Proceedings

THIRTY-FIRST ANNUAL INSTITUTE ON MINING HEALTH, SAFETY AND RESEARCH 2000

Edited by
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PROCEEDINGS

THIRTY-FIRST ANNUAL INSTITUTE ON MINING HEALTH, SAFETY AND RESEARCH

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INTRODUCTION

This *Proceedings* contains the presentations made during the program of the Thirty-First Annual Institute on Mining Health, Safety and Research, held at the Hotel Roanoke and Conference Center in Roanoke, Virginia, August 27-30, 2000.

The Thirty-First Annual Institute was the latest in a series of conferences, the first twenty-seven of which were held at Virginia Polytechnic Institute and State University in Blacksburg, Virginia. The Institute enjoys wide support from the mining community and was co-sponsored by the following organizations:

- Department of Mining and Minerals Engineering, Virginia Tech
- Department of Mining Engineering, University of Utah
- Mine Safety and Health Administration, U.S. Department of Labor
- National Institute for Occupational Safety and Health, Centers for Disease Control and Prevention
- National Mining Association
- National Stone Association
- Virginia Aggregates Association
- Bituminous Coal Operator's Association
- Center for Organizational and Technological Advancement

The Institute provides an information forum for mine operators, managers, superintendents, safety directors, engineers, inspectors, researchers, teachers, state agency officials, and others with a responsible interest in the important field of mining health and safety. In particular, the Institute is designed to help mine operating personnel gain a broader knowledge and understanding of the various aspects of mining health and safety, and to present them with methods of control and solutions developed through research.

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WELCOMING REMARKS

THE OPPORTUNITIES FOR DEVELOPING A SAFETY CULTURE IN THE MINING INDUSTRY

Frank Cone, CMSP

Safety Manager - Minerals Division
Tarmac America, Inc.

I appreciate the opportunity to be a part of Virginia Tech's 31ST Annual Institute on Mining Health, Safety, and Research. I work for Tarmac America, Inc. which produces and manufactures heavy building materials. The mining of granite rock, limestone, and sand & gravel is our core business. Most of my comments this morning are regarding this segment of the mining industry.

I started working in this industry purely by chance. I had been aware of the local quarry as I was growing up in the area. In fact I had spent some time with my friend, whose parents leased the property to the mine operator, trespassing on the quarry property. We would climb the screening tower to survey the landscape, or stand near the edge of the pit and try to throw rocks far enough over the wall to see the ripples in the water where the pit pump was located, or drive a four wheeler onto the stockpiles. We also used an abandoned quarry in another part of the county which had become a popular swimming hole. But even with those experiences, until I actually began working at the quarry, I didn't really understand what was being done to produce the products that I took for granted.

Many people aren't familiar with our industry and the impact it has on our way of life. They may also have a poor image of our industry. They see and hear negative comments on the television or read about it in the newspapers. Perhaps they even live near a mine site that has a high volume of truck traffic, or the plant is dusty and noisy, or the shot blast rattles the windows, or all of the above. But as you know, we need our industry...if you can't grow it then you have to mine it. We need our crushed stone for road construction, concrete bridge decks, footings for home and commercial buildings, the nations rail system, etc. However, most people still don't appreciate the mining industry in general and the construction aggregates industry in particular. If you were to ask people on the streets of Virginia what type of mining is done in our state, I believe the answer from the overwhelming majority would be coal. Although the mining of coal occurs only in Virginia's six western counties, there is a stone quarry, sand and gravel pit, or a related industry that uses our products, such as a ready mix plant, asphalt plant, or a block plant, in nearly every community across the state. Also, some reports indicate that the coal reserves in Virginia could be depleted within the next three decades. Conversely, the aggregates industry is in the midst of tremendous growth. But most importantly, we need the people who are working at these mine sites everyday and going about the business of producing the products so necessary to our way of life. We need to be diligent in finding ways to protect their safety and health and improving their work environment.

I don't think it is really any surprise that the people working in our industry are some of the best people in the world. I am reminded of two people that I met when I first started working in our industry. One was the company carpenter and the other was a farmer from a rural area that later became the plant superintendent. Neither had much formal education, but typical of the people working in our industry, they were very smart people with a strong work ethic and a "can do" mentality. I mention these two people because as I later discovered after getting to know them, that the carpenter had helped all of his children to build their own home and the plant supervisor had provided a college education for all of his children. Our industry has many similar type "heroes" who come to work each day, work hard, do a good job, and go home to take care of their families. They deserve our efforts to provide them with a safe and healthy workplace.

These are the same people when responding to surveys about their job and their workplace will not put their pay rate at the top of their list of priorities. Rather, they want some type of meaningful work. They want some influence in how their job is done. They want a good relationship with their peers. They want a good relationship with their supervisors. They want good benefits and a good work environment which encompasses a safe and healthy workplace. The people sitting in this room have the opportunity to help provide them with that good work environment. It doesn't matter whether you are employed by a state or federal agency, a union member, a nonunion member, or part of the mine operator's management team, you have a role to play.

However, I do have a problem when I perceive that one or more of the other groups begins to question my own commitment to employee safety. We can agree on protecting employees but seriously disagree on how we go about providing that protection. The number of new significant regulations now being imposed on the industry in such a short time frame has caused many operators to question MSHA's ability to manage this part of the process. The new Part 62 noise standard becomes effective September 13th; the Part 46 training rule on October 2nd; the HazCom standard is now being reviewed by the OMB and will probably be released in the next 30-60 days; and the diesel particulates standard, once it is reviewed by the OMB, will probably be released sometime next year. In my opinion, the most difficult part of rulemaking is when the mine operator begins the task of complying with the new standard. Does the data or scientific evidence really call for a new rule to be promulgated? Does any group have an agenda? Are we promulgating a new rule to advance the influence and sphere of an agency? Are there any political motives? These are legitimate questions. Once an advance notice of proposed rulemaking begins for any new standard, it takes on a life of its own. It becomes very difficult to stop the process even if evidence becomes available that the new standard may not really be necessary.

I'd like to speak for a moment about the much anticipated Part 46 Training Rule, which MSHA will soon begin enforcing on October 2, 2000 in the previously exempted industries of shell dredging, sand, gravel, surface stone, surface clay, colloidal phosphate, and surface limestone. This is an example of successful rulemaking. These same industries believed that for 20 years they were regulated by a training rule that didn't allow them to provide the training they thought was most appropriate for their operations. With a mandate from Congress, all affected parties were able to come together and "partner" to make this rule. No one side got everything they wanted, but it is a good common sense regulation. There are still some naysayers and doubters in the industry that don't agree with the new rule and there's nothing wrong with a little healthy skepticism to keep a balanced perspective. However, it appears to me that a training rule that is tailored for specific types of mines and will now be enforced by MSHA is infinitely better than a training rule that many thought was wrong for those same types of mines and wasn't being enforced by MSHA. Now the proof will be in the pudding and the people most likely to gain from the new training rule are the thousands of employees who will soon begin receiving safety training at operations that did none or very little during those past 20 years.

Fortunately, there is plenty of help for mine operators who are basically starting from scratch with their training program and also for those operators who have been providing training but want to improve their existing program. MSHA has done an outstanding job in developing and providing safety training materials through both the National Mine Health and Safety Academy and the Educational Field Services Office. MSHA's Starter Kit is well done and shows how simple it can be to develop a training plan and includes a comprehensive list of the training materials available from the Academy. Other materials available from MSHA which make it easier for an operator to prepare for the new rule are the "Part 46 Training Packet - Introduction to the Work Environment" and "30 CFR Part 46 Instructors Guide with Lesson Plans" (Instructors Guide Series IG 31 and IG 37). There are many other sources where you can acquire excellent training materials including the State Grants Program, industry trade associations such as the National Stone Association, independent contractors and consulting firms, and even the large mining companies that are willing to share their expertise and the training materials they use at their operations.

There are now two groups that must step up to the plate and deliver the goods. First, the mine operators must have their training plan ready to be posted two weeks prior to the effective date of the rule and then begin to effectively train their employees. With the easy availability of good training materials, the most difficult task for operators who have not been providing the training will be to now make the time available to get the training done and having the competent people prepared to deliver the training. We have some of the best people in the world working in the stone industry. I'm convinced that every operation has people that are very knowledgeable of the mining process. They know the safe way to do each job at their operation, and they care about the welfare of their coworkers. Using the training materials as a guide these people will provide good training and their employees will know how to recognize and avoid hazards and be prepared to work safely each day.

Secondly, MSHA needs to work with the mine operators during this transition period and be more concerned with helping the operators provide good training to their employees rather than simply issuing citations for violations. Operators who fail to meet even the most basic parts of the new rule must face the consequences. However, operators who appear to be training in good faith, but may have done some things incorrectly or omitted some items, should be given assistance rather than citations. I believe that MSHA should establish a protocol for mine inspectors to use when reviewing the training plans, lesson plans, or actually monitoring the training. Inspectors enforcing the rule differently from one District or field office to another will only cause more confusion among the operators. Part 46 was developed as a partnership between MSHA, the mine operators, and some labor unions. That philosophy needs to be maintained during this transition period.

I am hopeful that the new training rule will not only address the immediate concern, which is that every miner deserves to receive good safety training, but that it will also be the beginning of a cultural change. Only time will tell if the enforcement of training will actually reduce fatalities and serious injuries. But it should begin to demonstrate to employees that their well being is not some catch phrase but that mine operators and owners will be placing a high value on miner safety. The mining industry needs to continue to pursue a change in our safety culture. Culture can be defined as the development of the intellect through training or education and the refinement resulting from such training. That definition of culture would demand that safety become a value to any organization. Safety would become a principle, standard, or quality considered worthwhile or desirable. That type of culture would also demand that safety become a team effort involving all employees.

Several years ago, Tarmac initiated leader led teams at a cost of several million dollars. Full time "coaches" were taken from the existing work force and their jobs were filled. These coaches, with the appropriate training, were charged with helping the designated team leaders develop a consistent process throughout the organization that would allow all employees to have a direct influence on how their workplace was managed. The concept was that any employee working on the belt conveyor, the front end loader, or the shop floor would know that their contribution is appreciated and their ideas are incorporated into the work process. Conversely, the vision that senior management has for the organization and the direction that the organization is taking will be accurately communicated to all employees without it being diluted through the layers of organizational bureaucracy. It is a slow process. Anything of any value takes time to develop. It is an ongoing process. It will always be an ongoing process. However, employees soon began to feel empowered and they know that they are of value to the organization and are respected for their contribution to a successful organization.

I believe we must continue to make a cultural change at our mine sites to further reduce the rate of fatalities and serious injuries. I am reminded of a recent Dilbert cartoon that showed the boss announcing the company's safety goal for the coming year to the management team. The goal was zero disabling injuries. He went on to explain that in retrospect, last year's goal of 26 disabling injuries was a mistake. They had to injure nine employees to meet their goal. This was the beginning of a cultural change at Dilbert's workplace.

Let me ask you a few questions about your concept of a safety culture. Are zero injuries an achievable goal? Are zero inspection violations an achievable goal? As safety professionals can we advise management that there should be any other established goal? As many mine operators do, we have multiple operations in multiple states. It's not uncommon for many of those operations to go one year without any reportable injury accident. Quite frequently they report no injuries at all. Also, during a recent two year period, we had 30% or so of all of our MSHA plant inspections resulting in zero citations. Could that happen at all of our facilities during the same year? It's possible. But the goal has to remain zero. If not, we're tacitly agreeing that some injuries to some people are acceptable.

How many people in this room have ever held a baby in their arms? How many people in this room have ever dropped that baby? Everyone here has held an infant in their arms or maybe taken that child and gently placed it into the arms of their older sibling and cautioned them to be careful as they held their younger brother or sister. The reason we don't drop babies is that our culture doesn't allow us to drop babies. Can a culture be created that doesn't allow workplace injuries to occur?

Would you send your own son or daughter, grandson or granddaughter to work at your mine site? Or one that you supervised? Or even one that you just know about? Would you rather they worked at a place that had a zero injury record or one where injuries occur throughout the year?

There are several areas which I think can help change the way employees view their workplace and are probably the same type changes that you are making at your workplace. In my opinion there are really only two main components of a successful safety program. It is simple. You make certain that the physical plant machinery and mobile equipment are maintained in good safe condition with multiple types of safety inspections. Secondly, you make sure that employees are well trained and know the proper safe work procedures to use these machines and equipment to produce their product. The following are some time tested initiatives and the reasons why they will help to create a culture where safety is an integral part of the production process:

Hourly Employee Safety Committees

1. Employees feel they have a vested interest in their workplace.
2. Hourly employees are technically competent. They are the people that know the most about their workplace. Safety committees validate that they are respected for their expertise and are an important part of the organization.
3. They must be empowered to do a thorough inspection.
4. The hazards must be corrected in a timely manner. Employees need to see that their identification of safety hazards are important and will be addressed.

Preshift Inspections Of Stationary Machinery And Mobile Equipment

1. Employees need to be educated to the relevance of the preshift inspection to their own safety and who has the most to gain by doing a thorough inspection.
2. Repairs must be performed when defects are reported. It's very frustrating to employees to continually report the same problems repeatedly without any corrective action being taken.
3. Preshift inspections are not limited to mobile equipment operators. The plant operators need to perform the preshift inspection similar to equipment operators. The preshift inspection process utilizes the expertise and knowledge of employees. It also minimizes potential downtime which is demoralizing to employees.

Weekly Tool-Box Training

1. Safety is on the front burner continually. You can have the most effective one day 8 hour annual refresher program but what about the other 364 days. Tool-Box meetings provide the opportunity for employees to continually address safety in their workplace.
2. Supervisors become directly involved. Their knowledge is invaluable.
3. Supervisor must enforce what is being taught during the meetings.
4. A culture of producing "safe tons" is created.

Task Training

1. Possibly the most important training done.
2. Specific to a persons work.
3. Needs to be user friendly for supervisors and trainers.
4. Needs to be easily documented.

Hazard Training for Contractors

1. What is our obligation to independent contractors?
2. They're hired for their expertise.
3. Train for site specific hazards in their workplace.
4. Explain our expectations that they are expected and required to work safely.
5. Don't turn a blind eye to contractors working unsafely.
6. We cannot create a safety culture for our contractors. They must create their own safety culture.

Also, supervisors must be held accountable to continually and consistently enforce all safety rules and policies. Senior management must be held accountable to provide an unambiguous atmosphere where supervisors know they really are expected to enforce the rules and address all safety issues. These are examples of initiatives that can become part of the culture at any mining operation. Again, I believe there must be some type of cultural change at our mine sites to further reduce the rate of fatalities and serious injuries.

In closing, the obvious challenge before us is to try and further reduce fatalities and serious injuries in our industry. We've brought the rate down over the years but I feel it has basically remained flat for some time. But there must be a cultural change in the way we view safety and the value it adds to our industry to reduce it any further. This change can only happen if the mine operators create the atmosphere for it. MSHA's efforts in the training arena certainly contribute to helping an operator create this cultural change but the challenge still remains that of the operator. We must "do the right thing" and make these cultural changes in the coming years.

Again, I want to thank the Mining Engineering Department of Virginia Tech for the opportunity to speak at this year's conference and I have enjoyed being with you.

TECHNICAL SESSION I

NOISE STANDARDS AND PREVENTION

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PRACTICAL ADVICE ON COMPLYING WITH THE NEW MSHA NOISE STANDARD

Kelly F. Bailey, CIH

Occupational Health Manager, Vulcan Materials Company

What did you say? Am I hearing you right? These are just a couple of the many questions mine operators are asking in light of the recently passed Mine Safety and Health Administration's noise standard. There are many different items to comply with in the new noise rule, but two primary areas are sampling to determine noise exposure and audiometric testing. In this article, I will provide some guidance based on Vulcan's twenty-four years of noise sampling and administering hearing tests to its miners. There are some things you need to do, even if the standard does not require them, and there are good and bad ways to do some of the things the standard does require. There are also some things you need to know before you sign on the dotted line with your audiometric testing provider.

EXPOSURE MONITORING

Vulcan's exposure monitoring programs are designed to prevent occupational illness **and achieve compliance with applicable standards.** Many would consider this sentence to be redundant since exposure standards are promulgated to prevent illness. In the case of MSHA's noise standard, however, this is not true. The scientific literature on hearing loss demonstrates that hearing impairment can occur in unprotected persons routinely exposed to levels as low as 85 dBA. Consequently, if a Vulcan employee is exposed to a noise level of 85 dBA or higher, hearing protection is strongly recommended and in many cases mandatory.

Noise dosimeters are used to measure a person's exposure. The dosimeter works by accumulating the "dose" or amount of noise exposure and comparing it to the allowable level. The allowable level is a dose of 100%. You can get a 100% dose by being exposed to a noise level of 85 dBA for 16 hours, 90 dBA for eight hours, 95 dBA for four hours or 100 dBA for two hours. As you can see, when the noise level increases by 5 dBA, the allowable exposure time to reach 100 % dose is cut in half.

To accurately determine the noise exposure at 85 dBA, the settings on the noise dosimeter need to be set so that noise levels between 80 and 140 dBA are measured and integrated into the average noise exposure result. Under the new MSHA standard (and OSHA's existing noise standards), an employer is permitted to measure ***compliance with the permissible exposure limit (PEL)*** of 90 dBA by measuring noise levels between 90 and 140 dBA. For purposes of determining if an individual's exposure is at or above 85 dBA and therefore should be in a hearing conservation program, both OSHA and MSHA require that the employer integrate noise levels between 80 and 130 dBA. Basically, this means that you have to have a dosimeter that can measure both ranges simultaneously or that you use two dosimeters set with different ranges. At Vulcan we set our dosimeters at 80 – 140 to determine compliance with the PEL, and all persons are included in the hearing conservation program regardless of exposure (more about that later). Using a single range is less confusing, results in more accurate measurement of the actual noise exposure level and is more protective.

To protect the hearing of miners, the allowable exposure limit needs to be adjusted ***downward*** if the work shift is longer than eight hours. The reason for this is that a person needs at least 16 hours of "recovery time" if he/she is exposed to noise at an average level of 90 dBA. If a person is exposed for 10 hours instead of eight, the 90 dBA average 8-hr. limit needs to be lowered to 88 dBA. If the work shift is 12 hours, the limit should be an average of 87 dBA. In other words, if the shift is longer, the "recovery time" is shorter and, therefore, the allowable exposure level needs to be lower.

Finally, if for some reason, the noise sample cannot be collected for the entire shift, the dose on the dosimeter readout should be ***extrapolated to cover*** the unsampled period (dosimeter readout in % dose X shift minutes/sampled minutes). If this extrapolated dose is greater than 100%, then it is likely that the person would have been over the PEL. In order to have a sample that can be compared to the MSHA limits (other than the ceiling limit), the sampling duration should cover as much of the shift as possible. However, it is my recommendation that sampling should not be less than two-thirds of the shift.

The MSHA standard tells you to establish a "*system of monitoring*" but does not tell you how many samples you need to collect, what type of equipment to use, how to analyze the data with the exception of comparing it to the different exposure limits the agency has established. The rule does ***not*** require you to test every employee at your work site, but does require that the *"system of monitoring"* be adequate to enable you to identify individuals who may be exposed above the various limits the agency has established. So what should this *system of monitoring* include?

When Vulcan began noise monitoring of its operations back in 1980, we identified those jobs in a typical quarry or sand and gravel operation with the highest **potential** for noise exposure without regard to the use of hearing protection. Some examples of these jobs included drillers, mobile equipment operators in equipment without air conditioning, plant operators that did not have air-conditioned control houses or booths, plant helpers and various maintenance assignments.

This list of jobs became the targeted sampling list; we focused on these jobs first. (To address the two highest health risks in the aggregates business simultaneously, we also targeted those jobs with the highest potential for respirable dust exposure. In many cases they were the same as those for noise with some exceptions.) The sampling approach was designed to measure variability among the different seasons and among different days, so samples were collected at three separate times over the spring (March – May), summer (June-August), and fall (September – November) for each operation.

Noise dosimeters have an inherent measuring error of plus or minus two decibels associated with integrating the many different sound levels in the dBA frequencies. This is why MSHA will not issue a citation until the exposure exceeds 92 dBA or 132% dose. By applying this policy, MSHA is confident (95%) that the measured exposure was above 90 dBA. Although MSHA does not require it, good industrial hygiene practice applies this measuring error to both sides of the PEL. Therefore, exposures above 88 dBA (76% dose) for an eight-hour shift could actually be over the 90 dBA limit. At Vulcan we create an "exposure case" when the extrapolated dose is above 76%. Exposure cases require that the exposure circumstance be evaluated and feasible controls be installed to lower the dose below 76%. Subsequent monitoring must confirm that the exposures are under control before the case can be "closed."

The final part of the *system of monitoring* at Vulcan is the plant sound level meter monitoring program. In this program, sound level meters are available at each production site for routine noise assessment. All mobile equipment cabs are tested at high idle at least once a year by trained plant personnel. A record of the dBA and dBC levels is maintained and all equipment above 85 dBA are designated as hearing protection-required vehicles. Those above 88 dBA are examined for feasible controls (e.g., floormats, door seals, mufflers, insulation, etc.). Having an SLM on site allows line supervisors the ability to address noise cases with instant feedback without waiting for dedicated safety and health staff personnel. The plant SLMs are also used to survey the physical plant to identify areas where hearing protection should be mandatory. Once a plant believes that the noise levels have been reduced to acceptable levels, safety and health personnel conduct monitoring with more sophisticated equipment and either confirm or deny that the case can be "closed."

These basic elements of Vulcan's "system of monitoring" (i.e., use of hearing protection at 85 dBA and above, measuring noise dose between 80 and 140 dBA, adjusting the exposure limit downward for shifts longer than eight hours, extrapolating the dose to cover the unsampled portion of the shift, engineering or administratively reducing noise exposure at the 76% dose level, confirming an exposure problem has been resolved with subsequent sampling and establishing a plant SLM program) all contribute to an effective "system of monitoring." In 1998, Vulcan had a 98% compliance level for MSHA noise samples; the level in the aggregates industry was 84%.

Finally, before I leave exposure monitoring, the standard requires that you inform miners of their noise exposure results within 15 days if they are at or above the 85 dBA action level. I recommend that you also inform them of the good news when they are below 85 dBA, which hopefully will be more often than not.

AUDIOMETRIC TESTING

The MSHA (and OSHA) hearing conservation provision of its noise standard requires that all miners whose work-shift average exposure is above 85 dBA be enrolled in a hearing conservation program that includes hearing testing and training among other requirements. Participation in audiometric testing is voluntary; however, the operator must make it available. At Vulcan, all employees are included in the hearing conservation program regardless of their typical exposure level. This is very important for a number of reasons.

First, if you segregate your work force into hearing conservation jobs and non-hearing conservation jobs, you essentially reduce the flexibility you have of assigning people to different jobs unless they have had a hearing test and have received the mandatory training. Second, by testing all employees at a location, you create your own "control group" from which you will be able to compare potentially high exposure with low exposure jobs. If you are protecting your workers with engineering controls, administrative controls and/or hearing protection, there should not be a big difference in the hearing levels between the two groups (e.g., those with average exposures over 85 dBA and those under 85 dBA). Even for folks that have an average workplace noise exposure below 85 dBA, there will be some hearing loss associated with just being a member of society (chain saws, motorcycles, race cars, loud music, etc.). This is even more prevalent in a blue-collar work force.

The MSHA standard allows the use of mobile testing vans for conducting baseline and periodic audiometric testing. If a mobile service is used, the mine operator has 12 months to establish the baseline for all miners in the hearing conservation program. The standard also requires that miners exposed to 85 dBA, and who have not yet taken their baseline hearing test, wear hearing protection daily until the hearing test can be administered. The standard also requires any miner who shows a significant threshold shift (STS) in hearing for the worse, and is exposed to levels above 85 dBA, be required to wear hearing protection. These are two good reasons for simply mandating the use of hearing protection for everyone exposed to 85 dBA or more. Trying to determine on a daily basis who has an STS or who hasn't had a baseline audiogram so that hearing protection can be enforced would be an administrative nightmare.

The standard mandates several training requirements for persons enrolled in the hearing conservation program and for people with STSs and hearing loss. To optimize training efforts, it is recommended that all new hire training include the mandatory training elements identified in the standard (§ 62.180) and that annual refresher training for noise be conducted after the audiometric results are received. The reason for this timing is that any miner with an STS or hearing loss (you won't know who has one until after the second audiogram) must receive re-training of the elements in § 62.180, be allowed to select a different hearing protector and review the effectiveness of noise controls. By conducting this training for everyone at this time, the operator won't be singling out certain individuals, and the same message will be conveyed to all miners. Prior to the day for hearing testing, a safety meeting should be conducted where information is conveyed as to when testing will be made available, the proper recording of information on forms for the testing service, the avoidance of noise 14 hours prior to being tested, etc.

The selection of an audiometric testing provider is very critical to the success of the overall program. Vulcan has used the Industrial Health Council (IHC), a non-profit, mobile medical testing service, since 1985 and is confident that IHC will meet the new MSHA provisions. Other providers may also satisfy compliance. There are several requirements that you should expect from your provider. First, the standard requires that the audiogram test be conducted by qualified technicians and supervised by a physician or audiologist, and that the test be administered using valid testing procedures. MSHA does not specify those procedures therefore, I recommend you follow OSHA guidance on testing procedures, since many operators have MSHA and OSHA regulated sites. Second, the results must be interpreted and provided to the operator within 30 days after the test was administered, and miners must receive their results in writing no later than 10 days after the operator gets the results. Third, the service needs to be able to manage multiple baselines simultaneously. For example, a person can have an original baseline (the first hearing test), a revised baseline (that can be different for each ear with respect to degree and date) after a permanent STS is identified, a baseline that is used for reporting hearing loss to MSHA if different from the original baseline (also either ear for different dates) and an improved baseline and revised baseline when an STS shows permanent improved hearing. Fourth, the provider needs to be able to quickly and accurately respond to requests for data from the employer, MSHA, a new employer and the employee. Fifth, written correspondence to employees and employers needs to be understandable and satisfy MSHA and OSHA regulations. Not as easy as it sounds.

If you are considering using local or regional audiometric providers, you need to make sure that the standards they use, equipment, procedures, qualifications of the technicians, etc., are as close to the same as possible and meet the standards otherwise, you will end up with a company-wide program so variable in quality that the hearing conservation status of your work force will never be able to be determined.

Let me leave you with one finally word of caution. There are only so many qualified audiometric testing providers currently available to satisfy the need for the US mining industry. If you have not firmed up your plans, and if you delay too long, you will be left with the leftovers and that may or may not be what you want.

There are other provisions of the new noise standard and more details that need to be taken care of to assure compliance. I have tried to address the two largest concerns—noise monitoring and hearing testing.

CROSS-SECTIONAL SURVEY OF NOISE EXPOSURE IN THE MINING INDUSTRY

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ABSTRACT

Prolonged exposure to noise over a period of years generally causes permanent damage to the auditory nerve and/or its sensory components. This irreversible damage, known as noise-induced hearing loss (NIHL), is the most common occupational disease in the United States today. Workers suffering from NIHL have difficulty understanding human speech and hearing other workplace cues. Despite the use of regulations and efforts by government and industry to reduce NIHL, the problem today is as prevalent as it was more than two decades ago. Recently, the Mine Safety and Health Administration (MSHA) promulgated a new regulation that is designed to reduce NIHL in the mining industry. One of the more significant provisions is the elimination of MSHA's past practice of giving "credit" for the use of personal hearing protection, thereby reestablishing the primacy of engineering and administrative controls.

However, there is a knowledge gap that is impeding the development and implementation of engineering and administrative controls. Although significant data exist on the exposure to noise by occupational code, little is known about the noise sources that contribute the most to the worker's dose. This is problematic in a workplace with multiple noise sources and workers who travel among noise sources. Yet without this knowledge, it is difficult to focus control efforts in any practical manner. Thus, it is important to characterize noise sources sufficiently well so that the sources most hazardous to hearing are identified and those conditions of exposure that are most amenable to engineering controls are pinpointed as well. The Pittsburgh Research Laboratory (PRL) of the National Institute for Occupational Safety and Health (NIOSH) is conducting a cross-sectional survey of noise sources and worker noise exposures in the mining industry to address this deficiency. The initial effort, conducted at a coal preparation plant and results are described in this paper. Preliminary analyses indicate that the noise levels on all floors exceeds 90 dBA in most areas, and that levels as high as 115 dBA were recorded. In addition, the one worker whose responsibility is to monitor the equipment and "house clean" the plant is slightly overexposed, even though he spends only half the shift in the plant. General information on the hearing loss problem in mining, a review of hearing protection use and noise regulations in mining, and other background materials are also presented.

INTRODUCTION

Noise is often regarded as a nuisance rather than as an occupational hazard. However, overexposure to noise can cause serious hearing loss. In 1996, NIOSH reported that occupational hearing loss is the most common occupational disease in the United States today, with 30 million workers exposed to excessive noise levels (NIOSH, 1996). The problem is particularly severe in all areas of mining (surface, processing plants, and underground), with studies indicating that 70% to 90% of miners have a NIHL large enough to be classified as a hearing disability (NIOSH, 1976; Franks, 1996). This alarming prevalence of hearing loss among miners is shown in Figure 1. For example, the median hearing threshold of retired miners was 20 decibels (dB) greater than that of the general population. By age 60, over 70% of miners had a hearing loss of more than 25 dB, and about 25% had a hearing loss of more than 40 dB. Franks (1996) review of a private company's 20,022 audiograms indicated that the number of miners with hearing impairments increased exponentially with age until age 50, at which time 90% of the miners had a hearing impairment.

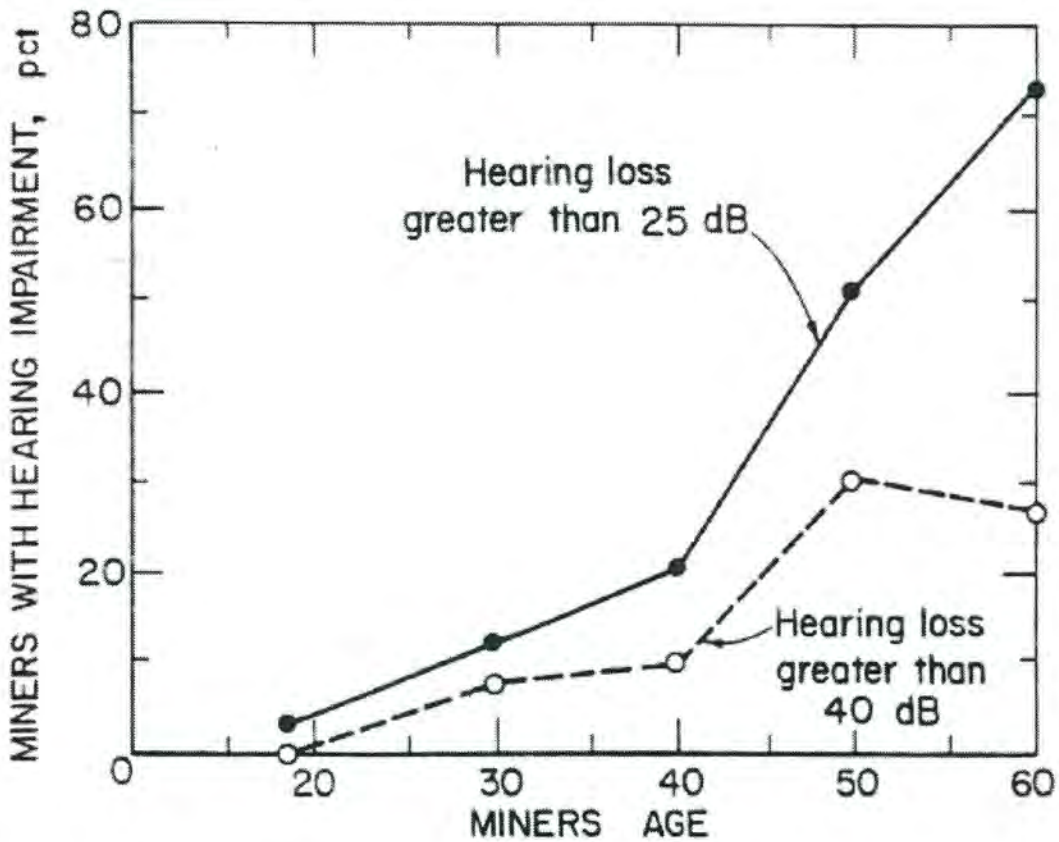

Figure 1: Hearing loss as a function of age (NIOSH, 1976).

Since the passage of the Federal Coal Mine Health and Safety Act of 1969, there has been some progress in controlling mining noise. Machinery manufacturers have incorporated design changes to reduce noise levels. At the same time, however, many of these gains have been diminished by the use of ever larger, more powerful, and sometimes noisier machines.

Thus, the number of miners overexposed to noise, as defined by federal regulations, still exceeds their overexposure to all other health since the 1970s, although the percentage of miners overexposed to current MSHA noise regulations remains high (Seiler, et al. 1994). MSHA found that the percentage of coal miners with noise exposures exceeding federal regulations, and unadjusted for the wearing of hearing protection, was 26.5% and 21.6% for hazards. Data from more than 60,000 full-shift MSHA noise surveys show that the noise exposure of selected occupations has decreased surface and underground mining, respectively. Table I lists recently published data from MSHA noise surveys of exposures in the coal and metal/nonmetal mining industries (Federal Register, 1999).

Table I - MSHA noise samples exceeding specified TWA_8 sound levels.

Industry segment	TWA_8 sound level, dBA[1]	90-dBA threshold		80-dBA threshold	
		Number of samples	Percent of samples	Number of samples	Percent of samples
Coal	90 (PEL)[2]	1075	25.3	------------	------------
	85 (Action Level)	-------------	-------------	3268	76.9
Metal/ Nonmetal	90 (PEL)	7360	17.4	------------	------------
	85 (Action Level)	-------------	-------------	28,250	66.9

[1]TWA_8 is the sound level, if constant over 8 hours, would result in the same noise dose as measured.
[2]Pel-Permissible exposure level

Despite the extensive work done in the 1970s and 1980s, NIHL is still a pervasive problem. MSHA has published new Noise Health Standards for Mining (Federal Register, 1999). One of the changes will be the adoption of a provision similar to OSHA's Hearing Conservation Amendment. MSHA concluded in a recent survey that if an OSHA-like hearing conservation program (HCP) were adopted, hypothetically 78% of the coal miners surveyed would be required to be in a hearing conservation program (Seiler and Giardino, 1994). Based on full-shift time-resolved dosimeter measurements at six U.S. longwall operations, Bartholomae and Burks (1995) found that all the longwall face workers surveyed in these mines would be required to be in a hearing conservation program. These data are corroborated by data collected in the National Occupational Health Survey of Mining (NOHSM) during the 1980s (Greskevitch et al., 1996). Based on this survey, the projected mine workers potentially overexposed to noise was approximately 200,000 workers, or 73% of the workforce.

PERSONAL HEARING PROTECTION

At first glance, personal hearing protection devices (earplugs, earmuffs, etc.) seem to be a relatively cheap and simple solution to almost any noise problem. However, good industrial hygiene and safety practices suggest that hearing protectors should be considered only as an interim or secondary noise control solution and that engineering and/or administrative controls should be first employed. There are several reasons for this. First, earplugs and earmuffs generally do not provide the same degree of protection in the mining workplace as they do in the laboratory or other types of workplaces (NIOSH, 1996; and Giardino and Durkt, 1996). The use of personal hearing protection (PHP) was studied by Stewart and Burgi (1980) and Berger (1983), who found that earmuffs have serious limitations when worn under mine conditions. These include much less real work noise attenuation than that measured under laboratory conditions and the possibility of reduced hearing causing a safety hazard (AIHA, 1986). The effectiveness of PHP can be improved through proper fit, but the possible hazard from overprotection while wearing PHPs is unresolved.

Second, miners often refuse to wear hearing protectors because they are uncomfortable, annoying, or prevent them from perceiving signals such as the sounds that precede a roof fall ("roof talk") or backup alarms on moving equipment (NIOSH, 1996). Often miners simply do not appreciate the risk presented by excessive noise, nor do they believe that using PHPs will protect them.

Finally, spot surveys have shown that miners believe that they are wearing hearing protectors more than they really are. For example, a research group in New South Wales, Australia, surveyed one mine where 75% of the miners stated that they used hearing protectors "regularly" (55%) or "all the time" (20%). In fact, the investigators found that only 40% of the miners wore hearing protectors regularly and 20% wore them some of the time (O'Malley and O'Beirne, 1993).

The limitations of PHPs underscore the importance of using engineering and administrative controls to the fullest extent practicable. At the same time, however, PHPs can offer some reasonable measure of protection, especially when fit and worn correctly. As such, their importance in an overall hearing loss prevention program should not be underestimated.

HIGHLIGHTS OF NOISE REGULATIONS IN MINING

Regulation of noise in mining is covered in Title 30 of the Code of Federal Regulations (30 CFR). The Federal Coal Mine Health and Safety Act of 1969 established requirements for protecting coal miners from excessive noise and, subsequently, the Federal Mine Safety and Health Act of 1977 broadened the scope to include all miners, regardless of mineral type (CFR 30 1977). The regulations allowed a permissible exposure level PEL) of 90 dBA TWA over 8 hours (TWA_8). Exposure below the criterion of 90 dBA is unregulated, while continuous exposure to levels greater than 115 dBA is not permitted. Many noise sources are not continuous, and movement by the worker generally results in exposure to various levels of noise for differing periods of time. This problem of exposure versus duration of exposure is evaluated using the well-known noise exposure index (NEI); the worker is out of compliance if the NEI exceeds unity. In practice, the dose received is most often determined using a type 2 personal noise dosimeter, as defined by American National Standards Institute (ANSI) S1.25-1991(R1997), American National Standard for Personal Noise Dosimeters (ANSI, 1991). Despite allegations that personal noise dosimeters are not as accurate as sound level meters or that they read erroneously with impulse noises, research has found that they are as accurate as sound level meters (Valoski et al., 1995); moreover, they correctly weigh impulse levels (Evans et al., 1991).

The new rulemaking efforts undertaken by MSHA, adopted in September 1999 and scheduled to go into effect in September 2000, retain the PEL of 90 dBA TWA_8, and include a new action level which is a noise dose of 50%, or equivalently a TWA_8 of 85dBA. The new regulation requires the mine operator to enroll a miner in an HCP if, during any work shift, the miner's noise exposure equals or exceeds the action level. Moreover, the new rules establishes the primacy of engineering and administrative noise controls, and explicitly eliminates credit for the use of personal hearing protection. Additional criteria include, a dual hearing protection level of 105 dBA TWA_8, and no miner is permitted to be exposed to sound levels exceeding 115 dBA. Specific details of the new regulations are listed in Table II.

Table II - Details of Part 62 - Occupational noise exposure measurements.

Type	TWA_8, dBA	Dose	Sound levels integrated, dBA	Exchange rate, dB	Weighting	Response
Action level	85	50%	80 to 130	5	A	Slow
Permissible exposure level	90	100%	90 to 140	5	A	Slow

CROSS-SECTIONAL SURVEY OF NOISE EXPOSURE

Methods

NIOSH is conducting a study to obtain multi-shift worker noise exposure and equipment noise levels to develop an up-to-date comprehensive profile of miners' noise exposures as a function of equipment and activity-specific measures. This study is a crucial component in the effort to develop noise controls because it will define the sources of miners' dosages and the characteristics of those sources. Once this information is available, efforts can focus on the development and application of appropriate engineering and administrative control measures that will result in reduced exposures for mine workers. Data collection will be performed at underground and surface coal and metal/nonmetal mines and in mineral processing plants. Although an exact study population has not been defined at this time, it is necessary to survey all segments of the mining industry because workers across the industry continue to have a significant risk of hearing impairment, as illustrated by the MSHA inspector noise survey data published in the Federal Register (1999) (see Table III).

Table III - MSHA inspector noise samples exceeding specified TWA_8 sound levels.

Mining sector	Occupation	Number of samples	90-dBA threshold	80-dBA threshold
			Percent of samples >90 dBA (PEL)	Percent of samples $85 dBA (action level)
Metal/ Nonmetal	Front-End-Loader Oper...........................	12,812	12.9	67.7
	Truck Driver...	6,216	13.1	73.7
	Crusher Oper...	5,357	19.9	65.1
	Bulldozer Oper..	1,440	50.7	86.2
	Bagger..	1,308	10.2	65.0
	Sizing/Washing Plant Oper....................	1,246	13.2	59.7
	Dredge/Barge Attendant..........................	1,124	27.2	78.7
	Clean-up Person......................................	927	19.3	71.3
	Dry Screen Oper.....................................	871	11.7	57.6
	Utility Worker...	846	12.4	60.6
	Mechanic...	761	3.8	43.9
	Supervisors/Administrators....................	730	9.0	32.2
	Laborer...	642	17.1	65.7
	Dragline Oper..	583	34.0	82.5
	Backhoe Oper..	546	8.4	52.6
	Dryer/Kiln Oper......................................	517	10.5	55.5
	Rotary Drill Oper. (electric/hydraulic)....	543	39.6	83.1
	Rotary Drill Oper. (Pneumatic)..............	489	64.4	89.0
Coal	Continuous Miner Helper.......................	68	33.8	88.2
	Continuous Miner Oper..........................	262	49.6	96.2
	Roof Bolter Oper. (Single).....................	234	21.8	85.5
	Roof Bolter Oper. (Twin).......................	92	31.5	98.9
	Shuttle Car Oper.....................................	260	13.5	78.5
	Scoop Car oper..	94	18.1	74.5
	Cutting Machine Oper.............................	22	36.4	63.6
	Headgate Oper...	20	40.0	100.0
	Longwall Oper..	34	70.6	100.0
	Jack Setter (Longwall)...........................	25	23.0	68.0
	Cleaning Plant Oper................................	107	36.4	77.6
	Bulldozer oper..	225	48.9	94.2
	Front-End-Loader Oper...........................	244	16.0	76.6
	Highwall Drill Oper................................	83	21.7	77.1
	Refuse/Backfill Truck Driver..................	162	13.6	78.4
	Coal Truck Driver...................................	28	17.9	64.3

The plan of research is comprehensive and is designed to include all workers at each site investigated. The data collected will include worker noise dose, equipment noise, and other worker, mine, and equipment-specific information necessary for characterizing the noise sources. At each site, mine workers will wear time-resolved dosimeters. During the shift, a task-based exposure assessment methods (T-Beam) approach studies will be used to correlate each mine worker's tasks, the noise dose received, and the noise source responsible for that incremental contribution to the miner's total exposure. Noise profiling of mine machinery will be conducted using hand-held sound level meters. This will consist of A-Weighted Equivalent Continuous Sound Levels (Leq) measurements on a uniform grid pattern to develop detailed noise contours and "area sweeping" of mine machinery to calculate sound power. The instruments that will be used to make these measurements include Quest Technologies Model Q 400, Noise Dosimeters, and Quest Model 2900,Integrating and Logging Sound Level Meters (fig. 2). Finally, site-specific parameters, such as characteristics of the mine plan, will be documented to support subsequent analyses. The bulk of the data collection activities are completed over five shifts. Typically, one or two site visits are made in advance of the data collection to gather information for the design of the site-specific data-collection activities.

