is also installed at each well head to ensure against any back flow of gas which could theoretically occur between holes if the pipeline became plugged. A flexible tubular member is installed between the well head and the pipeline. The flexibility eliminates the possibility of a transverse force being applied to the well head as a result of pipeline movements—a condition that could be brought about by floor heaving. The pipeline routing in outside entries is kept minimal to reduce the exposure to floor heaving. These flexible hoses are housed inside triple-braided stainless steel wire cages that afford significant protection against crushing, impact, and abrasion.

Some well heads have gas-water separators and automatic dump valves installed. The removal of water from the pipeline enables the gas to flow from the seam more readily. The pipelines are always laid in return airways to eliminate the possibility that any leaked gas could reach the working places. The return airways are coursed directly from the mine.

A sensitive methane detector is located outby (or downwind) of the pipeline. The nucleus of the methane detection system, where the continuous signals from all the individual sensors are monitored, is located in fresh air. The cylinders contain an inert gas that is used to pressurize the small diameter plastic line along the entire pipeline route. The electronics contained in the larger enclosure continuously monitor the signals generated by the individual methane sensors. Should the methane concentration increase beyond a preset level, the gas pressure is released from the small diameter plastic lines and all well heads automatically shut in. The system has been approved by the United States Bureau of Mines for use in gassy coal mines and resets itself automatically. That is, should the methane concentrations later drop below the preset level, the inert gas cylinders will be reopened via a solenoid valve and the holes again allowed to produce. The system was designed and manufactured by Lee Engineering, McMurray, Pennsylvania, and has performed flawlessly since its installation nearly one year ago.

The pipeline is firebossed daily by certified personnel. The system is proving quite satisfactory, but is somewhat more complex than those normally encountered in coal mines and does require some additional training.

At the termination point of the underground pipeline it connects to the vertical vent hole where the gas flows to the surface. Security procedures require that surface installations are always fenced and locked. The discharge stack where the gas is released to the atmosphere is covered with insulation to reduce the likelihood of freezing. The produced gas is water saturated and quite a significant amount of condensation occurs when the warmer gas contacts the cool steel pipe at the surface. A flame arrestor was installed to eliminate the possibility of any flame propagation back down the vent.

Poles set on either side of the vent stack support a grounding system to offer lightning protection. The system was installed in accordance with the MESA recommendations and the National Fire Protection Code.

A check valve to prevent any gas back flow into the mine is installed.

The feasibility of coal gas utilization has been investigated and found to be economically unaffordable at the present volume of gas production. The coal gas produced form Continental Oil projects is not acceptable as a pipeline gas. It contains quantities of water and carbon dioxide in excess of the maximums specified by most pipeline agreements. The economics have also been found to be highly sensitive to transportation costs. The economic viability of coal gas marketing from Conoco's projects is under continuing review and is improving as continued horizontal drilling yields greater levels of gas production.

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SAFETY ASPECTS OF LONGWALL MINING IN THE ILLINOIS COAL BASIN

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Under the Bureau's Advanced Mining Technology Program, the Bureau of Mines and Old Ben Coal Company concluded a cost-sharing agreement in April 1975 with the objective of demonstrating that the Herrin No. 6 coalbed can be mined safely by longwall methods. Earlier attempts at longwall mining the Herrin No. 6 coalbed ended in failure on account of inadequate roof support by chocks, which could not maintain control of hazardous roof shale breaking in large blocks. However, in recent years longwall mining has appeared to be attractive again if novel designs, such as roof shields, can solve the instability problem at the face and achieve control of a hazardous roof where conventional powered roof supports failed.

The cooperative effort was guided by the following program, which emphasized safety in design and operation:

1. A premining study as a basis for equipment selection.
2. Roof support designed to control hazardous roof.
3. Face equipment with many safeguards.
4. Operation mode with safe face procedures.
5. Panel-to-panel transfer involving safe recovery, move, and reinstallation of equipment.
6. A ground and subsidence study to monitor the effect of mining on rock strata and surface.

Mine Design

Under the 1975 agreement between the Bureau of Mines and Old Ben, three panels, each 460 feet by 1740 feet, are to be mined by longwall methods at Old Ben No. 24 mine near Benton, Franklin County, Ill. The panel entries, in sets of three on 60-inch and 100-foot centers, are being developed by continuous miners and are 14 feet wide, except for the center entry, which is widened to 18 feet to accommodate the longwall power pack and the power center along the conveyor belt. The roof of the entries is supported by 5-foot resin bolts on a 5-foot grid. A sealing coat is applied to roof and ribs to shield them from the effect of the mine atmosphere.

The chain pillar, on 60-foot centers, is mined along with the longwall face, adding to its length. This pattern is unique in the industry and is designed to increase the recovery of the valuable resource. Three gateroads lead to the face: the headgate; the supply gate, tracked for men and supply transportation; and the tailgate.

Pillar extraction by continuous miner of the panel adjacent to the tailgate preceded the longwall mining and could affect roof control at the tailgate because the pillar is only 86 feet wide and is intersected by crosscuts.

Premining Investigation

Dames and Moore, geotechnical consultants, conducted the premining effort to lay the groundwork for selecting a roof support system designed to control hazardous ground and came up with the following results:

1. A minimum of six inches of floor coal should be left because the soft underclay fails at 300 psi, when wet, while the floor coal can bear at least 1000 psi, according to tests. Therefore, the con-

tinuous miner in the development and the shearer along the face must be directed to cut to the roof rock and to leave enough floor coal in order to maintain a mining height of seven feet in the 8.5 to 9-foot-thick No. 6 coalbed.

2. The rock strata columns, according to cores from boreholes, indicate that the immediate roof is formed by a very strong, fine-grained shale up to 70 feet in thickness that contains 20 to 30 per cent siltstone. The main top is composed of sandstone, limestone, and sandy shale. The floor consists of a soft underclay, two feet in thickness, underlain by limestone. A possible strata separation could occur between shale and limestone at 70 feet above the coalbed. Considering strata separation and cantilever action in the very stiff strata, a roof support system with a mean load density of nine tons/sq. ft. was recommended.

The direction of the main rock joints, N80°E, roughly parallels the planned longwall faces, and therefore favors caving. A large fault system, the Rend Lake Fault, was projected from earlier exposures. Trending north-south, it probably strikes within several hundred feet to the west of the proposed mining area but may affect the operation to some degree.

3. Stress analyses of chocks damaged during the operation of a longwall face in No. 21 mine indicated that a roof support must provide stability against lateral thrust and continuous canopy overhead.

Roof Support Designed for Hazardous Ground

The results of the premining investigation pointed to the selection of roof shields on account of their stability compared with other forms of powered roof supports. Shields can easily advance with the canopy contacting the roof. Other advantages of operation with powered shields follow:

1. Cavities do not have to be blocked by cribbing above the canopies.

2. The shields fully shelter the space where men work from falling roof rock.

3. Not much loose material is left on the floor, so manual cleanup work is not needed.

4. The shields advance fast because the space is free of rubble and the structure is stiff.

5. The "trampel effect" is less severe. German miners use this expression to describe roof deterioration above powered supports due to frequent setting and lowering. Shields take fewer steps walking from gob to face, because shield canopies are shorter than chock canopies.