Figure 2: Dosimeter and sound level meter for conducting noise surveys.

Results

Progress to date includes completion of pilot studies at an underground coal mine and underground limestone mine, and a full-scale study conducted at a coal preparation plant. The pilot studies served both as training exercises for the field crews and for refining the data collection and analysis procedures. The study at the preparation plant included surveying the noise on all eight floors and a control room (fig. 3). The data collected included A-Weighted Leq , as well as Linear 1/3rd Octave Band Sound Pressure Levels (SPL's) around all major pieces of processing equipment.

The plant was a modern/multicircuit coal preparation plant. It was constructed of steel I-beams for internal support with corrugated steel walls (fig. 4), except for the first floor, which had walls constructed from concrete block. All floors were constructed of 4 inches of concrete, except on the second and sixth floors, which were made of open steel grating. In addition, there were many open spaces that extended from one floor to the next, or in some cases, from the ground floor to the top floor. The processing equipment included classifying cyclones, sieve bends, magnetic separators, flotation cells, banana screens, heavy media cyclones, D&R screens, coal spirals, centrifuges, clean coal and refuse conveyors, and pumps.

Figure 3: Noise measurement being made with a sound level meter.

Figure 4: Example of wall and building construction including open spaces.

The measured Leq levels ranged from 83 to 115 dBA, with most floors averaging in the upper 80s and above (see Table IV). Although Table IV lists the dominant noise sources, characterization of noise sources in the plant was a complicated task for several reasons. First, the sheer number of pieces of equipment and their close proximity to each other made separating specific noise sources extremely difficult, and process considerations made it impossible to operate equipment independently. Next, the openness of the building allowed noise to propagate between floors, as did the floor-to-floor connections of the equipment. Finally, the measured noise came from several sources, most often a combination of airborne and structure-borne noise paths (fig. 5). Airborne noise was present as direct sound, generated by the equipment, the process, and motors, and as reverberant sound reflected by the building's walls and floors. Structure-borne noise paths resulted from equipment vibration and transfer of that vibration to the building's structural components. The vibrant energy was then radiated as airborne sound into the surrounding area. An example of a contour plot of the noise levels is illustrated in Figure 6.

Table IV - Summary of Leq levels.

Floor	Leq Range, dBA	Major equipment	Dominant noise source (Leq, dA)
1	91 - 99	pumps, pump motors	classifying cyclone pump (99.4 dBA)
2	92 - 96	Conveyors	clean coal and refuse conveyors (93.5 dBA)
3	93 - 103	dewatering screens, centrifuges, mag separators,	fine refuse dewatering screen (101.6 dBA)
4	94 - 101	sieve bends, D&R screens, coal spirals	clean coal and refuse D&R screens (100.4 dBA)
5	91 - 101	heavy media cyclones, banana screens, flotation cells, sieve bend	raw coal banana screens (99.8 dBA)
6	89 - 115	sieve bends, cyclones	fine clean coal sieve bends (104.6 dBA)
7	89 - 92	raw coal conveyor, sieve bends, mag separator	None
8	88 - 91	Cyclones	15-inch dia. Classifying cyclones (91 dBA)
Control Room	74 (Inside) 90 (Outside)	plant controls, monitors, etc.	None

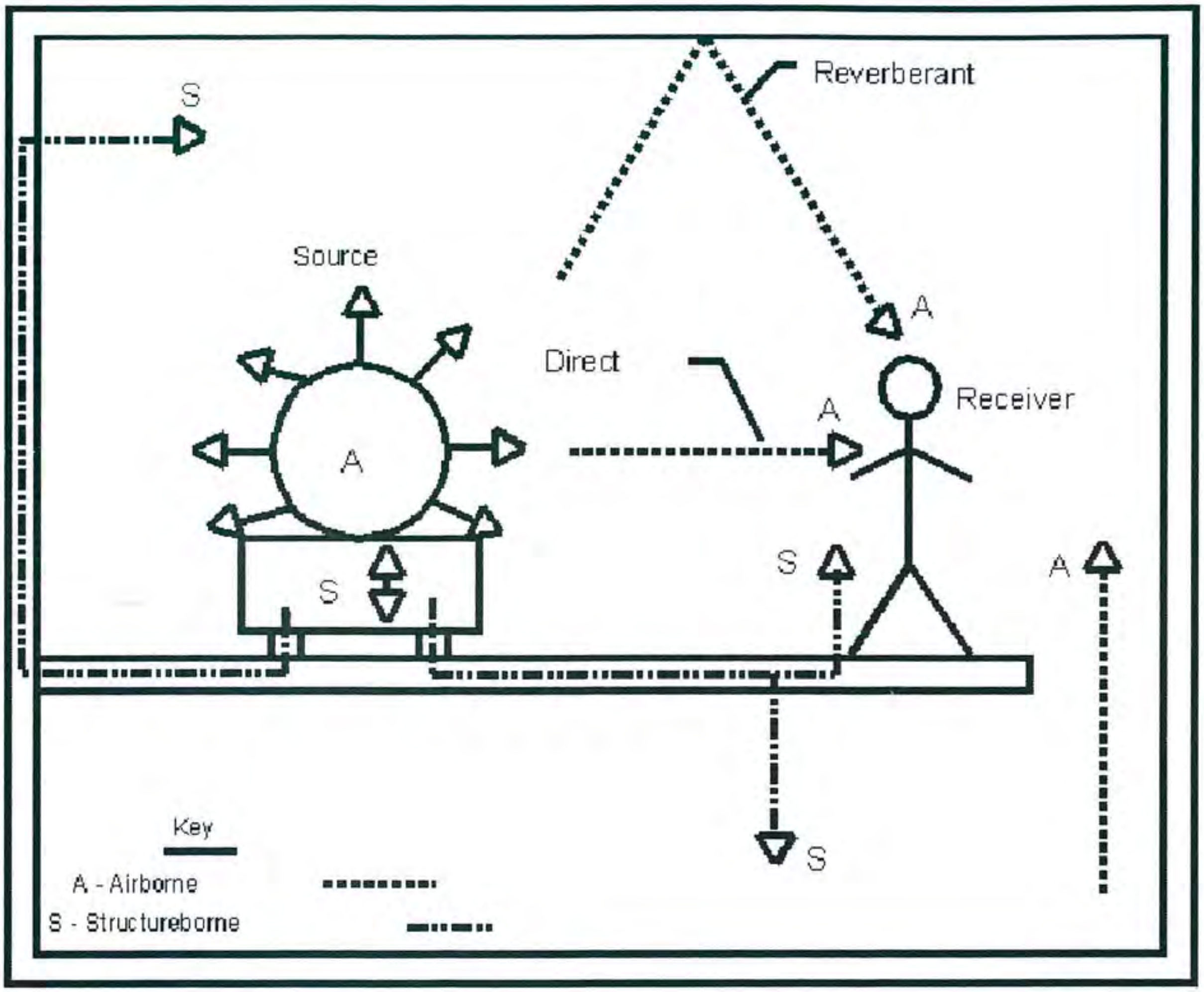

Figure 5: Example of airborne (A) and structureborne (S) noise paths.

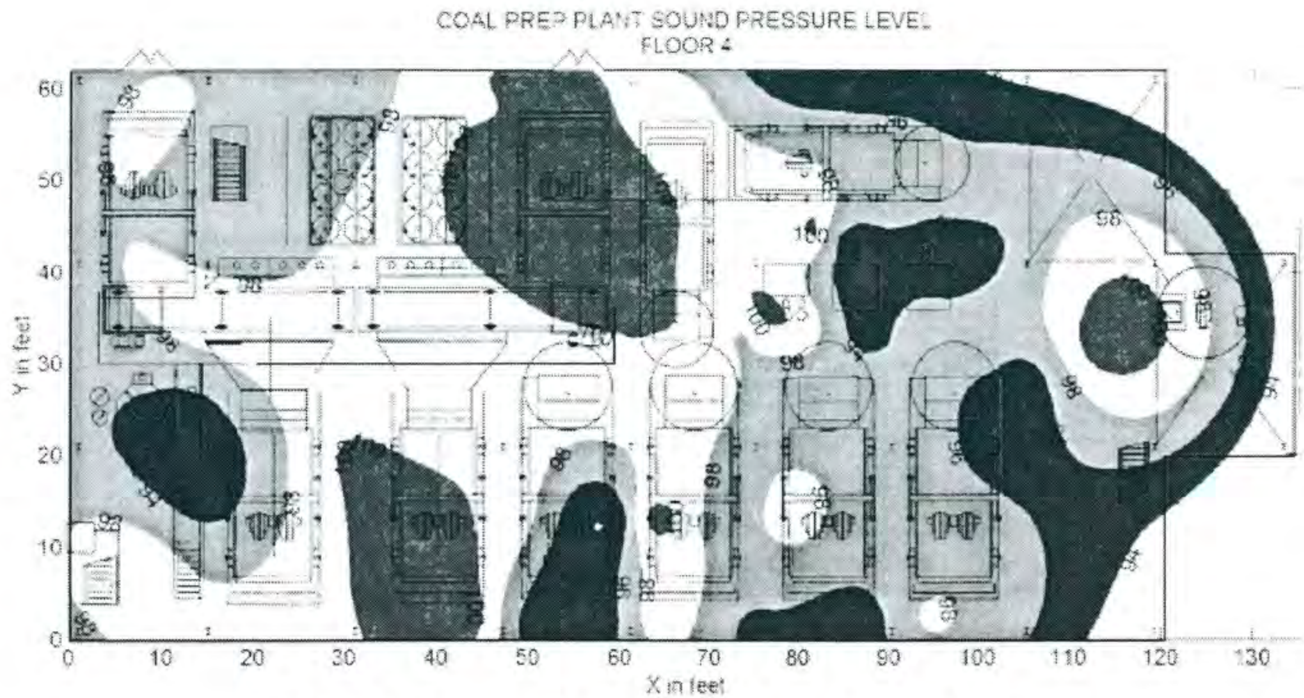

Figure 6: Example of noise contours in prep plant.

A few general observations of the noise levels on all floors can be made. (1) Although the highest noise levels were recorded on floor 6, floors 3 and 4 are considered to be the noisiest floors overall because the noise was consistent throughout the entire floors. (2) Vibration is certainly a factor in generation of noise throughout the plant. (3) Reverberant noise from the building walls is likely a significant component of the noise throughout the plant. (4) The openness and construction of the plant is conducive to noise propagation between floors. This likely resulted in "smearing" or "blending" of the noise from floor to floor.

In addition, several man-shifts were spent following the Plant Controls Man, documenting his work activities while he wore a personal dosimeter. He wore a personal dosimeter for parts of two shifts (8 a.m. to 3 p.m.), while a NIOSH Researcher performed a time and motion study as he traveled throughout the plant. Table V summarizes the Plant Control Man's location throughout the shifts. Table VI presents the projected dose and time-weighted average. The projected dose, in percent, is computed by measuring the dose for a specified time period (in this case, approximately 7 hrs) and extrapolating it to a different time period (8 hrs). The time-weighted average is the average sound level computed over an 8-hour time period.

Figure 7 is a plot of the cumulative dose for the measurement period. The sections of the graph with the steepest slope indicate the periods that the Plant Controls Man was in the plant and receiving most of his measured noise dosage. In contrast, the flat slope sections of the graph are the minimal dosages accumulated while he was in the control room or traveling between the control room and plant.

Table V - Location of Plant Controls Man			
Location	**Duration, min.**	**Time, pct.**	**Percent Dose**
Control Room	210.20	51	12[1]
Plant	189.25	46	88
Traveling between plant and control room	11.50	3	ND[2]
Total	**411**	**100**	**100**

[1]Although no control room was under 80 dBA, some higher noise levels occurred because of equipment and the door being opened.
[2]ND - Not determined. Since time period was small and because the old plant was not running, the dose is included in the Control Room dose

Table VI - Projected 8-Hour Dose for Plant Controls Man (one shift)		
MSHA Designation	Projected Dose, pct	Time Weighted Average (TWA_8), dBA
Action Level	159.96	93.4
Permissible Exposure Level	152.36	93.0

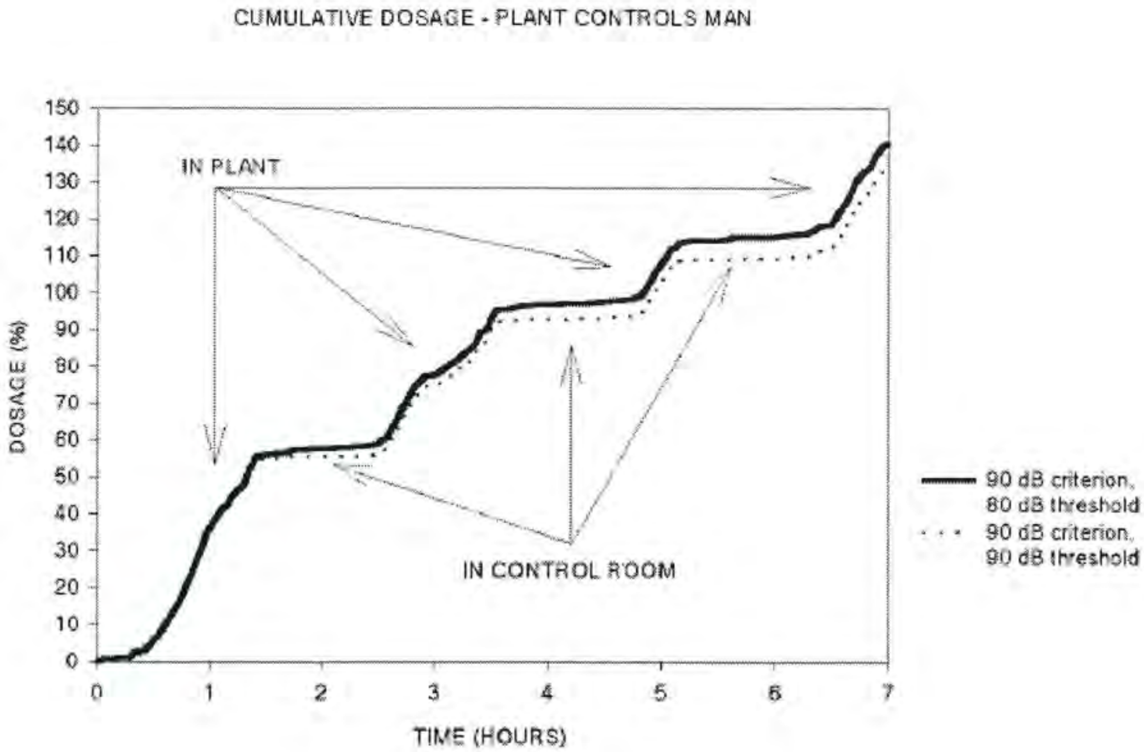

Figure 7: Plot of cumulative noise dose for plant controls man.

SUMMARY

Noise-induced hearing loss is a concern in the mining industry. One study revealed that more than of 90% of miners have a hearing impairment by the age of 50 (Franks, 1996). In addition, based on thousands of inspector noise samples, MSHA has suggested that miners in all sectors of mining and occupations continue to have a significant risk of NIHL over a working lifetime. Despite government and industry efforts over the past three decades, hearing loss remains relatively unchanged in the industry. It is apparent that it is a complex problem that will require an understanding of its underlying causes. Although engineering and administrative controls represent the desired means of protecting workers from excess exposure, it will be necessary to understand where mine workers receive their exposure and the specific characteristics (frequency, duration, level) of the offending noise sources. The NIOSH cross-sectional survey project will establish valid worker noise exposure and equipment noise level data for formulating intervention strategies that target high-risk equipment and activities with the noisiest exposures for mine workers.

The coal preparation plant study highlighted in this paper illustrates the nature of the study and the complexities of the data analysis. The ultimate value and application of the findings from this plant will be in aggregated form when it can be examined as part of a larger sample of plants and mines. However, there is specific value to these findings as well. Careful study and review of the contour plots revealed "hot spots" of higher noise levels. These can be the starting points for applying engineering and administrative controls in an attempt to reduce noise and worker exposures.

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THE AGING WORKFORCE: AN EMERGING ISSUE IN THE MINING INDUSTRY

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ABSTRACT

According to the Bureau of Labor Statistics (BLS), workforce estimates of median age suggest that the median age of the mining workforce, which has been experiencing overall declines in numbers of employees, is rising more rapidly than for the overall U.S. civilian labor force. In the absence of detailed demographic data for the mining industry, the current study uses injury and illness data reported to the Mine Safety and Health Administration (MSHA) to examine differences, over time and by commodity, in the proportions of injured or ill workers aged 45 years and older. These data indicate that from 1988 to 1998, the percentage of injured or ill older workers (aged 45 years and older) have been steadily increasing. The most notable increase occurred at coal mining operations where the proportion of injured/ill older workers increased from 24 to 44 percent. A more detailed breakdown of the industry by commodity for 1998 showed the highest proportions of older injured/ill workers (over 40%) occurring in coal, iron ore, alumina mills, cement, and trona operations. In coal operations, as the employment size of the mine operation increased, so did its proportion of older injured/ill workers. Also, higher proportions of older injured/ill workers were observed at surface coal work locations than at underground locations. The distribution of older injured/ill miners by occupation for several select commodities shows the highest proportions for supervisors, electricians, mechanics, and surface equipment operators, and the lowest proportions for surface laborers and for underground coal roof bolters and scoop operators. With few exceptions, older injured workers have the highest median number of days lost per injury when examined by type of mining operation and by occupation. Given the high proportions of older injured or ill workers in many sectors of the mining industry, health and safety programs must consider the physiological changes associated with aging when evaluating job tasks and the working environment.

INTRODUCTION

According to the Bureau of Labor Statistics (BLS), the median age of the U. S. civilian labor force is projected to reach a record high of 40.7 in 2008 (Fullerton, 1999). This is a slight increase over the previous high of 40.5 attained in 1962, before the baby boomers entered the work force. After 1962, the median age declined steadily until 1980 and has been rising since. However, for industry segments such as mining, which have been experiencing overall declines in employment, statistics provided by BLS indicate that the median age for mining is rising even more rapidly (see Table I). Further, among the major sectors of the mining industry, the median age of the coal mining work force, estimated at 45.2 in 1998 (see Table II), is already well beyond the median age projected for the civilian labor force in 2008. For comparison, also provided in Table II are the median ages of injured and ill workers reported to the Mine Safety and Health Administration (MSHA) in 1998. Although the BLS estimates of median age within the mining industry are based on relatively small sample sizes obtained from the Current Population Survey (CPS), note that both the CPS estimates and the median ages of injured or ill workers reported to MSHA, show the same relative order of differences in age between the major sectors of the mining industry.

Table I. Median age of the U. S. civilian labor force (Fullerton, 1999) and the mining industry, including oil and gas (BLS, 2000), for selected historical years and projected 2008.

	1962	*1978*	*1988*	*1998*	*2008*
Labor force	40.5	34.8	35.9	38.7	40.7
Mining	42.1	34.9	37.8	41.2	?

Table II. Estimates of the median age for sectors of the mining industry (BLS, 2000) from the Current Population Survey (CPS) and the median age of injured/ill workers reported to MSHA in 1998.

	Median age	
Mining sector	**CPS**	**MSHA**
Oil and gas..............	41.1	NA
Coal........................	45.2	43
Metal......................	41.3	42
Nonmetallic	39.1	39
Nonmetals..........	NA	40
Stone	NA	39
Sand and Gravel.	NA	37

These statistics are of concern for several reasons. Although most research studies indicate that occupational injury rates appear to decline with increasing age, the severity of these injuries appear to increase and injured older workers tend to require longer recovery periods (WHO, 1993).

Concerns about the problems of an aging workforce are not confined to the U.S. In many industrialized and developing countries, the increases in life expectancy and declining birth rates, has resulted in unprecedented increases in the mean age of the population. These increases have impacted the mean age of the working population, which is rising rapidly and expected to continue to rise (WHO, 1993). Increasing concerns about aging and its impact on worker capacity relative to worker demands prompted the World Health Organization (WHO) to form the Study Group on Aging and Working Capacity. The Study Group's published report defines work capacity to include the physical, mental, and social functional abilities necessary to perform a given type of work. The report examines those issues related to aging that may diminish worker capacity, because when job demands exceed the work capacity of the individual, decreased productivity, job-related stress, disease and disability are likely to result.

It is therefore important for employers to consider the needs of their aging workers when evaluating the safety and health aspects of the workplace.

Although detailed demographic data for various sectors of the mining industry are not available, the comprehensive set of injury and illness data collected by MSHA under 30 CFR Part 50 can be used to examine differences in the proportions of older injured or ill workers (aged 45 years or older) in specific sectors of the mining industry. The current study uses the MSHA accident/injury/illness files from 1988 through 1998 to examine trends in the proportions of older injured or ill workers for major sectors of the mining industry. Data from1998, the most recent reporting year for which close-out data are available, are used to profile differences in the proportions of older injured/ill workers for the various commodities. Within coal mining operations, differences in the proportions of these workers are examined by work location and by operation employment size. Additionally, the proportions of older injured/ill workers within various occupations are presented for several select commodities, with numbers adequate for comparison. And, finally, differences in recovery time from injuries are examined for three age groups, using the median number of lost workdays per incident. These differences are presented within commodities and within select occupations.

METHODS AND DEFINITIONS

MSHA mine operator accident/injury/illness files for the years 1988 through 1998 were used to examine trends in the proportions of injured or ill older workers by year and commodity. MSHA mine operator employment and accident/injury/illness files for 1998, the most recently released reporting year, were used to summarize mine-level information and to characterize injured and ill workers by commodity. The total numbers of employees were computed by summing the average annual number (averaged across four quarters) of employees reported for all operational subunits except office locations. Similarly, the total number of employee hours, used in the computation of incidence rates, was obtained by summing the hours reported for all subunits except office locations. Injuries and illnesses occurring in office locations were also excluded from all analyses. Additionally, the computations for the percentage of injured or ill employees included only reports of fatal and nonfatal injuries and illnesses (designated by MSHA as degrees of injury 1 through 7) to mine operator employees with a reported age of 18 through 79. Missing or invalid ages resulted in the exclusion of 2.9% of cases of injuries or illness in 1998. Similarly, 9% of the reports of injuries or illnesses had missing data for the years of total mining experience of the injured/ill person in 1998, and thus were excluded from computations involving years of total mining experience.

The median number of days lost reported for three age groups was examined only for those nonfatal injuries resulting in lost workdays during the three-year period from 1996 through 1998. The three most recent reporting years were examined to ensure adequate numbers of cases. These lost workday cases included those nonfatal injuries that resulted in partial or total permanent disabilities, as well as incidents involving actual days away from work and/or days of restricted work activity (designated by MSHA as degrees of injury 2 through 5). The number of lost workdays was computed by summing the actual days away from work and days of restricted work activity. Statutory days charged for permanently disabling injuries were used when days lost were not reported or when the statutory days exceeded the reported number of days lost. Lost workday cases with an invalid closing document number (indicating no return-to-work date reported) were excluded from the analyses. The invalid closing document number is used by MSHA to designate those cases for which the days lost were estimated using an algorithm that computes the average number of days lost for similar injuries. This estimate is used when a follow-up report from the operator of the actual number of days lost is not received by MSHA prior to a close-out of the file for the report year. These cases accounted for about 10% of all lost workday cases used in the analyses for the three-year period from 1996 through 1998.

Mining industry segments were examined separately using MSHA's designation of canvass class, which differentiates the five primary commodities of coal, metal (metallic minerals), nonmetal (nonmetallic minerals), stone, and sand and gravel. The commodity classification codes assigned by MSHA were used to further differentiate specific commodities within canvass classes (e.g., differentiating iron ore, copper, and gold within the metal mining operations).

Consistent with recent literature on older workers, with an emphasis on workplace interventions to prevent injury, illness or disability among these workers, the term 'older workers' is used to designate workers aged 45 years and older (Wegman, 1999).

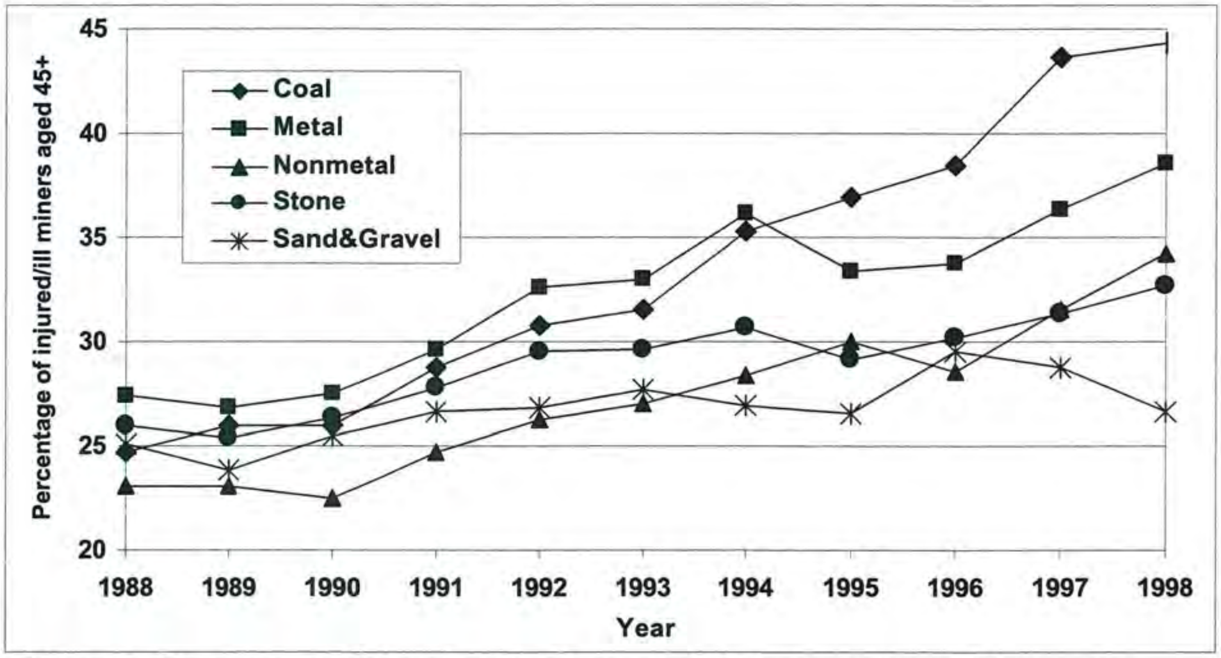

Figure 1. Percentage of injured or ill miners, aged 45 years and over, by operator canvass class and year, MSHA, 1988-98.

DISTRIBUTION OF OLDER INJURED OR ILL WORKERS IN MINING

The annual proportion of injuries and illnesses accounted for by older workers (aged 45 and older) from 1988 to 1998 is illustrated in Figure 1 for each of the five classes of mine operators. In 1988, the percentage of injuries and illnesses accounted for by older workers differed by less than 5% among the five types of operators, ranging from 23% for nonmetal operators to 27% for metal operators. Since 1988, the gaps between these operators have widened considerably, ranging from a low of 27% for sand and gravel operators to a high of 44% for coal mine operators. During this same period, the incident rates of injuries and illness have been steadily declining for all five classes of operators. Overall, the incidence rate of injuries/illnesses declined by about 36%, from a high of 10.6 injuries/illnesses per 200,000 employee hours in 1999 to a low of 6.8 in 1999.

A more detailed breakdown of the major types of mining operations, their general employment characteristics, and demographic information of the injured/ill workers reported from those operations in 1998 is presented in Table III. Among the metal, nonmetal, and stone operations, the proportions of injured/ill older workers vary considerably. Iron ore operations and alumina mills report the largest proportions of injured/ill older workers, at 46.3% and 44.9 %, respectively. Conversely, the lowest proportions of injured/ill older workers occur at dimension stone (17.8%) and sand and gravel operations (26.7%). These operations are also among the smallest in terms of average mine size, employing fewer than 10 employees per operation. Additionally, these two types of operations report the highest proportions of injured/ill workers with one year or less of total mining experience. In fact, in this same category, at dimension stone mines the reported proportion (42.4%) is over three times the proportion of 12.3% reported for the mining industry overall. Conversely, the proportion of inexperienced injured/ill workers at coal mining operations (3.9%) is less than one-third of the overall rate. These differences are certainly substantial, but should be interpreted with caution. Lower proportions of injured/ill inexperienced workers in coal may reflect its declining employment (down 40% since 1998) as well as the availability of experienced miners for hire, and/or low employee turnover. Conversely, the high percentage of injured/ill inexperienced workers at dimension stone mines may reflect increasing employment rates and high employee turnover. In fact, both stone and sand and gravel operations are most likely to be seasonal operations, reporting lower employment in the winter months, which may contribute to a high employee turnover. However, it is also possible that these high rates reflect a higher rate of injury for inexperienced workers at these operations.

Distribution of Older Injured or Ill Workers in Coal Mining Operations

With over 7,500 injuries and illnesses reported by coal mining operations in 1998, 44% of which involved workers aged 45 or older, these operations accounted for the largest numbers of older injured or ill workers. Within these operations, the proportions of older injured/ill workers vary with the work location and with the employment size of the operations (see Table IV). Based on the distribution of these workers, proportionately more injured/ill older workers are found in surface work locations than in underground locations, and among surface workers proportionately more older injured/ill workers are in preparation plants than at surface production operations. It also appears that as the employment size of a coal mining operation increases, so do its proportion of older injured/ill workers.

These differences in the proportions of older injured/ill workers may reflect the tendency of older workers to select out (transfer or terminate) or to be selected out (e.g., due to disability) of particular occupations and work environments. Research indicates that older workers are less likely to be involved in occupations requiring excessive physical demands or in stressful and dangerous work environments (WHO, 1993). In general, underground work locations exhibit all the elements of a stressful and dangerous work environment, particularly at small underground operations, which are more likely to operate in thin coal seams where the working height of the mine further restricts the posture, mobility, and visibility of the worker. In terms of risk of injury, historically, both fatal and nonfatal injury rates are highest in underground mines than in surface mines or preparation plants. Differences in the proportion of older workers by employment size may reflect the ability of larger operations to be more flexible in relation to job assignments or task selection within jobs, such that the physical demands on the older worker are reduced. Additionally, larger operations may invest more resources in ergonomic programs that reduce worker stress related to both job task and the workplace in general.

Table III. Mine-level characteristics and characteristics of injured or ill workers by commodity, MSHA, 1998.

Canvass class/ Commodity	Mine-level characteristics			Injured or ill workers		
	Total no. of employees (thousands)	No. of mines	Average employee size	No. injuries/ illnesses	% aged 45+	% 1 yr. exp. or less
Coal	**85.2**	**2,459**	**35**	**7,543**	**44.4**	**3.9**
Anthracite	1.7	202	9	143	44.8	9.8
Bituminous	83.5	2,257	37	7,400	44.4	3.8
Metal	**40.0**	**337**	**119**	**2,440**	**38.5**	**11.1**
Iron ore	7.4	30	246	549	46.3	6.0
Alumina mill	2.9	8	363	287	44.9	16.7
Lead/zinc ores	1.9	28	68	122	42.6	8.2
Silver ores	1.7	12	141	134	38.1	20.4
Copper	11.5	45	255	460	37.4	8.0
Gold	11.3	158	71	568	35.9	6.8
Other metals	3.3	56	60	320	24.4	22.1
Nonmetal	**23.5**	**789**	**30**	**1,164**	**34.2**	**17.7**
Trona, potash, borate minerals	4.5	17	262	204	44.1	8.7
Phosphate rock	3.4	29	117	129	39.5	16.7
Clay, common	6.8	236	29	305	31.5	19.1
Other nonmetals	8.9	507	18	526	30.6	19.9
Stone	**68.0**	**3,808**	**18**	**4,262**	**32.7**	**21.8**
Cement	10.8	101	107	917	43.8	12.5
Lime	3.2	66	49	256	37.9	8.5
Crushed stone	49.2	3,138	16	2,768	30.2	24.4
Dimension stone	4.7	503	9	321	17.8	42.4
Sand&Gravel	**35.2**	**6,403**	**5 – 6**	**1,627**	**26.7**	**26.7**
Totals	**251.9**	**13,796**	**18**	**17,036**	**38.2**	**12.3**

Table IV. Percentage of injuries and illnesses for workers aged 45 years and older by work location and employment size for Bituminous Coal operations (MSHA, 1998).

Work location/Employment size	Total no. of injuries/ illnesses	% aged 45+
Underground	**5,397**	**42.2**
Fewer than 20 employees	271	29.5
20 – 49 employees	1,202	28.9
50 – 99 employees	816	27.9
100 – 249 employees	1,269	43.2
250 or more employees	1,839	58.5
Surface (excluding prep plants)	**1,509**	**49.0**
Fewer than 20 employees	150	44.0
20 – 49 employees	324	36.7
50 – 99 employees	278	36.7
100 – 249 employees	459	58.8
250 or more employees	298	61.1
Preparation Plants	**488**	**53.5**
Fewer than 20 employees	105	34.3
20 – 49 employees	159	47.8
50 – 99 employees	128	70.3
100 – 249 employees	35	60.0
250 or more employees	61	62.3

Distribution of Older Injured or Ill Workers by Occupation

The proportion of older injured/ill workers by occupation in 1998 was examined for a select set of occupations within five work locations with numbers sufficient for reliable analysis. The proportions of older injured/ill workers by occupation are listed in Table V for underground bituminous coal, surface bituminous coal, surface crushed stone, cement, and sand and gravel operations. Surface operations include preparation plants and mills as well as surface extraction operations such as strip mines and quarries. Occupations are listed in order of decreasing proportions of older injured/ill workers.

In general, the proportion of older injured/ill workers is substantially higher for supervisory occupations and electricians than is observed for the operation overall. Although less substantial, the proportions of older injured/ill mechanics and surface equipment operators (operators of cranes, bull dozers, and front-end loaders) are also higher than the proportions of older injured/ill workers observed for the operation overall. Conversely, the lowest percentages of injured/ill older workers are observed for laborers in all but underground coal mines. In fact, the proportions of older injured/ill laborers in crushed stone, cement, and sand and gravel mines are substantially lower than the overall proportions of older injured/ill workers at these operations. This is particularly evident in cement operations where 43.8% of all injured/ill workers were aged 45 or older, but only 14.2% of the injured/ill laborers were older workers. In underground coal, the lowest proportions of injured/ill older workers are observed for roof bolters and scoop operators. Roof bolting is physically demanding work, often requiring awkward postures and body movements. Additionally, risk of injury to roof bolters by falls of roof is higher than for any other occupation (Peters and Randolph, 1991). The lower proportion of injured/ill older scoop operators may be due to the higher prevalence of scoops in thin-seam mines, where the low working heights require the low clearance afforded by scoops. And, as noted previously, the low working heights associated with thin seam mines make these environments particularly stressful for the older worker.

In a study of age and occupational change among underground coal miners, Powell (1973) reported considerable differences in the proportion of older workers for different mining occupations. The study found proportionately fewer older workers in the most physically demanding and highly paced occupations. An increase in the number of miners leaving work or moving from heavier to lighter work was markedly noticeable at about the age of 45.

AGE DIFFERENCES IN INJURY RECOVERY TIMES

The total number of valid lost workday incidents for the three-year period from 1996 to 1998 was 29,227. The number of lost workdays for these incidents varied from one day to 6,000 days for cases of total permanent disability. Because a single extremely high score can increase the mean dramatically, the median number of days lost for nonfatal injuries resulting in lost time was used to examine differences in the number of days lost for three age groups of injured workers (18 - 34, 35 - 44, 45+). The median number of days lost for the three age groups is presented by canvass and commodity in Table VI. The total number of valid lost workday incidents With the exception of anthracite coal, the median number of days lost for injured older workers (aged 45+) consistently exceeds that observed for both groups of younger workers (aged 18 - 34 and 35 - 44). With minor exceptions, this trend persists within occupational groupings as well (see Table VII). Additionally, the median number of days lost shows significant variations among the different types of operations and occupations as well as among age groups.

Although the focus in this study is on workers aged 45 years and older, it is also worth noting that the median number of days lost for injured workers aged 35 to 44 exceed those for younger workers in most of the commodities and occupations examined. In a study conducted in the New South Wales underground coal mining industry, the mine worker's age was also identified as a significant factor associated with occupational injury severity as measured by the number of lost workdays (Hull et al., 1996). The study also found that the part of the body injured, the type of accident, the source of the injury, and the mine worker's activity at the time of injury contributed to the severity of the injury as well. However, what remains unclear is the degree to which the biological aging process contributes to the decreased resilience of older workers to recover from injury versus other factors that may accelerate the effects of aging, such as the cumulative effects of a poor working environment and the prevalence of chronic health conditions (Wegman, 1999).

Table V. Percentage of injured or ill workers aged 45 years and older by occupation for five types of mining operations (MSHA, 1998).

Type of operation/Occupation	Total no. of injuries/ illnesses	% aged 45+
Underground Bituminous Coal.........	**5,403**	**42.3**
Supervisor	427	54.8
Electrician/helper/wireman	279	53.4
Mechanic/repairman/helper	500	48.8
Belt/conveyor man	326	45.4
Laborer/utility man	942	45.1
Continuous miner op	342	38.9
Shuttle car/ram op	399	38.1
Continuous miner helper	107	32.7
Roof bolter op	1,075	29.6
Scoop/load-haul-dump op	245	26.5
Surface Bituminous Coal	**1,997**	**50.1**
Crane op	83	77.1
Electrician/helper/wireman	83	63.9
Supervisor	107	61.7
Prep plant worker	108	52.8
Bulldozer op	219	52.1
Mechanic/repairman/helper	464	51.3
Truck driver	115	49.6
Welder	123	44.7
Laborer/utility man	192	41.7
Surface Crushed Stone	**2,682**	**30.1**
Supervisor	146	45.9
Bulldozer op	136	41.2
Frond-end loader op	161	34.8
Welder	166	33.1
Mechanic/repairman/helper	562	31.5
Truck driver	283	27.9
Mill worker	437	26.8
Laborer	371	16.2
Cement	**917**	**43.8**
Mechanic/repairman/helper	332	52.4
Mill worker	163	36.8
Laborer	162	14.2
Sand&Gravel	**1,627**	**26.7**
Supervisor	105	42.9
Front-end loader op	140	33.6
Mill worker	211	28.4
Bulldozer op	131	28.2
Mechanic/repairman/helper	253	27.3
Truck driver	105	26.7
Welder	82	20.7
Laborer	282	14.5

Table VI. Median number of days lost by commodity and age group for lost workday cases reported to MSHA from 1996 to1998.

Canvass class/ Commodity	Age group 18 - 34	35 - 44	45+	Total
Coal	**8**	**16**	**21**	**16**
Anthracite	12	23	22	18
Bituminous	8	15	21	16
Metal	**8**	**12**	**18**	**12**
Iron ore	7	12	17	13
Alumina mill	12	14.5	16	14
Lead/zinc ores	11	25	32.5	22
Silver ores	6	10	19	8
Copper	11	18	25	17
Gold	8	8	12	8
Other metals	8	12	18	12
Nonmetal	**7**	**11**	**17**	**10**
Trona, potash, borate minerals	9	19.5	23	17
Phosphate rock	13	17	21	18
Clay, common	7	8	15	8
Other nonmetals	7	10	13	8
Stone	**7**	**10**	**13**	**9**
Cement	8	11	13	11
Lime	10	14	18	14
Crushed stone	6	10	12	9
Dimension stone	6	5	11	6
Sand&Gravel	**6**	**8**	**11**	**8**
Totals	**7**	**13**	**18**	**12**

Table VII. Median number of days lost by age group, commodity, and occupation for lost workday cases reported to MSHA from 1996 to 1998.

	Age group			
Type of operation/Occupation	**18 - 34**	**35 - 44**	**45+**	**Total**
Underground Bituminous Coal	**8**	**16**	**22**	**16**
Supervisor	20	14	25.5	20
Electrician/helper/wireman	6	14	12.5	13
Mechanic/repairman/helper	11	17	22	19
Belt/conveyor man	5	16	20	15
Laborer/utility man	7	18	21	17
Continuous miner op	20.5	15	22	18
Shuttle car/ram op	7	11	20.5	12
Continuous miner helper	7	21	21	19
Roof bolter op	9	14	24	15
Scoop/load-haul-dump op	6.5	19	21	15
Surface Bituminous Coal	**6**	**14**	**19**	**14**
Crane op	NC	15	24	18.5
Electrician/helper/wireman	10.5	24	22	20
Supervisor	NC	19.5	17	18
Prep plant worker	7	18	21	17
Bulldozer op	7	11	19.5	13
Mechanic/repairman/helper	6	15	17	14.5
Truck driver	6	16	28	15
Welder	11	13	20.5	17
Laborer/utility man	5	13	18.5	10
Surface Crushed Stone	**6**	**10**	**12**	**9**
Supervisor	14	7.5	14	10.5
Bulldozer op	6	12	8.5	8.5
Frond-end loader op	7	7.5	10	7.5
Welder	8	12	13.5	12
Mechanic/repairman/helper	7	11	15.5	11
Truck driver	6	9	10	8
Mill worker	7	11	12	9
Laborer	5	9	12	7
Cement	**8**	**11**	**13**	**11**
Mechanic/repairman/helper	7	11	13	11
Mill worker	10	10	14.5	11.5
Laborer	7	12	11.5	9
Sand&Gravel	**6**	**8**	**11**	**8**
Supervisor	6	8	22	10
Front-end loader op	6.5	11	18	8
Mill worker	7	8	10	8
Bulldozer op	6.5	7.5	24	10
Mechanic/repairman/helper	6	9	10	8
Truck driver	6	8	9	8
Welder	9	9	8.5	9
Laborer	5.5	9	9.5	7

NC – not computed (fewer than 10 observations)

SUMMARY AND DISCUSSION

The current study identified increasing trends in the proportions of older injured or ill workers over the past decade for various sectors of the mining industry. The most recent release of the MSHA injury/illness data, the 1998 reporting year, indicates that the highest concentrations of older injured or ill workers occur in coal, iron ore, alumina mills, cement, and trona operations. Coal operations account for a substantial portion of the injuries and illnesses reported for older workers. Within these operations, the distribution of older injured/ill workers indicates that the highest proportions are found at large surface operations, while the lowest proportions occur in small underground mines.

In general, among the different occupations examined, higher proportions of older injured/ill workers were observed for supervisors, electricians, mechanics, and surface equipment operators (excluding truck drivers). Conversely, the lowest proportions of older injured or ill workers occurred for the occupations designated as laborers at surface operations. And in underground coal mines, the occupations of roof bolters and scoop operators had the lowest percentages of older injured/ill workers. Regardless of occupation or type of operation, the median number of days lost from work due to injury was higher for older workers than for younger workers.

Given the relatively high proportions of older injured/ill workers in the mining industry, particularly in coal, occupational health and safety programs need to address the problems of an aging workforce. Physiological changes associated with aging that may impact the capacity of older workers include decreases in the sensory functions (particularly auditory and visual senses), in the motor functions (muscular strength and endurance, reaction time), and in cardiorespiratory functions (aerobic power) (Robertson, 1998). However, researchers are quick to point out that these changes with age are highly variable and should not be applied indiscriminately to all aging workers (Brant et al., 1994). To prevent premature declines in work capacity among aging workers, the World Health Organization (1993) recommends that health and safety programs target three primary factors for intervention:

1. Excessive physical demands including static muscular work, lifting and carrying, repetitive movements, and awkward postures;
2. Stressful and dangerous work environments with a high risk of injury, or that are poorly lit or expose workers to extreme temperatures;
3. Poor work organization resulting in conflicts of responsibility and poor work planning or rigid working conditions.

Designing and applying effective interventions for older workers should be of critical importance in mining health and safety. Although the issue of an aging workforce is more urgent for some sectors of the mining industry than others, as workers continue to age, the health and safety of aging workers will be of increasing concern to all segments of mining.

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LUNCHEON SESSION

PROFESSIONAL AWARD FOR MINING HEALTH, SAFETY & RESEARCH

PROFESSIONAL AWARD FOR MINING HEALTH, SAFETY & RESEARCH

Dr. Michael Karmis is the Stonie Barker Professor and Head of the Mining and Minerals Engineering Department, Virginia Tech (VT) and the Director of the Virginia Center for Coal and Energy Research (VCCER), established by the Virginia Legislature to support educational, research and public policy programs in coal and energy. He earned his B.S. and Ph.D. degrees in Mining Engineering from Strathclyde University, Scotland. He joined VT in 1978, has served as Head of Department for about 12 years and as Center Director for two years.

His broad research interests are in mine safety, ground control, mine systems and in the development of energy and natural resources. He has authored more than 130 scientific papers and reports and edited 21 *Proceedings* volumes and two Textbooks published by the Society for Mining, Metallurgy and Exploration (SME). Dr. Karmis has directed or co-directed 37 major research projects and served as major advisor to 26 graduate students.

Dr. Karmis served as Chairman of the Annual Institute on Mining Health, Safety and Research for almost 20 years. Through his leadership, this national meeting, now in its 32nd year, is recognized internationally as a unique forum in this topic. Dr. Karmis was also successful in diversifying the original Coal Mining Institute, by attracting the interest of the metal mining, quarrying, industrial minerals and aggregate industries, which are now important participants and contributors to the success of the meeting. In addition, the venue now alternates between VPI&SU and the University of Utah.

Two years ago, Dr. Karmis assembled a group of mine health and safety professionals who undertook the task of developing a state-of-the-art and encompassing health and safety textbook, devoted to this field. After a tremendous collective writing and editing effort *Concepts and Processes in Mine Health and Safety Management* was completed and will be published by SME in March 2001. This textbook is the product of a dedicated effort of 52 co-authors and numerous technical reviewers.

Dr. Karmis has been active within the Society for Mining, Metallurgy and Exploration and served for 12 years on the SME Board of Directors. A Professional Engineer in the U.S.A., and a Licensed Engineer (Eur Ing) in Europe, Dr. Karmis has been active in consulting with the minerals industry. He is a Distinguished Member of the SME, a Fellow of the Institute of Quarrying and a Fellow of the Institute of Mining and Metallurgy. Dr. Karmis has received many national awards, including the 1982 Publication Award of the SME, the 1987 Educational Excellence Award of the Pittsburgh Coal Mining Institute of America, the 1988 Distinguished Service Award of the Mining and Exploration Division of the SME and the 1995 Rock Mechanics Award. He was selected as the

1997 National Stone Association (NSA) Professor of the Year for his contribution to his discipline and profession. In 1998, he was presented with the Outstanding Faculty Award from the Old Timers Club for his contribution to mining education.

In recognition of his long career and extensive contributions to mining safety, the 31st Annual Institute on Mining Health, Safety and Research is pleased to present to Dr. Michael Karmis the Professional Award for Mining Health, Safety and Research.

GOOD SAFETY IS GOOD BUSINESS

Don Blankenship

Chairman, CEO & President
A. T. Massey Coal Company, Inc.

First, I'd like to say don't expect much from my talk. You see, the town in which I grew up had perhaps the worst school in a county ranked 55th of 55 state counties in a state ranked 49th or 50th in the country. Yes, I was raised (or should I say "reared"?) in a small Southern West Virginia town where the sign reading "SLOW CHILDREN PLAYING" had nothing to do with driving slowly or safely.

Regardless, today my subject is safety. Where I was raised (reared) being safe meant primarily two things: For children, don't play on the railroad tracks. And for adults, stay out of the beer joints.

Unfortunately, it is popular for the media to portray greedy businesses taking shortcuts to avoid safety. In fact, the reality is just the opposite: i.e., even if businesses were all greedy, safety adds to profit.

Coal companies who strive for top profitability, literally, cannot afford not to have the safest mines. Fiscal year to date, Massey's NFDL is 2.32, reflecting 67 accidents. For the same time period, the industry average NFDL is 4.75.If Massey's NFDL were the same as the industry average, Massey would have incurred 138 accidents during this time period—more than twice as many as actually occurred.

If Massey had had the industry average NFDL rate last year, it would have paid at least $2,840,000 to $3,550,000 more in workers compensation costs. This amount does not include the time and production lost with each accident, nor the medical cost of the accidents. Fatalities cost well over $1 million for each occurrence.

Today, I want to discuss the following:

- How Massey's safety program (S-1) works.
- Some examples of the program in action.
- Some results of the program (will do this first).

Massey runs a NFDL rate of 2.32 – again, twice as good as the industry average, and it does this despite having one of the highest, perhaps <u>the</u> highest, ratios of underground workers (and face workers) of any major coal company

For those of you who are not familiar with Massey, it operates the largest number of underground mines in the US and possibly in the world. Last year, it was the fifth largest coal company in the US, based on revenue, and the seventh, based on tonnage. It employs approximately 3300 members, operates 15 mines and 56 underground sections.

Massey began its S-1 program (Safety is Job 1) in the early 1990s. At that time, Massey's NFDL rate was better than (about 2/3 of) the industry average -- about where the coal industry average is today.

The program is, fundamentally, a three-part effort. The first part consists of a set of standards (the S-1 manual, including mandates developed by Massey's SDG group -- the Safety Development Group).

The second is a continuing series of audits by a (mostly) outside safety audit team. At least once each year, every operating company is visited for a detailed (weeklong is typical) review of its operations to check for compliance with S-1.

The third is a continuous improvement plan and a commitment to engineer out hazards and to learn from our mistakes. As part of this process, the SDG investigates every lost-time accident in Massey and makes recommendations to prevent these from reoccurring. Their recommendations have never been rejected because of financial considerations.

As an example of the improvement process, several years ago, at one of the Massey mines, a rock that fell onto an open jeep and injured a rider. The recommendation/mandate from the SDG was that Massey companies would operate only covered rail vehicles, which Massey has done now for the past five years. Covered vehicles insured that we would not have any more injuries like this anywhere in the company.

Massey has a second program, well known in the industry, called P-2 (production is next to safety). While this is also a set of standards as well as a continuous improvement program, what is striking is that, for many of the provisions of both the S-1 and P-2 programs, it is difficult (sometimes impossible) to decide whether they belong in the S-1 or the P-2 books. Indeed, a number of the provisions appear in both books. One example is the use of forklifts on all underground CM sections. They could just as easily have been mandated by the safety development group as they could have been dictated by a P-2 provision.

At Massey, virtually all section supplies arrive at the mine site in pallets that have been sized to fit the clearance over the track entry for that mine. The pallets are loaded onto the supply cars by the outside forklift, and are unloaded by the section forklift. The supplies are also delivered to the point of usage by the forklift and handled manually just once. This reduces the likelihood of slips and falls, as well as back and finger injuries. It also greatly reduces the number of man-hours spent loading and unloading supplies.

Essential to any safety program is the commitment of upper management. At Massey, we believe that this commitment must be shown by decisive action. The action may be directed toward personnel or it may involve physical improvements (spending).

For example, if a person is observed going out from supported roof, he is fired. No ifs, ands or buts. At the same time, when Massey made the decision to go to forklifts on every section it was a \$3 million decision (50 sections at \$60K each) and was done within a year.

There are also certain things that Massey's safety program is <u>not</u>. It is not based on slogans or banners or catch phrases. (Though good communication is a critical part of any management process. The S-1 program, for example, includes having the auditors interview our hourly members.)

It is not "touchy-feely". Attitudes are important, but we believe that these can better be changed by having the members see positive management actions rather than being given feel-good speeches.

It is not a "no-consequences" program. There are definite penalties to management for failing to comply with the S-1 program.

It wouldn't be appropriate to visit Virginia Tech in the fall and not mention football. In fact, last night's football game cancellation when the thunderstorm arose was a good example of safety overruling profits and popularity.

But my football trivia question is: What college football team won more games than any other in the 90's? What college team beat Clemson by the same margin last year that Florida State did? What team has the longest Division I winning streak at home? The longest current winning streak in total? What team had Heisman candidate finalists 2 of the past 3 years? What school did Stan (Suboleski) say I graduated from?

The answer is <u>Marshall University</u>.

Now I'd like to leave you with a few thoughts before I present a short film that we put together on the S-1 program.

Nothing happens unless first a dream.
- *Carl Sandburg*

Not everything that is faced can be changed but nothing can be changed until it is faced.
- *James Baldwin*

There's a way to do it better… find it.
- *Thomas Edison*

See things as you would have them instead of as they are.
- *Robert Collier*

All you need is a plan, a road map, and the courage to press on to your destination.
- *Earl Nightingale*

Great results cannot be achieved at once; and we must be satisfied to advance in life as we walk, step by step.
- *Earl Nightingale*

Never, never, never, never, never, never give up.
- *Winston Churchill*

Thank you for the opportunity to speak about safety. I'll be happy to answer any questions.

TECHNICAL SESSION II

MINING HAZARDS

Session Chairs

Frank Linkous
Chief, Division of Mines
Virginia Department of Mines, Minerals and Energy

Rick Sink
Manager of Business Services
WHAM Inc.

FIELD ASSESSMENT OF RETROFITTING SURFACE COAL MINE EQUIPMENT CABS WITH AIR FILTRATION SYSTEMS

by J.A. Organiscak[1], A.B. Cecala[1], W.A. Heitbrink[1], E. D. Thimons[1], M. Schmitz[2], and E. Ahrenholtz[2]

[1]National Institute for Occupational Safety and Health. [2]Clean Air Filter[7]

ABSTRACT

Operator cabs on a front-end loader and a rotary rock drill were retrofitted with ceiling mounted heating/AC units and air filtration systems. Subsequently surface coal mine field studies were conducted to evaluate the respirable dust protection these retrofitted cab systems offer to the equipment operator. A significant 10:1 respirable dust protection factor (ratio of outside to inside cab dust levels) was measured for the front-end loader cab with positive pressurization of the cab interior. Whereas an insignificant 3:1 respirable dust protection factor was measured for the drill cab without positive pressurization of the cab interior. These results indicate that achieving positive interior cab pressurization with retrofitted cab filtration systems is a key element to their dust control effectiveness.

INTRODUCTION

The Mine Safety and Health Administration (MSHA) permissible dust exposure for coal mine workers is a shift average of 2.0 mg of airborne respirable coal mine dust per cubic meter of air (2.0 mg/m^3) as defined by the Mining Research Establishment (MRE) Criteria (U. S. Code of Federal Regulations, 1998). If the airborne respirable dust (ARD) sample contains more than 5% crystalline silica, the dust standard is reduced to the quotient of 10 divided by the percentage of silica in the dust, limiting the respirable crystalline silica exposure to a maximum of 100 ì g/m^3 (MRE equivalent) for the working shift. Compliance with these respirable dust standards is expected to significantly reduce a worker=s risk of occupational lung disease over an average life expectancy.

MSHA=s dust enforcement program includes both inspector and coal mine operator dust sampling. MSHA=s surface coal mine dust program focuses its sampling efforts at designated work positions (DWP=s). These are particular areas or occupations that have been historically shown to either exceed 1 mg/m^3 of respirable dust or have high silica exposure. The local MSHA official has the authority to classify DWPs based on an operation=s dust sampling history or to classify non-designated work positions (NDWPs) based on a history of competent dust abatement. The most frequently sampled and classified DWPs at surface coal mines are the highwall drill operator, bulldozer operator, refuse/backfill truck driver, and highlift operator. MSHA dust exposure data from 1985-1992 showed that the percentage of the DWP dust samples containing more than 5% silica ranged between 81% for the highwall drill operator and 25% for the highlift operator [Tomb et al. 1995]. The percentage of these DWP dust samples that exceeded the 100 ì g/m^3 silica limit ranged between 77% for the highwall drill operator and 26% for the highlift operator [Tomb et al. 1995]. These data suggest that overexposure to silica dust is an ongoing surface coal mine dust problem for the highwall drill operator, bulldozer operator, refuse/backfill truck driver, and highlift operator.

An engineering control measure for surface mining equipment is enclosed operator cabs with air filtration systems. These cabs usually re-circulate and re-condition a majority of inside cab air with a smaller portion of the air added from the outside as makeup air. In order for the enclosed cab to protect the operator from the dust generated during excavation, the inside cab air must be efficiently filtered and the cab enclosure must be maintained under a positive ventilation pressure.

The agricultural industry has developed quality performance specifications for tractor cab enclosures. These enclosures are designed to protect equipment operators from pesticide exposures during their application. The premise for this standard is that a cab must act as an acceptable substitute for a respirator that adheres to the Worker Protection Standard (WPS) of the U.S. Environmental Protection Agency (EPA) [Heitbrink et al. 1998]. The American Society of Agricultural Engineers (ASAE) Standard S525 specifies that a cab enclosure will provide a 50:1 reduction in particles (commonly referred to as a protection factor of 50) with an aerodynamic diameter larger than 3 ì m [ASAE 1997]. This enclosed cab performance standard is equivalent to the protection offered by a full face respirator. The ASAE standard also specifies a minimum positive differential static pressure of 6 mm of water gauge for the cab enclosure and recommends ambient aerosol test procedures with optical particle counters to evaluate the performance of these enclosures [ASAE, 1997]. However, field evaluation of these agricultural cab testing procedures has shown that low ambient aerosol concentrations outside the cab can notably bias the evaluation to yield lower protection factors [Heitbrink et al. 1998; Heitbrink et al. 1999].

Recent surface mining dust surveys conducted by the National Institute for Occupational Safety and Health (NIOSH) on drills and bulldozers have shown that enclosed cabs can effectively control the operator=s dust exposure, but enclosed cab integrity problems still exist [Organiscak and Page 2000]. The enclosed cab protection factors measured on rotary drills ranged from 2.5 to 84, and those measured on bulldozers ranged from 0 to 45. Some of the newer equipment cabs tended to be better sealed and cleaner, while some of the older equipment tended to be more poorly sealed and dirtier. One of the least protective drill cabs did not include any heating, air conditioning, and air filtration systems. Some of the older surface mining equipment also possesses enclosed cabs with heating, but no air-conditioning, and/or air filtration systems.

To evaluate the respirable dust protection provided by retrofitting enclosed cab improvements, NIOSH recently conducted several dust control field studies of retrofitting old enclosed cabs with air-conditioning, heating, and air filtration systems. This paper describes these field studies conduced on a CAT 980B front-end loader and a Davey M8B rotary drill.

RETROFITTED CAB DEMONSTRATIONS

An enclosed operator cab field dust evaluation was conduced on a CAT 980B front-end loader and a Davey M8B drill before and after the cabs were retrofitted with roof mounted air- conditioner/heater units and external filtration units. This work was conducted as part of a mine demonstration project to improve the dust control integrity of enclosed operator cabs on mobile mining equipment. The two pieces of equipment studied originally possessed enclosed cabs with floor heaters and no air-conditioning and filtration systems. A baseline cab dust study was initially conducted on each piece of equipment for 3 to 4 production shifts before any cab modifications were made. A follow-up dust study was repeated for 4 to 6 production shifts after the cab modifications.

Each enclosed cab was retrofitted with a new ceiling-mounted Red Dot air-conditioner/heater unit (Model R-9757) with an external make-up air fan and Clean Air Filter[7] filtration system.[3] The external filter for the make-up air was a 2-stage Clean Air Filter[7] cartridge with a cellulose paper medium for the first stage and a final respirator medium as the second stage. A Clean Air Filter[7] housing with cyclonic inlets contained the filter cartridge and was connected to a centrifugal fan for blowing make-up air into the Red Dot air-conditioner/heater unit. A single-stage respirator media filter was also mounted on the Red Dot unit=s inside cab re-circulation inlet. The respirator media filter performance specifications are at least 99 percent capture efficiency for 0.1 ì m of mono-dispersed sodium chloride particles, as determined from TSI test method 85 lpm/dm^2.

Installation of each piece of equipment for the ceiling-mounted system took about a day, while another half day was invested on sealing the cab. The original floor heaters in the cabs were either removed or disconnected from operation, so that the Red Dot ceiling units and Clean Air Filter[7] units would consistently be used for both heating and air-conditioning functions. Cab enclosure structures on both pieces of equipment had numerous holes and cracks and thus positive inside cab air pressure was difficult to achieve. To enhance cabin air pressure, the CAT 980B cab enclosure cracks were sealed with silicon caulk and the door gaps were sealed with dense foam weather strip. A positive static cab pressure of 0.01" to 0.015" water gauge was achieved after several hours of sealing all the visible gaps/holes in the cab. The Davey M8B drill cab structure was in very poor condition, with large holes in the cab for the mechanical drill control linkages and a loosely fitted bi-folding door on the drill table side of the drill. Because of the numerous holes and gaps present in the cab enclosure, positive static cab pressure was not achieved on the Davey M8B drill. However, the discharge of the air-conditioning/heating unit was directed over the operator position in an attempt to provide him with a clean-air zone within the cab.

[3]Mention of any company name or product does not constitute endorsement by the National Institute for Occupational Safety and Health.

FIELD SAMPLING PROCEDURES

Dust sampling was conducted during multiple working shifts inside and outside the operator cabs to measure the enclosed cab dust control protection factor before and after the cab modifications were made. Data collected included: personal respirable dust samples; personal impactor dust mass size distributions; optical particle counter (OPC) size distributions; miniature real-time aerosol monitor (MINIRAM) respirable dust levels; weather conditions (wind speed, direction, temperature, humidity, etc.); and qualitative documentation of equipment operation.

***Gravimetric Sampling*:** Area airborne respirable dust sampling was conducted with personal samplers located inside the operator cabs and near the dust source outside of the cabs to assess the protection performance of these cabs in relation to operator dust levels. Each personal dust sampler included a Mine Safety Appliance (MSA) Flow-Lite pump, operating at 2.0 liters/min, with a 10-mm Dorr-Oliver nylon cyclone classifier to collect a respirable dust sample (U. S. Code of Federal Regulations, 1998). The respirable dust was deposited on a 37 mm MSA coal dust filter cassette. Three personal gravimetric dust samplers were used at each sampling location so that adequate amounts of dust could be collected for silica analysis. One

Sierra 298 personal impactor, operating at 2.0 liters/min, was also placed with the 3 personal samplers at the fixed sampling locations on each piece of equipment. Impactor dust size fractions of 21.3, 14.8, 9.8, 6.0, 3.5, and 1.55 ìm were collected on silicon greased filter substrates with smaller dust particles collected on a polyvinyl chloride final filter. The areas sampled are described below for the two pieces of equipment:

1. Highwall Rock Drill: 1) Inside front of the operator cab above control panel; 2) Outside back of cab; 3) Mobile tripod kept in the downwind dust plume near the drill hole shroud (*see figure 5, an impactor is not commonly used here due to dust overloading*).

2. Front End Loader: 1) Inside right side of the operator cab above bucket controls; 2) Outside the cab behind the left entrance door.

The MSA filters and impactor substrates were pre- and post-weighed to the nearest 0.001 mg on a microbalance. Personal dust sampling was usually conducted for more than 7 hours during the shift. The impactors were operated between 4 and 8 hours during the shift, depending on the dust concentrations or loading activity at the particular locations. Three shifts of baseline dust samples (March 29 - 31, 1999) and 6 shifts of controlled dust samples (Aug. 11 and 12, Sept. 14 and 15, and Oct. 20 and 21, 1999) were collected on the CAT 980B front end loader. Four baseline shifts of dust samples (March 29 - April 1, 1999) and 4 shifts of controlled dust samples (Aug. 11 and 12, and Oct. 21 and 22, 1999) were conducted on the Davey M8B rock drill.

The respirable dust mass collected on the filter cassettes were analyzed by PRL=s in-house analytical lab for crystalline silica by MSHA=s P-7 infrared spectrophotometer method [Ainsworth et al. 1989]. The silica content of the dust (in percent) is reported for at least 0.25 mg of dust mass and 25 ìg of silica. Many of the filter cassettes did not have 0.25 mg of dust mass, so multiple filter cassettes from the same sampling location were composited for silica analysis. Finally, some dust levels were so low that even the composite filter mass did not meet the minimum reportable range.

***Instantaneous Sampling*:** Supplemental airborne dust sampling was also conducted in the enclosed cabs using instantaneous dust monitors. A MINIRAM instantaneous dust instrument connected to a Metrosonics 331 data logger was placed with the gravimetric samplers in the operator cabs to examine real-time respirable dust level variations during the shift. The MINIRAM measures respirable dust by light-scattering techniques and was operated in the passive sampling mode. The Metrosonics 331 data logger recorded analog voltage output from the instrument which was downloaded to a personal computer for data analysis [Cecala et al. 1988]. Since the MINIRAM was not individually mass calibrated to the different dusts sampled, the instantaneous dust data is expressed in relative MINIRAM units to identify corresponding dust level changes influenced by particular operator practices during the shift.

As in agricultural cab testing procedures, instantaneous OPCs (GRIMM Technologies, Inc.) were also used during part of the sampling shift on the front-end loader on Aug 11th and on the drill on Aug 12th in similar fashion to the agricultural cab testing procedures [Heitbrink et al. 1998; Heitbrink et al. 1999]. OPC sampling was simultaneously conducted inside and outside the cab during a portion of the shift with the instruments alternated between these positions to remove any instrumental bias from the outside and inside particle counting. The OPC=s size data were used to determine the percentage and size of dust particle penetration into the enclosed cab systems.

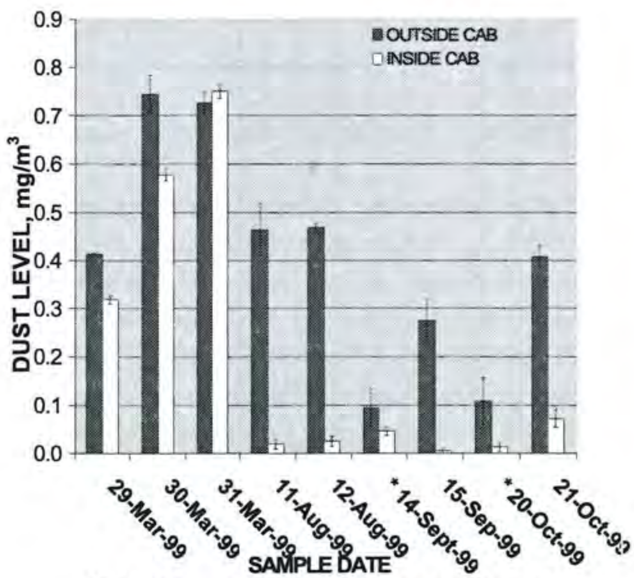
***Wind and Weather Parameters*:** A wind speed and directional instrument was positioned near the drilling operation to document the migration direction of drill dust. A Brunton Pocket Transit was used to align the instrument for proper azimuth readings. This instrument was also connected to two Metrosonics 331 data loggers to record both the speed and direction of the wind. Again the output stored in these loggers was downloaded to a personal computer for data analysis [Cecala et al.1988]. The drill location and orientation during the shift was drawn on a map to identify the dust migration of the drill dust with respect to the operator cab. Also, a description of weather conditions was recorded with wet and dry bulb temperatures taken during the shift. Because the loader is mobile, constantly changing its orientation with the wind, its cab dust protection effectiveness was anticipated to be a good long-term representative average for the various wind directions. Therefore, wind measurements were not taken around the front end loader.

FIELD STUDY RESULTS

***CAT 980B Front-End Loader*:** Figure 1 shows the CAT 980B front-end loader loading coal from a stockpile with the Red Dot and Clean Air Filter[7] units operating on top of the cab enclosure. The 980B loader was used for loading stockpiled coal into the portable crusher, removing the coal seam from the pit, and loading coal trucks for transport from the mine. The loader was commonly rotated among these multiple tasks during the same sampling shift. The operator kept the doors and windows shut during all the sampling shifts (baseline and modified cab), except to enter and exit the loader cab.

Average respirable dust levels (taken by 3 personal samplers) measured inside and outside the loader cab are shown in figure 2 with their respective standard error bars for the multiple samplers. Results from the 3 baseline shifts (March 29 - 31) and six controlled shifts (Aug. 11 and 12, Sept. 14 and 15, and Oct. 20 and 21) show that the refurbished enclosed cab had noticeably reduced respirable dust levels inside the cab as compared to outside the cab. Wet ground conditions on Sept. 14 and Oct. 20 from early morning rain lowered the outside loader cab dust levels as compared to the other four controlled shifts with dry ground conditions.

Figure 1 - CAT 980 loader studied.



·Note: the ground was wet from morning rain on Sept. 14 and Oct. 20

Figure 2 - Dust results from loader cab evaluation.

A typical instantaneous dust level history (MINIRAM) recorded inside the loader cab is shown in figure 3 for the March 31st baseline shift and the Aug. 12th controlled shift. These dust level histories illustrate the notable inside cab dust reductions achieved from the loader cab modifications. They also show that the pressurized cab air cleaning system maintained lower dust levels with very little variation during the work shift.

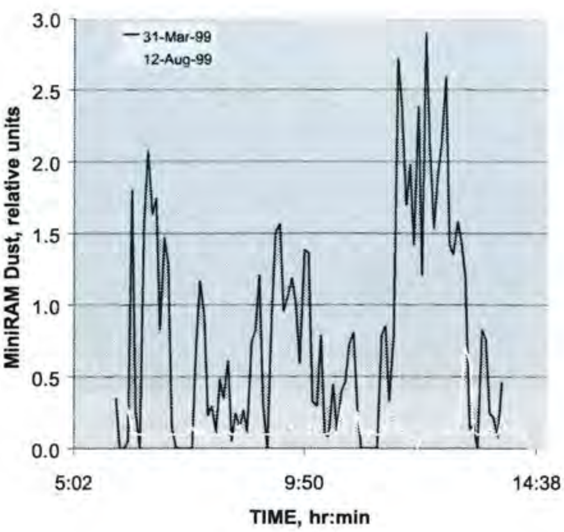


Figure 3 - Loader cab dust level histories.

Table I shows the average dust levels and cab protection factors with daily ranges for the baseline and modified cab on the CAT 980B front-end loader. The CAT 980B front-end loader showed a significant improvement in operator cab dust control effectiveness with the addition of the Red Dot and Clean Air Filter[7] units to the cab enclosure and the sealing of all the visual openings or cracks found in the cab enclosure. The inside cab dust levels were reduced by about one order of magnitude after these changes were made. The enclosed cab dust protection factors (outside/inside) was increased on average from 1.1 to 10.1. Similarly, the percentage of cab penetration of dust ((inside/outside) H100) was reduced from 87% to 10%, while the percentage of dust reduction (((inside ! outside)/outside) H100) increased from -13 % to -90 % with the cab improvements.

This cab improvement is considered to be statistically significant. The one-tailed t-test probability (*p*-value) that the inside and outside cab dust levels are the same notably decreased from 0.328 to 0.006 with the cab modification. Although this average improvement was significant, it must be noted that the day-to-day protection factors for the modified cab ranged from 2.1 to 50.1 (see table I), while the inside cab respirable dust levels consistently remained under 0.07 mg/m^3. This supports the work of others that field evaluation of cabs with low exterior dust levels tends to bias the cab=s protection factor towards lower levels because the relative differences approach the background levels inside the cab [Heitbrink et al. 1998; Heitbrink et al. 1999].

The particle size percentage of dust penetration (% of particle count inside the cab as compared to outside the cab) into the modified loader cab by particle size count can be seen by the OPC data collected on Aug 11 in figure 4. This figure shows that a somewhat higher percentage of smaller sized dust particles was observed to penetrate the loader cab. Never the less, the loader cab appeared to provide effective protection throughout the particle size range, with less than 5 % of the 1 ìm or larger particles penetrating the loader cab enclosure. The average shift cab protection factor and the average shift percentage of cab penetration of respirable dust measured with the personal samplers during this particular day was 24.2 and 4.1%, showing reasonably good agreement with the OPC data.

Although the original cab provided negligible control of respirable dust, the original cab enclosure kept a notable amount of the larger sized dust particles from entering the cab. The average mass median diameter (MMD) of the outside dust particles and the inside dust particles for the original cab was 27.3 ìm and 11.5 ìm, respectively. This indicates that the CAT 980B front-end loader had a fairly good enveloped cab structure for resealing and retrofitting with a filtered air conditioning system. Finally, the silica percentage of the respirable dust for the CAT 980B front-end loader was found to be very low, commonly below 5%, both inside and outside the

TABLE I - DUST LEVEL SUMMARY OF CAB STUDY

PARAMETERS	CAT 980B FRONT-END LOADER		DAVEY M8B ROTARY DRILL	
	BASELINE 3 Shifts Average [*Range*]	MODIFIED CAB 6 Shifts Average [*Range*]	BASELINE 4 Shifts Average [*Range*]	MODIFIED CAB 4 Shifts Average [*Range*]
Outside Cab Respirable Dust Level, mg/m^3	0.63 [*0.41, 0.74*]	0.30 [*0.09, 0.47*]	0.72 [*0.10, 1.13*]	0.23 [*0.12, 0.46*]
Inside Cab Respirable Dust Level, mg/m^3	0.55 [*0.32, 0.75*]	0.03 [*0.01, 0.07*]	0.13 [*0.09, 0.19*]	0.08 [*0.02, 0.17*]
†*p*-value ($t_{statistic} \leq t_{critical}$, one-tail test)	0.328	0.006	0.042	0.078
††Cab Protection Factor, (Out/In)	1.1 [*1.0, 1.3*]	10.1 [*2.1, 50.1*]	5.4 [*0.5, 13.2*]	2.9 [*2.0, 7.9*]
††Percentage of Cab Penetration, (In/Out) $\times$100	87 [*77, 103*]	10 [*2, 48*]	19 [*8, 182*]	34 [*13, 49*]
††Percentage of Dust Reduction, ((In − Out)/Out) $\times$100	-13 [*-23, +3*]	-90 [*-98, -52*]	-81 [*-92, +82*]	-66 [*-87, -51*]
‡Mass Median Diameter--Outside Cab, ìm	27.3 [*25.1, 30.6*]	25.6 [*19.9, 36.9*]	31.3 [*24.9, 36.6*]	32.5 [*24.2, 48.6*]
‡Mass Median Diameter--Inside Cab, ìm	11.5 [*9.2, 15.4*]	8.1 [*1.7, 21.7*]	28.7 [*18.3, 46.2*]	18.3 [*13.1, 25.4*]
‡‡Silica Percentage of Respirable Dust--Outside Cab	3.5 [*3.0, 4.1*]	2.0 [*1.6, 3.1*]	35.4 [*28.0, 39.9*]	35.4 [*18.2, 46.6*]
‡‡Silica Percentage of Respirable Dust--Inside Cab	3.9 [*1.8, 5.6*]	N.E.S.	22.1 [*15.2, 31.6*]	29.3 [*N.E.S., 29.3*]

† Null Hypothesis (H_o): (Average Inside Cab Dust Level - Average Outside Cab Dust Level) = 0
Alternative Hypothesis (H_a): (Average Inside Cab Dust Level - Average Outside Cab Dust Level) < 0

†† The average cab protection factor, average percentage of cab penetration, and average percentage of dust reduction are based on the average dust level measured outside and inside the cab. The range is based on individual shift measurements.

‡ The mass medium diameter is the dust particle size where half of the airborne dust mass is above and half below this size (measured with Sierra personal impactors).

‡‡ Silica analysis of respirable personal samplers by MSHA Standard Method No. P-7, with infrared determination of quartz in respirable coal mine dust. Analysis usually performed on sample composites to obtain enough mass. N.E.S. - Not Enough Sample.

cab during all the baseline testing and modified cab testing.

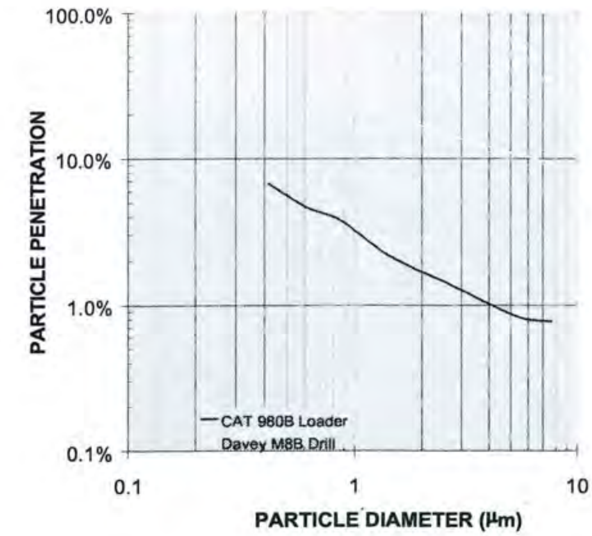


Figure 4 - Percentage of aerosol penetrating into the cabs.

***Davey M8B Rotary Rock Drill*:** Figure 5 shows the Davey M8B rotary drill with the Red Dot and Clean Air Filter[7] units operating on top of the cab enclosure during the highwall drilling operation. The drill operated throughout most of the shift, except for a morning break and short non-production time periods needed for minor drill maintenance. This drill operation utilized two employees working as a team. One employee operated the drill from within the cab (drill operator) while the other employee worked outside the cab (drill helper), changing drill steels and clearing the cuttings from the hole. Throughout the shift the drill operator and helper switched work details (positions). During the baseline testing, the two cab doors were constantly left open all shift so the employees could visually and verbally communicate with each other. During the modified cab testing, the two cab doors were typically closed during most of the drilling activities. The bifold door facing the drill table was opened for drill steel changes and both doors were opened for drill placement during the modified cab sampling shifts. The employees open these cab doors at these particular times so that they can observe and communicate (visually and verbally) to each other during the manual changing of drill steels and drill machine placement on the bench.

Figure 5 - Davey M8B drill studied.

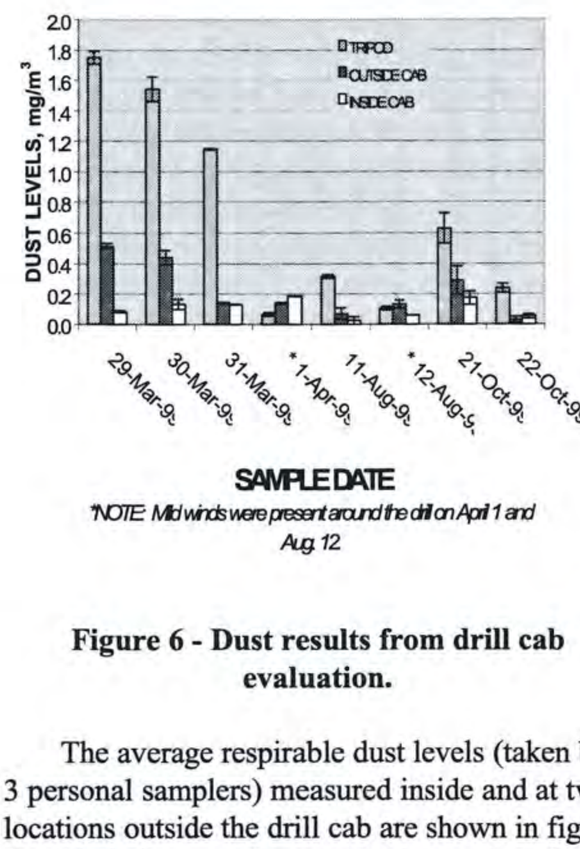


Figure 6 - Dust results from drill cab evaluation.

The average respirable dust levels (taken by 3 personal samplers) measured inside and at two locations outside the drill cab are shown in figure 6, with their respective standard error bars for the samplers. The inside and outside cab dust samplers were at fixed positions on the drill, while the mobile tripod dust samplers were placed on the downwind side of the drill hole shroud (see figure 5). The dust levels measured inside the operator cab were commonly lower and noticeably more consistent than those levels measured at the tripod and outside of the cab during the study. The tripod dust levels were more variable and typically higher than the outside cab dust levels, during both the baseline and modified cab testing. The tripod dust levels were also significantly higher during 3 of the baseline sampling shifts as compared to the modified cab sampling shifts.

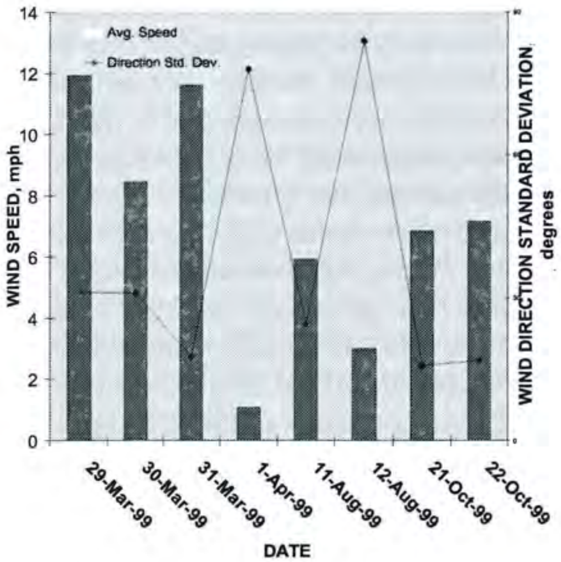


Figure 7 - Summary of daily wind speed and direction measurements around the drill.

The key factors influencing these day-to-day dust level variations were wind speed and direction. Figure 7 shows the average wind speed and the standard deviation of the wind direction around the drill for the sampling shifts, while figure 8 shows the typical wind direction histories observed during a strong windy day (March 31) versus a calm mild day (August 12). As evidenced by these data, on the strong windy days the wind direction deviations were notably less than for the calmer days. In comparison tripod dust levels were notably higher than the outside cab dust levels for the strong windy days, while they were more comparable on the calm days (see figure 6). This most likely occurred because the dust plume for the windy days was more directional towards the downwind tripod sampling location, while it was more dispersed among the tripod and outside cab sampling locations for the calm days (see figure 8). Figure 9 also shows the strong positive relationship between the tripod dust level and wind speed. The wind speed irrespective of directional variation was also believe to be a key factor in this tripod dust and wind speed relationship, since notably more dust entrainment around the drill was visually observed on the extremely windy days during the baseline sampling in March.

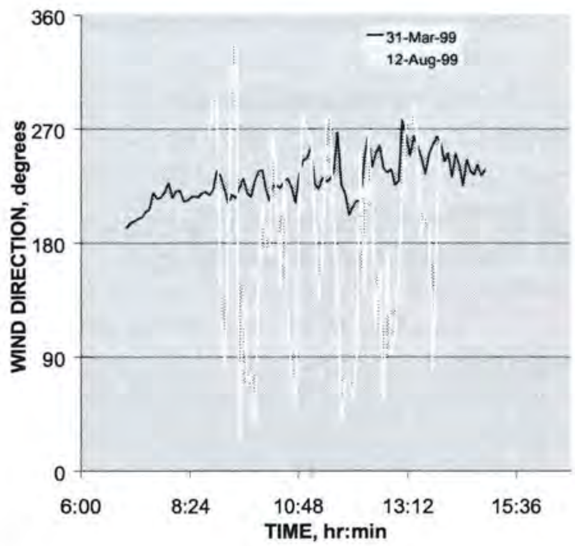


Figure 8 - Strong and mild wind direction histories around the drill.