6. The short span of exposed roof and little delay in supporting it achieve control of weak strata where earlier attempts at face mechanization by powered supports failed.

The Thyssen lemniscate-type roof shields satisfied selection criteria for the safest possible roof support as follows:

1. A mean load density in excess of nine tons/sq. ft.

2. A hydraulic travel from six feet to 10 feet in one stroke without the need for extension members so that cavities can be supported quickly.

3. A yield pressure of less than 6000 psi, providing a more reliable yield cartridge function than at higher pressures.

4. A span of 12 inches between canopy tip and face before the shearer pass.

5. Each half of the base can be lifted individually to overcome obstacles or steps in the floor.

The canopy tip of the lemniscate-type shield moves up and down in a nearby straight line which is the center part of a lemniscate or figure-eight curve; therefore, the span between canopy tip and face remains nearly constant and approximates one foot. The shield weighs 17.5 tons, and its overall length is 16.5 feet. The power pack supplies a hydraulic working pressure of 4350 psi, and the shield is designed to yield at 5500 psi, when spring-loaded relief cartridges open up. Thus, the working pressure amounts to 80 per cent of the yield pressure to maintain a strong thrust against the roof right after exposure and to prevent bed separation.

Great power is concentrated in the legs, owing to a special design. The two legs together can exert a thrust of 518 tons and yield to a load 656 tons. However, owing to the geometric design of the shield, a mechanical disadvantage reduces the forces that act on the canopy hinge. At seven feet of mining height, the shield can exert a thrust of 404 tons and sustain a load of 512 tons at yield.

In the one-web-back mode of operation, each shield is advanced by a stroke of two feet, equal to the cutter drum width after the shearer has cut by. However, the entire shield can be brought two feet closer to the face and operated in the none-web-back mode by lowering the canopy extensions, which are two feet in length and called "flippers" among Old Ben longwall miners. In the closed position the shield is rated to yield at a mean load density of 12.5 tons/sq. ft. before the shearer pass and at 10 tons/sq. ft. after the shearer has cut by. The tendency of the shield base to dig into the floor is counteracted by having the advancing ram mounted in an inclined position.

Each shield is controlled from the adjacent unit to protect the shield operator while he actuates the controls. The shield can be advanced while automatically maintaining a soft roof contact to keep the roof intact after exposure. But a manual override control allows him to retract the legs to overcome roof irregularities.

The hydraulic system must have enough capacity to coordinate the speed of the shield with the shearer haulage speed and to keep the delay of supporting the roof after exposure to a minimum. Hydraulic pressures in the legs are a measure of roof loads. Therefore, each leg is fitted with a piston-type load indicator to monitor roof loads and to provide early warning of impending approach to the yield point.

A safe travelway along the face is provided.

Dust originating from the gob during shield advance is controlled by automatic sprays mounted on the canopies, and the working space is sealed by sideplates fitted between the shields and maintained by springs and hydraulic cylinders.

To assure the safe performance and reliability of the shield and to discover flaws in design, components and entire prototypes were subjected to a battery of tests in the facilities of the German Research Center for Rock Mechanics and Roof Support, in Essen.

Face Equipment

The shield line consists of 95 units installed on five-foot centers. Three shields are placed across the headgate and two across the tailgate. A power pack comprises two pumps, each powered by a 75-hp motor and supplying 25 gpm of five per cent oil-in-water emulsion to a dual pressure system, one mixing tank, and two filters. A total of 150 single hydraulic props, with a range of seven to 10 feet, were procured and placed into the three gateroads and crosscuts to within 80 feet from the face. They are extended by connecting them to the hydraulic system via hose and quick coupling and yield at 44 tons if properly blocked. According to European and U.S. experience, rows of single props, by virtue of early bearing capability, control convergence in gateroads better than late-bearing cribbing.

Several props, properly blocked and instrumented with hydraulic gages, provide an early warning system against excessive abutment pressures, which have never occurred.

Following mining, transportation and communication equipment is in use:

1. A modified Eickhoff* EDW 2 x 100 L double-ranging drum shearer with two air-cooled 100-kw motors; 54 by 24-inch drums with two-start spiral vanes turning at 63 rpm; each drum equipped with a cowl to clean the shearer track; dust control by water sprays both integrated with the drums and mounted on the machine frame; 26-mm haulage chain, with hydraulic tensioners mounted on the

*Reference to specify equipment does not imply endorsement by the Bureau of Mines.

head and tail terminals of the conveyor and connected with the hydraulic system of the shields. The tensioners compensate for the elastic stretch of the haulage chain and provide uniform chain loading regardless of the position of the shearer along the face.

2. An Eickhoff EKF 3 single-chain conveyor with a 150-hp drive at each end. Controls of belt conveyor, stage loader, and face conveyor are interlocked in sequence so that belt stoppage will shut down stage loader and face conveyor. But the face conveyor can only be restarted by activating a reset switch located at the stage loader after a warning is sounded over the page phone system. Emergency cutoff pushbutton stations are placed every 50 feet along the panline. If the conveyor is shut off at such a station, it cannot be started except at the same station. Before the conveyor is restarted by activating the switch, a warning must be sounded over the nearby page phone.

3. A set of furniture: spill plates, ramp plates, and the Bretby cable handler which protects the shearer cable and water hose from mechanical damage.

4. A stage loader mounted on rubber tires, attached to the face conveyor head and swivelled on the belt tailpiece, which allows retracting of the conveyor belt in 12-foot increments. A chunk breaker is provided.

5. A 42-inch retractable belt conveyor with solid structure suspended from the roof by chains.

6. A power center for 440v.

7. A communication system by page phones placed at 100-foot intervals along the shield line.

Face Installation, Training and Longwall Operation

The face conveyor with its furniture was installed first in the staging room, which was tracked with three rails to facilitate moving the shields. The shields were assembled in a 12-foot-high rigging chamber at the head of the staging room as they were hauled in from the shaft bottom in parts. After assembly, each shield was maneuvered into place with a S & S Unatrack battery-powered front end loader. The shearer was introduced at the head end of the panline. Head drive and stage loader completed the installation. Face operation started on September 3, after the entire electric system was inspected and approved by MESA.

Prior to initiating the face operation, Old Ben conducted an intensive training program for face crews, UMWA officials, supervisors, and maintenance personnel at the nearby Rend Lake College. Management staff, including the vice-president of operations and corporate director of safety, explained the reasons behind Old Ben's decision to reintroduce longwall mining. A three-dimensional model of the longwall served to demonstrate the sequence of operations. Hazards peculiar to longwall mining were identified, such as whipping shearer haulage chain, running face conveyor, advancing conveyor and shields, coal and rock fall from face, flying particles of coal and rock, bursting hydraulic hoses and fittings, and handling single props.

At the face, manufacturers' service personnel and industrial engineering staff instructed each crew member individually to operate each piece of equipment with hands-on training under one-on-one supervision.

Maintenance personnel received instruction in troubleshooting techniques and in operation, maintenance, lubrication, and electric and hydraulic circuits of the shearer. The shearer manufacturer provided an instructor and supplied a cutting motor and a haulage box wired for hands-on training.

Safety and job skill training and retraining have been conducted since the start of the operation.