Table I shows the average dust levels and cab protection factors, with daily ranges for the baseline and modified cab on the Davey M8B drill. The drill=s outside cab respirable dust levels reported in table I are inclusive averages of both the tripod and outside cab sampling locations. A negligible change in operator cab dust control effectiveness was measured by adding the Red Dot and Clean Air Filter[7] units to the cab enclosure. The enclosed cab dust protection factor was decreased on average from 5.4 to 2.9. Similarly, the percentage of cab penetration of dust was increased from 19% to 34% and the percentage of dust reduction decreased from -81% to -66% with the cab improvements.

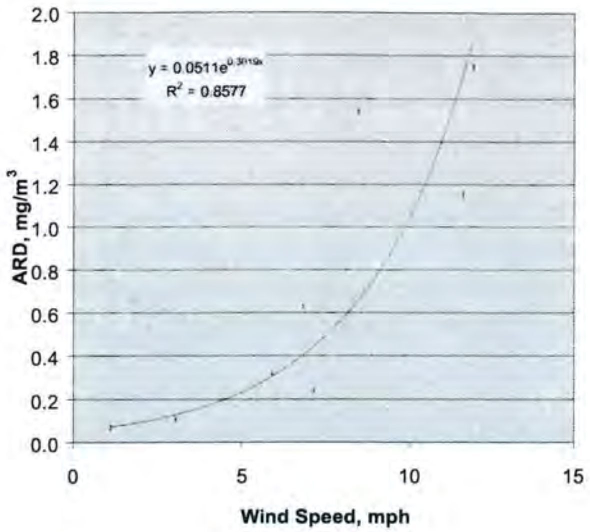


Figure 9 - Tripod dust level and wind speed relationship around the drill.

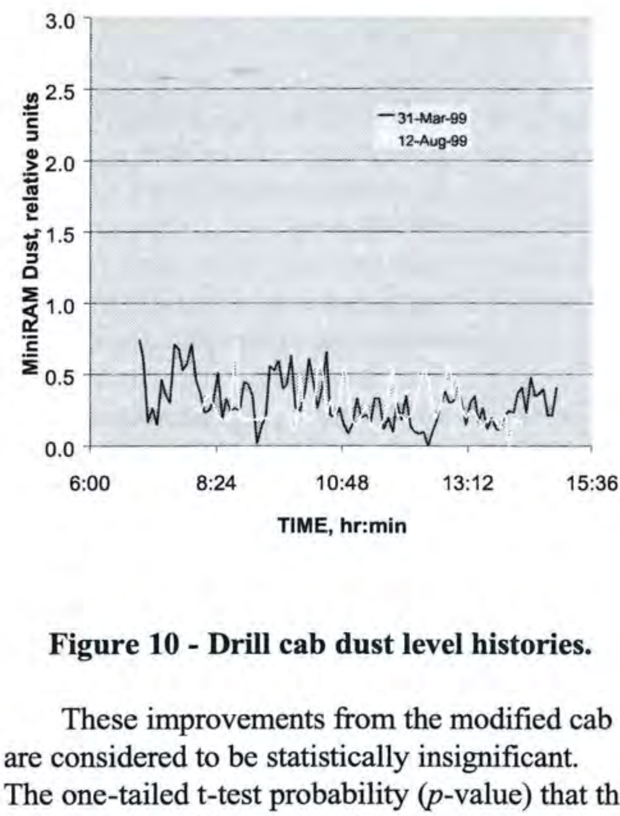


Figure 10 - Drill cab dust level histories.

These improvements from the modified cab are considered to be statistically insignificant. The one-tailed t-test probability (*p*-value) that the inside and outside cab dust levels are the same increased from 0.042 to 0.078 with the cab modification. Typical instantaneous dust level histories (MINIRAM) recorded inside the drill cab are shown in figure 10 for the March 31st baseline shift and the Aug. 12th modified shift. These dust level histories illustrate that no notable inside cab dust reductions were achieved from the drill cab modifications.

Problems encountered with the drill cab enclosure were large openings in the cab structure for the mechanical control linkages and a loosely fitting bi-folding door facing the drill table. Positive cab pressure was not achieved on the drill because of these enclosure sealing problems. Also, the operator usually opened the cab doors to communicate with the helper, during drill steel changes and drill moves. Furthermore, the wind direction played a very important role in the dust measurements made during these field studies, especially for the baseline conditions. These variations can be seen in the shift ranges of the cab protection factors, the percentage of cab penetration, and the percentage of dust reductions measured (see table I). The baseline cab protection factors ranged from 0.5 to 13.2 (percentage of cab penetration from 8 to 182 and percentage of dust reductions from -92% to + 82). The modified cab provided less variation in protection factors, ranging between 2.0 and 7.9 (percentage of cab penetration from 13 to 49 and percentage of dust reductions from -87% to - 57%). Although the drill cab=s clean air system did not show an average improvement in its effectiveness, the variation caused by wind effects were somewhat reduced.

The particle size percentage of dust penetration (percentage of particle count inside the cab as compared to outside the cab) into the drill cab by particle size count is demonstrated by the OPC data collected on Aug 12 in figure 4. As can be seen in this figure, the drill had a notably higher percentage of dust penetrating the cab as compared to the loader. The drill cab dust penetration was one order of magnitude higher as compared to the front-end loader. Ten to 30% of the 7 to 1 ìm dust particles, respectively, penetrated the drill cab enclosure. The average shift protection factor and the average shift percentage of cab penetration of respirable dust measured with the personal samplers during this particular day were 2.0 and 49.3%, showing reasonably good agreement with the OPC data.

The drill cab=s air cleaning system did provide some improvement in the median dust size that entered the cab. A negligible difference was observed in the mass median diameter (MMD) of dust outside the cab as compared to inside the cab during the baseline tests (31.3 ìm outside and 28.7 ìm inside). However, the MMDs of the outside dust and the inside dust for the improved cab were 32.5 ìm and 18.3 ìm, respectively. This indicates that the filtered air made some improvement within the cab, but the effect was diluted by outside air infiltration into the cab by gaps in the structure (no positive cab pressure) or by the door being opened frequently. Finally, the silica percentage of the respirable dust for the Davey M8B drill was found to be notably higher than that of the front-end loader, commonly above 20%, both inside and outside the cab during all the baseline testing and modified cab testing.

CONCLUSIONS

These field studies show that two key elements are needed to control dust levels in enclosed operator cabs. First, the cab needs to have a high quality of re-circulating and incoming filtered airflow; secondly, a cab structure needs to be adequately sealed to achieve positive static pressure with the incoming clean air flow. Both of these key elements were accomplished with the CAT 980B front-end loader, providing on average a 10:1 cab protection factor for the operator. The cab structure on the Davey M8B was not adequately sealed, diminishing the overall effectiveness of the cab air filtration system. By not achieving positive pressure inside the drill cab, the outside dust was able to penetrate the cab structure.

Relative cab performance measures determined in the field were noticeably affected by the wind and outside dust levels. The cab protection factor, percentage of cab penetration, and percentage of dust reduction measures were observed to change noticeably during day-to-day operations, while the respirable dust levels remained consistently low inside the cab. Exterior cab dust levels affected by weather (wind and precipitation) were found to considerably change the relative cab performance measures, especially if the exterior cab dust levels were very low. However, the primary goal of enclosed cab performance should focus on consistently achieving inside cab dust levels below worker compliance levels, while ensuring adequate protection from high dust levels outside the cab.

MSHA=s dust enforcement data show that enclosed cabs on drills, bulldozers, refuse/backfill trucks, and high lifts (front-end loaders) continue to be suspect in providing adequate operator protection from silica dust at mining operations. Therefore, some of the enclosed cabs used on mining equipment need to be updated, reconditioned, or better maintained. To resolve enclosed cab performance problems and improve mine worker health, MSHA is currently pursuing enclosed cab seminars around the U.S. to build partnerships between health specialists, labor, mining companies, mining equipment manufacturers, heating & air-conditioning equipment manufacturers and filter media companies [MSHA2000]. NIOSH is currently studying field measurement quality control procedures for prompt determination of an enclosed cab=s dust protection capabilities.

ACKNOWLEDGMENTS

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INVESTIGATING THE TRANSPORTATION OF DIESEL EXHAUST FUMES IN MINES TO RESOLVE EXPOSURE ISSUES THROUGH TRACER GAS TECHNIQUES

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ABSTRACT

Tracer gas techniques, employing sulphur hexafluoride (SF_6), can provide both quantitative and qualitative information[1]. These techniques, as offered by CANMET, range from the simple and accurate determination of mine airflows through to sometimes complex surrogate applications for tracking pollutants carried by the air. In this latter regard, and as will be shown in this paper, they can supply invaluable information that can assist an environmental engineer to gain a greater understanding of how the airflow/ventilation performs, and where applicable make an informed decision on any corrective action.

In the first example, a large underground room and pillar mine was experiencing relatively high levels of nitrogen dioxide (NO_2) in its exhaust corridor that contained milling operations and also served as the main access route to the production areas. Here, a 2.7ppm concentration of NO_2, lower than the 3ppm TLV-TWA[2], was used as an "action" level to reduce diesel usage. For this mine, a simple tracer gas test showed that the average travel time for air through the exhaust system was in excess of 24hrs as a result of the size, length and number of exhaust routes. Consequently, nitric oxide (NO) concentrations of the order of 4ppm leaving the production area could easily result in values greater than the 2.7ppm in the main exhaust. Overall, this investigation showed that it might be impossible to avoid "action" levels by increasing the airflow in the mine and that NO monitoring closer to the diesel activity might be a more appropriate control.

In the second example, a bucket shovel operator in an open pit mine was complaining of physical discomfort due to diesel exhaust fumes from "dirty" haulage trucks. Here, a preliminary study, which focused on the standard gaseous components, carbon dioxide (CO_2), carbon monoxide (CO), NO, NO_2 and sulphur dioxide (SO_2), and particulate (soot/ RCD) failed to find any significant exposure that might account for the complaints. The follow-up study, which looked at the exhaust components in more detail, doped a "dirty" truck's exhaust with tracer gas and demonstrated its potential to reach the shovel operator. Consequently, it was shown that shovel operators were possibly exposed to "aldehyde" concentrations that could cause the reported discomfort. As a result of this study the mine modified the exhaust systems of the offending trucks.

INTRODUCTION

Understanding ventilation can be very complex, especially when there is the need to determine its effectiveness to dilute and remove pollutants and how that may relate to the exposure of workers to contaminants within that environment. On the macroscopic scale, the use of anemometers has long been the standard practice to measure average air velocities and determine the bulk airflow, but their application fails to provide the detail required on the dispersal of pollutants by turbulent and differing velocity airstreams within the bulk flow. To a limited degree, the use of smoke as a visible indicator, can provide some insight into flow regimes on the microscopic scale but their application is limited and the observed results often not quantifiable. It is because of these limitations that tracer gas techniques have been developed.

The Tracer Gas Principle And Its History

The use of a tracer gas is very simple. The method typically entails releasing into the air a known quantity of a gas not normally present in the environment under investigation, and then determining the resultant concentration of that gas in air. Applications detailing the use of a tracer gas in the evaluation of ventilation in dwellings are documented as early as 1940[(3)] and in mining related studies date back to 1957[(4)]. In the early 1970's it was becoming a regular "tool" used by the U.S. Bureau of Mines[(5)]. CANMET's interest in the method started in 1978[(6)] and has continued through to the present.

Over the years, a variety of tracer gases have been used including: coal gas, carbon dioxide, hydrogen, argon-41 and krypton-85 radio-isotopes, nitrous oxide, acetone, oxygen, helium, methane[(7)]; propane[(8)]; Freon 12 and Freon 13B1[(9)]; naturally occurring radioactive gases[(10)]; but the most popular is sulphur hexafluoride, SF_6. This is an ideal gas for tracer studies as it can be measured at low concentrations in air (i.e. down to the part per trillion level). It is inert, non-toxic and non-allergenic, it is odourless, and neither flammable nor explosive. Also for the most part, it will not be absorbed or attach to surfaces and it is thermally stable. Lastly, it is not normally present in the air. This gas is only dangerous to humans if it displaces oxygen. The only negative aspects of this gas are its relatively high molecular weight, so care has to be taken to ensure that it mixes with air, and lastly, and possibly of greatest concern is that it is a greenhouse gas. However, in this environmental regard, it should be noted that the amount of SF_6 used in ventilation studies is trivial when compared to its other industrial uses.

CANMET's Tracer Gas Methods.

Over the last 20 years CANMET has devoted considerable resources to the development of tracer gas techniques and analysis. This has included the development of stable continuous release systems[(1)], a real-time rapid sequential analyzer for SF_6 that has been used in mines[(11)], the use of multiple gases[(9)], sampling and calibration systems[(12)]. Today, CANMET routinely uses tracer gas methods for both the quantitative and qualitative assessment of mine ventilation as highlighted in the following two lists and subsequent examples:

Airflow applications -

- Direct determination of flow, i.e. in vertical shafts, raises[(1)] and other inaccessible routes[(13)],
- In-situ fan and auxiliary duct testing[(1,12)],
- Measurement of gob[(9)], inter-mine and other leakages[(14)] or recirculation[(12)],
- Evaluating the ventilation systems of tunnel boring machines[(8)] and surface processing buildings[(1)], and
- Determining airflow distributions along longwalls[(9)], and airflows and residence times of leaching stopes[(11,13)].

Surrogate applications –

- Mine fire/escape route[(1,12)] and stench gas warning system simulations[(15)],
- Evaluating blast fume and diesel equipment exhaust[(1)] clearance times,
- Propagation of dust and airborne radiation[(10)] throughout mines and mills[(1)], and
- Evaluating ventilation strategies for methane dispersal in metal mines.

MINE CLEARANCE TIMES

The first example of a tracer gas method was designed to determine the clearance time from the mine, through its exhaust system, of diesel fumes generated in production areas.

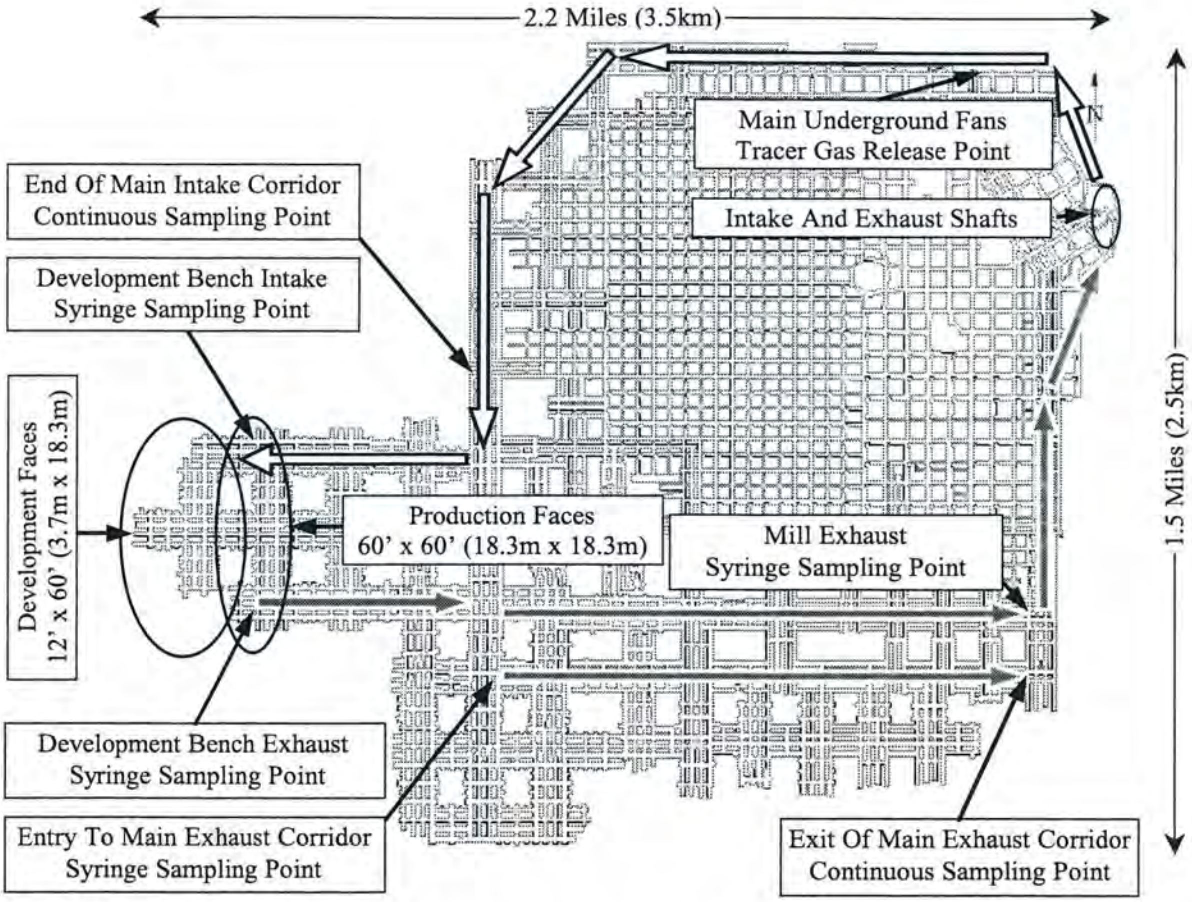

Figure 1. General Mine Schematic and Tracer Gas Test Set-Up

Background

A large underground room and pillar mine wanted to determine if increasing the overall ventilation would dramatically reduce the NO_2 levels experienced in the mine's exhaust ventilation system. The mine, which covered an area of 1.5 x 2.2 miles (2.5 x 3.5 km), extracted rooms of 2400-3600 ft^2 (220-300 m^2) and used multiple ventilation routes, was prone to areas of low air velocity even in the primary arteries. As a consequence of protracted clearance times for diesel fumes from the mine, it was believed that a significant portion of the NO generated by the production machinery had ample time to convert to NO_2. Due to this time dependent relationship, NO_2 levels, as measured with electro-chemical cell sensors in the exhaust route could reach 2.7ppm. This concentration, which is less than the ACGIH 8-hr TLV, was a prescribed "action" level for the mine to reduce the number of diesel units operating and hence production within the mine.

Test Methodology

The general layout of the mine is shown in Figure 1. The air enters the mine via two intake shafts on the shore of a lake in the NE quadrant, and then travels around the northern and western perimeters to the production area at the far west extremity of the mine. Within the intake route, in the north, are the mine's main fans. The air then returns through the southern part of the mine and eastern perimeter to leave the mine via a single exhaust shaft adjacent to the intake shafts. With the exception of the shafts, the rest of the mine is under a lake. The total length of the ventilation system from intake to exhaust collars through the production area, is of the order of 6.9 miles (11km).

To determine the clearance time of fumes through, and from the production area, the ideal location to introduce a surrogate of pollutants would have been just prior to the face. Unfortunately, this was not feasible due to the number

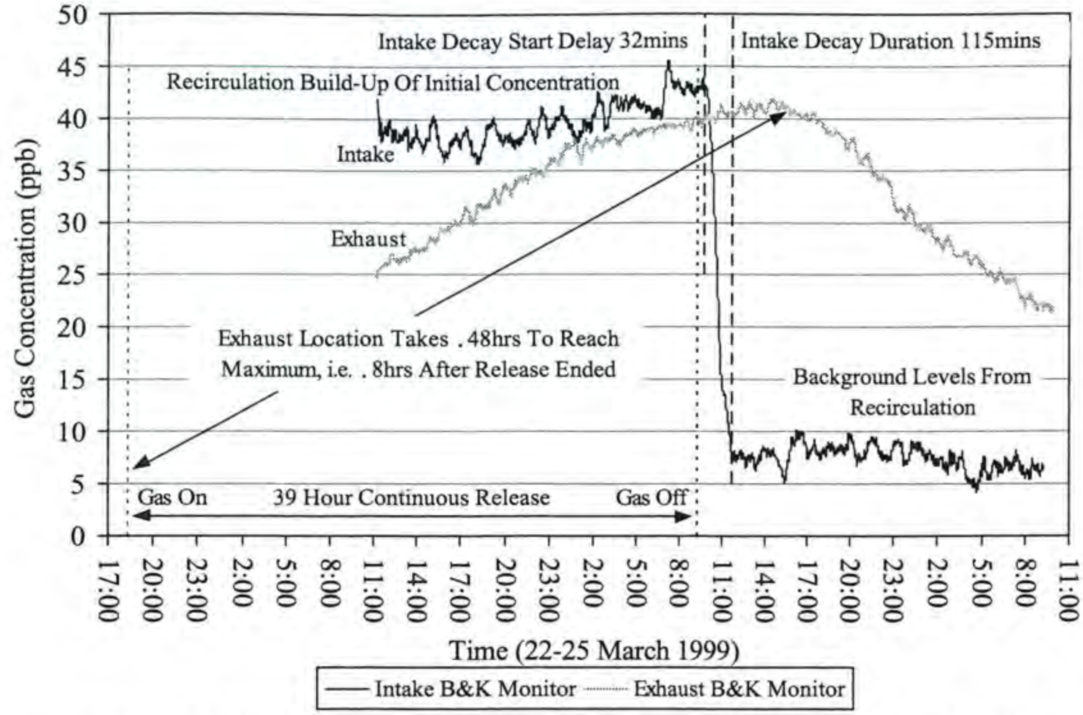

Figure 2. Tracer Gas Profiles At The Continuous Monitoring Locations

and size of the intake routes to the production area, and the inability of the system to ensure complete mixing of the gas with the host air stream before it reached the face. Because of the need for mixing, the only viable gas release location was back at the intake fans (Figure 1).

The release rate of the gas was designed to produce a concentration of 40-50ppb SF_6 once mixed with the intake air to accommodate the continuous gas monitors. Because of the size of the mine and low transit velocities through the system, a release duration of the order of 35-40 hours was chosen to allow the release and the ventilation system to reach equilibrium.

After determining the gas release point, six primary sample locations were selected, at which to monitor either continuously or through extended manual sampling periods the arrival, peak and decay of the gas release. These were:

- Near the end of the main intake corridor,
- The intake entry to the development bench,
- The exhaust of the development bench,
- Entry of the main west-east exhaust corridor,
- The end of the mill exhaust, and
- End of the main west-east exhaust corridor.

Of these locations, the first and last, were each monitored continuously with calibrated Brüel and Kjær (B&K) 1302 multi-gas monitors. These instruments which operate upon an infra-red photo acoustic principle were both fitted with the most appropriate optical filter for SF_6. At the four intermediate locations, air samples were collected in 30cc disposable syringes fitted with air-tight caps, at regular time intervals for post study analysis on an electron capture detector gas chromatograph.

A sampling duration of the order of 50 hours, beginning . 12 hours after the release started was chosen, to check for an equilibrium state and then monitor the flushing of the gas from the system once the release was terminated. During this period, the B&K1302 monitored the gas concentration continuously, and regular manual sampling was performed throughout two 10-hour day shifts plus an additional 2 hours on a third day shift.

Test Results

Figures 2 & 3, present a selection of the gas monitoring and sample analysis results plotted against time. Air transit time results derived from

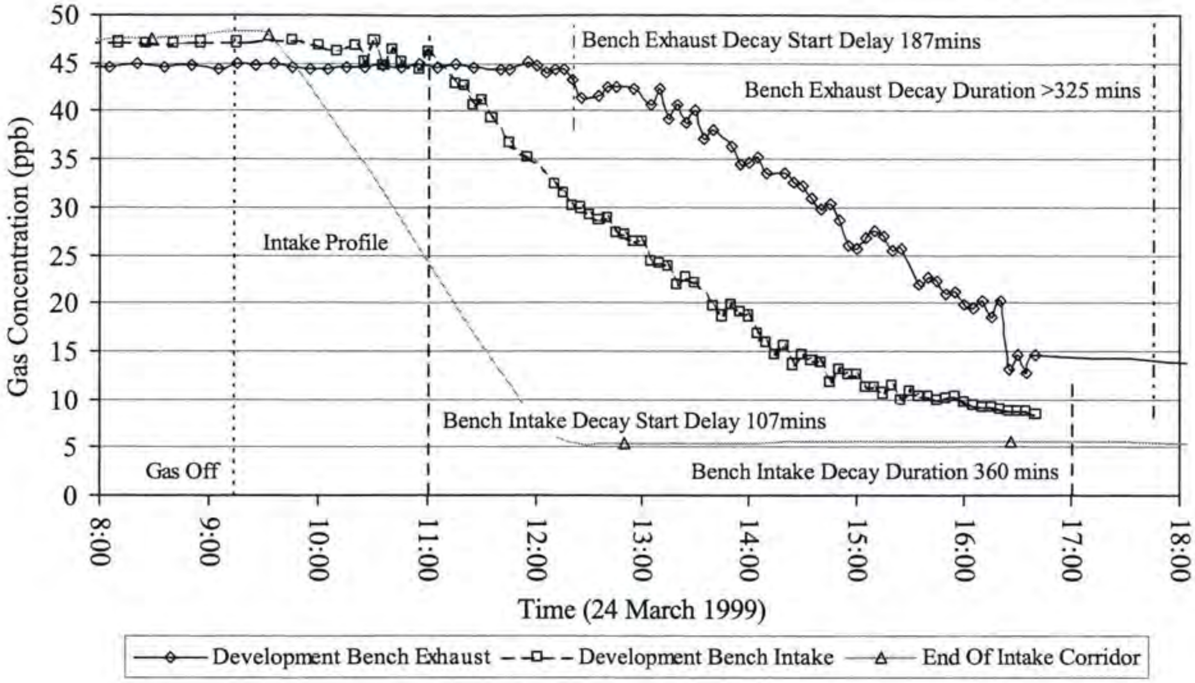

Figure 3. Tracer Gas Profiles At the Syringe Sampling Locations

such profiles are presented in Table I. For the monitoring location near the end of the main intake corridor, Figure 2 shows that the gas release and ventilation system had already reached some level of equilibrium by the time the monitor was turned on. However, the gas concentration continued to gradually increase under the influence of recirculation induced by the underground fan placement. Upon termination of the gas release, the gas concentrations started to decay after . 30 minutes, and the decay was completed after . 150 minutes. After this time, the monitor indicated some background levels of tracer gas due to the recirculation in the system.

For the monitoring location at the end of the main south exhaust corridor, Figure 2 shows that the system failed to reach a comparable steady equilibrium level before the concentrations started to decay. However after . 48 hours it did reach a maximum that was comparable to the initial concentrations at the intake prior to the influence of recirculation. This maximum occurred . 8 hours after the gas release was terminated, and in the remaining 18 hours of monitoring the concentration only decayed to . 40-50% of the maximum.

Together, these two continuous tracer gas profiles start to show that although fresh air travels through the main intake corridor relatively

Table I. Derived And *Estimated* Air Transit Time / Age of Air Results

Destination	Air Transit Time From Gas Release Point (min.)			Age Of Air From Bench Mid-Point (min.)		
	Min.	Max.	Avg.	Min.	Max.	Avg.
End of Main Intake	30	150	90			
Development Bench Intake	110	470	290			
(Avg. Bench Intake/Exhaust)	(150)	(655)	(403)	0	0	0
Development Bench Exhaust	190	840	515	40	185	112
Entry Main Exhaust	290	1740	1015	140	1085	613
End of Main South Exhaust	520	2880	1700	370	2225	1298
NO_2 Monitoring Location	*610*	*3090*	*1850*	*460*	*2435*	*1448*

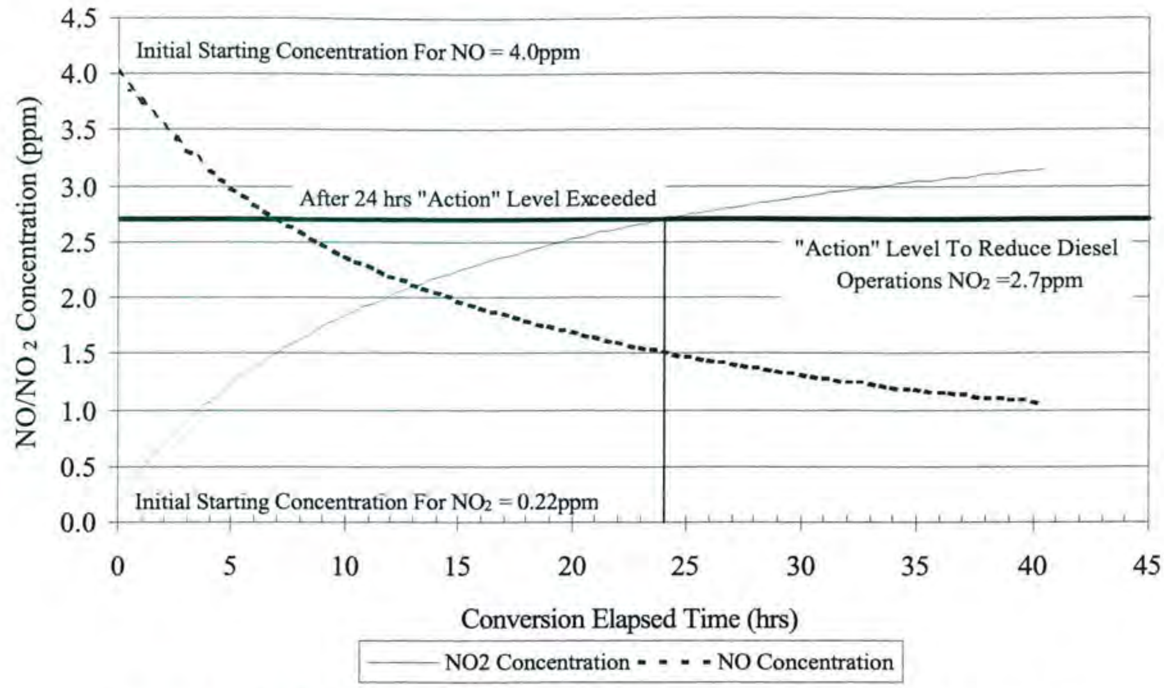

Figure 4. NO/NO2 Time Dependant Conversion From Fine's Equation

quickly, beyond that point the ventilation system's efficiency to remove pollutants drops dramatically resulting in total mine clearance times of days. However to get a better understanding of the NO/NO_2 conversion, transit times have to be related to the source of the pollutants, namely the face rather than mine-wide.

For the sampling location at the intake to the development bench, Figure 3 shows that a comparable steady equilibrium concentration level to that in the main intake was achieved, and that the decay started . 110 minutes after the release was terminated. The decay then lasted . 360 minutes, for a total completion time of 470 minutes, before reaching background levels.

For the sampling location at the exhaust from the development bench, Figure 3 also shows that a comparable steady equilibrium level to the main intake was achieved, and the decay started . 190 minutes after the release was stopped, but failed to reached background levels within the sampling shift of March 24th. Although not shown, this location had reverted to background levels by the start of the next day's sampling shift. On comparing this profile with those of the intake locations, a full decay time estimate from the termination of the release of . 840 minutes can be obtained.

A similar analysis of profiles at other sampling points has produced the data presented in Table I, which includes estimates of the transit times to a NO_2 monitor in the main exhaust back towards the shafts. In this table the average age of the contaminated air has been calculated from the mid-point of the development region. Table I shows that the age of the contaminated air at the continuous monitoring location for NO_2 ranges from 460 minutes (0.32 days) through to 2435 minutes (1.7 days) with an average of 1448 minutes (1.0 days). These values were used to determine the time dependant conversion of NO to NO_2 within the mine's main exhaust.

Figure 4 was derived from Fine's equation[16]:

$$NO_2 = \frac{2.28 \times 10^{-7} \times t \times (NO_{init})^2 \times O_2}{2.28 \times 10^{-7} \times t \times NO_{init} \times O_2 + 1}$$

where, NO_2 : concentration of nitrogen dioxide (ppm) at a given time, t

NO_{init} : initial concentration of nitric oxide (ppm),

O_2 : oxygen concentration (%), and
t : conversion time (sec.).

In Figure 4, it can be seen that an initial harmless concentration of 4.0ppm NO in the ambient air leaving the production area without further dilution, can generate 2.48ppm NO_2 after 1 day. This produces a total NO_2 concentration of 2.70ppm on including a recirculated background of 0.22ppm, which is above the prescribed "action" level for the mine to reduce production. Similarly, it can be shown that if the mine were to increase its airflow by 50%, so reducing its transit times by 33%, the air leaving the production area has still only to be 4.5ppm NO to generate an "action" level after 16 hours.

Based upon the results of this study, it can be stated that although increasing the airflow in the mine will help to combat high NO_2 levels in the mine's main exhaust, the fact remains, that a significant time period must elapse before an "action" level is noted and corrective action taken. For example under the current regime, an "alarm" condition would not be noted until 1 day later, the average residence time. At this point the number of diesel-powered production units would be significantly reduced and then it would take up to 1.7 days, the maximum clearance time for the whole of the mine's exhaust system to be flushed.

To be more pro-active, a better indicator of a potential NO_2 problem would be to monitor NO. For example, at the entry to the mine's main exhaust system with a 3 to 4ppm "action" level. However, as this location still has average and maximum transit times of 6 and 18hrs respectively, monitoring locations even closer to the diesel activity may be considered if such suitable locations exist.

Clearance Time Test Conclusions

This application of tracer gas has highlighted that clearance times out of the mine, from the production areas are on average 24 hours and that complete flushing could take 40 hours. The use of Fine's equation has then shown that these times are of sufficient length to enable a considerable portion of any NO, generated in the production areas, to convert to NO_2 before leaving the mine. Therefore, it is predominantly this time dependant relationship and not diesel activity that results in the "action" alarm level.

To counteract the "action" alarm, which necessitates a reduction in production, the mine could increase the overall flow through the mine, thereby reducing the transit times. However this does not guarantee that alarms will not continue to occur at the end of the exhaust system. Furthermore, the requirement for more air purely to solve this problem, in light of there being no apparent gas problems elsewhere appears to be unjustified. In this instance, the mine would be afforded better control of NO_2 concentrations by monitoring the NO concentrations with an appropriate alarm level before the conversion starts. Although this may also signal a reduction in production, the duration of such reductions will be significantly less than the 40 hours currently needed to flush the complete exhaust system and resume normal operations.

DIESEL EXHAUST SURROGATE

The second application of a tracer gas method was designed to explore the potential exposure of a bucket-shovel operator to exhaust fumes from 2000hp diesel powered 200 tonne haulage trucks during their loading (Figure 5).

Background

As a result of worker complaints of eye irritation, nausea and headaches CANMET was asked on two occasions to evaluate the exposure of shovel operators to diesel exhaust in an open pit metal mine. An investigation, in 1995, focused on the shovel operator's exposure to the primary gaseous diesel exhaust contaminants: CO_2, CO, NO, NO_2 and SO_2; and the exposure of both shovel and truck operators to diesel particulate (soot). This first investigation used electrochemical cells and an infra-red detector to monitor the gas concentrations in real time. The particulate exposures were assessed through an

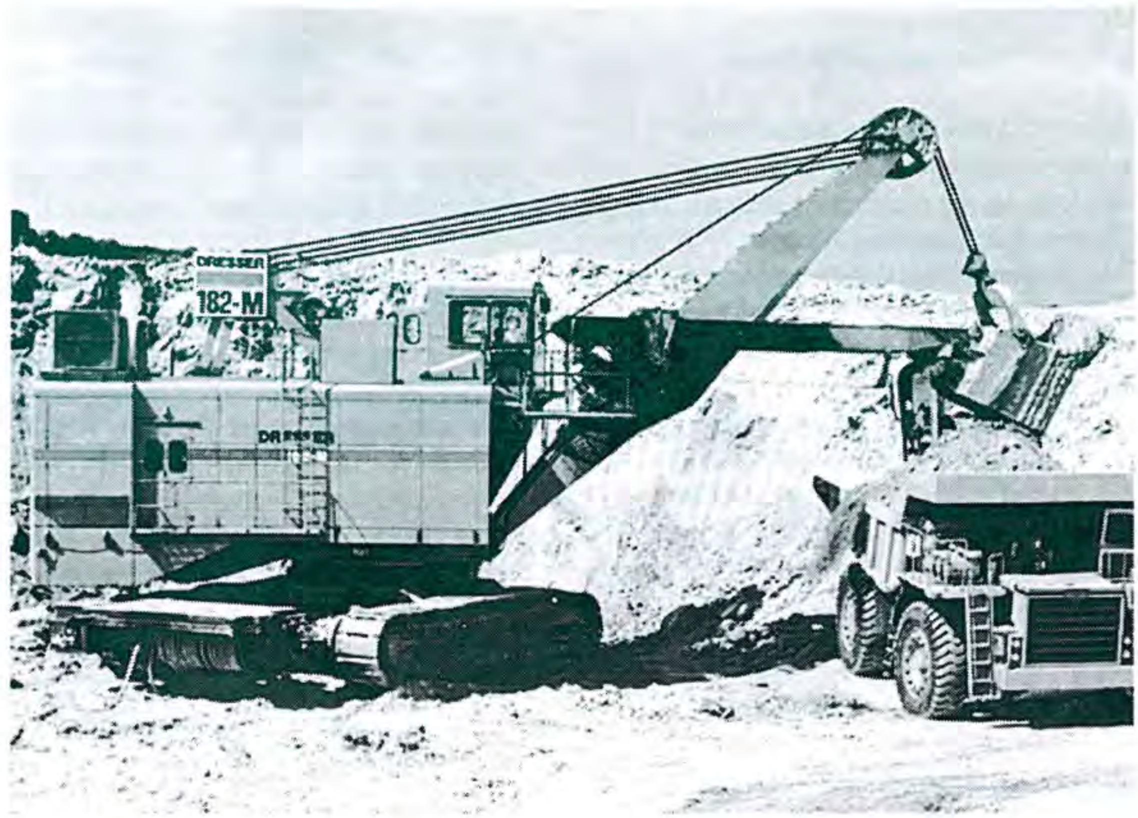

Figure 5. An Example Of A Bucket-Shovel Filling A Haulage Truck

ashing process of dust samples collected on a filter through 10mm nylon cyclones. The results of the first investigation were as follows, in each instance the exposure is also referenced as a percentage of the American Conference of Governmental Industrial Hygienists (ACGIH)[(2)] occupational exposure value unless otherwise stated:

- For CO_2, the average exposure level of the shovel operator during loading was 650ppm (13% of the 5000ppm 8-hr TLV) with only minor variations.
- For CO, the average exposure level was 1.2ppm (5% of the 25ppm TLV) with peaks up to 3.9ppm.
- For NO, the average exposure level was 0.4ppm (<2% of the 25ppm TLV) with peaks up to 1.23ppm.
- For both NO_2 & SO_2, no measurable concentrations were detected.
- For diesel particulate, time weighted average sampling indicated exposures of 0.018mg/m3 (<2% of the Canadian *ad hoc* Diesel Committee's recommended 1.5mg/m^3 limit[(17)], or potentially 36% of the ACGIH notice of intended change (NIC) of 0.05mg/m^3 for total carbon.

Based upon the above results, the study failed to find any of the most commonly monitored components of diesel exhaust to be close to any legislated limits. However, it was found that the relative humidity inside the cab of the shovel was low, at 15%, which could account for some of the eye irritation. Despite these findings, it was noted that some diesel smells could be detected inside the cab of the shovel and that these coincided with loading operations. Consequently a more detailed investigation was performed to investigate the potential of formaldehyde being the cause of the worker's complaints.

Formaldehyde is the primary aldehyde, or partially oxidized hydrocarbon, found in diesel exhaust where it normally occurs in the gas phase. Aldehydes, especially formaldehyde, are noted for their irritancy to the eyes and respiratory tract and are also suspected carcinogens. The ACGIH recommended STEL for formaldehyde is 0.3ppm [(2)].

Table II. Haulage Truck Exhaust Analysis

Engine Classification	Engine Speed	Engine Loading	Exhaust "Aldehyde" Concentration (ppm)	Tracer Gas Concentration (ppb)
"Clean"	750	No-load	151 ± 17	
	1900	Load bank	161 ± 31	
"Dirty"	750	No-load	621 ± 2	$50{,}700 \pm 25\%$
	1900	Load bank	362 ± 27	$28{,}800 \pm 25\%$

Formaldehyde is difficult to measure in the field in raw exhaust (i.e. directly from the tail-pipe of a operational vehicle) and even more so in ambient air when it is further diluted as most analysis methods require a timed sample collected over at least 5 minutes.

Test Methodology

For the purpose of this study an instantaneous method of determining the ambient "aldehyde" concentration was required to evaluate the effects of short-lived pollutant sources. To facilitate this, two methods were employed to assess formaldehyde concentration levels.

Raw exhaust. A calibrated B&K 1302 multi-gas monitor was used to determine directly the "aldehyde" concentration of a vehicle's exhaust from air samples collected in multi-layered gas sampling bags. This infra-red photo acoustic instrument, although fitted with the most appropriate optical filter for formaldehyde, was still subject to interference from other substances, including acetaldehyde which can also be found in diesel exhaust. Therefore the results quoted for "aldehyde" although being predominantly formaldehyde probably include an acetaldehyde component.

Ambient sampling. For the detailed ambient determination, an indirect method with a surrogate, namely SF_6, whose concentration could be determined at any specific instant, was employed. This gas was released at a controlled rate into the exhaust pipe of a truck just after the engine to promote mixing. Air samples were then collected in 30cc syringes on a regular time base inside the cab of a shovel and analyzed post-study with a gas chromatograph to determine their operator's exposure profile.

Air samples were also collected from the raw exhaust and analyzed to determine the discharge concentration. The ratio of the exhaust's SF_6 values and "aldehyde" was then used convert the samples collected inside the cab resulting in the generation of "aldehyde" concentration versus time graphs necessary to assess the operator's exposure.

Test Results – Truck Exhaust

Five trucks' engines were evaluated for their "aldehyde" production using the B&K analyzer. Of the trucks selected, two were reported by the operators to have "dirty" engines due to the smoke they produced at low idle, were common to one manufacturer, and the three other engines that were believed to be "clean" comprised one by the same manufacturer plus two by another manufacturer. The results of this analysis are given in Table II.