The operation mode is unidirectional. Crews remain in intake air while the shearer is cutting, and the shearer cleans its track before the conveyor is rammed over to the face. The intake air, at a volume of 25,000 cfm, reaches the face via the supply gate from where the main split of 20,000 cfm travels to the tailgate and the remaining 5000 cfm goes to the headgate. The major part of the return air reaches the bleeder via the gob. The sensor of the methane monitor is mounted on the conveyor taildrive. Power is cut off if CH_4 concentration reaches one per cent.

Roof safety in longwall mining depends on adequate load-bearing capacity, stability, and reliable function of the roof support. The characteristics of shields, such as advance under soft roof contact, the speed of advance, great thrust against the roof when set, and a load density at yield up to 12.5 tons/sq. ft., helped to provide roof control at the face. The shields maintained roof control even when the face struck a roll which displaced the entire coalbed and caused rock falls 20 feet in height, well above the maximum shield range of 10 feet.

The effect of pillar extraction by continuous miner preceding the longwall mining in the panel adjacent to the tailgate was not noticeable to any extent at the longwall face. But the tailgate came under some roof pressure, which was sustained by the hydraulic jacks, in addition to the two shields in the tailgate, and by cribs built across the crosscuts.

The occurrence of coalballs, sideritic limestone concretions that ranged in size from golfballs to footballs, posed by far the greatest problem to mining. At first the tailgate development struck two large pods. The retreating longwall face encountered solid accumulations at the tail end and in the center. Unlike pillar extraction, a longwall cannot bypass such problem areas. It took 7½ weeks in November and December to work through the pods, which had to be blasted where they displaced the entire coalbed or stuck to the floor. Illinois state mine law permits blasting only when nobody is in the mine. Many shearer picks were consumed, and the drum was damaged and had to be replaced.

The nine-man longwall section crew consisted of: two shearer operators, five shield men, one mechanic, and one foreman. The first panel delivered a total of 205,000 tons of coal on a 1734 foot length during 162 working days. The occurrence of coalballs during 7½ weeks caused a serious setback in productivity. Daily averages in April were 8.2 cuts, 14.5 feet of advance, and 1708 tons. At best, the shearer achieved 15½ cuts, or 3317 tons, in 24 hours.

Panel-to-Panel Transfer

Recovery of longwall roof support is always onerous and requires careful planning. But shield recovery poses an unusual problem because part of the structure is under the caved rock. The weight of the Thyssen shield, 17.5 tons, adds to the difficulty of moving. The method of equipment recovery in a recovery room prepared with a very strong roof support was unique and was designed to handle the heavy shields in the safest possible manner. In the recovery room, rails were installed in the roof shale to reach over the longwall block. The inby ends of these rails would come to rest on the canopies of the shields at the approach of the face prior to holing through. The outby ends were supported by hydraulic props set in the recovery room. Track was laid to facilitate removal of the shields.

As the face approached the recovery room within 10 cuts, or approximately 20 feet, wire mesh was threaded over the canopies to form a protective mat under which the recovery could take place unimpeded by debris from the caved roof rock. As expected, the rails installed in the roof shale above the coal and supported by hydraulic jacks under the outby ends helped to secure the roof during the critical moment on the evening of May 11, 1977, when the face holed through into the recovery room. The majority of shields yielded a little until they could be advanced to the bolted roof of the recovery room.

The face equipment was removed in the following sequence: shearer, panline, and shields. The shearer was taken outside for overhaul. The panline, in lengths of three pans, was transferred to the staging room for the new face. Each shield, refitted with longer hoses and under adjacent control, was advanced from the shield line, step by step, under its own hydraulic power. Once on the track of the recovery room, the shield was disconnected from the hydraulic net and pulled by Unatrack battery-powered front end loader to the headgate, where it was laid on its side on a rail dolly for transportation to the new face. Roof control during shield recovery was maintained by a moving bulkhead formed by two shields advancing side by side along the rib side of the recovery room. A crib took the place of each shield removed.

Each shield, lying on the rail dolly, was hauled on the track by Unatrack to the tail end of the staging area, where it was uprighted and set on the staging room track under control of another Unatrack. It was then maneuvered into its place in the new shield line. Introduction of the shearer over the tail end of the panline and connecting the tail drive with the conveyor concluded the move. Operation started on August 1, 1977.

Ground Control and Subsidence Study

The rock mechanics monitoring program was designed to achieve the following objectives:

1. To provide early earning capability by measuring strata movement and pressures during mining so that corrective measures could be taken quickly should an unusual situation arise.

2. To evaluate the adequacy of the roof support system and to derive design criteria for future longwall mining efforts.

The program included the following tasks:

1. Measuring convergence in gateroads and differential rock and floor strata movement to detect bed separation.

2. In situ pressure measurement by installing stressmeters of the vibrating-wire-gage type in coal and floor of the longwall panel and the chain pillars.

3. Tracing the loading profile in the roof support system by measuring hydraulic pressure with load indicators, gages, and recorders.

4. Determining vertical extent of caving by boreholes from surface to coalbed by Time Domaine Reflectometry.

The Illinois State Geological Survey was also involved in providing early warning of unusual conditions by mapping the longwall panel area as it was developed and mined.

Surface subsidence studies were initiated over the longwall mining area to measure vertical and horizontal movements, including tilting of monuments. The shape of the subsidence trough and the angle of draw are to be determined to indicate whether the mining will affect surface structures.

Conclusion

So far results have been encouraging. A low accident rate is an indication that the longwall system may be safer than room-and-pillar mining. Of course, Old Ben's intensive training paid off. But the use of shields appears to be the major factor in maintaining roof control. The characteristics of the selected shield satisfied criteria by deploying the highest yet mean load density for any shield. On the other hand, panel-to-panel transfer of the shields posed an unusual problem.

The rock mechanics study indicated the extent of caving in the strata profile but did not reveal any damaging rock mass movements and loads.

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A NEW CONCEPT FOR WATER SPRAYS ON CONTINUOUS MINING MACHINES

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Prevention of face ignitions is a continuing item of research in Consolidation Coal Company's Southern Appalachia region. In the past, machine-mounted diffuser fans were our main deterrent to face ignitions. Diffuser fans mix and dilute liberated methane with incoming ventilation air to eliminate potentially explosive pockets of methane. These fans, however, are a major source of maintenance downtime and noise.

Research by the Bureau of Mines and others seemed to indicate that chances of frictional ignition of concentrated methane pockets were reduced at cutter bit speeds under 400 fpm. British research found a limiting bit speed of 250 fpm for severe steel-on-sandstone frictional conditions. Lower bit speeds combined with diffuser fans have reduced but not completely eliminated ignitions.

Extremely high downtime on diffuser fans in low coal made finding an alternate method of methane and ignition control at the face mandatory. Reports of successful use of venturi water sprays by United States Steel Corporation prompted us to investigate water sprays versus diffuser fans for the prevention of face ignitions on continuous miners. This program was begun in September 1976. Venturi water sprays are available commercially, so for comparison purposes, several continuous miners were equipped with the sprays in addition to diffuser fans.

A second and more important investigation into water sprays for face ventilation control was brought to our attention by a report to the USBM by Foster-Miller Associates Inc., entitled "Guidelines for Optimum Methane Control at the Mine Working Face through the Installation of Diffuser Fan Systems and Modified Water Spray Systems on Continuous Mining Machines." Foster-Miller indicated that the large effect of conventional continuous miner water sprays being "on" versus "off" during diffuser fan optimization tests prompted the USBM to alter the contracted program to include a parallel test program using just water sprays for methane control at the face instead of diffuser fans.

Discussions with Foster-Miller revealed that open hollow cone water sprays were effective in inducing air movement and were, in Foster-Miller's opinion, more effective than venturi sprays. A visit to Foster-Miller's test mine mock-up for methane control testing revealed that water sprays were indeed very effective in moving air. We were particularly impressed with the ability of the hollow cone water sprays on the sides and rear of the machine in moving fresh air up the rib and across the face to the curtain.

Since Foster-Miller's water spray system exhibited capabilities of not just methane diffusion but also means of increased air circulation across the face for methane evacuation, it was agreed among those directly involved that we would take the necessary steps to verify Foster-Miller's guidelines on an actual continuous miner in our Southern Appalachia Region.

The two mock-up test mine entries at Foster-Miller were designed to give testing flexibility under varying simulated mining conditions as follows:

1. The test entries are 100 feet long, 16 feet wide and six feet and four feet high respectively.

2. Main ventilation is variable from 3000 to 20,000 cfm, exhausting either through 18-inch vent tubing or line brattice as required.

3. Methane is introduced through a perforated manifold extending full height across the face and two inches from the face along each rib. Methane emission is controllable from 0-100 cfm.

4. Diffuser fan delivery variable to 7000 cfm (1000 cfm was generally used).

5. Twenty-four water spray nozzles were used at a pressure of 200 psi and 41 gpm total.

6. The 11CM continuous miner mock-up is capable of cutter head rotation and cleanup motion.

7. Both the miner and the entry can be converted from full face to a two-pass mining test mode.

In order to compare the effectiveness of different methane control configurations, Foster-Miller derived a term, FVE—face ventilation effectiveness, defined as the average methane concentration in the return (calculable) divided by the average methane concentration (measured). An FVE of 60-80 per cent is good, 100 per cent means perfect diffusion, greater than 100 per cent would indicate little recirculation and nearly all of the available ventilation air moving across the face so that the average methane concentration at the face approaches half that behind the curtain (concentration at the face is zero on the upstream side). Average concentration at the face does not entirely preclude high concentration pockets, however.

As mentioned previously, the USBM should be complimented on changing the course of the optimized diffuser fan program to include optimized water spray tests for methane control. This came about upon their noting a two to one improvement in FVE in diffuser fan tests "with" vs. "without" normal water sprays. Better FVEs were finally realized with water sprays than with diffuser fans, probably because of increased evacuation of diffused methane.

A series of volumetric efficiency tests of various commercially available spray nozzles, cfm/gpm, indicated that fewer, smaller size, hollow cone type nozzles were the most effective at moving air at 200 psi water pressure. Volumetric efficiencies of 190 (150 cfm ÷ 0.8 gpm) were observed per nozzle, the water sprays moving air by virtue of entrainment and momentum transfer.

Included in some of Foster-Miller's original nozzle testing was a venturi-type spray nozzle which showed less efficiency than bare nozzles. Foster-Miller recently tested a single unit flat type venturi spray unit and also a cylindrical type venturi spray unit versus the identical bare nozzles, and found less air was moved if the spray pattern touched the venturi shroud, otherwise no difference in efficiency was apparent.

Findings and Guidelines

In addition to showing that better face ventilation effectiveness is possible with water sprays than with diffuser fans, Foster-Miller also advanced the following findings and/or guidelines for water sprays:

1. FVE increased with increased spray water pressure (200 psi ÷ 100 psi yields 117 FVE ÷ 78 FVE) but not in direct proportion.

2. Water flow rate through the nozzles varies with pressure-about in the same ration as FVE varies with pressure.

3. No effect on FVE could be observed with gathering arms operating or not.

4. An increase in methane liberation results in a proportional increase in methane concentration at the face.

5. An increase of three in main ventilation results in less than a factor of two decrease in average methane concentration at the face. The FVE is actually less for 9000 than for 3000, since the average methane in the return is reduced by a ratio of three.

6. A vent tube or curtain set back of 20 feet instead of 10 feet results in only a 40 per cent decrease in FVE.

7. Air circulation across the face results in recirculation when it cannot be captured in the return.

8. The first modified water spray system out-performed the optimum diffuser fan system at the good ventilation condition (9000 cfm at 10 feet tube set back), but not at a poor ventilation condition (3000 cfm at 20 feet tube set back).

9. The optimum water spray system at 100 psi and 15 gpm performs as well as the optimized diffuser fan system at 1000 cfm. At 200 psi the optimum water spray system will out-perform the optimized diffuser fan at 1000 cfm.

Conclusion: A continuous release of more than 20 cfm of methane makes it impossible to meet the one per cent requirement on methane concentration with more than five feet set back of the tube withut auxiliary means to move the air to the face, even with good cfm of air.

The following are guidelines for water spray installations:

1. For minimum water consumption, use hollow cone spray nozzles with 70 to 90 degree spray angles that use less than 1.5 gpm at 200 psi.

2. Orient nozzles to aid and complement the normal air flow pattern and create a circulation that sweeps across the face.

3. Keep nozzles away from high energy air streams caused by other nozzles.

4. Locate nozzles as far back on the machine as practical to pull fresh air to the face.

5. Stop adding nozzles to the system when air recirculation approaches the operator's position.

6. Use small nozzles near the floor to block back flow of recirculating air.

Verification of Optimized Water Sprays on 120L Heliminer

Following the decision on November 1, 1976 to try the F-M optimized water spray system on a rebuilt miner, various plans were considerred to verify our potential effectiveness compared to Foster-Miller's lab results.

It was decided that a less sophisticated test program, initially above ground, in which air directions and velocities could be compared would provide sufficient verification of water spray effectiveness to warrant underground tests and/or request for MESA approval. We surmised that if air flows, velocities, and directions were comparable, then methane concentrations should also be comparable.

A test section 48 feet long, 20 feet wide ad four feet high was constructed adjacent to the Pocahontas Central Shop. The last 16 feet of test entry wasa constructed of plywood with three 9 x 12-inch hand holes in each 4 x 8-foot plywood sheet so that air directions and velocities could be measured by reaching in the top, face, and rib areas with a hand-held conventional propellor-type air velocity meter. While we recognize that the accuracy of air velocity measurements taken with an anemometer in and near water sprays is far from perfect, this method worked and was better than other methods of velocity measurement which we tried or considered. A line curtain was constructed two feet off the right hand rib to within 10 feet of the face. Main ventilation was provided by a 24-inch diameter, 20 hp, 3450 rpm Joy section fan rated at 8000 cfm at 6-8-inch water gauge. We estimate that this provided only about 5000 cfm on our test section, considering the excess leakage in our system. Air velocities behind the curtain roughly verified this.

In general, air flow directions were determined by smoke introduced from a Bernz-O-Matic insect fogger using vaporized mineral oil as smoke. In some of the latter tests, colored balloons were hung across the entry, and their increased declination with air flow via the water sprays was recorded on movie film. Some of the smoke flow patterns were also recorded on film.