From this table it is apparent that under both load conditions, the "dirty" engines produce significantly more "aldehyde" than the clean engines especially at no-load, and that there was little difference between the "clean" engines irrespective of the manufacturer.

The discharge concentration of SF_6 in one of the "dirty" engine's exhaust was determined at four engine speeds and found to be reasonably linear. The high and low speed SF_6 concentrations given in Table II have been given a high tolerance beyond the normal ±5% accuracy of SF_6 analysis, due to three factors:

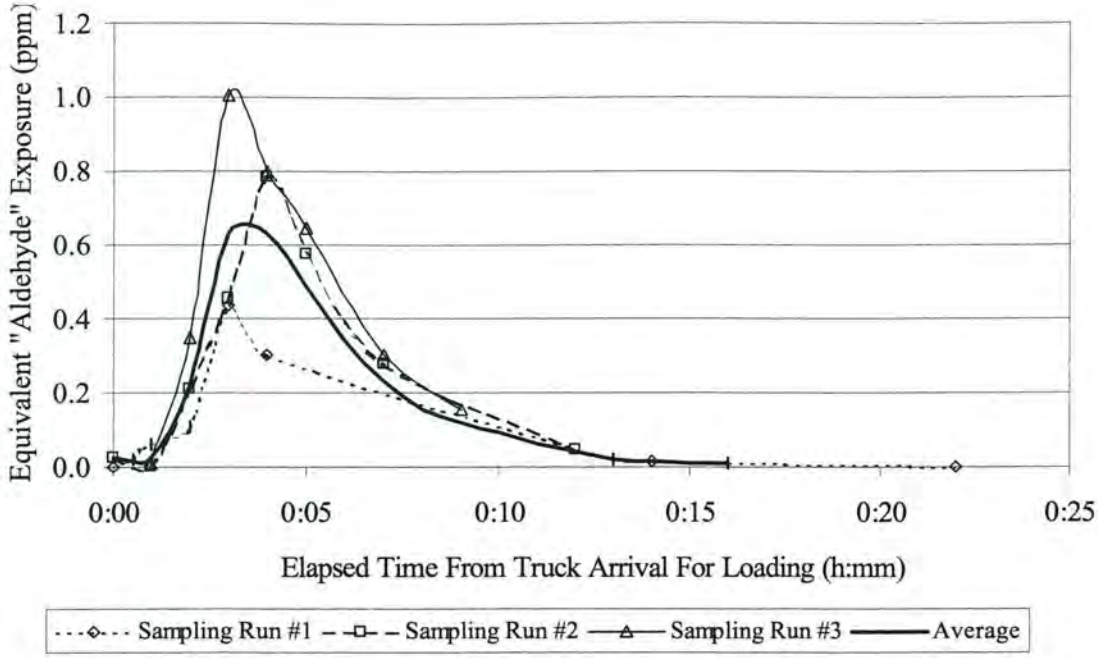

Figure 6. Shovel Operator Exposure Profiles From A Single "Dirty" Haulage Truck

A. Fluctuations in the exhaust flow due to the engine's cycle,
B. Fluctuations in the tracer gas flow due to pressure changes in the exhaust again due to the engine's cycle, and
C. Dilution errors as a consequence of diluting the exhaust concentration down to a calibrated range of the gas chromatograph.

Test Results – Shovel Cab Environment

From Table II it can be seen that a stationary idling truck has the greatest potential to produce a local environment containing significant levels of "aldehyde". Hence, three tests for contamination inside the shovel operator's cabin were performed while the stationary "dirty" truck was being loaded. Each test comprised collecting air samples in 30cc disposable syringes fitted with an air-tight cap at time intervals over a 9 to 22 minute period. Post study analysis of these air samples with a electron capture gas chromatograph optimized for SF_6, upon conversion to equivalent "aldehyde" concentrations produced the results displayed in Figure 6.

Firstly, all three sample runs show distinct "aldehyde" contamination peaks, potentially as high as 1.01ppm, inside the shovel operator's cab, but their magnitude varies by as much as a factor of 2. From these, the 15-minute average "aldehyde" exposure can be estimated for each subsequent run as 0.14, 0.23 and 0.29ppm. These values show that the "dirty" truck has the potential to expose the shovel operator to concentrations close to the 15-minute STEL.

Secondly, this graph shows that the contamination from a single truck can persist for 10-15 minutes. This is important to note as the loading cycle is typically one truck every 4-5 minutes, so there is the possibility that the shovel operator could be experiencing the cumulative effects of the exhausts from more than one vehicle.

Result Extrapolation - Multiple Vehicles

The combined effect of multiple truck exhausts on the shovel operator's exposure as shown in Figure 7, was obtained as follows:

1. The average contamination peak profile (Figure 6) was used for the "dirty" truck.
2. For "clean" trucks, the average peak profile was reduced by 76%. This correction was obtained from the relative exhaust concentrations of "aldehydes" of the "clean" and

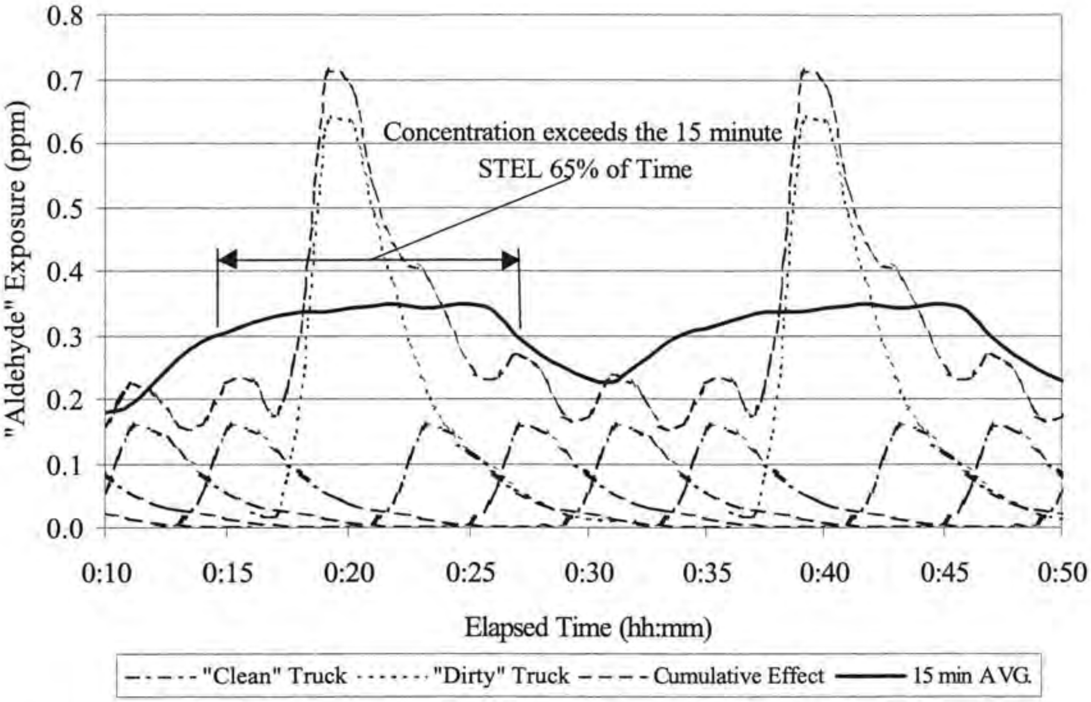

Figure 7. Shovel Operator Exposure Profiles From Multiple Trucks, Assuming A 4 Minute Loading Cycle And Every 5th Haulage Truck Is "Dirty"

"dirty" trucks at idle conditions (Table II).

3. From process observation, the highest truck loading frequency was every 4 minutes.
4. Typically five trucks serviced one shovel.

Figure 7 now provides an indication of the cumulative effect of five trucks and especially the influence of the one "dirty" truck. Furthermore, after producing a continuous profile of 15-minute averages, it can be seen that the "aldehyde" concentration could exceed the 0.3ppm for 65% of the time. A similar analysis assuming five "clean" trucks would have produced a continuous 15-minute average profile just below 0.2ppm.

Surrogate Test Conclusions and Post Mortem

Again a simple application of a tracer gas method has provided a valuable insight into a potential concern in an open pit mining operation that might not otherwise be obtained. Using SF_6 as a surrogate and extrapolating the cumulative effects of multiple vehicles has indicated the possibility of prolonged exposure to "aldehydes" that may be in excess of the 0.3ppm STEL for formaldehyde. Furthermore, it can be stated that the major contributor to this excursion is one "dirty" truck engine.

As a consequence of this investigation, the engine manufacturer in conjunction with the mining company, have modified the exhaust system of the "dirty" trucks.

CONCLUSION

This paper, through two selected examples, has shown how tracer gas methods can be used to provide mining engineering staff with an insight to a specific problem.

In the first example, SF_6 was used to determine air transit times in an underground mine with large, long and numerous air routes. Here, it was shown that increasing the airflow, at significant cost, might not stop the time dependant conversion of NO to undesirable NO_2 concentrations in the mine's main exhaust. In this instance, a better form of control might be NO monitoring in advance of the gas's conversion.

In the second example, SF_6 was used as a surrogate of a pollutant that might not otherwise have been measurable, namely formaldehyde. Here it was shown that in an open pit operation, a single haulage truck with a "dirty" engine even though only present for 4 minutes had the potential to contaminate a shovel operator's environment.

These examples continue to show the value of tracer gas techniques in mine environmental studies, such as offered by CANMET, as they can invariably provide definitive information that may not otherwise be obtainable.

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Addressing the Human Aspects of Mining Safety: From Behaviors to Attitudes to Culture Change

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This paper combines the technology of applied behavior analysis with theories of motivation, learning, and social influence to provide a conceptual framework and evaluation system for implementing a cost-effective, long-term, employee-driven process for directing and motivating the occurrence of injury-prevention behaviors in mining operations.

Within the mining industry, tremendous progress has been made to improve on-the-job safety, particularly in ergonomic intervention (Langton, 1995). The psychological and behavioral aspects of mining safety represent the next necessary step in keeping miners safe. In this regard, our NIOSH-supported research has focused on the development of an intervention process to reduce at-risk behavior in underground and above-ground mines. This includes the development of: a) evaluation procedures to measure the progress of a behavior improvement process; b) survey techniques to assess people's readiness to actively care for the safety of others from an interdependent perspective; and c) a metric to track property damage as a predictor of unintentional injury.

BACKGROUND AND SIGNIFICANCE

Injury is the principle cause of lost-person years of productive life in the U.S., accounting for more years of lost potential than cancer and cardiovascular disease combined (Waller, 1987). More specifically, injuries kill more than 142,000 Americans and require an estimated 62.5 million dollars in medical attention each year (U.S. of Labor Statistics, 1997). Annually, more than 80,000 Americans are permanently disabled by work-related injuries. In fact, injuries are the leading cause of death among individuals aged 44 and less in the U.S. (Baker, Conroy, & Johnston, 1992). Clearly, injuries occurring on the job due to unsafe (or at-risk) work behaviors remain a significant nationwide problem (Baker et al., 1992).

Social and Economic Ramifications

In addition to the traumatic personal consequences experienced by employees and their friends and families due to unexpected industrial injuries and deaths, there are also critical social and economic consequences to consider. Although the severe pain and suffering caused by these misfortunes cannot be quantified, the social and economic costs can be estimated. These costs include lost wages, medical expenses, insurance claims, production delays, lost-time of coworkers, equipment damage, fire losses, and indirect costs (National Safety Council, 1988). The overall cost of work-related injuries incurred back in 1989 has been estimated at $48 billion. This figure is an increase from the 1987 estimate of 42.4 billion dollars and is dramatically larger than the 1985 estimate of $34.6 billion (National Safety Council, 1988). Although all of these estimates are enormous, the numbers also indicate the cost of industrial injuries is increasing at an alarming rate. And, it is likely these estimates underestimate the true impact of industrial injuries because of problems with current surveillance techniques and the fact that many injuries are not reported (Baker et al., 1992; The National Committee for Injury Prevention and Control, 1989; U.S. Bureau of Labor Statistics, 1997).

Significance to Mining

In 1998, 148 workers were killed in both surface and underground mines (U.S. Bureau of Labor Statistics, 1998). This accounts for more than 6% of all work-related fatalities in 1998, yet mining employees represent only .5% of the total workforce (U.S. Bureau of Labor Statistics, 1998). Also, there were more than 30,100 injury reports by miners in 1998 (U.S. Bureau of Labor Statistics, 1998). It is likely these injury numbers grossly underestimate the actual number of miners hurt on the job because so many workplace injuries are unreported or hidden (Miller, 1997).

Due to the frequency and severity of injuries, the U.S. Department of Health and Human Services has identified unintentional injury prevention as a priority for attaining the goals outlined in *Healthy People 2010: National Health Promotion and Disease Prevention Objectives* (2000). *Healthy People 2010* has 28 focus areas and 437 specific objectives. Focus Area 20 (Occupational Safety and Health) has two objectives directly related to mining: 20-1b. -- a reduction in fatalities in mining faculties, and 20-2f. -- a reduction in work-related injuries in mining facilities.

Much has been accomplished to make engineering improvements in mining. For example, video camera back-up monitors and sophisticated Doppler radar alarms have been used to try and reduce injuries involving haulage trucks (Boldt & Baker, 1995). Although such technologies are improving with laboratory and field experiments, mining experts have pointed out that the technological improvements must be supplemented with a focus on the human aspects of safety, including the application of employee surveys and behavior-based incentive programs (Peters, 1995). With this in mind, "the U.S. Bureau of Mines identified fundamental psychological principles that could be employed to enhance the ability of miners to recognize and respond to threats in their dangerous work environment" (Kowalski, Fotta, & Barrett, 1995, p. 95). This defines the direct relevance of behavior-based safety (BBS) to mining.

It's impossible to protect the miner completely with environmental manipulations. Therefore, mining experts have pointed out the need to complement technological advancements with a focus on the human dimensions of safety, including the application of employee surveys, training series, incentive programs, and feedback presentations (Langton, 1995; Peters, 1995; Peters, Bockosh, & Fotta, 1997). In fact, the U.S. Bureau of Mines referred to the need to apply fundamental psychological principles in the enhancement of miners' ability to recognize and react appropriately to threats in their risky work environment (Kowalski, Folta, & Barrett, 1995).

The leading-edge technology in applying psychological principles to keeping workers safe is behavior-based safety. As Boling (1995) points out, "Today the progressive, productive and safe companies around the world have one common denominator, an innovative safety program that is behavior-based" (p. 2). The components of behavior-based technology go beyond behavioral observation and feedback methods and statistical techniques for charting safety performance improvement. They include ways to enhance interpersonal communication, individual responsibility, teamwork, morale, involvement, peer support, follow-up intervention, and other factors needed for continuous safety improvement (Geller, 1996, 1997b, 1998a, in press).

APPLICATIONS OF BEHAVIORAL SCIENCE TO IMPROVE MINE SAFETY

The term "behavior-based safety" (BBS) has become quite popular among safety professionals, consultants, and members of safety steering committees. It is commonly used to reflect a proactive upstream approach to safety by focusing attention on at-risk behaviors that can lead to an injury and on safe behaviors that can contribute to injury prevention. Beyond this general definition, however, there seems to be substantial misperception, misunderstanding, and misapplication. Here we offer a brief review of BBS principles, procedures, and benefits,.

Principle 1: Focus Intervention on Observable Behavior

Whatever intervention strategy is used to improve a human dimension of safety, the process should focus on behavior. Whether using training, feedback, injury analysis, coaching, or incentives to benefit safety, target behavior. Why? First you can be objective and impersonal about behavior. You can talk about behavior independently from people's opinions, attitudes, and feelings. Behavior varies according to factors in the external world, including equipment design, management systems, the behaviors shown by others, and various social dynamics. An open discussion about the environmental and interpersonal determinants of safe versus at-risk behavior can lead to practical modifications of the work culture to encourage safe behavior and discourage at-risk behavior.

Behavior-based intervention ***acts people into thinking differently,*** whereas person-based intervention ***thinks people into acting differently***. The person-based approach is used successfully by many psychiatrists and clinical psychologists in professional therapy sessions, but it is not cost-effective in a group or organizational setting. To be effective, person-focused intervention requires extensive one-on-one interaction between a client and a specially-trained intervention specialist. Even if time and facilities were available for an intervention to focus on internal and nonobservable attitudes and person states, few safety professionals or consultants have the education, training, and experience to implement such an approach. Internal person factors can be improved indirectly, however, by directly focusing on behaviors in certain ways.

The key is to focus on behavior and you'll be on the right track, whatever the intervention approach it's **behavior-based** commitment, **behavior-based** goal-setting, **behavior-based** feedback, **behavior-based** training, **behavior-based** recognition (Geller, 1997a), **behavior-based** incentives/rewards, and so on.

Principle 2: Look for External Factors to Improve Performance

Internal person dimensions like attitudes, perceptions, and cognitions are difficult to define objectively and change directly. In fact, most of us don't have the education, training, experience, nor time to deal with people's attitudes or person states directly. Instead, look for external factors that influence behavior independent of individual feelings, preferences, and perceptions. When you empower people to analyze behavior from a systems perspective and implement interventions to improve behavior, you will indirectly improve their attitude, commitment, and internal motivation.

Careful observation and analysis of ongoing work practices can pinpoint many potential causes of safe and at-risk behaviors. Those causes external to people -- including reward and penalty contingencies, policies, or management mandates -- can often be altered for the improvement of both behavior and attitude. In contrast, internal person factors are difficult to identify, and if defined, they are even more difficult to change directly. So with BBS the focus is placed on external factors -- environmental conditions and behaviors -- which can be changed upstream from a potential injury.

Principle 3:Focus on Positive Consequences to Motivate Safety Improvement

The ABC three-term contingency is a basic tool of BBS. "A" stands for "activator," or the antecedent events that precede behavior (B) and direct it. "C" refers to "consequence," or the environmental stimuli or events that follow behavior and determine it's future occurrence. We do what we do to gain a positive consequence or to escape or avoid a negative consequence. And, we stop doing what we're doing when our behavior results in an immediate negative consequence.

The most powerful motivating consequences are "soon" and "certain." That's why most at-risk behavior occurs. Compared to safe behavior, at-risk behavior provides the performer with such soon and certain consequences as comfort, convenience, and faster job completion.

As this third BBS principle specifies, using positive over negative consequences is critically important. It's relevant to "attitude," and many other internal feelings of people. Think about it. How does a reward, personal recognition, or a group celebration make you feel compared to a reprimand or criticism? Both types of consequences can have similar direct effects on behavior. There is usually a big difference, however, in the accompanying attitude or feeling state.

As detailed elsewhere (Geller, 1996, 1997a, 1998b), when positive recognition is delivered correctly, it does more than increase the frequency of the behavior it follows. Through stimulus and response generalization it also increases the probability other safe behaviors will occur (Ludwig & Geller, 1997, 2000). The popular belief that we learn more from our mistakes than our achievements is actually wrong. Our errors tell us what to stop doing, but our successes show us what to keep doing. So recognizing people's safe behavior will facilitate more learning and positive motivation than will criticizing people's at-risk behavior. Only with positive consequences can you improve both behavior and attitude at the same time.

Don't Rely on Common Sense

More explanations for these and additional principles of BBS are presented elsewhere (Geller, 1996, 1997b, 1998b). These three principles are most pertinent with regard to developing interventions to improve the psychological aspects of mine safety. Do they seem like plain old common sense to you? If so, congratulations. But please note that others will not necessary feel the same. Common sense or intuition is often incorrect. What sounds good to one person will not necessarily sound right to another. Consider, for example, the following strategies managers or safety leaders have implemented intuitively in an attempt to deal with the human dynamics of safety.

- Punish a person who returns to work after a lost-time injury.
- Implement a safety incentive program whereby everyone in an organization gets a prize if no one reports an injury.
- Set up a "safe employee of the month" program in which one individual in a large facility is publicly recognized for having the "Best Safety Attitude."
- Establish an observation system whereby employees must observe one unsafe condition or behavior each day and "stop" it.
- Invite a motivational speaker to address all employees with themes like "Try Harder," "Change Your Attitude About Safety," "Self-Affirmation is the Key to Motivation," or "Safety Awareness and a Positive Attitude are Key to Behavior Change."

> Post signs with slogans like "Think Safety," "Safety is a Condition of Employment," "Zero Accidents is Our Goal," "Safety is a Priority," or "All Injuries are Preventable."

Do any of these psychological tactics sound familiar? All of these approaches are ineffective in the long run, and run counter to the three behavior-based principles described above. Some of these techniques can actually do more harm than good to the human aspects of mining safety. Yet, I'm sure you've seen, perhaps even experienced, some of these motivation or persuasion techniques. Why? Because they seemed like good common sense to someone.

The development and implementation of an improvement intervention requires guidance from empirical investigation, not common sense. This is true for repairing a bridge, constructing a building, designing an underground mine, or administrating a safety incentive program. "Contrary to popular belief, there is not too little common sense in business, there is too much" (Daniels, 1999, p. 10).

As reflected in the three principles described above, BBS is based on 50 years of rigorous research. And with additional research, the methods and tools of BBS will continuously improve. Let's consider a general behavioral safety method which has been used by several researchers to evaluate the effectiveness of specific intervention techniques to prevent injury (e.g., Fellner & Sulzer-Azaroff, 1984; Geller, 1988; Komaki, Heinzmann, & Lawson, 1980; Ludwig & Geller, 1997; Sulzer-Azaroff & DeSantamaria, 1980). It has also been used by numerous organizations to improve their safety performance (cf. Geller, 1996, 1998b, in press).

The Do It Process

The DO IT process puts people in control of improving behaviors directly related to injury prevention. It's a basic tool for addressing the behavioral aspects of a safety problem. It provides an objective way to analyze why certain safety-related behaviors occur or don't occur (Geller, 1999b), and to evaluate the effect of interventions implemented to decrease at-risk behavior or increase safe behavior. If an intervention does not influence the desired impact, it is either refined or replaced with a completely different behavior-change technique.

"D" for Define. The DO IT process begins by defining specific behaviors to influence – the targets of the continuous improvement process. They are safe behaviors that need to occur more often, or undesirable at-risk behaviors that need to occur less often. Usually the focus can be on certain safe behaviors that need to be substituted for particular at-risk behaviors. The definition of a safe behavior could be as basic as wearing particular personal protective equipment (PPE) or "walking within pedestrian walkways." Alternatively, the safe target could be a process that requires a designated series of safe behaviors, as when parking a truck for unloading, locking out energy sources, or lifting a package.

Arriving at a precise definition of a DO IT target is facilitated with the development of an observation checklist to determine how safely a certain target behavior or process is being performed. Just developing the precise behavioral definitions for such a checklist is a valuable learning experience. And, when workers get involved in developing a behavioral checklist, they own a training process that can improve the human factors of occupational safety on both the outside (behaviors) and the inside (feelings and attitudes) .

"O" for Observe. When miners observe each other for certain safe or at-risk behaviors, they learn that everyone performs at-risk behavior, often without even realizing it. This observation stage is not fault-finding, but is a fact-finding process to facilitate the discovery of behaviors and conditions that need to be altered to prevent injuries. In addition, safe behaviors and conditions are identified for interpersonal support or recognition.

Behavioral observations are only made with the permission of the worker being observed. Although unannounced observations might give a more realistic picture of the frequency of at-risk behavior, such audits would reduce interpersonal trust and give the impression of a negative "gotcha" program. Furthermore, from a performance-improvement perspective, interpersonal observations without permission cannot raise safety "mindfulness" (Geller, 1999a; Langer, 1989). It's likely the mindfulness developed and increased from up-front and voluntary use of a behavioral checklist is critical for improving safety-related behavior and preventing unintentional injury.

It's easy to fall into a mindless job routine, and be unprepared to handle unanticipated events in a safe and timely manner. Also, some mindless activity can put miners at significant risk for personal injury. A behavior-based observation and feedback process provides the mechanism for increasing peoples mindfulness on the job.

The same observation procedure is not suitable for all situations. In fact, the customization and refinement of an observation process for a particular setting should never stop. Often it's best to begin with a limited number of behaviors and a relatively simple checklist. This reduces the possibility of some people feeling overwhelmed and frustrated. Starting small also facilitates the broadest range of voluntary participation. Subsequently, the process is successfully expanded in the number of behaviors and work areas covered.

"I" for Intervene. This is the heart of the DO IT process. Now interventions are designed and implemented in an attempt to increase safe behavior and/or decrease at-risk behavior. As reflected earlier in Principle 2, intervention refers to changing external conditions of the work environment in order to make safe behavior more likely and at-risk behavior less likely. When developing interventions, use Principle 3 as your guide. That is, the most motivating consequences are soon, certain, and sizable, and positive consequences are preferable to negative consequences.

The process of observing and recording the occurrences of safe and at-risk behavior on a checklist provides an opportunity to give individuals and groups constructive behavioral feedback. When the results of a behavioral observation are shown to individuals or groups, they receive the kind of information that enables practice to improve performance. Considerable research has shown that providing workers with feedback regarding their safe and at-risk behaviors is a very cost-effective intervention approach for improving safety performance (e.g., Geller, 1996; Krause et al., 1996; Reber, Wallin, & Chhoker, 1990; Sulzer-Azaroff & DeSantamaria, 1980; McAfee & Winn, 1989; Petersen, 1989).

Besides behavior-based feedback, researchers have found a number of other intervention strategies to be effective at improving safety related work practices. These include worker-designed safety slogans, near miss analysis and corrective action, individual and group goal setting, safe behavior promise cards, actively caring thank-you cards, behavioral safety coaching, as well as incentive/reward programs that target specific behaviors of individuals or groups. These intervention strategies are described elsewhere (Geller, 1996, 1998b, in press), some having been applied in mining settings (Fox, Hopkins, & Anger, 1987; Rhoton, 1980).

"T" for Test. This fourth phase of DO IT furnishes work teams with information they need in order to refine or replace a behavior-focused intervention, thereby improving the process. If observations indicate the target behaviors have not improved, the work team analyzes and discusses the situation, and alters the intervention or selects another intervention approach. On the other hand, if the behavior-focused goal is reached, the team members turn their attention to another set of behaviors. New critical behaviors might be added to the observation checklist, thus expanding the domain of behavioral observation and feedback. A new intervention procedure might be added that targets only the new behaviors.

Each time a work team tests the impact of an intervention approach, they learn more about how to improve safety performance. In this way they have become behavioral scientists, using the DO IT process to: a) analyze and diagnose a human factors problem, b) monitor the impact of a behavior-focused intervention, and c) refine interventions for continuous improvement. The results from this testing process provide motivating consequences to support this learning and keep the team members involved.

APPLICATIONS OF BBS IN ABOVE-GROUND ROCK QUARRIES

Our recent attempts to apply BBS and the DO IT process at three above-ground rock quarries in southwest Virginia taught us valuable lessons and suggested directions for future research. We have developed a behavior-based training program for mining environments, and found miners very receptive to the basic principles. However, training alone was not sufficient to increase safe work practices. Nor was our training sufficient to motivate participation in a behavioral observation and feedback process. But, ongoing work practices did become more safe as the result of research assistants making periodic observations of miners' safe versus at-risk behaviors.

Procedure

At each mining site, the employees were given four hours of BBS training. The training included an educational component designed to teach employees the basic principles of BBS, as well as a training component to give the miners an opportunity to practice a) observing a coworker's safety-related behavior, b) completing an observation checklist, and c) providing feedback regarding a coworker's safe and at-risk behaviors.

After the training sessions, the employees participated in focus groups to help develop an observation checklist tailored for each site. This important component helps to develop a sense of empowerment. By involving miners in the actual development of their safety process, they feel a sense of ownership, and are therefore, more likely to work for the survival of the process.

Safety leaders at each site designated a convenient place to display the behavioral checklists, and they provided a drop-box nearby for the employees to deposit completed checklists. It is important to note that the checklists are anonymous; they cannot be traced to any individual worker. This is to ensure honesty and to reassure the miners that this is not a tattle-tale program.

The completed checklists were collected by research assistants and analyzed. The behavioral data were provided to each site periodically, updating the safety leaders and the miners on their safety performance.

Results

The data from the mining sites indicated that the BBS was effective at increasing safe work practices. However, training alone was not sufficient to increase safe work behaviors. For example, the percentage of employees wearing safety goggles was nearly identical during Baseline (M=52% usage) and after Training (M=51% usage) at Site A. However, the percentage of employees wearing safety goggles increased substantially following the implementation of the observation and feedback process (M=68% usage).

Consistent with past research, the behavior-based observation and feedback process in the current field study led to clear improvements in subsequent safety performance. Based on 3,898 observations over an eight-month period, overall percent safe scores increased from 80.6% to

90.1%. The current results support the use of a BBS process to increase safe work practices.

Participation Difficulties

Limiting the impact of behavioral safety at the mining sites was the lack of employee participation. The employees never took ownership of the process. Some of the managers were seemingly against the BBS process and stressed production ahead of safety. Also, employees frequently completed the observation process incorrectly (e.g., by marking down only at-risk behaviors, tattling on fellow employees, and sometimes handing in blank observation cards).

To increase participation, a participant-focused incentive/reward program was implemented. During the incentive program, the number of observation cards completed increased by 600% and the number of different employees completing observations cards increased by 500%. Although this program did increase participation, this increase was short lived.

A focus group was held at one site to explore reasons for the lack of sustained involvement. When asked about the value of the rewards, the inexpensive items -- a tool kit and food coupons -- were valued by the employees, but most employees were under the impression that a select few employees were "stuffing" the box. In other words, some employees thought other coworkers were filling out bogus checklists to increase their chances of winning the reward.

Another problem discussed during the focus group was the perception of "just more paper work." Over half of the employees indicated they were observing their coworkers and providing behavioral feedback, but they just didn't have the time or the desire to complete the checklist. It seems they didn't believe the BBS process was worth their time.

Other employees were still concerned about getting in trouble or getting their coworkers in trouble for reporting at-risk behaviors. The employees were reassured on several occasions that this was not a tattle-tail process, however, this concern remained an issue. Our prior BBS research indicated that interpersonal trust was a critical factor needed for this process to work (DePasquale & Geller, 1999), and these focus-group discussions verified that finding.

Procedural Limitations

One factor contributing to a ceiling effect observed across all sites was our inability to observe all possible work behaviors. Since our research assistants were dependent on the quarry foreman to chaperon them around the sites, we were only allowed to make observations when he was present. If a piece of equipment broke down, or the miners had a problem that needed his assistance, we were unable to make behavioral observations that day. As such, the data we collected only reflects occasions when the mining operations were running smoothly.

Future Directions

After making several visits to the mines it became apparent that miners in above-ground rock quarries are essentially solitary workers. Many mining tasks are accomplished by a lone worker. The only time employees would get an opportunity to observe and give peer feedback to a coworker was when some piece of equipment broke down. There's typically no supervisor or coworker around to hold a miner accountable for performing the job safely. So the challenge for safety professionals and corporate leaders in mining environments is to build the kind of work culture that enables or facilitates responsibility or personal accountability for safety (Geller, 1998a). Thus, a broad-based and long-term set of guidelines are needed for designing practical intervention strategies to reduce the risk of unintentional injury in occupations where employees work alone and cannot be held accountable for their actions by another individual. Geller and Clarke (1999) refer to such a process as safety self-management. Our research suggests that this would likely be a more appropriate intervention approach at the four mining sites we studied.

Self-Management Interventions

Self-management (Mahoney, 1971, 1972) is a behavioral improvement process whereby individuals change their own behavior in a goal-directed fashion by: a) manipulating behavioral antecedents, b) observing and recording specific target behaviors, and c) self-administrating rewards for personal achievements (Geller & Clarke, 1999; Kazdin, 1993; Watson & Tharp, 1997).

The practical benefits of a self-management process has been demonstrated in numerous clinical settings, including the reduction of alcohol consumption (Garvin, Alcorn, Faulkner, & Kim, 1992; Sitharthan, Kavanagh, & Sayer, 1996; Sobell & Sobell, 1995), the control of weight (Baker & Kirschenbaum, 1993), and the cessation of smoking (Curry, 1993; Shiffman, 1984). However, the potential benefits of using self-management techniques to improve safety-related occupational behaviors have not been systematically evaluated.

Research indicates that five self-management procedures can facilitate behavioral improvement; including: a) activator management (Heins, Lloyd, & Hallahan, 1986), b) social support (Stuart, 1967), c) goal setting (Latham & Yukl, 1975; Locke & Latham, 1990), d) self-observation and self-recording (Ericsson, Krampe, & Tesch-Romer, 1993; Lau, Bradley, & Parr, 1993), and e) self-rewards (Sohn & Lanal, 1982). See Geller (1998a) and Geller and Clarke (1999) for details on how these safety self-management techniques can be applied to improve safety performance.

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DEVELOPMENT AND EVALUATION OF A TRAINING EXERCISE FOR CONSTRUCTION, MAINTENANCE AND REPAIR WORK ACTIVITIES

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ABSTRACT

Recent studies have shown that miners performing construction, maintenance, and repair (CMR) work activities in the conduct of their jobs incur from 39 to 65 percent of all reported injuries in the mining industry. The number is particularly high at surface aggregate operations; however, the problem exists at all mining locations and commodities. To address this issue, an interactive, (3-D) slides training exercise, Hazard Recognition Training Program for Construction, Maintenance and Repair Activities, was developed. The purpose of the exercise is to teach workers to recognize CMR hazards in the workplace and to deal with them using accepted safe work procedures. It was field tested using a total of 340 persons from surface mining operations in six states. The subjects were tested before and after the training intervention to determine if objectives of the instruction were achieved. Results indicated that 71 percent of the participants showed improvement in their test scores. Following the posttest, subjects responded to a seven question Likert scale. These questions related to the validity of the exercise and the utility of the training program. More than 93 percent of the miners reported that they "learned something new from the training" and over 94% said they "would use these practices to work more safely".

INTRODUCTION

A review of 1995 injury data by a large aggregate mining company in the United States showed that a high percentage of incidents within their operation occurred to miners who were performing CMR work activities in the conduct of their jobs [1]. It was thought that similar findings may also exist throughout other segments of the mining industry. To investigate this issue, a group of mine safety practitioners from The National Institute for Occupational Safety and Health (NIOSH), the Mine Safety and Health Administration (MSHA), and the mining industry, including the large aggregate mining company, met at NIOSH's Pittsburgh Research Laboratory. At the meeting, the extent to which injuries may be attributed to construction, maintenance, and repair work activities, and possible strategies for reducing these numbers were discussed.

In order to proceed, however, it was deemed necessary that everyone agree on a single definition of "construction, maintenance and repair". After reaching a consensus, the definition (see Appendix A) was applied to narratives of 604 incidents that occurred over a period of three years at the large aggregate producer's mining operations. It was concluded that 65% of these incidents resulted from employees performing CMR work activities. A follow up inter-rater reliability assessment showed the level of agreement among the four raters (two representatives from NIOSH, one from MSHA and one from industry) to be 94%.

Other evidence of the extent of CMR injuries among miners was documented in a NIOSH-funded investigation by Lehman and Layne [2]. In this study, the consensus definition was applied to narratives of 21,024 injuries for all commodities (both surface and underground locations) throughout the U.S. mining industry during the same three-year period. It was determined that 39% of these injuries occurred to employees who were performing CMR work activities.

After reviewing narratives of incidents and discussing factors that contributed to these injuries, the group of mine safety practitioners agreed that an appropriate strategy for reducing CMR injuries would be to develop a meaningful training intervention. The goal of the training would be to increase employee awareness of hazards and also advise safe construction, maintenance and repair practices for workers to follow. An interactive, 3-D slides CMR training program called <u>Hazard Recognition Training Program for Construction, Maintenance, and Repair Activities</u> was subsequently developed to accomplish these objectives. [3] The purpose of this paper is to document its' development and evaluation.

THE "HAZARD RECOGNITION TRAINING PROGRAM FOR CONSTRUCTION, MAINTENANCE, AND REPAIR ACTIVITIES"

Overview

The <u>Hazard Recognition Training Program for Construction, Maintenance, and Repair Activities</u> is an 80-page teaching document that includes a set of three (3-D) slide reels, each containing seven scenes. The slides depict various construction, maintenance and repair work activities at non-coal, surface mining operations. They provide visual references from which class discussions emanate as trainees focus on the hazards of the particular CMR work activity being depicted.

The concept of *degraded images* is incorporated into the (3-D) slides. Both instructional aids, (3-D) slides and *degraded images*, have been shown in earlier studies to be effective for training miners to recognize and respond to hazards [4, 5, 6].

Three-dimensional slides were reported by the former U.S. Bureau of Mines (The safety and health research functions of the former U.S. Bureau of Mines were transferred to NIOSH in 1996) to be effective for teaching miners to recognize various geologic and mining-induced irregularities that may cause ground failures. As such, they serve as an excellent proxy for training miners to recognize cues that distinguish various types of hazards.

The *degraded image* concept was originally developed and used for military target detection training. *Degraded images* are scenes where the subjects are partially hidden from view, observed from an eccentric angle, viewed through haze or dust, inadequately illuminated, or otherwise obstructed so as to camouflage the target. Military research has shown that pilots who were trained with less than ideal (or degraded) visuals were more successful in subsequent identification of targets than those trained using ideal (or highlighted) pictures of targets.

In the 80-page CMR training program, 61 pages constitute a comprehensive Instructor's Guide and the other 19 pages contain information that trainers may use for handouts and overheads. The Instructor's Guide includes the following sections: Introduction; Performance Objectives; Instructor Guidelines; Classroom Format; Key Concepts; Materials; Instructional Method; Instructor's Notes; Key Concepts by Scene; and List of Hazards by Scene. The materials for handouts and overheads include: the definition of CMR work activities; a classroom format; inter-rater reliability results; the Pretest/Posttest (with answer key); and a student handout containing safe work practices.

Exercise Development

The initial step in developing the exercise was to identify key concepts (topic areas) to be incorporated into the training program. This was accomplished, in part, through discussions with, and recommendations from, a multi-disciplinary group of professionals who provided expertise in choosing the concepts. Individuals in this group are proficient in one or more of the following areas: mining, industrial, and safety engineering; education and training; enforcement; ergonomics; and mine labor/management. Seventeen key concepts were identified. They include: Communications; Confined Spaces; Electrical; Elevated Work; Ergonomics; Excavation and Trenching; Falling Materials; Fire Safety; Hand Tools; Hazard Communication; Health Hazards; Lockout/Tagout; Machine Guarding; Material Handling; Mobile Equipment; Personal Protective Equipment; and Welding/Cutting.

The next step was to prepare a broad account of information about each of the key concepts. This material also includes best practices for dealing with CMR hazards and explicit discussion notes to serve as a resource for trainers.

Concurrently, visuals depicting construction, maintenance and repair activities at noncoal surface mines were obtained using specialized (3-D) photographic equipment. The slides, which correspond to the 17 key concepts, demonstrate various hazardous conditions and situations relating to all of the concept areas.

A first draft of the CMR training program was then prepared and sent to various industry, academia and MSHA representatives for authentication. Their comments and recommendations were considered and some of them were incorporated into the draft.

The next step in the development process was to pilot test the exercise. The purpose of the pilot study was to use the instructional materials and evaluation procedures in a "trial run" with a small number of subjects and make any necessary changes prior to field testing. Two pilot tests were conducted. The first included representatives from industry, MSHA, NIOSH and academia. The pretest was administered, the exercise presented, and the posttest and evaluation followed. The second pilot test was conducted at the MSHA Mine Academy with representatives from MSHA's Field Services Division. Identical presentation procedures were followed for both pilot tests. Based on comments from the participants, some changes were made to the exercise content, particularly use of appropriate terminology for mining equipment, work processes, and mine conditions. The pilot test experiences also helped to establish consistency in presenting the exercise for field testing. In particular, they helped to structure parameters of the training program with regard to allocation of total time for each concept and specific time for follow up discussions. After considering all recommendations and suggestions, those that were judged to improve the exercise were incorporated into a final draft.

EXERCISE EVALUATION

The CMR training program was developed as a synergistic exercise in which trainees actively participate throughout the entire instructional period. As such, the resulting discussions vary with each training class because the information being shared is directly related to the knowledge, experience, skills, and interests of those in the class. This exchange is essential as it contributes to achieving the stated learning objectives; however, because of this interaction, it is inherently impossible to attribute any "pretest to posttest" improvement in test scores entirely to the training. Even using a control group to which no training intervention is applied, scores for the experimental group are still affected by the major competing explanation, i.e. variability of class discussions. Internal validity, therefore, cannot be achieved regardless of the experimental design and, it would be unrealistic to propose that any improvement in scores is tied entirely to the CMR training exercise. However, this is not to suggest that higher test scores may not be partially due to the training. It is reasoned that if results show a substantially large number of subjects scoring higher in the posttest, then the CMR training must have had a positive effect.

Experimental Design

Because of inability to eliminate the *confound* described above, a simple one-group, repeated measures pretest-posttest experiment was designed to determine if miners' test scores would improve following the CMR training intervention. A non-probability haphazard sample consisting of miners representing the noncoal, surface mining industry in six States were selected. They were measured both before and after the CMR training exercise (independent variable). The dependent variable is the change in scores between the pre- and posttests.

Subjects

The subjects were voluntary participants from mine training classes, safety seminars, or conference workshops at various locations in six states. This "sample of convenience" consisted of 340 persons in 12 nonequivalent groups, ranging in class size from 18 to 61 individuals. They were located in PA, WV, VA, NC, AL, and WY.

Their job classifications varied from hourly employees to supervisors. There were 119 job classifications represented. For reporting purposes, the subjects were categorized as miner-laborer; technical; and supervisory. Examples of each include welder, laborer, and truck driver for miner-laborer; project manager, engineer, and trainer for technical; and foreman, quarry supervisor, and plant manager for supervisor.

Field Tests

Twelve field tests were conducted. Each was structured so that the same sequence of events occurred at all locations. The chronological succession began with the pretest, followed by the CMR training exercise, in which persons viewed slides and discussed hazards, and ended with the posttest. No direct, follow up discussions of the pretest questions were held. The total training time needed for each field test was approximately two and one-half hours. The pretest and posttest contained twenty identical true or false questions. Each question was grounded in one or more of the focus areas identified as content material.

After the posttest, subjects completed (1) a demographic information form which asked for job title, years experience in the job, age, and years experience in mining, (2) a seven question Likert scale which related to the validity of the exercise and utility of the training program, (3) a strong point/weak point query which asked for their opinions on the overall strengths and weaknesses of the CMR training program. Table I depicts these results.

Table I: Demographics of Subjects						
	N	Range	Minimum	Maximum	Mean	Std. Deviation
Age	223	44	19	63	38.89	10.49
Yrs. Exp. In Job	202	35	0	35	9.55	8.30
Yrs. Exp. In Mining	212	42	0	42	13.59	9.58
Yrs. At Mine	195	35	0	35	10.51	8.87
Valid N	173					

Results

The results of the field tests are presented in three parts. The first part looks at subjects' improvement in scores from pretest to posttest following the training; the second part describes miners' self-reporting evaluation of the training; and the third part summarizes participants' opinions on the strong and weak points of the program. Improvement in test scores is defined as subjects getting one or more additional correct answers in the posttest than in the pretest. If the number of correct answers from pretest to posttest decreased or stayed the same, then no improvement was recorded.

Test Scores: 71% of the subjects (241 of 340) showed improvement in their posttest scores following the manipulation (training). The mean score among all subjects (N=340) in the pretest was 14.49 correct answers; standard deviation was 2.57. The mean score among all subjects in the posttest was 16.01 correct answers; standard deviation was 2.28. Of subjects who just showed improvement in the posttest (N=241) the mean score was 16.47 correct answers; standard deviation was 2.13. Tables II, III, IV, and V show these results.

Of the 241 subjects who showed improvement, 34.9% increased their scores by 1 correct answer; 29% increased their scores by two; and 19.9% improved by three additional correct answers in the posttest. Table VI shows these results.

The job category of miner-laborer had the highest posttest score improvement following training with a mean of 2.4 additional correct answers (s.d. = 0.35). Improved scores for subjects classified as technical (T) averaged 2.0 more correct answers (s.d. = 0.61) and, for those classified as supervisory (S), subjects increased their scores by 1.9 additional correct answers (s.d. = 0.83)

Table II: Pretest/Posttest Comparison		
	Pretest Scores	Posttest Scores
N	340	340
Mean	14.49	16.01
Std. Deviation	2.57	2.28

Table III: Pretest scores

Correct Answers	Frequency	Percent	Cumulative Percent
1	1	.3	.3
5	2	.6	.9
6	1	.3	1.2
7	1	.3	1.5
8	2	.6	2.1
9	7	2.1	4.1
10	10	2.9	7.1
11	16	4.7	11.8
12	17	5.0	16.8
13	38	11.2	27.9
14	61	17.9	45.9
15	60	17.6	63.5
16	54	15.9	79.4
17	41	12.1	91.5
18	19	5.6	97.1
19	8	2.4	99.4
20	2	.6	100.0
Total	340	100.0	

Table IV: Posttest scores

Correct Answers	Frequency	Percent	Cumulative Percent
7	1	.3	.3
9	1	.3	.6
10	3	.9	1.5
11	11	3.2	4.7
12	7	2.1	6.8
13	26	7.6	14.4
14	35	10.3	24.7
15	38	11.2	35.9
16	57	16.8	52.6
17	71	20.9	73.5
18	46	13.5	87.1
19	34	10.0	97.1
20	10	2.9	100.0
Total	340	100.0	

Table V: Overall Test Results		
	Pretest Scores	Posttest Scores
N	241	241
Mean	14.10	16.45
Std. Deviation	2.58	2.13

Table VI: Test Score Improvements										
Improved Score	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10
Number of subjects	84	70	48	17	11	4	4	2	0	1
Percent showing improvement	34.9	29.0	19.9	7.1	4.6	1.7	1.7	0.8	0.0	0.4

Trainees' evaluation: The analysis of the trainee rating scale responses show that more than 93% reported "learning something new from the exercise" and over 94% said they "would use some of the ideas presented to work more safely". (Percentages were determined by combining ratings of "4" and "3") Table VII shows these results.

The utility of the CMR exercise was estimated to be high as more than 94% of the subjects reported that "the way the material was presented is a good way to learn". The final measure of the exercise validity was judged to be high as more than 92% of the subjects indicated that "the visuals (3-D slides) helped explain concepts".

Strong Points/Weak Points: Nearly one-half of the participants (153 of 340) commented on the strengths and weaknesses of the training program. The leading remarks regarding strengths addressed the high degree of group discussion and class participation exhibited throughout the training sessions. Several participants commented: "a new interactive way to present material...I like the 3-D effect and interaction with students...allowed us to discuss and share information." Another common theme was the scope of realism brought to the visuals through the use of 3-D scenes. "The scenes offered a good perspective of general work areas." Approximately one-forth (84 of 340) of the subjects commented on the weak points of the training program. The consensus suggested that there were too many hazards introduced to adequately address all of them in the allotted time. "had to rush through scenes....too little time dedicated to all the hazards." Also, many of the comments suggested that additional visuals (overheads or videos) be used to support the concepts seen in the 3-D slides; "...if supporting images could be placed on a screen for all to see, it would be easier to explain hazards and safeguards." Others stated that the 3-D scenes were too restrictive and did not allow one to see the entire work environment.

CONCLUSION

The CMR training program was designed to teach hazard recognition skills and to present safe work procedures for miners who perform construction, maintenance, and repair work activities. These were addressed as the instructor and trainees proceed together through a series of (3-D) visuals showing CMR work activities at surface noncoal mines. During training classes, miners can vicariously experience workplace conditions because the slides realistically portray various construction, maintenance and repair activities that typically occur at surface operations. The instructor leads the participants in discussions as they consider the key points depicted in each scene.

How effective is the training? Experimental results indicate that the training program helped mine employees recognize CMR type workplace hazards and also increased their knowledge of accepted safe work practices. Approximately seven out of ten subjects increased their test scores following training.

Table VII: Self-reporting Measure Results - Subjects' Rating of CMR Exercise

		Likert Scale					
		Strongly Agree			Strongly Disagree		
		4	3	2	1	Mean	Std.Dev.
The directions for working this exercise were clear	(N=335)	226	95	12	2	3.63	.59
The slides did not show actual working conditions	(N=335)	33	60	84	158	1.90	1.02
The visuals (3-D slides) helped explain concepts	(N=336)	192	118	16	10	3.46	.72
The safe work practices presented will not help me work safely	(N=336)	48	26	65	197	1.78	1.09
I will use some of the ideas presented to work more safely	(N=338)	222	96	9	11	3.57	.70
I learned something new from this exercise	(N=338)	221	92	20	5	3.57	.67
The way the material was presented is a good way for me to learn	(N=335)	215	101	16	3	3.58	.63

Were the subjects sensitized because of the pretest? Possibly; but somewhat less than suspected for two reasons. One, a moderately long period of time (more than two hours) lapsed between the pre- and post- tests and, two, the ensuing class discussions did not concentrate directly on the test questions; instead they focused on the visuals.

The self-reporting evaluation gave a clear indication of the meaningfulness of the CMR training. More than nine of ten subjects reported that they "learned something new from the exercise"; "would use some of the ideas presented to work more safely"; and "the (3-D) slides helped to explain concepts".

The CMR training program presents a realistic opportunity for miners to become more cognizant of the many hazards associated with construction, maintenance, and repair work activities and to learn about safe work practices for dealing with them. This experience may help workers to recognize unsafe situations and conditions in the safety of the classroom and prepare for events that are likely to occur in the real mining world.

To date, approximately 2,100 copies of the exercise have been distributed to the mining industry. The program is nonperishable and may be reused in training classes. The interactive format is favored by many trainers because it provides for active classroom participation as opposed to the traditional, passive teaching of facts and reviewing of injury data and incident narratives. However, the true impact of the CMR training program probably has not yet been realized. Follow-up observations to evaluate the impact of this program could improve the quality and effectiveness of future training materials. The lessons learned in the development of the training program, as well as the content of the exercise itself, should help to improve the health and safety of our nation's miners.

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APPENDIX A

GUIDELINES FOR CLASSIFYING CONSTRUCTION, MAINTENANCE AND REPAIR ACCIDENTS

Accidents may be classified as construction, maintenance or repair type accidents if they meet **at least one** of the following criteria: The definition of "construction" work activities: the building, rebuilding, alteration, or demolition of any facility or addition to existing facility at a surface mine, surface area of an underground mine or underground mine; including painting, decoration or restoration associated with such work, and the excavation of land connected therewith, but excluding shaft and slope sinking and work performed on the surface incidental to shaft or slope sinking. (36CSR23, Board of Coal Mine Health and Safety, West Virginia)

* The definition of "maintenance/repair" work activities: the constructing, installing, setting up, adjusting, inspecting, modifying, and maintaining and/or servicing machines or equipment. These activities may include; lubricating, cleaning or un-jamming of machines or equipment and making adjustments or tool changes, where the employee may be exposed to the unexpected energization or startup of the equipment or release of hazardous energy. (29CFR Part 1926. Lock out/tag out procedures, OSHA)

* All welding and cutting activities, use of non-powered and powered hand tool and those activities involving the use of both mobile and fixed cranes.

* All activities involving the assembly, disassembly, setting up and dismantling of equipment, machines and related components therein.

* All those activities including walking/ running/crawling and climbing if the activity was within the performance of construction, maintenance or repair work.

Note:
Classification of construction/maintenance/repair activities are made independent of employee occupation or job title.

TECHNICAL SESSION II-A

DEVELOPMENTS IN UNDERGROUND MINING

Session Chairs

Richard Stickler
Director
Pennsylvania Bureau of Deep Mine Safety

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A REVIEW OF OCCUPATIONAL SILICA EXPOSURES ON CONTINUOUS MINING OPERATIONS

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ABSTRACT

Data on dust control practices, geology, and occupational exposures were gathered for approximately eighty underground continuous mining units. Despite silica contents in excess of 5%, nearly forty units successfully maintained silica concentrations at or below 100 $\mu g/m^3$ on a majority of occupational dust samples while the remainder had difficulty maintaining this level. These two sample sets were termed group A operations and group B operations, respectively.

Analyses of productivities, geologies, and dust control parameters revealed only minor differences between these two groups. Subsequent analyses of face ventilation design showed considerable differences in silica exposure and silica content between group A and group B at the continuous mining machine and roof bolter operator occupations. These differences were minimal when using exhaust curtain ventilation with a dust scrubber. This face ventilation system may benefit operations having difficulty controlling silica dust exposure and silica dust content.

Finally, the collected data showed that occupational samples from group B operations possessed generally higher silica exposures and silica content than similar samples from group A. The single head roof bolter (helper) possessed among the highest silica exposures and silica contents in both groups.

INTRODUCTION

The Federal Coal Mine Health and Safety Act limits the respirable dust exposure of mine workers to a time weighted average of 2.0 mg/m^3 for a working shift (1). If the respirable dust sample contains more than 5 percent silica by weight, the dust standard is reduced according to the formula 10 ÷ (%silica). This maintains silica dust levels at or below 100 $\mu g/m^3$.

The process of adjusting the respirable dust standard is discussed by others (2-4) and is based upon the silica content of dust samples collected by coal mine operators and by Mine Safety and Health Administration (MSHA) coal mine inspectors. Such sampling is conducted for eight hours at 2.0 liters/min using a 10-mm nylon cyclone preseparator. On continuous mining operations, samples analyzed for silica content are typically collected at mining machine operator and roof bolter operator occupations.

Typically the dustiest occupation at each production unit, mining machine operator samples are used to adjust the respirable dust standard for the entire mining unit with the understanding that the reduced standard will protect all other occupations. Due to a variety of factors (5), however, silica dust levels at roof bolter occupations often can exceed silica levels found at the continuous mining machine. To provide additional protection, MSHA may require more frequent sampling of roof bolter occupations by the mine operator.

A number of studies have defined trends in silica dust exposures for the underground coal mining industry (2,3). These analyses revealed that for the period July 1991 through 1992, more than 40% of the continuous mining machine operator and machine helper samples exceeded 5% silica. Between 25 and 30% of the operator and helper samples exceeded 100 Fg/m^3 for respirable silica. For this same period, roughly 50 to nearly 70% of the roof bolter operator and helper samples exceeded 5% silica. Thirty to forty percent of the samples exceeded 100 Fg/m^3.

Comparisons of two groups of operations provided insight into potential causes of silica dust exposures. One group, despite high silica dust content, successfully controlled occupational exposures. The second group could not successfully control exposures. This required examinations of dust control practices, work practices, geologic conditions, and corresponding occupational exposures for operations in each of these two groups.

Such information was available at MSHA field offices. Data on approved dust control practices and work practices were part of the dust control plan established for each underground coal operation. Actual operating conditions plus corresponding occupational exposures were found in reports filed by MSHA coal mine inspectors after sampling at these operations. This study only used exposure data from MSHA compliance sampling in lieu of exposure data from mine operator compliance sampling.

The two groups were identified through examinations of the MSHA coal mine silica data base that contains all compliance samples analyzed for silica after 1981. Nearly 97,000 silica records are represented in the period 1982-1996. Each sample record contains considerable information, such as MSHA mine identification number, mining unit designation, sample date, sampling time, occupation sampled, pre and post filter weights, and silica percentage. From this data, respirable dust and respirable silica dust concentrations were calculated.

Due to the large number of underground coal mining operations represented in the database and the need to gather dust plan and MSHA exposure data from each operation, only a very small subset of these operations could be considered for the study. For this reason, several restrictions were placed on the selection of a particular mining operation.

Only operations in southern West Virginia, southwestern Virginia, northeastern Kentucky, and southeastern Kentucky were considered. Prior to the start of this study, MSHA suggested that these areas be considered due to their high prevalence of silica exposure.

When this survey began, October 1997 compliance sampling data was the most current available from MSHA. Identification of operations for this study was based upon the results of MSHA compliance sampling at the continuous mining machine operator and roof bolter operator occupations for the period January 1997 to October 1997. Pre-1997 sampling data was not used to keep this information as current as possible.

For this study, two groups of operations were identified in each area. The first group contained those operations with a majority of samples exceeding 5% silica and having silica dust concentrations less than or equal to 100 Fg/m^3 (group A operations). The last group contained those operations with a majority of samples exceeding both 5% silica and 100 Fg/m^3 (group B operations). Eighty operations initially were considered in this study, forty able to maintain respirable silica dust levels on a majority of compliance samples at or below 100 Fg/m^3 and forty unable to maintain a majority of samples below this level. During visits to the various MSHA field offices, we discovered that some operations were no longer producing. Because current exposure data was not available for these operations, they were dropped from further consideration. The study examined 39 underground coal mining operations in group A and 36 operations in group B. These operations were selected from a population of nearly 700 operations in these geographic areas.

MSHA field offices were visited and data gathered from the approved dust control plan for each of the selected underground mining operations in that district. MSHA mine inspector reports for each operation were reviewed to note dust control practices and to note corresponding occupational exposures to respirable silica and coal mine dusts (table I). The data gathered from these visits produced a history of occupational exposure for these mining operations from January 1997 to an approximate end date of June 1998 to October 1998 (end dates differed because the districts were not visited in the same month).

Table I. Distribution of occupational exposures for two sample groups

Occupation sampled	Number of occupational samples in group A operations		Number of occupational samples in group B operations	
	≤ 100µg/m^3	> 100µg/m^3	≤ 100µg/m^3	> 100µg/m^3
Continuous miner operator	177	60	63	88
Twin head roof bolter (return side operator)	60	8	11	34
Twin head roof bolter (intake side operator)	50	11	11	35
Single head roof bolter (operator)	58	12	22	39
Single head roof bolter (helper)	15	4	5	14

EVALUATION OF GENERAL CHARACTERISTICS

General characteristics for group A operations and group B operations are given in table II. These include daily tonnage produced, coal thickness mined, and rock thickness mined during sampling by the MSHA mine inspector. Also given are values for various dust control parameters (face ventilation quantity, scrubber quantity, spray count, and water pressure) as measured by the coal mine inspector during sampling.

Maximum, minimum, and median values categorize the distribution of values for each characteristic. Differences in sample size between group A and B operations are attributed to the number of operations in each group and the extent of sampling at each operation.

Table II. Characteristics of operations in two sample groups

Characteristic	Group A Operations			Group B Operations		
	Number of samples	Range of values	Median value	Number of samples	Range of values	Median value
Production (tons)	384	96-2700	600	226	85-2520	572
Coal thickness (inches)	318	4-108	48	162	0-126	42
Rock thickness (inches)	315	0-60	8	162	0-96	8
Face ventilation flow (cfm)	325	1080-49500	6462	203	2024-45360	6500
Scrubber flow (cfm)	133	3000-11600	5034	53	2580-13099	5600
Spray count	313	18-50	27	188	14-58	25
Spray pressure (psi)	313	50-280	90	195	50-260	85

Notes:
1. Face ventilation flow is amount of air flowing to the continuous miner.
2. Spray count is the number of water sprays on the continuous mining machine.

This data shows small differences in dust control parameters and geologic conditions for operations in groups A and B. Group A operations produced roughly 5% more than Group B operations. Ranges of coal and rock thicknesses were greater for Group B operations than Group A operations. The range of face ventilation values was greater for group A than group B although median values were similar. While the range of scrubber airflows was similar, the median value was greater for B operations than group A operations. Water spray count and spray pressure were consistent between the two groups.

Although data was available on rock thicknesses, little information was available on the composition of rock present at the cutting face. Previous work (6,8) showed that the silica content in the roof, floor, or parting material could influence the amount of respirable silica generated during the mining process.

Previous work showed that a number of factors could potentially impact occupational silica dust exposures and silica dust contents in underground coal mining. Dust control practices such as ventilation airflow, flooded-bed dust scrubber quantity, water spray quantity and pressure, and water spray configuration influence silica dust exposures (4). Dust control practices operating in the ranges given in table II have more effect on silica exposures than silica contents. With MSHA's presence during sampling, it is likely that good execution of the dust control plan occurred. When evaluating the collected data, silica exposures were assumed as indicators of dust control effectiveness on that mining unit.

Work practices affect silica content, for instance, using a modified cutting scheme to avoid grinding of parting rock (4), avoiding the use of worn cutting bits (5), minimizing the time the roof bolter works downwind of the continuous mining machine (6), and maintenance of dust control systems. Geologic conditions at the mining unit (such as rock parting type, thickness, and silica content) affect silica content (7,8). For these analyses, silica content was assumed as an indicator of either work practices or geologic conditions on the mining unit.

ASSESSMENTS OF FACE VENTILATION

Four basic types of face ventilation systems were noted in reviews of MSHA mine inspector reports.

1. Exhaust ventilation curtain with dust control provided by a machine-mounted flooded-bed dust scrubber.
2. Exhaust ventilation curtain with water sprays on the mining machine arranged in either a sprayfan or other directional face spray design.
3. Combination system using intake curtains to ventilate faces on one side of the section belt entry and exhaust curtains to ventilate faces on the other side. This system reduced the number of drive-through check curtains between each face and the feeder-breaker in the belt entry. Dust control provided by a flooded bed dust scrubber.
4. Intake or blowing ventilation curtain with a flooded-bed scrubber for dust control. However, this face ventilation scheme was not widely represented in the accumulated data and, for this reason, was not evaluated further in this study.

Occupational exposures were noted when the continuous mining machine operator was sampled concurrently with the roof bolter operator. Table III compares minimum, maximum, and median values for silica dust exposure and silica content at the mining machine operator occupation for group A and group B operations.

This data in table III shows that all face ventilation designs were equally represented in group A operations while group B operations used combination curtain face ventilation by nearly two-to-one margin over other ventilation schemes. Group A silica exposures generally were much less for all face ventilation schemes with the exception of the exhaust curtain with dust scrubber. For group B operations, silica exposures were less using this ventilation scheme. Silica content levels were much less for group A operations than for group B operations.

Table III. Distribution of silica exposures and silica contents at continuous mining machine operator occupations for groups A and B operations.

Ventilation Scheme	Number of samples	Group	Silica exposures (µg/m^3)		Silica contents (%)	
			Range of values	Median value	Range of values	Median value
Exhaust curtain with scrubber	59 31	A B	16-312 1-532	90 150	1-17 0-16	6 9
Exhaust curtain with directional spray designs	57 31	A B	2-260 20-946	100 210	0-14 2-25	8 11
Combination curtain system	62 55	A B	3-549 40-571	90 180	2-14 7-22	8 11

These data suggest that dust control effectiveness for group A operations generally was superior to that for group B operations for all face ventilation schemes except when using exhaust curtain with a scrubber. The high silica contents of group B show that this group of operations likely suffered from poor work practices or inferior geologic conditions compared to group A operations.

Roof bolter exposures were more difficult to assess because their position could change with respect to the fresh airflow of the face ventilation curtain. While being sampled by the MSHA mine inspector, the roof bolter(s) could move upstream or downstream of the continuous mining machine, thus changing environmental conditions for roof bolter occupations. Unfortunately, information of this type was seldom available from MSHA reports.

A comparison of silica exposures and contents at roof bolter occupations for group A and B operations is given in table IV. This data show that, for all three face ventilation designs, bolter occupation silica exposures were highest for group B operations. The difference between A and B operations was greatest when using exhaust curtain face ventilation with directional sprays or combination curtain face ventilation. Differences were less when using exhaust curtain ventilation with a scrubber. Silica contents for roof bolter occupations in group A operations were less than those measured in group B operations. This suggests improved work practices or geologic conditions for group A operations.

Table IV. Distribution of silica exposures and contents at roof bolter occupations for group A and B operations.

Ventilation Scheme	Number of samples	Group	Silica exposures ($\mu g/m^3$)		Silica contents (%)	
			Range of values	Median value	Range of values	Median value
Exhaust curtain with scrubber	59 31	A B	0-364 17-381	80 150	0-24 5-21	9 11
Exhaust curtain with directional spray designs	57 31	A B	0-259 20-1771	80 210	0-14 5-77	8 14
Combination curtain system	62 55	A B	20-303 40-571	90 190	2-16 7-28	10 12

The data show that differences in group A and group B silica exposures for mining machine and roof bolter operator occupations were minimized when using exhaust curtain face ventilation with a dust scrubber. Using either exhaust curtain ventilation with directional sprays or combination curtain ventilation, group B operations were unable to control occupational silica exposures at either the mining machine or roof bolter occupations. It appears that operations able to control silica exposures for mining machine operator occupations were able to control exposures for roof bolter occupations.

The data also suggest that operations having difficulty controlling silica dust exposures may benefit from using exhaust curtain ventilation with a dust scrubber. This scheme is not affected as much by high intake air velocities as is blowing curtain ventilation. Past work showed that high air velocities issuing from the mouth of a blowing curtain could disrupt capture of respirable dust by blowing the dust cloud around the mining machine (9). Also, positioning of the mining machine operator is not as critical to controlling this person's occupational exposure when using exhaust curtain ventilation. The operator has some latitude in movement as long as this person remains outby the curtain mouth. However, the operator must remain within the mouth of the blowing curtain to control occupational dust exposures (10). Three, exhaust curtain ventilation with a dust scrubber does not produce high dust gradients around the mining machine as can occur with the sprayfan system. These high dust gradients also can affect occupational exposures of the roof bolter operator when working downwind.

ASSESSMENTS OF OCCUPATIONAL EXPOSURES

Data on silica exposures and silica percentages were categorized according to worker occupation for group A and B operations (table V). Much of the exposure data is given for the continuous mining machine operator. The remaining roof bolter occupations, with exception of the single head roof bolter helper, had similar numbers of samples in this study. Silica data were available for other non-bolting occupations. However, these samples were not very numerous and, consequently, were not reported.

Table V. Distribution of silica exposures and contents for continuous mining and roof bolting occupations in group A and B operations.

Occupation	Number of samples	Group	Silica exposures ($\mu g/m^3$)		Silica contents (%)	
			Range of values	Median value	Range of values	Median value
Continuous mining machine operator	321 192	A B	1-664 1-1149	90 100	0-37 0-34	8 9
Twin boom roof bolter (intake side operator)	77 52	A B	8-340 10-583	70 150	1-23 1-74	10 12
Twin boom roof bolter (return side operator)	70 54	A B	10-329 4-653	70 200	2-21 0-27	9 13
Single boom roof bolter operator	86 67	A B	0-263 1-1771	80 200	0-35 0-39	8 14
Single boom roof bolter (helper)	22 20	A B	7-644 4-740	90 280	1-17 2-21	9 14

This data show that silica exposures and silica contents at all occupations were higher for group B operations than group A operations. Variations in exposures and contents suggest that these differences were attributable to changes in dust control effectiveness, work practices, and geologic conditions.

The single head roof bolter (helper) possessed among the highest silica exposures and silica contents in group A and B operations. This suggests less effective dust controls, inappropriate work practices, and poorer geologic conditions at this occupation for group A and B operations. It is also possible that the bolter helper engaged in activities that put this person at risk for increased exposure to silica. These would include emptying the dust box and cleaning the dust filters.

SUMMARY

Data on dust control parameters, geology, and corresponding silica exposures was gathered for nearly forty operations that, despite silica content in excess of 5%, were able to maintain silica dust levels in a majority of occupational samples at or below 100 $\mu g/m^3$. Approximately forty other operations were identified that were unable to maintain a majority of occupational samples less than or equal to100 $\mu g/m^3$. These two subsets were termed group A and group B operations, respectively.

Evaluations of geologic and dust control parameters generally revealed only minor differences between group A and group B operations. Subsequent assessments of face ventilation design showed that group B operations suffered from higher silica exposures and contents at continuous mining machine and roof bolter occupations. However, these differences were minimized when using exhaust curtain ventilation with a dust scrubber. This face ventilation system may benefit those operations having difficulty controlling occupational silica exposures.

These evaluations also revealed that occupational silica exposures and silica contents were higher for group B operations than group A operations. In both groups, the single head roof bolter (helper) occupation possessed among the highest silica exposures and silica contents. It is possible that this occupation engaged in activities that put this individual at risk for overexposure to respirable silica.

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BEST PRACTICES TO MITIGATE INJURIES AND FATALITIES FROM ROCK FALLS

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INTRODUCTION

Falls of ground continue to be one of the most serious causes of injury to U.S. miners. Of the 256 fatal injuries that occurred in mining between 1996 and 1998, 52 (20%) were caused by falls of ground (Table I). Falls of ground affect some sectors of the mining industry more severely than others. For instance, nearly 40% of the 98 coal mine fatalities between 1996 and 1998 were caused by falls of ground. Underground miners are at much greater risk than surface miners. Nearly half (45 out of 101) of underground mine fatalities were attributed to roof, rib and face falls, while less than 5% of the 155 surface fatalities were caused by falls of highwalls or slopes.

Sources of Data

All of the injury data examined in this study were derived from MSHA's Fatal Investigation Reports and the MSHA accident database. Because falls of ground often result in serious injury, MSHA Fatal Alert Bulletins and Fatal Investigation Reports provide a useful snapshot of ground control issues in the mining industry. MSHA requires that mines file a report on every reportable accident that occurs, containing information on the accident's location, severity, classification, activity, and nature of injury, etc. A short narrative is generally included as well. Accident reports can be searched by many of the above fields.

From 1996 to 1998, miners suffered a total of 55,096 injuries, which ranged in severity from death (degree 1) to injuries with no days away from work nor restricted duty (degree 6). Six percent of the total injuries were from falls of ground, including machine accidents where caving rock was coded as the source. As the data in Table II indicate, 98% of all non-fatal fall of ground injuries occurred in underground mines, with underground coal mines accounting for 83% of the total.

Table II also shows the distribution non-fatal fall of ground injuries by severity and commodity. The injuries are classified into Lost Time Injuries that resulted in permanent disability (degree 2) or days off work (degrees 3-4), and Injuries Without Lost Time that resulted in no more than restricted duty (degree 5-6). Overall, groundfall injuries appear to be more serious than other types of mining injuries. 65% of all ground fall injuries resulted in lost time compared to 54% of all types of mining injuries.

Table I - Fatalities from 1996 to 1998 by commodity for both falls of ground and other mining classifications.

	1996		1997		1998		Total	
	Under-ground	Surface & Prep. Plants	Under-ground	Surface & Prep. Plants	Under-ground	Surface & Prep. Plants	Under-ground	Surface & Prep. Plant
Coal falls of ground	13	1	9	0	14	1	36	2
Coal total	33	6	22	8	22	7	77	21
Metal falls of ground	1	0	2	1	3	0	6	8
Metal total	5	3	7	5	5	5	17	13
Nonmetal falls of ground	0	0	0	0	0	0	0	0
Nonmetal total	0	1	1	2	2	4	3	7
Stone falls of ground	2	2	1	1	0	1	3	4
Stone total	2	25	2	26	0	23	4	74
Sand/Gravel falls of ground	0	0	0	0	0	0	0	0
Sand/Gravel total	0	11	0	17	0	12	0	40
Total falls of ground	16	3	12	2	17	2	45	7
Total Mining	40	46	32	58	29	51	101	155

Legal Framework

Laws governing mining in the U.S. are listed in the Code of Federal Regulations under Title 30 - Mineral Resources.

Underground coal mining roof control is covered in 18 subsections within Part 75. While each of these sections outlines an important step in controlling falls of ground, some sections are cited more frequently in Fatal Investigation Reports. Between 1996 and 1998, a total of 30 citations were given to mines following fatal accidents, citing 6 of the 18 sections (Table III). The most frequently cited subsection was 75.202 - Protection from falls of roof, face, and rib. Section 75.202 requires that ground support must protect persons from hazards related to falls of the roof, face or ribs and coal or rock bursts in areas where they work or travel. It also states that no person may work or travel under unsupported roof unless in accordance with special procedures. Another common citation listed was violation of the Roof Control Plan. The language also states that additional measures shall be taken to protect persons if unusual hazards are encountered.

Underground metal/nonmetal mining roof control is covered in 9 subsections within Part 57. Between 1996 and 1998, the most frequently cited subsection following fatalities was 57.3200, which requires that hazardous ground conditions be taken down or supported before other work or travel is permitted (Table IV). The law also requires that the affected area be posted with a warning against entry and, when left unattended, that a barrier be installed to impede unauthorized entry. Many of the Fatal Investigation Reports reveal that geologic structures contributed to the conditions referred to in subsection language. In four of the reports, inadequate examination of ground conditions was cited.

Table II- Non-fatal fall of ground injuries from 1996 to 1998.

Severity	Commodity	Under-ground	All other	Total
Lost time injuries (Degree 2 to 4)	Coal	1807	23	1830
	Metal	140	5	145
	Nonmetal	15	0	15
	Stone	14	9	23
	Subtotal	1976	37	2013
Injuries without lost time (Degree 5 and 6)	Coal	777	9	786
	Metal	269	3	272
	Nonmetal	23	1	24
	Stone	16	7	23
	Subtotal	1085	20	1105
All non-fatal injuries	Coal	2584	32	2616
	Metal	409	8	417
	Nonmetal	38	1	39
	Stone	30	16	46
	Total	3061	57	3118

Surface mining ground control is covered in nine subsections within Part 56 for metal/ nonmetal mines and 15 subsections with Part 77 for coal mines. Several violations cited subsection 56.3200, which requires hazardous ground conditions to be taken down or supported before work or travel is permitted. The directive states that until corrective work is completed, the area shall be posted with a warning against entry and, when left unattended, a barrier shall be installed to impede unauthorized entry.

GROUNDFALL HAZARDS AND BEST PRACTICES TO CONTROL THEM

The 51 Fatal Investigation Reports from 1996 to 1998 provide a window on the most significant groundfall hazards facing today's miners. Some of these hazards, such as geologic features, affect all miners to one degree or another. Others are specific to the commodity or mining method.

Geologic Discontinuities

Mines are unique structures because they are not constructed of man-made materials, such as steel or concrete, but are rather built of rock, just as nature made them. Thus, integrity of a mine structure is greatly affected by the natural weaknesses or discontinuities that disrupt the continuity of the roof and rib. Geologic discontinuities can originate while the material is being deposited by sedimentary or intrusive processes, or later when it is being subjected to tectonic forces. Depositional discontinuities include slips, clastic dikes, fossil remains, bedding planes, and transition zones. Structural discontinuities include faults, joints, and igneous dikes.

Table III - Violations of Part 75 from fall of ground Fatal Investigation Reports, 1996-1998.

Subsection Violated	Title	Number
75.202	Protection from falls of roof, face, and ribs	18
75.203	Mining method	1
75.204	Roof bolting	0
75.205	Installation of roof support using mining machine with integral bolter	0
75.206	Conventional roof support	0
75.207	Pillar recovery	0
75.208	Warning devices	0
75.209	Automated temporary roof support systems	1
75.210	Manual installation of temporary support	0
75.211	Roof testing and scaling	0
75.212	Rehabilitation of areas with unsupported roof	1
75.213	Roof support removal	3
75.214	Supplemental support materials, equipment, and tools	0
75.215	Longwall mining systems	0
75.220	Roof control plan	6
75.221	Roof control plan information	0
75.222	Roof control plan--approval criteria	0
75.223	Evaluation and revision of roof control plan	0

Table IV - Violations to Part 57 from fall of ground Fatal Investigation Reports, 1996-1998.

Subsection Violated	Title	Number
57.3200	Correction of hazardous conditions	6
57.3201	Location for performing scaling	1
57.3202	Scaling tools	1
57.3203	Rock fixtures	0
57.3360	Ground support use	0
57.3400	Secondary breakage	0
57.3401	Examination of ground conditions	4
57.3460	Maintenance between machinery or equipment and ribs	0
57.3461	Rock bursts	0

Discontinuities occur in many shapes and sizes and are generally difficult to recognize in advance of mining. They often contribute to fatal accidents, frequently in combination with other factors. Miners, and particularly roof bolt operators and face drillers, need to be trained to recognize geologic discontinuities as soon as they are exposed by mining. They must also be aware of the proper support techniques and have the necessary support materials available.

Underground Coal Mine Hazards

Between 1996 and 1998, 36 underground coal miners were killed in 33 separate incidents. Table V lists the hazards that contributed to these incidents and their frequency. In some cases, more than one hazard was involved. For example, 13 fatalities occurred during pillar extraction, with 3 of the accidents resulting from premature intersection collapses.

Unsupported roof. Roof bolts and the Automated Temporary Roof Support (ATRS) are the first lines of defense against roof falls in underground coal mines. When miners go under unsupported roof, they are completely unprotected. Between 1996 and 1998, approximately 25% of coal mine roof and rib fatalities occurred when miners were beyond roof supports. While there are no grounds for complacency, the recent record does represent an improvement from a decade ago, when nearly 50% of ground fall fatalities occurred beneath unsupported roof (Peters, 1992). The improvement was achieved through new equipment, enforcement and a persistent educational campaign.

During the early 1990's, the U.S. Bureau of Mines conducted an extensive series of interviews with miners to determine why they might go out under unsupported roof (Peters, 1992). The most common response was that they had unintentionally walked out beyond the supports. The most effective countermeasure, then, is to ensure that all areas of unsupported roof are clearly posted with highly visible warning devices. Training is also essential. Mallett et al. (1992) argue that verbal admonitions and threats of discipline are less effective than training that graphically imparts the severe consequences of roof falls.

Finally, the prevalence of dangerous behavior depends greatly on the miner's perception of the company's policy concerning going under unsupported roof, on how that policy is enforced, and on the attitude and behavior of his supervisor and coworkers. The best prevention programs involve high-level managers who directly communicate their commitment to the goal of keeping people away from unsupported roof.

Table V - Factors in underground coal mine fall of ground fatalities.

Factor	1996	1997	1998	Total
Pillar extraction	4	4	5	13
Inby roof support	4	0	5	9
Intersections	1	3	2	6
Geology	1	4	1	6
Rib	2	3	0	5
Construction	1	0	3	4
Skin Control	1	0	2	3
Longwall face	1	1	0	2

Roof bolter safety. Roof bolt operators are on the front line in the fight against ground falls. They are continually exposed to roof and rib hazards, and historically they have experienced more groundfall-related injuries than any other occupation in mining. While large roof and rib falls have been responsible for several fatalities, most injuries are caused by relatively small pieces of rock.

The roof bolt machine, with its ATRS and canopy, is the critical piece of safety equipment. It should always be in proper operating condition before use. The proper bolting sequence, as defined in the Roof Control Plan, must always be followed. Several fatalities have resulted when operators of single-boom machines installed bolts out of sequence and placed themselves under unsupported roof.

Rehabilitation of roof falls and construction of overcasts and boom holes present special hazards. In many such areas the roof is unusually high, and often the ATRS cannot effectively contact it. If the ATRS cannot be set against the top, it is necessary to set jacks for temporary support or use a manufacturer's approved ATRS extension. Two roof bolt operators have been killed in recent years while bolting high top during mine construction activities.

Roof bolt operators are also responsible for protecting the entire crew with high-quality bolt installations. Poorly installed roof bolts can be worse than none at all, because they provide a false sense of security. Manufacturers' recommendations regarding resin spin and hold times must always be followed. Holes must be drilled to the proper length (not more than one-inch deeper than the bolt's length). The torque on tensioned roof bolts must be checked as required by CFR 75.204(f).

Skin failures of roof and rib. Skin failures are those that do not involve failure of the roof support elements, but result from rock spalling from between roof bolts, around ATRS systems, or from ribs. They are of particular concern because they cause injuries and fatalities to workers who should have been protected by supports. In 1997, 98% of the 810 roof and rib injuries suffered by mine workers were attributed to skin failures (Bauer et al., 1999).

Roof skin failures almost always involve pieces of rock that are less than 0.6 m (2 ft) thick. About 40% of the 669 roof skin injuries in 1997 involved roof bolt operators, and occurred beneath the ATRS. The other roof skin injuries occurred beneath permanent support and involved workers in a wide variety of activities. Common roof skin control techniques include oversized plates, header boards, wood planks, steel straps, meshing, and (in rare instances) spray coatings (sealants).

Between 1996 and 1998, rib failures resulted in 6 fatalities in underground coal mines. Only one of these fatal injuries was to a face worker, the other five were all mechanics and electricians performing their duties well outby the face. Nearly 80% of the 128 rib injuries that occurred in 1997 took place beneath permanently supported roof. Non-fatal rib injuries resulted in an average of 43 lost workdays each, versus 25 days for the average roof skin injury.

The seam height is the single greatest factor contributing to rib failures. The seam height was greater than 2.5 m (8 ft) in all six of the fatalities, and was greater than 3 m (10 ft) in three of them. The incidence of rib injuries increases dramatically once the seam height reaches 2.2 m (7 ft). Interestingly, mines with the very thickest seams see lower rib injury rates, probably because most of them routinely use rib support. No rib support was used in any of the six fatal accidents, however. Rib failure is often associated with rock partings and/or discontinuities within the pillar, or with overhanging brows created by roof drawrock. The most effective rib supports employ full planks or mesh held in place by roof bolts.

Pillar recovery. Pillar recovery has always been an integral part of U.S. underground coal mining. It can be a less capital-intensive, more flexible alternative to longwall mining for small, irregular reserves. A recent study estimated that pillar recovery accounts for about 10% of the coal mined underground (Mark et al., 1997).

The process of pillar recovery removes the main support for the overburden and allows the ground to cave. As a result, the pillar line is an extremely dynamic and highly stressed environment. Safety depends on controlling the caving through proper extraction sequencing and roof support. Historically, retreat mining has accounted for a disproportionate number of roof fall fatalities, including 13 between 1996 and 1998. Three of the accidents during this period resulted in double fatalities.

Traditional roof control plans require that numerous timber posts be set during each stage of pillar recovery. Recently, Mobile Roof Supports (MRS) have become available that replace many of the timbers (Figure 1). MRS resemble longwall shields mounted on bulldozer tracks. They can have many safety advantages over timbers. In particular, they are more effective as roof supports, they do not require workers to approach the mined-out gob area to set them, and they reduce the potential for materials handling injuries (Chase et al., 1997).

Following the Roof Control Plan is absolutely critical to safe pillar recovery operations. Fatality investigations have frequently found that lifts were too wide, too deep, or out of sequence. The Plan may also specify the minimum dimensions of the remnant coal left in place called *stumps* and *fenders*. However, the Roof Control Plan is a minimum plan, and additional supports should be used at any indication of bad roof.