A 120L miner fitted with the same nozzle complement as Foster-Miller's "optimum water spray system" was located in the left-hand corner of the 20 foot wide entry. Air flow in the entry with main ventilation of about 5000 cfm but the water sprays not activated shows dramatically how the bulk of normal ventilation air short circuits to the curtain and does not sweep the face. These air flow directions were developed with smoke traces and appear very succinctly on movie film made during the test program.

A dual complement nozzle system was installed on the 120L Heliminer in question since the line curtain may be on either side of the machine. A complete mirror image dual system was installed with the necessary water valves to switch from the RH curtain system to the LH curtain system as required.

Foster-Miller indicated that better face ventilation was achieved with the optimized water sprays with the cutter head in the "up" position and that different air flow patterns occured with "head up" opposed to "head down" for a given set of main parameters. In view of this difference, our test data was was also collected in both "head up" and "head down" situations.

Air directions and velocities were observed with the 120L miner in the left-hand corner of the entry with the head up. Localized recirculation patterns were observed at the face; these were later found to be caused by the sprays immediately behind the cutting elements. Velocities were measured on top of the machine, roughly one foot below the roof.

In the "head down" position some differences were seen in recirculation patterns. The 540 fpm velocities at the face and approaching the right-hand corner, applied roughly over a four square foot. cross section, indicate an estimated air flow to that corner in excess of 2000 cfm.

Since two-pass mining is used very extensively in our Southern Appalachia region, a test configuration was developed with the 120L in a simulated four-foot deep sump cut on the right-hand side of the entry with the line curtain on the right—the full complement of water sprays still being used. The 500 fpm and 400 fpm velocities at the left-hand exterior corner of the sump cut, when applied to a conservative four square foot cross section, would indicate 1600-2000 cfm of fresh air moving into the sump cut and within reach of the cutter head sprays. A 640 fpm recirculation velocity in the left-hand interior corner of the sump cut would indicate around 600 fpm x 1-1/3 feet x 1-1/2 feet or 1200 cfm to sweep out the corner and diffuse any gas present.

An 11CM continuous miner was equipped with four-unit flat venturi spray manifolds on top of the cutter boom. One unit was mounted immediately behind the ripper chain, plus one each on the sides immediately to the rear of the cutter heads. Thre were also two single nozzle flat units on each side of the front, mounted on the cutter motors, and impinging on the outby ring of bits. Finally, two spray units were oriented so as to point forward and down into the throat of the conveyor where it exits from under the cutter boom.

The purpose of these 11CM tests was to provide a comparison of the two current state-of-the-art water spray systems. The smoke traces and visible reactions from the sprays were very similar to those exhibited by diffuser fans only on the 120L. The velocity directions showed that a large portion of the main ventilation air is short circuited to the curtain, but that the face nozzles do create recirculation and turbulence at the face for diffusion purposes. This system does not provide the large sweep of air along the side of the machine and across the face which we found is the main advantage of the Foster-Miller system. The addition of banks of side nozzles to the 11CM should be able to force more main ventilation air to the face area and perhaps compare to the Foster-Miller optimum spray nozzle system.

Summary and Conclusions

These tests were very educational in face ventilation principles as follows:

1. Main ventilation to the working face will, on its own, short circuit to the tube or curtain.

2. The set back of the tube or curtain (we used 10 feet in our tests) indicates how near to the face the unaided short circuit air currents reach.

3. At a given curtain set back, increased main ventilation will move the short circuit air currents closer to the face, but not in proportion to increased cfm.

4. To get effective face ventilation, we must pick up the incoming fresh air in the main air stream to the rear of the inby short circuit flow boundary and redirect the fresh air to the face. If the water spray nozzles of the diffuser fan intake are inby the short circuit air stream boundary, they cannot be totally effective. Excessive amounts of recirculation air at the face with either nozzles or diffuser fans will build up and roll back toward the operator.

5. The optimized water spray system will redirect the fresh air up the rib and across the face. The quantity of air will be a function of the number of air redirection nozzles employed and their capacities in cfm gpm.

6. As nozzles are needed on the front of the machine for dust control, we should endeavor to install them in such orientation as to complement the air flow across the face.

7. A trade-off exists between the degree of methane diffusion and the degree of methane evacuation for methane control at the face. The trade-off wil be affected by main ventilation levels and dust suppression requirements. We have verified the effectiveness of the optimized water spray for methane control; however, the optimized total package of methane control plus dust control may dictate modifications.

8. We are convinced that water sprays are as effective as diffuser fans for methane control at the face, and probably more effective. More importantly, the water sprays offer tremendous flexibility in customizing optimum systems for different mining conditions and configurations such as curtain or vent tube location and reversal.

At this writing, the Consolidation Coal Company has water spray systems in operation or in process at Itmann, Maitland, Bishop, Oak Park, and Loveridge Mines.

FEDERAL MINE HEALTH AND SAFETY PROGRAMS: WHAT IS PAST IS PROLOGUE

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Department of the Interior, Washington, DC

Our commitment to mine health and safety is a strong one. We fully believe that the right of the American worker to a safe, healthful workplace is fundamental and that the government's duty is to ensure that this right is a daily reality.

The past few years have brought into focus a long-developing energy crisis and, consequently, a crisis in the continued development of a strong industrial foundation for our nation. For years, responsible persons have warned that fossil fuel resources are finite. Last spring, President Carter warned that the alternative to changes in our life style—the ways in which we use, and too often waste, energy—may be a national disaster. The next few years will be critical in developing an independent energy base for this country and establishing the role coal will play in this base.

Two of the seven key objectives in the President's proposed national energy plan are particularly applicable to the coal industry. The cornerstone of the President's plan is energy conservation. The other objective is to increase coal production to more than a billion tons by 1985. It is the latter that I shall emphasize in this presentation.

The national energy plan would stimulate a conversion to the use of coal by: (1) prohibiting new factories and utilities from burning oil and gas; (2) providing tax assistance for converting industries to coal; (3) extending the five-year amortization for pollution controls through 1990; (4) giving high priority to the Federal government's coal research and development program.

As we turn more and more to coal for the energy to sustain and stimulate our economy, we can have no concerns more important than the health and safety of the American miner. As the coal mining industry expands its operations to meet the President's goal of greatly increasing coal consumption, the roles of the Department of the Interior's Mining Enforcement and Safety Administration and Bureau of Mines will become more vital than ever, both to coal miners and to all of us who will be depending on them to provide the major part of our increased self-sufficiency in energy. Our purpose will be to target on producing more coal while at the same time improving the health and safety of the coal miner. We must plan on increased production—the nation demands it; however, we cannot consider increased production at the cost of impaired working conditions.

The Department of the Interior has historically overseen the conservation and development of the natural resources in the United States. Its most important function has been to insure that these vast resources are developed in an orderly fashion to provide for their use and enjoyment by present and future generations. We are charged with orderly management of the physical resources of our land and, equally important, guarding the health and safety of the people who develop and use these resources.

Until 1973, the Bureau of Mines was charged with conducting mining research and with enforcing federal mine health and safety regulations. Today, the Mining Enforcement and Safety Administration is charged with enforcement of mine health and safety regulations.

As a result of a series of mine disasters and the gradual realization of the magnitude of coal worker's pneumoconiosis (CWP), black lung, increased public and congressional concern led to a gradual increase in the Federal presence and eventually to Federal dominance in the fields of coal mine health and safety. The movement culminated in the Federal Coal Mine Health and Safety Act of 1969 (1969 Coal Act).

The Federal government's role in mine health and safety began in 1910 when Congress, prompted by a series of disasterous coal mine explosions in 1907 which killed over 700 miners, created the Bureau of Mines. From 1910 to 1941 the Federal government's responsibilities under the law were primarily those of advising, conducting experimental work, and teaching various courses in first aid, accident prevention, and mine rescue. The Federal government's role in inspection and investigative work was limited.

In 1940, there were 18 coal mine explosions in the nation, accounting for 206 deaths. These explosions aroused public concern and in 1941, Congress reacted with Public Law 49 which gave federal inspectors the right to enter coal mines to make inspections and investigations. No set of rules or regulations was mandated by Public Law 49, nor did it give inspectors enforcement powers. The inspections conducted under this law were of an advisory nature only.

In 1946, a national coal strike occurred. Using emergency war powers, the Federal government seized and operated the coal mines for a short time. During the negotiations to settle the dispute, the first formal code of regulations for mine safety was formulated. These regulations served as a basis for inspections by union representatives and federal mine inspectors. The regulations were only advisory in nature.

On March 25, 1947, another coal mine explosion occurred at Centralia, Illinois, in which 111 men lost their lives. Another series of explosions in 1951, culminating in the deaths of 119 men at the Orient No. 2 Mine, Frankfort, Illinois, occurred before further legislative action was taken.

Again, Congress responded to the public outcry for new legislation by passing the Federal Coal Mine Safety Act, Public Law 552, which became effective on July 16, 1952. For the first time, Federal inspectors could issue Notices of Violation and Orders of Withdrawal when imminent dangers were found. The 1952 Act was designed specifically to prevent major mine disasters. The law excluded surface coal mines and mines employing less than 15 miners. It also did not make provisions for elimination of the day-to-day accidents that kill miners one at a time.

Following enactment of the 1952 Act, some operators were quick to find loopholes and, rather than open one large mine, elected to open a series of small mines employing 14 miners each. This subterfuge accounted for a sudden growth in small mine operations, and accidents at small mines took a sudden and significant upturn.

Following explosions at small mine operations, Congress again reacted and in 1966 passed amendments to the 1952 Act that removed the small mine designation.

Public opinion in the United States has led to legislative protection of the miner's health. One of the first major actions resulted from the large number of miners who died from silicosis as a result of dust exposure during the 1930's. Consequently, national guidelines were set in 1933 for compensation of silicosis victims, and many states passed legislation setting silicosis dust standards and providing compensation for miners.

During this time, medical evidence from around the world showed a relationship between the use of mechanical mining equipment in coal mining and an increased rate of miner respiratory diseases. Unfortunately, the high rate was attributed to silicosis. Not until 1942 was CWP recognized in Great Britain as a specific disease.

By the 1950's evidence was accumulating which indicated that pneumoconiosis was increasing among coal miners in the United States, and CWP was recognized as a distinct occupational disease. A United States Public Health Service study from 1963 to 1965 demonstrated that CWP was a serious health problem affecting about 10 per cent of the active miners and 20 per cent of the retired miners from 29 coal mines that were surveyed.

The passage of the Federal Metal and Nonmetallic Safety Act in 1966 was a major step in mine regulation because it provided the first federal health standards for industry.

The history of growing public concern over the health of coal miners and the growing federal authority and responsibility for coal mine health and safety came together in the Federal Coal Mine Health and Safety Act of 1969 (1969 Coal Act). A powerful motivator of Congressional action at this time was the concern about the incidence of pneumoconiosis in coal miners. Following the Farmington disaster in November 1968, 42 separate bills were introduced in Congress and the 1969 Coal Act was passed and signed into law. It provided vastly increased enforcement powers; covered surface mines for the first time, addressed health problems, provided benefits to miners disabled by black lung disease, set forth procedures for developing and promulgating health and safety standards, and substantially increased federal coal mine health and safety research. It further provided for civil and criminal penalties for infractions of statutory provisions and mandatory standards. Importantly, the 1969 Coal Act and the Department's administration of the Act are aimed not only at preventing disasters, but also at eliminating accidents that kill or maim individual miners.

This year, the Senate and House agreed in conference committee to new mine health and safety legislation. Known as the Federal Mine Safety and Health Act of 1977, the conference committee bill has cleared the Senate and action in the House is expected shortly. Most provisions of the Act will go into effect 120 days after the President signs it into law.

The bill transfers MESA to the Labor Department as a separate organization. It is a firm commitment of this Administration, no matter which Department responsibility for mine health and safety is placed, to guarantee the American mine worker his fundamental right—the right to a safe and healthful workplace. The keys to our success will be better education, stronger enforcement, and safer, more healthful technology.

The importance of education is clear. As we develop a more coal-reliant energy program, we will need to train thousands of new coal miners to work safely. Therefore, in terms of immediate payoff, the greatest impact will be felt through the acceleration of education and training.

As many of you know, reports on mining operations (both underground and surface) clearly show three phases of coal miner employment in which the miner is more susceptible to injury and death. These are: (1) the new, inexperienced miner; (2) the newly employed experienced miner; and (3) the miner with low task experience. Current data show that about 35 per cent of the victims of accidental deaths in coal mining had less than one year of task experience. Preliminary studies of disabling injuries, reported to MESA's Health and Safety Analysis Center, also point out the high percentage of injuries for miners with less than a year of total mining experience or with little experience at a specific task. Also, workers with five to 10 years of mining experience show a higher injury rate than those with over 10 years of experience. These statistics indicate that the lack of pre-entry training, lack of retraining, and lack of training before changing job assignments are significant factors in the number of injuries occurring in coal mines. Regulations that will address training will be published this year, and similar regulations for coal mine construction workers are being drafted.

The Administration has worked with the Congress to strengthen mine health and safety legislation and is implementing administrative reforms to improve procedures for setting standards, for enforcement, and for collecting penalties.

Enforcement has always been, and should remain, the primary focus of our programs to make mining a safer and more healthful occupation. However, our experience has shown that enforcement alone is not enough to insure a safe, healthful working environment for the nation's coal miners. We cannot station an inspector alongside every coal miner around the clock. Therefore, we must utilize our personnel in the most efficient way, which means directing resources toward critical areas. Because of this basic management decision, we have initiated a series of wide-ranging changes in coal mine enforcement efforts.

For example, we have recently emphasized in our regular enforcement program the concept of impact of "blitz" inspections. These are conducted by a team of inspectors who examine many or all of the working sections of a mine simultaneously. This enables us to get a more complete picture of health and safety conditions throughout the entire mine. We conducted 688 impact inspections in 1976 throughout the country. As a result of these inspections, we issued about 8,000 Notices of Violation and 460 Closure Orders.

To further increase the effectiveness of impact inspections as well as other inspections, we have intensified our review of ventilation and roof control plans submitted by coal mine operators to MESA District offices. MESA inspectors have redoubled their efforts to closely monitor training plans to make sure they are being followed by mine operators.

We have taken steps to direct our resources toward critical areas by assigning hazard analysis-accident prevention (HAAP) inspectors to reduce the potential for accidents and injuries in underground coal mines. This will be accomplished by applying hazard analysis and accident prevention techniques during inspections at selected mines.