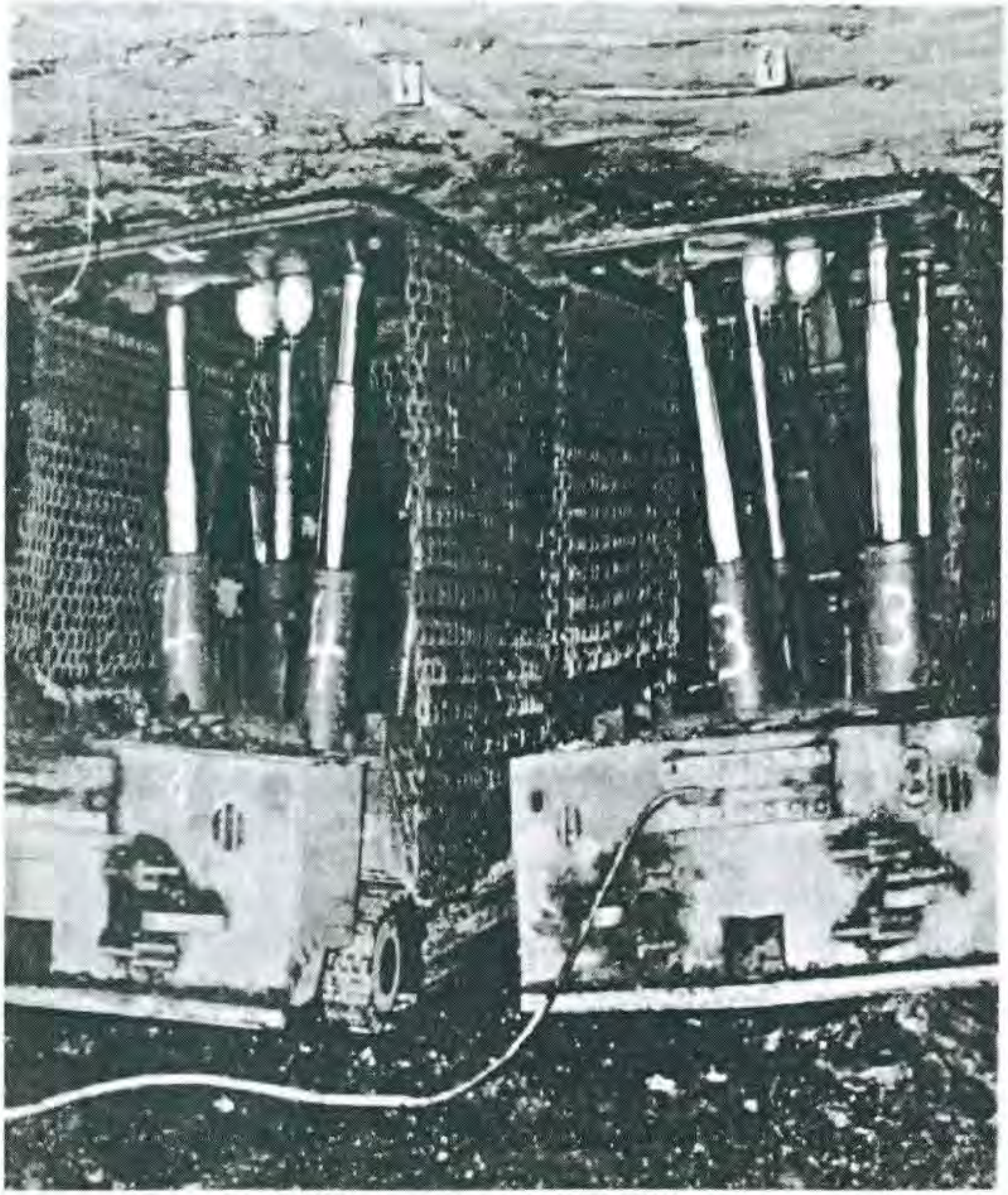

Figure 1. Mobile Roof Supports (MRS) for retreat mining

The recovery of the final stump, or pushout, is the most hazardous aspect of pillar recovery operations. During the past 20 years, nearly half of all fatalities during retreat mining have occurred while the pushout was being mined. The pushout should never be mined if conditions do not look safe or if adverse conditions arise during mining. All unnecessary personnel should remain outby the intersection at all times during pillar recovery, but especially while the pushout is being mined.

Extended cuts and remote control mining. Extended (deep) cut mining is where the continuous mining machine advances the face more than 6 m (20 ft) beyond the last row of permanent supports. The development of remote-control for continuous miners (Figure 2), spray fan systems, and flooded-bed scrubbers has provided the technology to enable deep cuts. By 1997, about 75% of all underground labor hours were worked at mines with extended cut permits. However, extended cuts raise a number of ventilation, ground control, and human factors issues. Between 1988 and 1995, extended cuts may have been a factor in 26% of all roof fall fatalities in underground coal mines (Bauer et al., 1997).

Figure 2. Miner operating a continuous mining machine by remote control

In practice, many mines with permits only take extended cuts when conditions allow for them. Where the roof is competent, extended cuts are routine. At the other extreme, when the roof is poor, miners may not even be able to complete a 6 m (20-ft) cut before the roof collapses. A premature roof collapse can trap the continuous miner or endanger the crew, or it can create uneven and hazardous conditions for the roof bolters. Where premature collapses are likely, additional roof supports (extra bolts, planks, mesh or straps) should be used within the last two rows of supports to prevent the fall from overriding these supports.

Remote control mining allows the operator to stay further back from the unsupported roof, but it also removes him from the protection provided by the canopy. The freedom of movement, combined with a lack of visibility, can tempt the operator to stray into dangerous locations. Several fatalities have occurred during the mining of the first cut in a 90 degree crosscut to operators who had gone inby permanent supports (Figure 3). In response, some companies have limited the length of the initial cuts in a crosscut to 20 ft, and others have angled the crosscuts to provide better visibility.

Hazards in Underground Metal and Nonmetal Mines

Between 1996 and 1998, nine fatal fall of ground injuries from eight different accidents occurred in underground metal and nonmetal mines (Table VI). Two major contributing factors were the failure to conduct proper roof and rib examinations and problems with removing loose rock. Overall, metal and nonmetal underground mines have lower ground fall injury rates than coal mines.

Large openings. Many metal/nonmetal mines have large openings, especially nonmetal stone and salt mines and metal mines with stopes Large mine openings have roof or back greater than 5 m (16 ft) high, with spans greater than 10 m (30 ft) wide. When the back is high, a miner's ability to observe the ground conditions is greatly reduced. Additionally, many metal/nonmetal mines use roof bolts on an infrequent basis. Ventilation of large openings is sometimes poorly controlled, promoting dramatic fluctuations in humidity, and sometimes fog. High humidity can cause even strong rocks to split and crack, creating hazards for miners. Because of these factors and others, mines with large openings rely on mining both a stable roof beam and a stable roof line to reduce ground control hazards (Iannacchione et. al, 1998(a)).

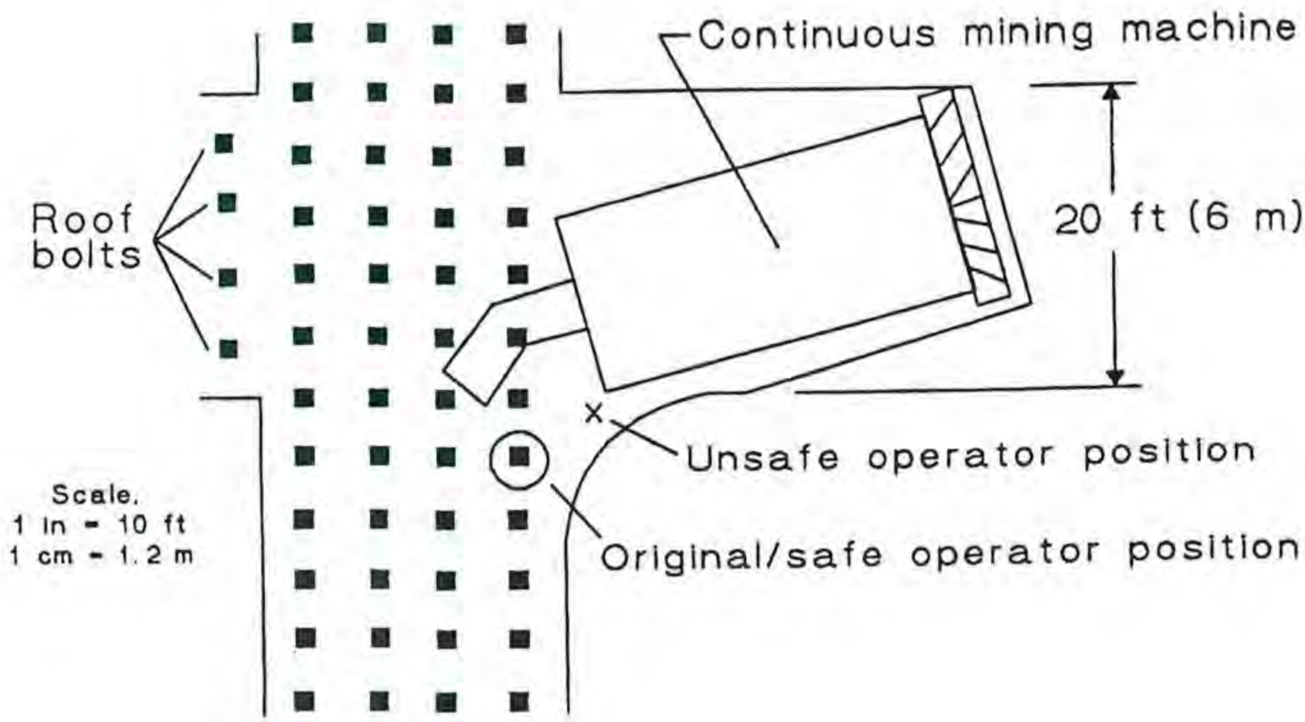

Figure 3. Unsafe location for a continuous mining machine operator while mining a crosscut

A stable roof beam is generally massive, strong, thick, and persistent. Natural laminations, bedding planes, or interfaces between rock layers often provide the best roof lines (Figure 4). If a natural smooth roof plane does not exist, special blasting procedures like pre-splitting or smooth blasting can be used to produce an artificial smooth roof plane. Conversely, poor blasting practices often have a negative influence on roof and rib stability. Overbreak can damage the roof and rib rock, while bootlegs (poor rock breakage at the end of a blasthole due to inadequate explosive burn) can leave broken rock along uneven rib and face surfaces.

Figure 4. Smooth roof line produced by a persistent bedding plane lamination within the roof rock beam

Table 6 - Factors associated with the 9 metal/nonmetal underground fatalities, 1996-1998.

Date	Commodity	Factor	State	Type of fall	Job	Mining height, m
11/4/98	Metal	inadequate examination/failure to remove loose ground	CO	Rib	Driller	3
3/4/98	Metal	failure to remove loose ground	AZ	Roof	Installing support	NA
1/19/98	Metal	failure to support or remove loose ground	MO	Roof	Surveying	4.9
4/1/97	Stone	inadequate examination/geology	TN	Rib	Driller	7.6
2/5/97	Metal	unsafe location	NV	Roof	Scaling	2.4
2/3/97	Metal	large span/geology	TN	Roof	Driller	5.5
7/24/97	Metal	unsafe practice/loose ground	NV	Rib	Driller	12.2
5/10/96 (double fatal)	Stone	failure to support loose ground	MO	Roof	Blasters	7.6

Scaling. Scaling is necessary to remove loose rock from the sidewalls (rib) and hanging walls (roof) of mine openings. It is particularly important when the rock and ore are removed by blasting, as in most underground metal and nonmetal mines.

Scaling may be conducted either with a hand-held pry bar or with mechanical equipment. Mechanical scalers usually remove the greatest portion of the loose rock, using an assortment of prying, or hammering scraping attachments. Hand scaling is often conducted by a worker mounted in a lift basket in high openings.

A study of accidents in underground stone mines between 1985 and 1994 found that nearly one-third of the ground control injuries involved scaling (Grau and Prosser, 1997). More than 90% of these involved hand scaling. Mechanical scaling generally affords greater protection, because the miner is positioned in a protective cab at a greater distance from the loose rock.

The data from this study also showed that the extremities and limbs were the body parts most often injured during scaling. Arm and leg padding, such as worn by athletes, may be one way to cushion the blow from falling rock and may also lessen the severity of an accident.

Global Safety Strategies

Best practices, as discussed in the previous section, generally address ground control safety in the immediate vicinity of the miner. Creating a stable mine environment begins much earlier, however, during the process of mine design. Ground monitoring can also be central to the creation of a ground control safety culture at a mine.

Safe Mine Design: Mine design includes pillar sizing, layout of drifts and entries, dimensions of openings, and artificial support. Mine planners seek optimum designs that balance the competing goals of ground control, ventilation, equipment size, production requirements, and costs. In recent years, a number of design aids have been made available to assist with the ground control aspects of design.

The role of pillars is to support the great weight of the overburden above the mine. No man-made supports (except filled stopes in metal mines) have anything near the tremendous load-carrying capability of mine pillars. Longwall panel extraction, pillar recovery, and multiple seam operations can all increase pillar loads, and benching can reduce pillar strength.

Mining layout can often be used to minimize the effects of geologic hazards. Traditionally, features such as joints, cleats, and faults have been considered in design. More recently, horizontal stress has become an important concern. Global plate tectonics are the primary source of horizontal stress in mines, and measurements have shown that horizontal stresses are often three times as great as vertical overburden stresses. Horizontal stresses have caused roof potting, cutter roof, and roof falls in coal and limestone mines (Mark and Mucho, 1994; Iannacchione, 1998b). Their destructive effects can be reduced by orienting the mine so that most of the drivage parallels the direction of the maximum horizontal stress.

The maximum stable size of mine openings depends greatly on the geology. The back in some stone and salt mines is so competent that it can routinely maintain spans of 15 m (45 ft), while 5 m (15 ft) spans may be unstable in the weak, fractured ground found in some coal and hard rock mines.

U.S. mines use more than 100 million roof bolts every year. Only mines with exceptionally competent country rock can do without pattern roof bolting, and even they require some spot bolting. A wide variety of rock bolts are available, but matching the proper bolt type and pattern with the ground conditions remains as much an art as a science (Mark, 2000).

Controlling Catastrophic Failures in Underground Mines: Catastrophic failures that create hazards for miners in coal, metal, and nonmetal underground mines include coal mine bumps, hard rock bursts, large collapses, and outbursts. Hazards to miners range from injuries associated with flying rocks to complete burial in ejected rock. Pressure waves from large collapses can throw miners into natural and manmade structures. When large quantities of gas are instantaneously released, gas ignition or asphyxiation can occur.

Coal mine bumps have presented serious mining problems since the early 1900's. In 1996, 3 miners were killed in two different bump events. Two Kentucky miners were fatally injured when six pillars suddenly failed violently during pillar recovery operations. The second event claimed the life of a Utah miner when coal along a longwall face violently ejected into the shields. Both of these events occurred in characteristic settings for coal bumps, with elevated overburden, proximity to a gob area, and a strong hanging roof.

Hard rock bursts have been occurring in deep metal mines for as long as records have been kept (White et al., 1995). Federal regulations have been developed mainly in the form of administrative controls (subsection 57.3461). When a rock burst causes miners to withdraw, impairs ventilation or impedes passage, MSHA must be notified. A rock burst control plan should then be developed and implemented. This plan is required to reduce the occurrence of rock bursts through monitoring and minimizing exposure. Monitoring can range from simple deformation measurements to mine-wide microseismic monitoring systems. Minimizing exposure can range from administrative controls to the use of remote controlled equipment.

A *pillar collapse* is a sudden, violent event that can pose a serious hazard in a room and pillar mine. A collapse occurs when one pillar in a mining layout fails, transferring its load to neighboring pillars, causing them to fail, and so on in a domino fashion. A pillar collapse can induce a devastating airblast which can disrupt the ventilation system and send flying debris that can injure or kill miners. In recent years, at least 13 coal mines and 6 metal/non-metal mines in the U.S. have experienced pillar collapses. Fortunately, only one fatality has resulted, following a collapse of hundreds of pillars at a Wyoming trona mine (Zipf and Mark, 1997).

Outbursts of gas and rock have occurred mainly in evaporite and to a lesser degree coal mines. With the occurrence of the multiple fatal explosion at the Belle Isle Salt Mine in 1981, domal salt mines in Louisiana and Texas were recognized as potential locations for large outbursts. Modifications to mining regulations were made in 1984 creating special levels of gassy metal/nonmetal mines (Subcategory II-A and II-B, 57.22003). Each advance in the gassy level requires additional operational safeguards. Outbursts in Canadian bedded salt and New Mexico potash mines have periodically created serious safety hazards.

Roof Monitoring. Roof falls seldom occur entirely without warning. Often, however, miners are not aware of the warning signals until it is too late. Most underground mines use observational techniques, primarily visual inspection, as a means of determining roof stability. Traditionally, miners have sounded the rock, listening for the drummy sounds that signal loose rock. Also the act of drilling exploration roof bolt or blast holes can provide much information about the rock. During drilling, blasting, and scaling operations, additional knowledge related to roof conditions can be gained. For example, a driller preparing to bolt may notice a sudden increase in the penetration rate, and then realize that possibly a gap or clay seam was encountered. Much of this "hands-on" information provides an overview of the general conditions related to roof stability. Observational techniques can be extended by monitoring the movement of the mine roof in boreholes using mechanical tools (Figure 5).

A comprehensive ground control plan not only includes the basic observational, visual, and hands-on components, but also uses supplemental observational and monitoring techniques and regularly reads, analyzes and displays information gained from these efforts. Mines that follow these practices and promote open communication and participation from everyone at the site are the mines with the most pro-active approaches towards ground control safety.

SURFACE HIGHWALLS AND SLOPES

Surface mines have relatively few serious falls of ground, with 6 fatalities in the period 1996 to 1998. However, six additional fatal falls of highwalls and slopes fatalities occurred in the first half of 1999. Two of these six were initially classified as Powered Haulage, but they were actually caused by slope failure beneath haulage equipment. The large jump in fatalities in 1999 is hopefully an aberration, but it may signal a new safety issue caused by a change in mining method or equipment, different enforcement practices, or a social issue such as the experience level of the mining workforce.

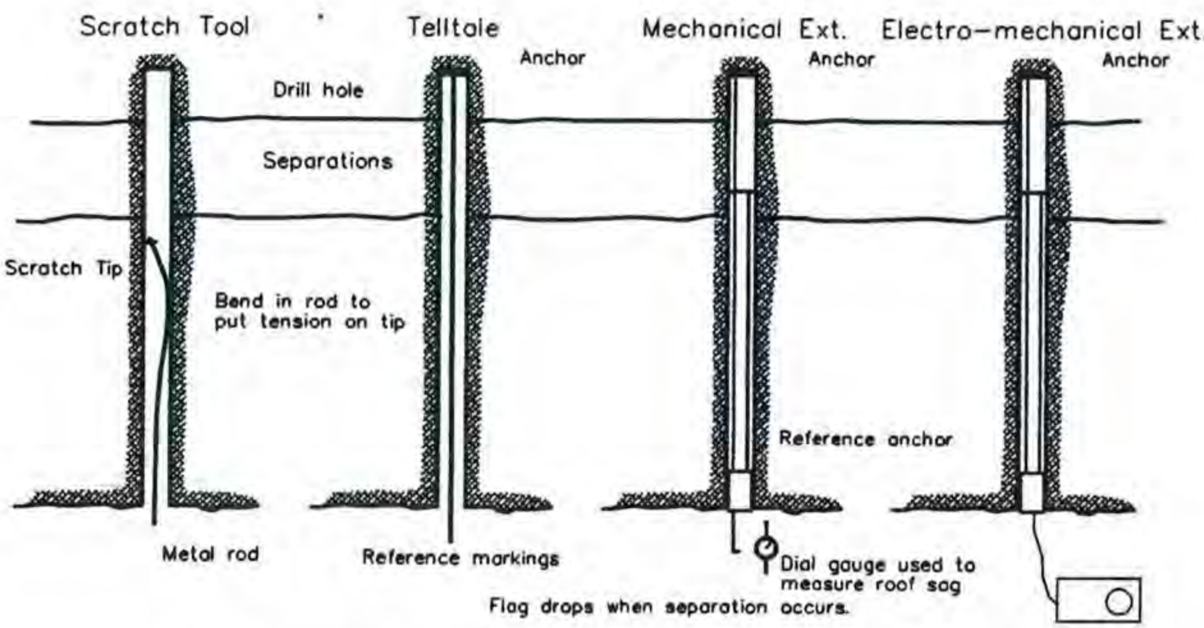

Figure 5. Four techniques used for roof monitoring

Most highwall injuries occur when loose pieces of rock fall on workers located below. Small pieces of rock can be dangerous when they fall from great height; even a fist-sized rock caused one recent fatality. At the other extreme, an entire section of a highwall or spoil pile may collapse, endangering miners working either on or beneath it.

Good basic design is essential to highwall safety. The height should be limited for stability and to allow scaling. Where the pit is deep, benches should be used to limit the slope height. Angling the highwall back from vertical also increases stability. Good blasting practices make for a smoother wall and reduce the need to scale. Drainage ditches should be used to divert springs and groundwater away from slopes.

Geologic features have contributed to many rockfall injuries from highwalls. Faults or "hillseams" (weathered joints) can create wedges of unstable ground that can slide into the pit. In dipping strata, the rock can also be prone to slide along bedding planes. Freeze-thaw action acts to loosen rocks, and has been cited in several fatality reports. A review of accident records indicates that highwall accidents are twice as likely to occur in December and January than they are in the summer months. The presence of abandoned underground mine openings in the highwall has contributed to three of the recent fatalities.

Rock faces should be monitored frequently to check for loose rocks, and scaling should be conducted as needed. As highwalls age, weathering may cause additional loosening. The surface at the top of the highwall should also be checked for tension cracks that could indicate pending massive slope failure. In very large pits, various kinds of electronic surveying and monitoring systems are in use to provide early warning.

CONCLUSIONS

This paper has presented an overview of the most significant ground control hazards facing today's mineworkers. Underground miners, particularly in coal mines, are at the greatest risk from ground falls. The six highwall and slope fatalities that occurred in the first half of 1999 show that surface miners are at risk as well.

The analysis of recent fatality investigations and accident statistics identified certain job categories, mining techniques, and geologic environments that appear to pose the greatest hazards. Best Practices have been developed through experience and research to reduce these risks. They combine engineering design, roof support, equipment, mining methods, and human factors to create safer workplaces and work practices. The Roof Control Plan is another valuable tool in this effort.

Unfortunately, recent trends indicate that ground fall injury rates have stopped decreasing, and may even be on the increase. A renewed effort by the entire mining community will be necessary to finally eradicate the groundfall hazard.

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REMOTE SEALING AS A MINE FIRE CONTROL TECHNIQUE

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ABSTRACT

Over the last 12 years there have been numerous underground coal mine fires that have resulted in the permanent sealing of the coal mine to extinguish the fire. Mine fires can best be fought from underground when the fire is still relatively small. However, once the fire has grown to a size were it has been determined that the fire can not be safely fought directly in the mine, then it must be fought from the surface. When fighting the fire from the surface, the task becomes more difficult because of the lack of updated information on the fire. The primary reason for this is due to limited access into the mine, (i.e. shafts, drifts, and boreholes around the fire area). This makes it difficult to determine the intensity, location and spread of the fire. Therefore, once fighting the fire from the surface becomes necessary, the first order of business is usually the drilling of strategically places boreholes to obtain updated information on the fire. The number and location of boreholes is often driven by physical factors (topography, depth of cover, overlying mines, and surface access) and determined optimum locations to obtain information on the fire.

In addition to using boreholes to collect information on the fire, boreholes have often been used to fight the underground fire remotely either by extinguishing and/or isolating the fire area. Boreholes have been used to deliver water and inert gas to the fire area to try to extinguish it. To isolate the fire area, a pumpable material is delivered through the borehole into the mine entry to create a remote plug or seal. In many cases, the results have been disappointing which have resulted in the permanent sealing of the coal mine and the laying off of its work force. This paper provides details on four attempts to use boreholes to create a remote plug or seal to isolate a mine fire.

INTRODUCTION

Underground coal mine fires continue to occur, but fortunately most of the fires are normally extinguished in their incipient stage. When the fire has not been able to be extinguished and personnel have been evacuated from the mine, the mine fire often has caused a loss of either the entire mine or a portion of it. Information on the fire's status becomes very limited once personnel have been evacuated from the mine.

Boreholes often become a critical link into obtaining information on the mine fire. However, the borehole is still a limited source of information because it is a single point in a multiple entry mine that indicates the conditions encountered were it enters the mine. Some of the conditions in the mine entry effecting information obtained from boreholes are water, roof falls, air quantity and air direction.

However, boreholes provide another useful function in fighting the fire remotely either via water injection, inert gas injection or isolating the fire area. Numerous attempts have been made to use boreholes for those purposes. The number and location of boreholes are often driven by topography, depth of cover, overlying mines, and surface access instead of the best location for fire fighting activities.

This paper provides a short review of four mine fires and the associated details on the attempts by mine management to use boreholes to create plugs for fire fighting. The four mine fires in which the attempt will be detailed are: BethEnergy Mines, Incorporated Marianna Mine No. 58 fire in March, 1988; Mathies Mine fire in October, 1990; Consolidation Coal Company's Loveridge No. 22 Mine fire in June, 1999; and Mountain Coal Company's West Elk Mine fire in January, 2000. The four mines used slightly different strategies for the creation of their plugs. Mathies mine management developed a concept of using multiple boreholes to plug two entries in its relatively shallow overburden mine. The other three relied on a single borehole to distribute the plugging material across the entry. Even though the technique of delivering the material was the same, the plugging material varied along with the distribution technique.

DISCUSSION

BethEnergy Mines, Incorporated's Marianna Mine No. 58

BethEnergy Mines, Incorporated's Marianna Mine No. 58 was located in Marianna Borough, Washington County, Pennsylvania. A fire was found at a belt drive in the 3 Northwest submains at about 10:20 P.M. on March 7, 1988. Initial fire fighting efforts made by miners were with fire extinguishers. As time passed and more miners became involved in fighting the fire, water, rock dust and foam generators were used. Those methods were ineffective, and on March 8 at 9:00 P.M. fire fighting activities ceased underground and everyone was evacuated from the mine. Fire fighting activities continued from the surface. Approximately 8 million gallons of water was pumped through a borehole into the fire area on March 13 through March 15. Nineteen additional boreholes were drilled into entries around the fire area to have material pumped into the mine entry to impound the water. On March 17, it was decided to convert the water retaining dams to seals in order to isolate the fire area from the rest of the mine. Twenty-two locations needed to be sealed to isolate the fire area, thus requiring an additional three locations to be drilled and pumped with material. Borehole depth varied slightly with most holes around 850 feet in depth.

Mine management used a mixture of limestone and cement to pump into the mine entries to initially form water retaining dams which were then reclassified to be used as seals. The cement mixture consisted of a ratio of 1 part cement to 3 parts sand. Initially, the aggregate size was between 1.0 to 1.5 inches but was changed to be < 1.0 inch with 25 percent passing through a #16 sieve. The cement mixture was pumped into each of the boreholes at rates ranging from 20 to 100 tons per hour. It was estimated that 160 tons of material was needed to be pumped into each borehole to form a 40 foot plug. The set time was estimated at 3 to 4 hours per hole. One of the entries was attempted to be plugged using polyurethane. Approximately 22,500 pounds of polyurethane was injected into the entry.

The method used to distribute the limestone and cement mixture into the mine entry was through a borehole drilled into the entry with a six inch diameter pipe inserted into the borehole. On the discharge end, a flexible elbow system which was able to rotate 160 degrees was used. The elbow was hinged to enable it to be lowered into the borehole to the mine entry. A cable, connected to the end of the elbow, was used to raise the end of the elbow to its proper position, 90 degrees from vertical. The distribution method employed was to allow the material to "spray" from the end of the elbow into the entry which was to be gradually rotated left and right to assure that the material was distributed throughout the entry to create the plug.

Nineteen of the twenty-two plugs initially were used as water dams. These 19 plugs were re-designated to be used as air seals with three additional plugs installed to create an air seal around the fire area. The amount of material pumped into each entry to create the plug varied considerably, ranging from 20 tons to 410 tons. It was determined from monitoring the mine atmosphere, inby and outby the plugs, that they were ineffective. On April 4, to further confirm the air communication inby and outby the fire area, nitrogen was injected into the fire area. Nitrogen was immediately measured in air samples collected outby the fire area.

The mine was re-entered on April 4 to physically examine the plugs. The rescue teams could not reach the plugged area because they encountered roof falls, roof and rib sloughage, water, and smoke which essentially prevented all routes of ingress to the plugged area.

Because of the ineffectiveness of seals and the underground conditions encountered by the mine rescue teams in their attempt to reach the plugs, a decision to seal the mine was made by the mine operator. On April 6, the surface openings of the Marianna mine were sealed.

Mathies Coal Company's Mathies Mine

Mathies Coal Company's Mathies Mine was located in Courtney, Washington County, Pennsylvania. A fire was discovered near the Thomas Portal in the track entry at approximately 5:00 A.M. on October 17, 1990. Fire fighting activities consisted of using fire extinguishers and water cars. An ignition of an unknown source occurred at 2:20 P.M. which caused injuries to seven fire fighters. Mine management decided to cease fire fighting activities and seal the mine openings.

On January 20, 1991, mine management decided to try to reopen the mine by isolating the fire area from the available coal reserves. This consisted of exploration and the construction of permanent seals both from underground and from the surface by the use of boreholes to pump material to create a plug. The remote pumpable seals were started during the week of March 4. On April 12, the permanent seal construction was determined to be successful and MSHA's role in the recovery operation ceased.

Mathies mine management's plan of installing two pumpable seals was designed with a knowledge that a shallow overburden depth, approximately 100 feet, needed to be drilled.. The entry width was between 10 to 14 feet. As shown in figures 1 and 2, the pumpable seal design required eight boreholes for each seal, plus an additional borehole inby and outby each seal for monitoring seal effectiveness.

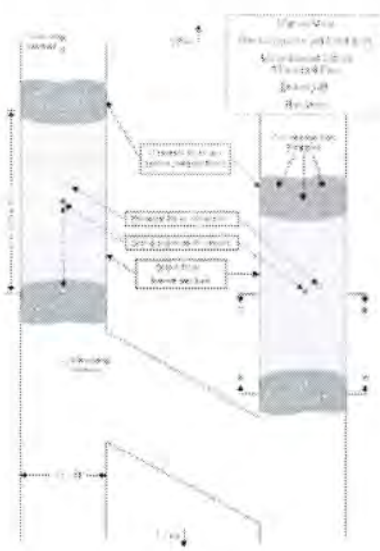

Figure 1. Plan View of Mathies Remote Seals

Each seal was designed to have two rows of boreholes separated by 30 to 35 feet. Each row consisted of three boreholes with a borehole in the center of the entry and a borehole placed approximately four feet on each side of the center borehole. These boreholes were pumped with a mixture of water, cement, sand and flyash. The ratio of cement, sand and flyash was approximately 15:2:5, respectively. The seals were used as end caps for the material injected in the middle.

Two boreholes were drilled between the end caps. One of the holes was used for material injection while the other was used for air and water pressure relief. The grout material injected into the void between the end caps consisted of a cement and flyash mixture. It was pumped through a pipe positioned in the bottom of the

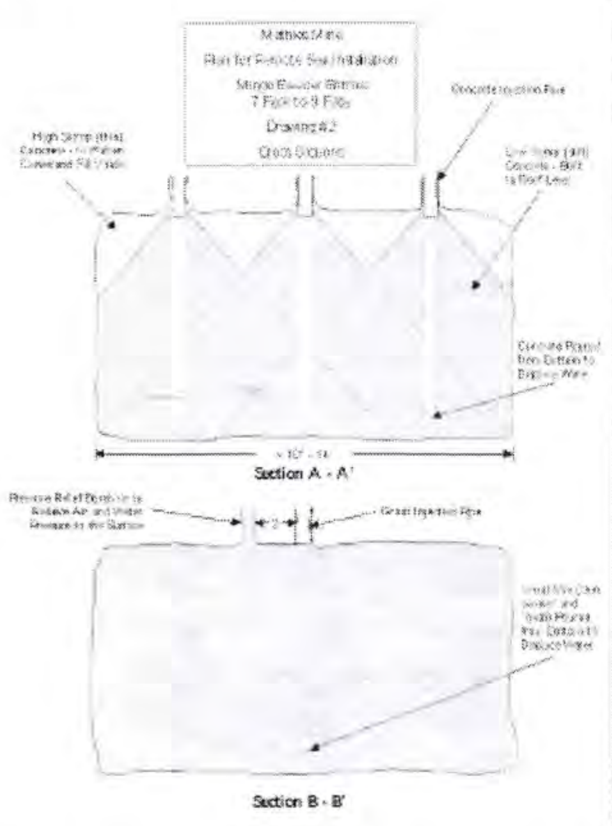

Figure 2 Cross-section of Mathies Remote Seals

entry to allow for the water and air to be displace through the other borehole. Atmospheric monitoring from inby and outby of the pumped seals indicated that they did function as intended and provided separation.

Consolidation Coal Company's Loveridge No. 22 Mine

Consolidation Coal Company's Loveridge No. 22 Mine fire was identified at 1:15 A.M. on June 22, 1999. The mine was located near Fairview, West Virginia. A fire was discovered in an entry adjacent to the gob of a retreated longwall panel completed in 1997. Efforts were made by miners using fire extinguishers and water to extinguish the fire. However, these efforts were unsuccessful and it was decided to evacuate the mine. The mine was evacuated by 2:30 A.M.. Mine management decided to seal the mine surface openings. Those openings were sealed by 9:30 P.M.

Mine management decided to pump inert gas into the fire area and try to isolate that area from the rest of mine by having boreholes drilled at strategic locations. Nitrogen was introduced into the fire area on June 24 at a flow rate of approximately 2,000 cfm as depicted in figure 3. Nitrogen flow into the fire area reached a rate as high as 3,000 cfm. When the supply of nitrogen started to become an issue do to its availability, carbon dioxide was then combined with nitrogen and injected into the borehole. A total volume of nitrogen and carbon dioxide injected into the fire area was 62,122,000 cubic feet and 13,754,000 cubic feet, respectively.

To isolate the fire area, a submain consisting of seven entries and three bleeder entries had to be plugged. A single borehole was drilled into intersections of each adjacent entry and used to pump material into each entry to form a plug. The average depth of each borehole was approximately 1,000 feet.

Figure 3 Inert Gas Injection at Loveridge Mine

Mine management developed a pumpable seal plan that used a single borehole per plug in conjunction with a slurry mixture that was induced at high pressure into the entry. The slurry mixture was made up of flyash, cement, water and sodium silicate. As shown in figure 4, a pipe line system was developed to direct the slurry from the mixing truck to the appropriate borehole. It was thought that the slurry would exit the borehole pipe under pressure, hitting the mine floor and splash in all directions. No more than approximately 660 cubic feet of slurry material would be pumped into a borehole at a time. Mine management thought that the mixture would take the form of a bowl. The slurry was allowed to set up for a minimum of 10 hours before pumping was continued again in the hole. However, after each time slurry material was pumped into the borehole, the line and hole was flushed with approximately 250 gallons of water. It was estimated that each hole would require approximately 5,200 cubic feet of slurry material per plug.

On July 5, the seven submain entries were plugged with material. A total of 1,590 tons of material was injected into those holes. On July 13, the three bleeder entry plugs were finished. A total of 990 tons of slurry material was injected.

Each set of plugs had an inby and outby borehole established to monitor the mine's atmosphere to make a determination if the plugs were successful at preventing air communication. Air quality measurements between these set of boreholes suggested that the plugs were unsuccessful, however they should provide restriction to air flow to the fire area when recovery of the mine is attempted in July, 2000.

Figure 4 Surface Seal Construction Site

Mountain Coal Company's West Elk Mine

Mountain Coal Company's West Elk Mine fire was identified on January 28, 2000. The mine was located near Somerset, Colorado. The topography of the area is mountainous. An elevated carbon monoxide measurement in the Lone Pine fan house provided the first indication to mine personnel that a problem existed. Mine personnel were sent to investigate and found elevated carbon monoxide in an air course coming from an area of the mine that was not travelable. Spontaneous combustion was believed to be a potential source of the elevated carbon monoxide concentrations. Mine personnel erected temporary seals (wood frame checks) on the outby side of the heating area to reduce the amount of air flowing toward and through the area in question. Upon completion of the temporary seals, mine personnel were withdrawn from the mine.

Since the fire area was not travelable, mine personnel decided on using strategically placed boreholes to identify the spontaneous combustion area (figure 5). The terrain over the suspected fire area was extremely rugged and presented quite a challenge to the operator in locating possible sites for boreholes. Borehole depth varied, depending on topography, but most

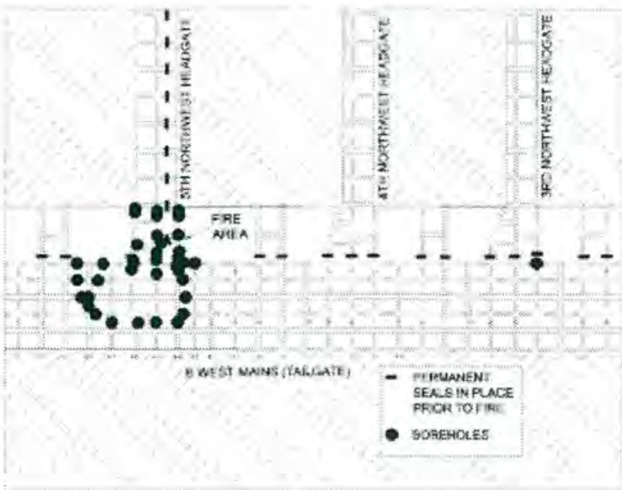

Figure 5 West Elk Mine Map Excerpt

of the holes were around 1,200 feet deep. Approximately ten boreholes were needed to be drilled to identify the fire area. Mine personnel decided on a course of action of using pumpable seals; water, nitrogen and foam injection to isolate and extinguish the fire. The pumpable seal was demonstrated on the mine's surface facilities as shown in figure 6.

The first borehole intersected the mine on February 9, 2000. A total of 43 holes were drilled over the course of the project. There were as many as three separate drill rigs in operation at one time. By analysis of the mine air samples taken from the boreholes, the location of the fire area was narrowed to the three gate entries between B West Mains and 5th NW and 6th NW longwall panels, which were mined in 1995. The mine had a dip of approximately three to five degrees into that longwall gob area from the B West Mains. Mine management determined that if the remote plugs could be pumped into these three entries, down dip from the fire area, water could be pumped into the mine to flood the fire area. The plugs that were pumped in would not have to be airtight, but rather hold enough water to flood the area. The system would work if the water could be pumped into the mine at a rate faster than the plugs leaked.

Figure 6 Surface Seal Pumping Demonstration

The remote plugs were pumped into single holes targeted for the center of the entry. The material consisted of a mixture of cement, fly ash, sodium silicate and water. The constituents were mixed on the surface at the borehole site and pumped through piping and the borehole into the mine entry.

The capacity for injecting water was initially around 700 gpm. After a couple of weeks, it was decided that an additional water line could be installed to double the pumping capacity. At this same time, it was determined that the plugs would be better able to dam up the water if bentonite, a clay like material, was added to the water. The bentonite would help to plug fissures in the remote plugs. It was after these changes were made that the water levels climbed and the water impoundment flooded the fire area. Since these two procedural changes both took place in the first week of April, it is hard to determine how much effect each one individually had on the flooding process.

While the remote seals and flooding process was underway, the company monitored the other available boreholes for temperature and where applicable, the water level. There was concern over the high temperatures and fire gasses in the holes located to the south of the suspected fire area toward the B West Mains. As a result of this concern that the fire area could be spreading south into the mains, additional remote plugs were planned to allow the water impoundment to extend in to the B West Mains. A total of sixteen remote plugs locations were needed to completely impound water in the expanded fire area (figure 7).

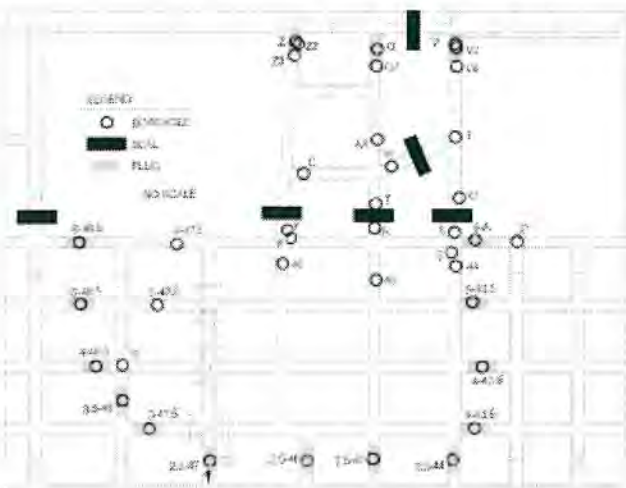

Figure 7 Final water impoundment area

The process of pumping each plug required from 50 to 350 tons of material. The minimum amount of material for a seal was pumped to location(s) that was on top of a fall with only a small gap in the strata to fill. The procedure was to spend a day stocking the borehole site with supplies, and then pump the material the next day. So the progress rate was about one remote plug every two days.

In addition to the water inundation, West Elk Mine pumped nitrogen gas into the boreholes and a mix of foam generating solution with some of the water being pumped into the mine. The nitrogen used was produced at the site of the boreholes by a nitrogen generator. The equipment utilized a process that enriched the airflow with a final nitrogen content approaching 95 percent. By April 2, the nitrogen flow rate was reported to be 700 cfm and was being injected into two boreholes. The flow rate was increased to 900 cfm by the next day. However, it was decided that the nitrogen would also increase the effectiveness of the foam mixture and it was diverted into the foam lines. From this point on, the majority of the nitrogen introduced into the mine was in the foam mixture. The target mixture for the foam and nitrogen was to have 1 cfm of nitrogen for each gallon per minute of foam.

On May 3rd the last remote seal was pumped and the impounded water had reached a level to completely cover the suspected fire area. The analysis of air samples indicated that the mine atmosphere had returned to normal ambient levels.

CONCLUSION

To date, there has reportedly been only two successful attempts to combat a mine fire by using boreholes to remotely pump material into an entry to isolate the fire area. The first application that worked was used at the Mathies Mine Fire. Again, it is important to note that eight holes were used to construct the plug in each of the mine's entries and the overburden was only approximately 100 feet.

The other successful application of using boreholes to fight a mine fire was the plugs constructed at the West Elk Mine fire. The remote pumped plugs impounded water sufficiently to allow the water to build to a level that flooded the fire area. This was accomplished after the doubling of the water volume being pumped into the fire area and the introduction of bentonite to reduce the amount of water leakage by filling suspected fissures in and around the seals.

Remote seals have proven in some cases to be an integral part of a successful mine recovery operation. They have allowed operators to reduce the amount of time and cost of associated mine recovery. Remote plugs have been used in areas of mines that were deteriorated and unsafe for travel for underground miners. As material and application technology improves, it is expected that remote seals will be used more effectively in combating mine fires.

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TECHNICAL SESSION II-B

DEVELOPMENTS IN SURFACE MINING AND QUARRYING

Session Chairs

Ronald Mullins
Safety Engineer Senior
Virginia Department of Mines, Minerals and Energy
Division of Minerals Mining

Polly Hester
Safety and Human Resources Coordinator
Salem Stone Corporation

UPDATE ON PART 46 TRAINING PROGRAMS

Cline Dooley

Boxley Company, Inc.