A "mine profile rating system" is another recently developed management tool which will ultimately help us to better identify problem mines by evaluating the disabling injury frequency rate, the record of compliance with federal health and safety regulations, and the health and safety management system of the rated mine. This system shows promise as a tool to assist us in deploying our inspection forces more effectively by concentrating them on mines having the most serious health and safety problems.

All of these specialized enforcement programs have helped us to pinpoint health and safety hazards and will continue to permit us to focus our resources on them in order to head off potentially dangerous situations for miners.

As we develop new regulations, we also are continuing to revise existing regulations so as to improve their effect on the health and safety of the miner. One of the main thrusts of our revisions is to eliminate ineffective wording and obsolescent regulations. Another reason for updating regulations is to take advantage of improvements in technology.

In 1977 and beyond, you can expect continued governmental efforts to improve and refine existing mandatory regulations. Illumination requirements for underground coal mines will go into effect, new and revised surface coal mining regulations will be developed, and new self-rescuer requirements will be set forth.

MESA is in the midst of a complete overhaul of all coal mine health regulations and policies based on the experience gained under the 1969 Coal Act—the first industry-wide federally mandated health standards. For example, we are (1) changing the emphasis of the Respirable Dust Program from a

sampling program to a dust control program and (2) providing for the monitoring of noise from mobile equipment with noise dosimeters newly developed through Bureau of Mines research.

One other enforcement tool which has been used in the Coal Mine Health and Safety Program is the assessment of civil monetary penalties. This program has been controversial and beset with difficulties since its inception. In August 1974 a very comprehensive assessment program was implemented to overcome adverse court decisions as well as ineffective portions of the program as it then existed. Since that time substantial improvements have occurred in the assessment and collection of civil penalties. This improvement is demonstrated by the fact that from April, 1970 to July, 1974 only $6.7 million in civil penalties were collected. From August 1974 to date, $30 million in civil penalties have been collected.

In support of the President's national energy plan, government agencies—including the Department of the Interior—will undertake a major expansion of coal research and development including environmental aspects and coal mining technology. The program will be conducted with the urgency required so that new technology will be available as needed. The Department of the Interior's Mine Health and Safety Research is a vital part of this total effort.

The Administration's goals contain what may appear to be conflicts; for example, production versus environmental protection, and productivity versus health and safety protection for mine workers. The latter pair, productivity versus health and safety, may even seem mutually exclusive. But safety and productivity can go hand in hand as evidenced by the experience in Great Britain over the past 25 years. In 1947, the British mining industry had a fatality rate of 0.34 per 100,000 man shifts. By 1976 that rate had been reduced by two-thirds; yet, during the period productivity had doubled. I believe that the technological improvements we seek through research can help to regain lost productivity without sacrificing health and safety.

During the recent House oversight hearings on the impact of the 1969 Coal Act, education and training received most of the emphasis. Although there is a substantial immediate impact to be realized by accelerating education and training, it cannot be expected to remove hazards from a production system, nor can it provide the best and lasting solutions to health and safety problems or productivity problems. Such solutions require advances in technology. The health and safety research and development program being conducted by the Department's Bureau of Mines is beginning to supply some of the solutions that are needed.

Significant progress has been made in the control of respirable dust. Main face ventilation and secondary ventilation techniques, machine-mounted dust collection systems, water sprays, air curtains, and modification of machine cutting parameters have all been used successfully to reduce dust concentrations in underground workings.

Bureau research has contributed directly to the development of mine illumination systems now being offered commercially by nine different manufacturers in this country. That same research made it possible for MESA to promulgate mine lighting regulations and gave industry the means to achieve compliance. As a result, an old and formidable enemy of the coal miner—darkness—is now being dispelled.

Electrical hazards, which studies have shown to be a significant factor in many disabling and fatal mine accidents, are now being reduced with "smart" circuit breakers, ground check monitors, and new and safer connectors, all products of Bureau research. Methane drainage in advance of mining has been demonstrated to have real potential, not only for making mining safer but also for augmenting our energy supplies.

Within the next few years, the Bureau's research will give us more of the kind of technology that can remove hazards from the mining system and protect workers more effectively against those hazards that remain. It is a comprehensive effort and it has started to pay off.

In coming months, we will see demonstrated new and more effective techniques for controlling noise on bulldozers, continuous mining machines, auger miners, and mantrips. We will have hand-held indicating detectors for carbon monoxide and oxides of nitrogen, ignition quenching devices, and new and more versatile fire extinguishing agents. Improved roof control techniques will reach the demonstration stage, along with many other important developments of research now in progress.

Meanwhile, we expect the rate at which the products of Bureau research are commercially adopted to accelerate. To date, more than 60 individual items developed through the Bureau's program have become commercially available to the mining industry. On the basis of the interest now being shown, we expect that number to increase by a hundred or more within the next two years.

The effectiveness of an accelerated pace of federal activities does not depend upon the government alone. Better regulations cannot be developed in a vacuum—we need and solicit meaningful input from all interested parties in order to develop effective and workable regulations. Enforcement of safe working practices and working conditions is our responsibility under the law, but it is the daily responsibility of industry and labor as well. We cannot rely solely upon the government to promote technological innovations in mine health and safety. We need the assistance of organizations such as this. Training in safe work practices can be prompted by the government, but the bulk of the day-to-day training is clearly now a corporate responsibility.

I read, into the present, signs of what the future will hold—a mounting effort by all of us to make a concerted and cooperative effort in reducing mining injuries and deaths. I believe that the milestones of our joint efforts are clearly pointing the way to what I choose to call a safety revolution. Our goal must be a mining industry that is free from disabling occupational injury or illness.

78 1976

ARMCO'S PRE-FOREMAN DEVELOPMENT PROGRAM

By JERRY PORTER
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coal mines organization

There is no subject more important than equipping the first line supervisor with the technical and managerial knowledge and skills necessary for him to properly perform his job.

To give you some background on the scope of our operations in West Virginia, we have three operating divisions within the state. We have approximately 1200 active employees, of whom 80 are classified as section foreman. Our production in 1976 was approximately 2.4 million tons of clean coal. We have added two new mines which will bring our annual production to approximately 3.5 million tons by mid-1978.

Within Armco Steel Corporation's Coal Operations Division, we have formulated a training program based on identified training needs of our first line supervisors in operations. It is safe to state, I believe, that we in the coal industry are experiencing similar problems: limited experience and mining knowledge within the work force and an expanded demand for our product. This situation practically forces us to develop educational programs that will serve to shorten the time period required for employees in new positions to gain the necessary knowledge and skills to satisfactorily carry out their responsibilities.

We have found that an employee who has many years of underground mining experience still has much to learn about managing a section. He has not viewed mining operations from the perspective of the section foreman who has been delegated the responsibility and the authority to produce coal safely and efficiently. With the use of high-speed sophisticated equipment and the necessity of possessing an understanding of complicated roof control plans and mining laws, managing a section is not a simple propositon. To be most effective, today's supervisor must be knowledgeable in both the technical and managerial areas of his job. Within the scope of the section foreman's duties and responsibilities lie his training needs.

In 1973, top management within our operations recognized an urgent need for an effective training program oriented toward first line operating supervisors. The resulting program required the section foreman trainee to spend eight months in training. He was assigned for training purposes to spend one month in each of the following areas: Engineering, Personnel Relations, Safety, Maintenance, General, and Operations. Additionally, he was assigned for two months in Industrial Engineering.

In late 1975 and early 1976, our training program underwent a major revision. A series of meetings were held with operating and staff department heads and managers to identify and define what a section foreman needed to know and be able to do in order to properly carry out his responsibilities. A result of these interviews was the development of five objectives of the training program and a specific program outline with time parameters defined for each course of study.

The objectives of the program are to enable the Section Foreman to:

1. Have a thorough knowledge of Armco's Accident Prevention Fundamentals program and both federal and state mine safety laws.

2. Gain knowledge of the technical aspects of the mining cycle and related

underground work.

3. Gain an effective knowledge of proper equipment utilization and maintenance.

4. Learn to manage the crew to maximize performance.

5. Be conversant with the requirements of the collective bargaining agreement.

The first day is devoted to familiarizing the new foreman trainees with the program. The vice president of coal operations personally welcomes the group to management and discusses with them the challenges and opportunities which lie ahead. This course is designed to acquaint the trainees with the work of a manager (planning, organizing, leading, and controlling) and the skills they must master if they are to succeed in their efforts to manage.

Armco's safety policies and procedures are emphasized during this course. It deals with why accidents occur, classification of accidents, how to fill out an an accident report, make a safety contact, keep proper safety records, hold a safety meeting, write a job safety analysis, make a safety observation, and other related internal safety procedures.

The class is sent to Charleston, West Virginia, where the Allied Health Agency instructs the trainee in emergency medical technician services. They are administered the state examination and certified by the state as emergency medical technicians.

A thorough knowledge of union relations and contract administration is required for a section foreman to be effective in managing his section. The National Bituminous Coal Wage Agreement is reviewed, with special emphasis on those sections which effect section foremen most directly. Grievance procedure is discussed and a mock arbitration hearing is staged, with half the trainees arguing the union's case and the other half arguing the company's posisiton.

A general introduction to electrical problems is given the trainees to increase their awareness in this area. Such subjects as permissibility, fusing, breaker settings, grounding, cable splicing, and various cable sizes are covered.

We found that because section foreman did not understand fully how to fill out maintenance reports or their significance to our preventive maintenance program, time needed to be devoted to the reports in our program. We spend two four-hour sessions discussing the importance of the reports to our preventive maintenance efforts.

Management of people resources is a prime responsibility of all supervisors. A knowledge of the uniqueness of individuals and the predictability of individual behavior will strengthen the foreman's ability to effectively manage his human resources. Upon completion of this subject, the trainees have a clearer understanding of why individuals act or react as they do. While the individual may respond to supervision in one manner when isolated from the work group, his behavior while in a work group may be entirely different. By understanding the reasons for this difference, the trainee is better equipped to effectively motivate and lead group behavior affirmatively toward the accomplishment of managerial objectives. Exposure to the content of this course provides the trainees with essential knowledge concerning the reasons for group behavioral patterns.

Functions of the Industrial Engineering Department and services provided section foreman are explained. Additionally, how to plan supply needs, eliminate waste, and obtain the proper tools is covered.

A general review of the flame safety light, anemometer, methane spotter, and use of gas masks are covered in the classroom and underground to make certain each trainee thoroughly understands how to properly use this equipment.

All aspects of the section foreman's reports are covered in detail. The trainees complete the reports, and they are checked by the instructor. In addition to the time spent here, each day the trainee is scheduled for instruction on our model mine he completes a report on conditions found on the model and submits this report to the instructor for review.

Basically, the program is centered around getting a man to do a job correctly, quickly, and conscientiously. After instruction on how to instruct an employee on a job, trainees are given the opportunity to demonstrate their proficiency by instructing other trainees.

The importance of a first line section foreman being a trained observer is paramount. The general behavioral objective of the course is to measurably improve the trainees awareness of people, processes, production, and the environment so he will recognize a variance (the way it should be, and the way it is) when one exists. A step-by-step approach is being taken to develop the trainee's ability to be a discerning observer.

An exposure to mining safety laws and proper procedures for working with explosives are covered during the program.

Primary consideration is given to what the section foreman needs to know about maintenance work and related problems. The objective is not to make him a maintenance person, but rather to familiarize him with those maintenance problems with which he must be knowledgeable to properly perform his job as a section foreman. A chief maintenance supervisor serves as instructor. Our model mine is used as a training aid in one segment of the program. Air flow requirements, requirements for man-doors, installation of overcasts, and mine safety law are taught. The trainee is instructed both in a classroom and underground.

One segment provides fundamental knowledge to the trainee needed for day-to-day decisions in underground mining. A booklet listing the sequencing of mining operations is given to each trainee, and extensive use is made of the model mine in learning proper sequencing and pillar extraction. Mining law and roof control plans are also used extensively. Classes are conducted both underground and in a classroom.

The trainee spends two weeks with one of our mine safety engineers to learn how safety inspections are conducted, what to look for, and corrective measures to be taken. Additionally during this time, the trainee accompanies federal and state inspectors on their inspections of the mine. A combination of classroom instruction on map reading and on-the-job training for survey work will provide the trainee with necessary engineering information needed by a section foreman. Particular emphasis is placed on the importance of identifying and maintaining accurate center lines during mining operations.

Trainees spend a part of their time with an industrial engineer learning to do a time study of a section. Then they continue during the week doing time studies which are checked for accuracy by an industrial engineer.

Transportation and haulage systems within our mining operations are explained to the trainees, and mining laws pertaining to transportation and haulage are covered in detail.

Again, the model mine is utilized as a training aid. Testing roof, controlling roof conditions, use of wood posts, cribs, or cross headers are covered during the course. And, mining laws pertaining to roof control are given extensive coverage.

During a portion of the training program the general mine foreman is responsible for insuring that the trainees receive instruction in track work, construction of overcasts, installation of trolley and feed line, installation of belt lines, handling supplies, and on-the-job training with a section foreman. Time is set aside for the trainees to return to the classroom for a brief review prior to their being assigned an operating section to manage. Any questions which they may have regarding any safety or operational subject are covered in detail to insure they are as well prepared for their job as possible.

The 812 hours of instruction occurs

during a 22-week period immediately after an hourly employee is selected for supervision. The first class of eight individuals entered the program in April 1976, and at present we have graduated 40 employees from the program. During the program and after the section foreman trainee has graduated, he is subjected to various questionnaires and inquiries regarding the relevance of the program to his knowledge and skill needs as a supervisor. As a result of these inquiries, the program has been modified to a minor degree.

Additionally, mine superintendents and division managers are requested to provide input into the program regarding subject matter and course content. Superintendents and division managers have been enthusiastic about our program and have a waiting list of section foremen in their mines who are presently operating sections whom they wish to enroll in the program.

In closing, I would only like to state that one vital ingredient to any successful training program is a firm commitment from top level management to the need for the program. In our operations, we have been fortunate enough to receive that needed support. The vice president of coal operations has provided as instructors some of our outstanding operating supervisors. This provides them an opportunity for individual development while serving to strengthen the entire program.

The situation within the coal industry today demands a positive approach toward the resolution of our training needs. How we address ourselves to the solution of our training problems will determine our future successes as companies and as an industry striving to satisfy America's needs for more energy.

78 1977

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