October 2, 2000 - "Good morning, my name is John Doe, an inspector with the Mine Safety and Health Administration. I am here to do an inspection of your mining operation, but first let's look at your Part 46 training plan and the training records and certificates that go along with your plan. You do have a Part 46 training plan, don't you?" If your response to this question is "Do what?", "Huh?", or "Are you crazy? You can't check my training records!", then you, the production operator, need to keep reading.

Beginning October 2, 2000, MSHA will enforce Part 46 training regulations for miners and other persons engaged in shell dredging, sand, gravel, surface stone, surface clay, colloidal phosphate, and surface limestone mines. MSHA has not been able to enforce the old Part 48 training rules for years because of a Congressional spending prohibition that did not provide funds for the enforcement of these rules. When the clock strikes midnight on October 2, surface mine operators around the country are going to need answers for those MSHA inspectors asking to see a Part 46 training plan.

THE TRAINING PLAN

MSHA requires that you develop and implement a written training plan for training newly hired experienced and inexperienced miners, training miners for new tasks, annual refresher training, and site-specific hazard awareness training. This plan has to be submitted to MSHA for approval unless it contains the following information:

- the name of the production operator or independent contractor;
- the name of the mine;
- the MSHA mine ID number or independent contractor ID number;
- name and position of the person or organization who is responsible for the health and safety training at the mine;
- a general description of the teaching methods and the course materials that are to be used including the subjects and the approximate time spent on each subject;
- a list of the persons and/or organizations who will provide the training, and the subject areas in which they are competent to instruct; and
- the evaluation procedures used to determine the effectiveness of the training.

Plans that contain all of this information are already considered approved and do not need to be submitted to MSHA for approval. The only requirement for plans not submitted to MSHA is that a copy be given to the miners' representative at least two weeks before the plan is implemented. If your mine does not have a miners' representative, then the plan needs to be posted where all miners will see it or a copy given to all miners. This training plan must be maintained at the mine site for MSHA inspectors to review. If the plan is not kept at the mine site, it must be made available within one business day upon a request from an MSHA inspector.

NOW WHAT DO I DO WITH THIS TRAINING PLAN?

Now that you have a written training plan, **<u>you must follow it</u>**! MSHA inspectors will not only be looking for a written training plan, but they will also be making sure that you are doing what your plan says you will do. For example, if your training plan says that your electrician is the only person competent to train on electrical hazards and you let your pit foreman do it, then you could run into some problems. You must also make sure that the training is done by a competent person and that all of the miners understand the training. Mines that employ persons that may be blind, deaf, speak a different language, etc. may have to make special arrangements to ensure the training is understood by these employees.

Part 46 does give you, the production operator, a lot of flexibility in what training methods you want to use. You may conduct your own training sessions or arrange to have another agency or organization do them for you. The only requirement is that the training be done by a competent person or organization and the training be documented.

NEW MINER TRAINING

Part 46 says that you have to give training to newly hired experienced and inexperienced miners. Inexperienced miners shall receive at least 24 hours of new miner training. At least 4 hours of this training has to be provided on the subjects in 46.5(b) before the new miner begins work at the mine. The subjects of 46.5(c) must be covered within 60 calendar days of the new miner beginning work at the mine. And finally, the remainder of the 24 hours must be provided within 90 days of the new miner beginning work at the mine. The flexibility here is that only nine subjects are mandatory to be taught to the new miner. The remainder of the training can be on any subjects that promote safety at the mine. It is important to note; however, that miners who have not completed the whole 24 hours of new miner training must work where an experienced miner can observe that the new miner is performing his/her work in a safe and healthful manner.

Part 46.6(a) says that you must give all newly hired experienced miners training before they begin work at your mine. There are seven subjects in 46.6(b) that must be covered before the experienced miner begins work, but there is no time limit on this training. Now what exactly is an experienced miner? Part 46.2d(1) defines an experienced miner as:

- A person who is employed as a miner on April 14, 1999;

- A person who has at least 12 months of cumulative surface mining or equivalent experience on or before October 2, 2000;

- A person who began employment as a miner after April 14, 1999, but before October 2, 2000 and who has received new miner training under § 48.25 or under proposed Part 46 requirements published April 14, 1999; or

- A person employed as a miner on or after October 2, 2000 who has completed 24 hours of new miner training under § 46.5 or § 48.25 and who has at least 12 cumulative months of surface mining or equivalent experience.

Once a miner meets one of the criteria listed above for an experienced miner, the miner will retain that status permanently.

A smart approach to new miner training might be to do all of the training "up front". Instead of trying to remember if the 60 or 90 calendar days are up, it would be a whole lot easier and simpler to do the training during the first 24 hours that the new miner works. Another smart approach to the newly hired experienced miner training is to treat all miners as if they are inexperienced miners. That way all new miners receive 24 hours of training no matter how experienced they are. This may save you from having to prove to an MSHA inspector that someone was an experienced miner when they came to work for you and that was why you only gave them 8 hours of training instead of 24.

NEW TASK TRAINING

"Go grab that loader and meet me in the bottom of the pit." If you have ever said this to one of your employees that you know has never operated a piece of heavy equipment before, then 30 CFR Part 46.7 needs to be part of your bed time reading before October 2, 2000. Part 46.7a states that you must provide any miner who is reassigned to a new task in which he or she has no previous work experience with training in the health and safety aspects and safe work procedures specific to that new task. In other words, "Go grab that loader" isn't going to fly any more.

If a miner has never performed a certain task before, then you must give him or her new task training before the miner performs the task. Also, if a change occurs in a miner's assigned task that affects the health and safety risks encountered by the miner, then you must provide the miner with additional new task training. You may give the miner hazard recognition training and then conduct the remainder of the training under the close observation of a competent person if you so choose. This is probably the best approach to task training a miner. It is hard to teach a miner in a classroom how to operate a pit loader and load shot rock. Put that miner in a loader in a remote part of the pit with some shot rock and let him practice while a competent loader operator is observing him. Eventually bring in a haul truck and let him or her practice loading the truck. Now that is effective training.

When do you not have to provide task training? Task training is not required when the miner has received training in a similar task, has previous work experience in the task, and can demonstrate the necessary skills to perform the task in a safe and healthful manner.

ANNUAL REFRESHER TRAINING

"If I provide a new miner with 24 hours of training and then task train him on any new tasks that he may perform, then I don't have to train him any more, do I?" You can find the answer to this question in Part 46.8a which states you must provide each miner with no less than 8 hours of annual refresher training:

- no later than 12 months after the miner begins work at the mine, or no later than March 30, 2001, whichever is later; and
- thereafter, no later than 12 months after the previous annual refresher training was completed.
- "Is there anything that I have to train on or can I pick my own subjects?" The only two things that Part 46 says you must train your employees in are:
- changes at the mine that could adversely affect the miner's health or safety; and
- other health and safety subjects that are relevant to mining operations at the mine.

For example, if you have put new blasting procedures in place at your mine, then that would definitely have to be covered in your annual refresher training. Part 46.8c contains a whole list of recommended subjects for the annual refresher training. This would be an excellent place to start if you are putting together an annual refresher training plan.

The most effective annual refresher training is site-specific training held on a routine basis. How many times have you heard this quote: "Here we go again with the same old training videos for eight hours. It is nap time!!" A training video with actors that no one knows does not have the same impact as a training video that was made at your mine site with your actual miners playing their roles. The same can be said for pictures from other mines that are showed to your miners for training purposes. Now show those same miners pictures of them from your mine and you have got their attention for the whole training session. Also, an eight hour day filled with videos or someone lecturing on safety is not as effective as monthly safety meetings that last approximately one hour. These monthly safety meetings are a constant "safety" reminder to your employees. Conducting all of your training in one 8 hour session can be seen by your employees as: "THEY are only doing this because MSHA is making THEM. THEY really don't care about us. THEY just want to show us these videos and get the 8 hours over with."

Part 46 gives you a lot of flexibility in the annual refresher training of your employees. Use this flexibility to provide your miners with site-specific training that will keep them the safest on your mine site.

SITE-SPECIFIC HAZARD AWARENESS TRAINING

Before a visitor on your mine site is exposed to mine hazards, you must provide them with site-specific hazard awareness training. You must provide this training to any person who is not a miner but is present at a mine site including:

- office or staff personnel;
- scientific workers;
- delivery workers;
- customers, including commercial over-the-road truck drivers;
- construction workers or employees of independent contractors who are not miners;
- maintenance or service workers who do not work at the mine site for frequent or extended periods; and
- vendors or visitors.

You must also provide site-specific hazard awareness training to miners who may move from one mine site to another while being employed by the same production operator or independent contractor. If a person is accompanied by an experienced miner who is familiar with the hazards at the mine site, site-specific hazard awareness training is not required.

Now how do you conduct the site-specific hazard awareness training and what do you use to train your visitors? There is no easy way to conduct site-specific hazard awareness training. Hazard awareness training can be provided through the use of written hazard warnings, oral instruction, signs and posted warnings, or walk around training. As long as you point out to your visitor the hazards that he or she may encounter while on your mine site, your requirements for site-specific hazard awareness training are complete.

RECORDS OF TRAINING

Now that you have conducted your training, how are you going to document it? Training received by a miner must be recorded and certified on a form containing the following:

- printed full name of person trained;

- type of training, duration of training, date training received, and the name of competent person who provided the training;

- name of mine or independent contractor, mine ID number or contractor ID number, and location of training;

- the statement, "False certification is punishable under § 110 (a) and (f) of the Federal Mine Safety and Health Act" printed in bold letters and in a conspicuous manner; and

- A statement signed by the person responsible for health and safety training, that states "I certify that the above training has been completed".

The MSHA form 5000-23 does not have all of the above information to meet the requirements of Part 46. It would be best to make up your own form that contains all of the information above for your company.

You have to have training records on all new miner training, annual refresher training, new task training, and site-specific hazard awareness training. Part 46.9 goes into more detail of when you have to have a record or certificate of the training being completed.

INDEPENDENT CONTRACTOR TRAINING

"You guys must have a Part 46 training plan and your miners must be trained before you can come in here to load our shot." If you tell this to your blasting contractor and he says, "What and the heck are you talking about?", then you need to read Part 46.12 to that blasting contractor. Part 46.12b(1) says that each independent contractor who employs a miner at the mine has the primary responsibility for complying with §§ 46.3 through 46.10, including providing new miner training, newly hired experienced miner training, new task training, and annual refresher training.

What responsibilities do the production operators have with respect to the training of independent contractors? The production operator has to provide site-specific hazard awareness training to the independent contractors and make them aware of their obligation to comply with MSHA regulations, including the new Part 46 training rule. The independent contractor must in turn inform the production operator of any hazards that may be created by the performance of their work at the mine site.

What do you do if an independent contractor who employs miners does not have a Part 46 training plan and the contractor's miners are not trained? Simple, find another contractor who has a Part 46 training plan and whose miners have been trained to do the work for you.

"HELP, HELP, WHERE CAN I GET HELP WITH MY PART 46 TRAINING PLAN?"

There are many resources to help both production operators and independent contractors develop an MSHA Part 46 training plan. The best place to look is on MSHA's home page (www.msha.gov). This page has everything you need to start a part 46 training plan. It has the Part 46 training standards, sample plans, compliance guidelines to answer any questions you may have, materials that MSHA uses to conduct training sessions on Part 46, and it also has some information that just pertains to independent contractors and their development of a training plan. The National Stone Association and the Virginia Aggregates Association also have web pages with Part 46 training assistance. Those web sites are www.aggregates.org and www.vaagg.org, respectively.

So now it is up to you. October 2 is quickly approaching and MSHA is ready to enforce your compliance with Part 46. If your company does not have a Part 46 training plan in effect right now or have not heard of Part 46 until today, you still have time. The best thing to do is get a copy of the Part 46 training regulations, read them, and understand them. Then use the resources provided to you to develop your own Part 46 Training Plan.

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DETECTING PROBLEMS WITH MINE SLOPE STABILITY

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ABSTRACT

Slope stability accidents are one of the leading causes of fatalities at U.S. surface mining operations. The Spokane Research Laboratory of the National Institute for Occupational Safety and Health (NIOSH) is currently conducting research to reduce the fatalities associated with slope failures. The purpose of this paper to discuss some of this research and to present potential new technologies for slope monitoring and design. The paper also briefly discusses various warning signs of slope instability, introduces the most common slope monitoring methods, describes the limitations of various slope monitoring systems, and presents some field results using some of this new technology.

CONSEQUENCES OF SLOPE FAILURES

Whether on the surface or underground, unanticipated movement of the ground can pose hazardous conditions which may lead to endangerment of lives, demolition of equipment, and the loss of property. In the five years since 1995, 33 miners have lost their lives as a result of surface ground control accidents (see Figure 1). As such, the National Institute for Occupational Safety and Health's (NIOSH) Office of Mine Safety and Health Research in Spokane, Washington has initiated a research program with the goal of reducing the number of injuries and fatalities resulting from slope failures at mines.

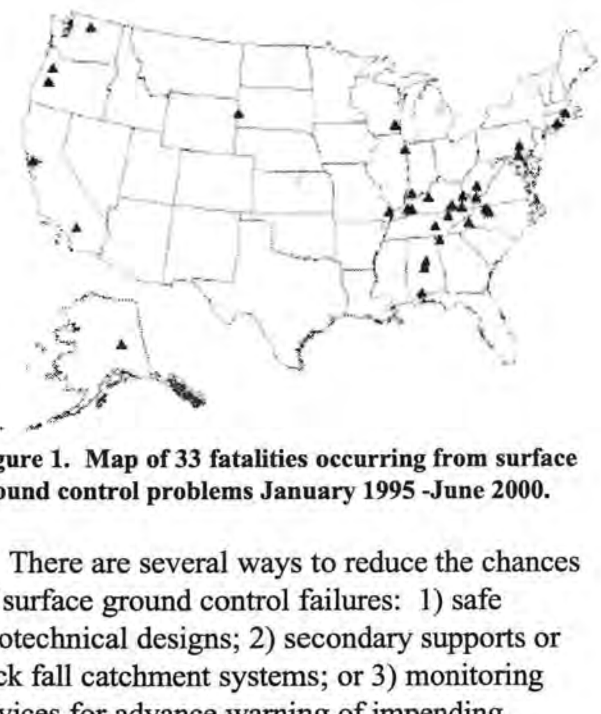

Figure 1. Map of 33 fatalities occurring from surface ground control problems January 1995 -June 2000.

There are several ways to reduce the chances of surface ground control failures: 1) safe geotechnical designs; 2) secondary supports or rock fall catchment systems; or 3) monitoring devices for advance warning of impending failures. While it is important to note that geotechnical designs can be improved to increase factors of safety, proper bench designs can be improved to minimize rock fall hazards, and certain support systems may enhance overall rock mass strength, diligent monitoring and examination of slopes for failure warning signs is the most important means of protecting exposed mine workers. Even the most carefully designed slopes may experience failure from unknown geologic structures, unexpected weather patterns, or seismic shock (Figure 2).

Figure 2. Consequences of unexpected slope failures.

SLOPE MONITORING SYSTEMS

Relative displacement measurements are the most common type of monitoring, complemented by monitoring of groundwater. The most important purpose of a slope monitoring program is to: 1) maintain safe operational practices; 2) provide advance notice of instability; and 3) provide additional geotechnical information regarding slope behavior (Sjöberg, 1996). The following is a list of the most common monitoring systems currently in use and is not intended to be an all-inclusive list of monitoring equipment. Readers interested in a more comprehensive list are referred to Szwedzicki, 1993.

Surface Measurements

Survey Network: A survey network consists of target prisms placed on and around areas of anticipated instability on the slopes, and one or more non-moving control points for survey stations. The angles and distances from the survey station to the prisms are measured on a regular basis to establish a history of movement on the slope. It is extremely important to place the permanent control points for the survey stations on stable ground. The surveys can be done manually by a survey crew or can be automated.

Tension Crack Mapping: The formation of cracks at the top of a slope is an obvious sign of instability. Measuring and monitoring the changes in crack width and direction of crack propagation is required to establish the extent of the unstable area. Existing cracks should be painted or flagged so that new cracks can be easily identified on subsequent inspections. Measurements of tension cracks may be as simple as driving two stakes on either side of the crack and using a survey tape or rod to measure the separations.

Another common method for monitoring movement across tension cracks is with a portable wire-line extensometer (Figure 3).

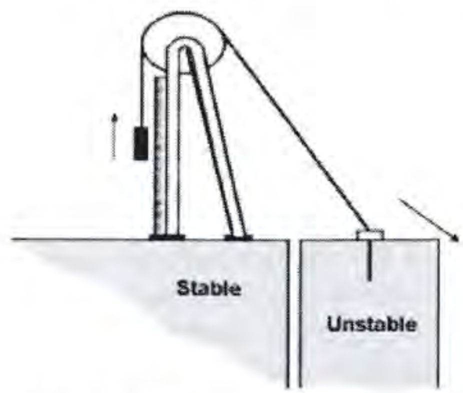

Figure 3. Portable wire-line extensometer for monitoring a tension crack.

The most common setup is comprised of a wire anchored in the unstable portion of the ground, with the monitor and pulley station located on a stable portion of the ground behind the last tension crack. The wire runs over the top of a pulley and is tensioned by a weight suspended from the other end. As the unstable portion of the ground moves away from the pulley stand, the weight will move and the displacements can be recorded either electronically or manually. Long lengths of wire can lead to errors due to sag or to thermal expansion, so readjustments and corrections are often necessary. The length of the extensometer wire should be limited to approximately 60 m (197 ft) to keep the errors due to line sag at a minimum (Call and Savely, 1990).

Subsurface Measurements

__Inclinometers:__ An inclinometer (figure 4) consists of a casing that is placed in the ground through the area of expected movements. The end of the casing is assumed to be fixed so that the lateral profile of displacement can be calculated. The casing has grooves cut on the sides that serve as tracks for the sensing unit.

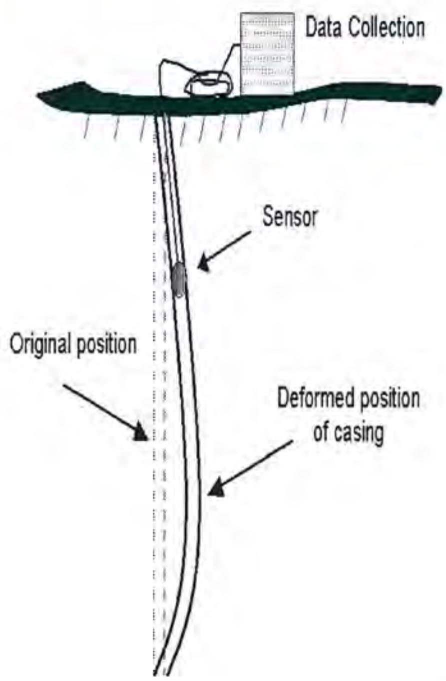

Figure 4. Cross-sectional schematic of typical traverse-probe inclinometer system.

The deflection of the casing, and hence the surrounding rock mass, are measured by determining the inclination of the sensing unit at various points along the length of the installations. The information collected from the inclinometers is important to slope stability studies for the following reasons (Kliche, 1999):

- to locate shear zone(s);
- to determine whether the shear along the zone(s) is planar or rotational;
- to measure the movement along the shear zone(s) and determine whether the movement is constant, accelerating, or decelerating.

__Time Domain Reflectometry (TDR):__ Time Domain Reflectometry is a technique in which electronic pulses are sent down a length of a coaxial cable. When deformation or a break in the cable is encountered, a signal is reflected giving information on the subsurface rock mass deformation. While inclinometers are more common for monitoring subsurface displacements, TDR cables are gaining popularity and have several advantages over traditional inclinometers (Kane, 1998):

- Lower cost of installation.
- Deeper hole depths possible.
- Rapid and remote monitoring possible.
- Immediate deformation determinations.
- Complex installations possible.

Recent advances have also been made in the use of TDR for monitoring ground water levels and piezometric pressures (Dowding, *et al.* 1996). A summary of applications of TDR in the mining industry is provided by O'Connor and Wade (1994).

__Borehole Extensometers:__ An extensometer consists of tensioned rods anchored at different points in a borehole (figure 5). Changes in the distance between the anchor and the rod head provide the displacement information for the rock mass.

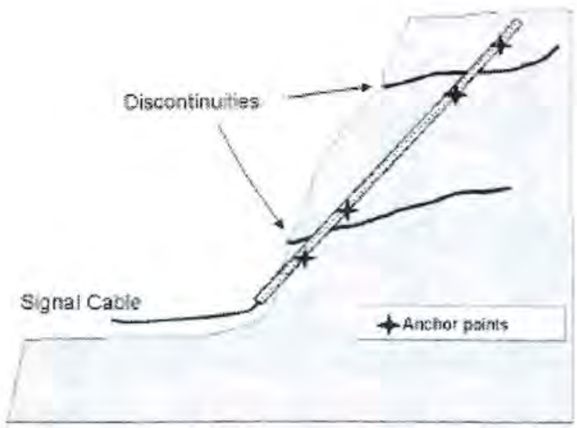

Figure 5. Multi-point borehole extensometer.

Piezometer: Piezometers are used to measure pore pressures and are valuable tools for evaluating the effectiveness of mine dewatering programs and the effects of seasonal variations. Excessive pore pressures, especially water infiltration at geologic boundaries, are responsible for many slope failures. Data on water pressure is essential for maintaining safe slopes since water behind a rock slope will decrease the resisting forces and will increase the driving forces on potentially unstable rock masses. Highwalls should be visually examined for new seeps or changes in flow rates as these are sometimes precursors to highwall failure. Additionally, pit slopes should be thoroughly examined for new zones of movement after heavy rains or snowmelts.

RESEARCH AND TECHNOLOGY

Stress, gravity loading, rock mass strength, geology, pore pressure, the presence of unknown underground workings, and many other factors contribute to slope failures. Because of the enormous surface area of many large open-pit mines, several varieties and scales of instabilities can occur. Complete vigilance to monitor each and every potential failure block is neither feasible, nor economical, and is certainly not attainable using today's most common point displacement monitoring techniques. Many of the current monitoring methods are also difficult to implement at quarries and surface coal mines, where near-vertical faces and lack of benching limit access to areas along the highwall. Additionally, as mining progresses, it is necessary to monitor different sections of the pit walls. Continually relocating devices is not only a costly and time consuming operation, but can also be dangerous -- especially with an unstable slope.

In an effort to make up for the shortcomings of point monitoring systems, NIOSH is examining new technology for slope monitoring that will look at the entire surface of the mine highwall for rock mass displacement and rock mass composition (Girard, 1998). Additionally, software has been created under a NIOSH contract to assist geotechnical engineers with bench designs to minimize rock fall hazards. A discussion of each technology follows.

Highwall Monitoring Using Radar Systems

Synthetic aperture radar (SAR) is a type of ground-mapping radar originally designed to be used from aircraft and satellites. SAR can be used to generate high quality digital elevation maps (DEM's) and to detect disturbances of the earth's surface. A variation of SAR – Interferometric Synthetic Aperture Radar (IFSAR) – uses differences in time-lapsed SAR images to generate maps of displacements (Fruneau and Achache, 1996). This technique has been successfully applied to produce displacement maps of ground movement caused by earthquakes, volcanic activity, and mine subsidence (Massonet, 1997; Canec, 1996). IFSAR can also be used to monitor displacement of unstable slopes or landslides (Reeves *et al.*, 1997; Sabine *et al.*, 1999).

IFSAR's have many advantages over current types of monitoring systems. Able to work in nearly all weather, an IFSAR can acquire imagery through fog, mist, rain, haze, or cloud cover, and can operate day or night. Also, an IFSAR can sample a large area for ground displacements, which gives them a tremendous advantage over survey networks, extensometers, and other instruments which sample movement on a discrete set of points. Recent developments in instruments such as prismless laser range finders partially address the problem of under sampling large areas for movement. However, the range and accuracy of these units can vary greatly depending on the reflectivity of the rock, the angle of the rock face, weather, and other factors. Manufacturers of prismless range finders generally claim a range of 500 meters or less.

The Microwave Earth Remote Sensing Laboratory at Brigham Young University (BYU) has recently designed and built a small synthetic aperture radar system capable of operating from light aircraft or from stationary ground-based stations (Thompson, 1998). The first field tests related to geohazards monitoring using this instrument were performed by BYU for the Canadian National Railways (CN). The railways were interested in finding a method to accurately detect rockfalls and washouts on railroad tracks before trains approached those dangerous areas. The initial results from those tests were positive, and BYU researchers are confident that their system can be adapted for NIOSH to monitor highwalls at mines. Field tests are anticipated to begin in the fall of 2000.

Imaging Spectroscopy

A contributing factor in many highwall failures is the presence of mechanically incompetent, argillically-altered rock. Major structures are generally well mapped, but weak rock units may be much more difficult to identify. Mine maps can vary greatly in quality and detail due to the subjectivity of various geologists and the extreme geologic complexity of many deposits. In addition, there are financial and practical limits to the number of samples that can be taken for geochemical or engineering analyses. Oftentimes, a large percentage of the data shown on geologic maps is an estimation by a geologist or mathematical interpolations of geotechnical results.

In order to help identify weak rock structures and remove ambiguity from geologic mapping, NIOSH researchers are testing applications of imaging spectrometers (Sabine, *et al.*, 1999). An imaging spectrometer is a device that can determine the composition of minerals from a distance by analyzing the diagnostic spectral absorption signatures (unique reflectance patterns of light that uniquely define each mineral). Like IFSAR, imaging spectrometers have been used from satellites and aircraft for geologic mapping for quite some time, but recent advances in technology have led to the development of smaller, portable, units that can also work from ground level.

One such instrument has been developed by a team of researchers at Carnegie-Mellon Research Institute (Denes *et al.*, 1997). The instrument (figure 6), known as a spectro-polarimetric imager (SPI), utilizes an acousto-optic tunable filter (AOTF) to detect the interaction of light and acoustic waves in certain crystals. The optical absorption or reflectance spectrum in the presence of ambient light then reveals the composition of objects in the image.

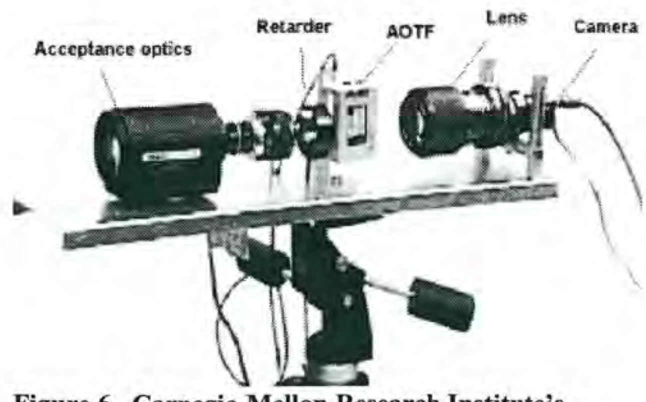

Figure 6. Carnegie-Mellon Research Institute's Spectro-polarimetric Imager.

In July 1999, NIOSH and Carnegie-Mellon Research Institute (CMRI) researchers conducted field-tests of the SPI for mine highwall imaging at Molycorp's Mountain Pass Mine near the California/Nevada border. This particular mine site was selected for field tests because the diagnostic spectral signatures of the ore (lanthanide series rare earths) are within the current spectral range of the prototype instrument.

Multispectral images were collected at twelve scenes at the Mountain Pass mine, including pit highwalls, outcrops, drill core, and hand samples. Figure 7a shows an area of interest (approximately 30 x 40 ft) on an exposed highwall. After the images were collected, substantial image processing, filtering, and computer analyses were performed. Figure 7b shows the results of the image analyses. The light areas are ore and the darker areas are waste rock. The field tests were successful in that the SPI images clearly illustrate the capabilities of the instrument to collect a multispectral image and discriminate materials within the image that may or may not be distinguishable by the human eye.

Figure 7a. Region of interest for field test of SPI.

Figure 7b. Results of SPI image analyses. Light areas correspond to bastnaesite ore, darker areas waste or unclassified spectra.

There are many advantages to using spectral analyses for geologic mapping. First of all, spectral identification of minerals would remove the human error and subjectivity of trying to visually determine the degree of alteration in a rock mass. Secondly, workers would be able to map mine highwalls, or other inaccessible or unsafe areas, from a safe distance. However, this instrument will only be useful for imaging weak, altered zones in pit highwalls if the spectral range is extended further into the infrared region. At this time, funding for this advancement in technology is not available at NIOSH or at CMRI.

Bench Design Software

Thorough engineering analyses of large slope cut in discontinuous rock masses often include investigations of bench stability. If kinematically viable rock failure modes are present in benches, it is unlikely that actual bench widths will match the original slope geometry plan. Consequently, rock fall hazard assessment and related slope stability safety issues must consider the operational ("as-built") catch-bench geometry and not the idealized ("as-designed") geometry.

Minor bench instabilities and rock falls adversely impact mine safety in two key areas. First, as failures break back along the top of a bench, storage capacity for holding rock fall debris is significantly reduced and falling rock from above may not be caught and retained on the bench (see figure 8). Secondly, large amounts of rock fall debris on benches may even trigger multiple-bench failures. Therefore, the ability to predict the volumes of bench-scale failures is essential for properly designing wide enough catch benches.

In order to assist mine operators with bench design, computer software has been developed for NIOSH's Spokane Research Laboratory (Miller, 2000). The software computes the probability of bench stability for plane shear or wedge failures using basic geological engineering input on bench dimensions, rock mass characteristics, and fracture data. After all of the potential failure geometries are computed, the probability of sliding is computed using a stochastic shear strength model (Miller, 1988; Miller and Girard, 2000.)

Results from a bench stability study can be used to help select inter-ramp slope angles and overall slope angles. Bench geometry has a direct influence on the overall slope angle as expressed by:

$$\tan(A)=1/[(W/H) + (1/\tan B)] \qquad (1)$$

Where: A = overall (average) slope angle;
B = bench face angle
H = vertical height of bench; and
W = horizontal width of bench.

Using this relationship and output from the software, geotechnical engineers can design overall slope angles and bench configurations to minimize extensive loss of catch bench width and thus minimize rockfall hazards.

Figure 8. Small, unexpected rock falls may indeed be more hazardous than massive failures that involve slow displacement of material over a longer period of time.

CONCLUSIONS AND RECOMMENDATIONS

Steps need to be taken to reduce the number of mining deaths resulting from slope instability. Diligent monitoring and safe design by qualified geotechnical engineers at mine sites is crucial. Additionally, proper catch bench design, blasting patterns that minimize overbreak, effective highwall scaling (where appropriate), and dewatering of potentially unstable zone are also important to minimize hazards related to highwall failures.

There are many new technologies being explored, but remote sensing, at the present time, cannot replace conventional geotechnical methods of investigation. A great deal of research is still needed to design and test new systems in order to make certain they are scientifically valid and economically viable options.

The NIOSH Slope Stability Hazard Recognition Team is pursuing a variety of research options to minimize the dangers associated with surface ground control problems. Results of this research may also benefit others involved in studies of landslides, rock falls, avalanches, volcanic activity, and other geohazards.

For more information about this or any other NIOSH project call 1-800-35-NIOSH. Or visit the website at: http://www.cdc.gov/niosh

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SURFACE HAULAGE ACCIDENTS: THE ROLE OF HAUL ROAD DESIGN

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ABSTRACT

The layout or condition of the haul road is a factor in many surface haulage accidents. This paper reviews recent accidents and examines the role played by such features as haul road grades and berms. Categories of haul road accidents are identified and lessons to be learned from them are discussed. Recommendations are made for measures that mine operators can take to improve haul-road and surface-haulage safety.

INTRODUCTION

Surface haulage accidents continue to be a leading cause of deaths and injuries in the mining industry. Leading causes of these accidents include operating equipment with defective brakes, improper use of retarders, driving too fast for the conditions, being struck while in equipment blind-spot areas, going over the edge at dump points, and getting caught in conveyors.

For the five year period from 1994 to 1998, 442 fatal accidents occurred in the mining industry and 156 of them, or 35 percent, were classified as "powered haulage." The "powered haulage" category covers accidents involving the motion of powered haulage equipment. This includes accidents involving mobile equipment such as haulage trucks and front-end loaders, as well as accidents involving conveyors. For the five-year period, 58 of the 442 fatal accident victims were truck drivers. Fatal accidents to truck drivers represented 13 percent of the total fatal accidents and 37 percent of the powered haulage accidents.

For the same five-year period, of the 66,441 non-fatal mining accidents involving lost work days, 7307 of them, or 11 percent, were classified as powered haulage. There were 3329 non-fatal accidents involving truck drivers. So truck drivers were involved in 5 percent of all mining accidents, but 13 percent of the fatal accidents.

The focus of this paper is on those surface haulage accidents where the layout or condition of the haulage road was a factor. The information contained in this paper is based on two sources: the accident investigations performed by the Mine Safety and Health Administration (MSHA); and the accident data submitted to MSHA by mine operators as required by 30 CFR, Part 50. Accidents that have occurred since 1994 were examined.

FATAL ACCIDENTS WHERE HAUL ROAD CONDITION WAS A FACTOR

The fatal accidents that have occurred since 1995 were examined to identify those cases where the condition of the haul road was a factor. The main categories of these accidents are described below and some lessons to be learned from them are discussed.

Accidents While Equipment Was Descending a Steep Grade

--A 26 year old truck driver died when he lost control of a 30-ton capacity articulated haul truck as he descended a grade that varied from 16 to 20 percent. The empty truck plowed through a berm. The accident happened at night on the victim's eighth haul of the shift. He had radioed that he had no brakes. The accident investigation revealed that at least 50% of the service braking system had not been working and that the truck was being operated too fast for the retarder to have been effective. The victim had less than four months of mining experience and had worked for two weeks as a truck driver at this mine. He was wearing his seatbelt when the accident occurred.

--A truck driver lost control while descending a grade. He died when he apparently attempted to jump from the truck as it turned on its side. Factors contributing to the accident included the steepness of the grade (up to 18 percent); the overloaded condition of the truck (45 versus 20 tons); and the inexperience of the driver (this was only his second trip down this haul road and his first trip in that particular truck). Unfortunately the truck driver did not attempt to take advantage of any of three runaway escape ramps once he lost control of the truck. There were signs at the ramp entrances and the ramps should have been clearly visible to the driver. When the accident occurred the truck turned on its side at a point below one escape ramp and just above another ramp. This accident points out the need for adequate training not only on truck operation, but also on haul road safety features.

--A front-end loader operator died when he lost control of the loader on a short stretch of a haul road that was at a 25-percent grade. The loader rolled over. The accident investigation revealed that the loader's brakes were defective.

--The operator of a 170-ton capacity haul truck was killed when his truck went out of control as he descended an 8-percent grade. The accident investigation revealed that the truck's retarder system was defective and the inadequately maintained service brakes could not provide enough braking force to stop the truck.

--The driver of a highway-type haulage truck was killed when he lost control of the truck while descending a half-mile long section of road that was on a 10-percent grade. The truck ran up the hillside and turned over. The driver was found pinned beneath the truck. The truck had been loaded to more than twice the manufacturer's specified maximum payload. Furthermore, examination of the truck revealed that only two of the truck's six service brakes were functioning. The victim had one year of experience as a truck driver and had received hazard training only.

--An over-the-road haul truck went out of control as it descended a haul road where the grade varied from 12 to 17 percent. The truck was found to have defective brakes. The victim was on the first shift of his first day at this mining operation.

--A driver was killed when he lost control of his hauler while descending a 10-percent grade. The victim had only 2 months of mining experience, with one month on the hauler. He was descending the grade in neutral, which allowed the vehicle to gain too much speed. When he attempted to shift into first gear, the engine stalled. Mechanical defects with the truck left it without emergency steering or braking. The hauler turned on its side and crushed the driver who had apparently attempted to jump clear.

--A haul truck went out of control as it descended a 19-percent grade leading into a switchback. The road was wet and slick. The truck went through a substantial berm. The victim, who was not wearing a seatbelt, was thrown through the windshield when the truck impacted a hillside. The truck had been traveling too fast for the conditions (slick and steep). Defects in the brakes, with some of the brakes being inoperative, would have increased the tendency for the truck to go into a skid.

Lessons to be learned: As indicated by the examples above, a scenario common to many haulage accidents is a combination of steep grades and defective equipment brakes. In some cases the accident investigation revealed that equipment operators were attempting to compensate for known mechanical problems. Of course, the effect of any mechanical problem is going to manifest itself most noticeably in more demanding operating conditions, such as on steep grades. These cases point out that well-designed and maintained haul roads—compatible with equipment characteristics and capabilities—can help prevent accidents, but good roads must be combined with a competent and systematic mobile-equipment maintenance program, a comprehensive driver training program, and effective supervision.

Accidents While Equipment Was Climbing a Steep Grade

--A road grader stalled as it was making a sharp right turn near the top of a 27-percent grade on an access road. The brakes were defective and were not capable of holding the grader on this steep of a grade. The grader rolled back, went through a low berm, and turned over, crushing the operator.

--A truck stalled while making a right turn near the top of a ramp where the grade steepened to 16 percent. The truck rolled backwards and turned on its side. Apparently the driver attempted to jump from the truck. He was found underneath the truck. The accident investigation revealed that the ability of the brakes to hold the truck on the grade was marginal.

--A loaded scraper stalled as it approached the top of a 320-foot long ramp on a 17-percent grade. The operator was killed when the scraper rolled backwards, went over the edge of the roadway and fell to a bench below. There was no berm for a distance of 65 feet in the area where the vehicle went over. The service and parking brakes were found to be out of adjustment.

Lesson to be learned: While the potential hazards of driving haulage equipment down steep grades are obvious, many accidents also occur going up such grades. In 1998 and 1999, at least 15 fatal and nonfatal accidents occurred when an equipment operator lost control while climbing a grade.

Accidents Involving Inadequate Berms

--A construction foreman died when his ¾ton pickup truck went off a roadway and rolled about 70 feet down a hillside. The road was at a grade of about 13 percent and the roadway width varied from 10 to 12.5 feet. An adequate berm had not been provided along the edge of the road. The accident investigation revealed no problems with the service brakes or the steering system on the truck.

--A truck driver was killed when his 50-ton capacity truck climbed over a berm and fell 40 feet to the quarry floor. The grade at the accident site was 9 percent. The truck was returning empty to the quarry. Concrete barriers, anchored by cables, were used for the berm. The mid-axle height of the truck was 40 inches while the concrete berms were 30 inches high. The accident occurred at a point where the road narrowed to 36 feet. An eyewitness saw the right front tire of the truck contact the concrete berm and climb up and over the barrier. The truck traveled for about two truck lengths on the berm then went over the edge. No mechanical problems were found with the truck. The driver was not wearing the seatbelt.

Lessons to be learned: One of the prime purposes of maintaining at least axle-height berms along the edge of elevated roads is to provide drivers with a well-defined and highly visible indicator of the location of the roadway's edge. Berms that are at least axle height also act to impede vehicles from going over the edge, and give the driver the opportunity to get the vehicle back onto the roadway.

Accidents Involving Boulder Berms

--A 25-ton capacity articulated haul truck went out of control coming down a 17 percent grade. When the loaded truck struck the boulder berm, the driver, who was not wearing a seatbelt, was thrown through the windshield. The truck's bed turned on its side but the cab portion remained upright. The service brakes were found to be fully functional and effective. The driver, age 72, had one-and-a-half years of mining experience and had been driving at this mine for two weeks.

--A truck driver died when his 40-ton capacity articulated haul truck struck the rock berm along the edge of a road. Although the victim was wearing a lap-type seatbelt, he received severe head injuries when he struck the cab structure. The accident occurred at a gradual curve in the road as the victim was returning to the quarry for another load. The right side of the truck struck the berm. No mechanical defects were found with the truck.

Lesson to be learned: Although the boulder berms served the function of keeping the vehicles from leaving the roadways, these instances illustrate one of the drawbacks of using boulders, by themselves, for berms. Depending on the size of the boulders, a vehicle's contact with the berm can be, in itself, like a collision. In contrast, berms constructed of a compacted pile of spoil consisting of material of various sizes can provide the characteristics of providing restraint while helping to redirect the vehicle and absorb some of the impact energy of the vehicle. Mine operators who use boulder berms should consider using the boulders in combination with soil and broken rock to obtain berms with improved safety characteristics.

Accidents Where Equipment Went into a Body of Water

--A truck driver drowned when his 30-ton capacity articulated truck went off the edge of an elevated road and plunged into a pond. The truck went off the road when it failed to negotiate a 90-degree right-hand turn. Even though there was a 10-foot drop-off into the pond, there was no berm for a distance of 100 feet. No mechanical defects were found with the truck. A co-worker, who operated the same type of truck, said that it usually took two attempts to make the right hand turn. The road was 20 feet wide at the bend.

--Returning to the pit empty, a driver lost control of a 40-ton capacity truck as he descended a 10 percent grade. At a point where the road curved to the right, the truck went off the road and into a 30-foot deep water-filled pit. The driver's body wasn't recovered from the pit until the next day. He was found about 90 feet from the completely submerged truck. The accident investigation found skid marks on the road, which was wet and slippery. No problems were found with the brakes but the gear selector linkage was defective. There was no berm along the edge of the roadway where it passed the water-filled pit.

Lessons to be learned: Haulage roads need to be widened at sharp curves and bends to accommodate haulage trucks. Roads need to be wide enough to provide a reasonable margin for error for truck drivers.

Adequate berms need to be provided along the edge of elevated roadways. When a haul road passes near a body of water there is added danger to a truck that leaves the roadway. Mine operators should consider providing extra large berms at more critical areas, such as where the potential exists for equipment to end up in water.

Accidents at Railroad Crossings

--An over-the-road truck driver was killed when her loaded truck was struck by a freight train. The railroad crossing was posted with warning and stop signs. Sight distance down the tracks was not restricted. Other workers reported that the train was blowing its whistle prior to the accident. The victim was found 180 feet from the point of impact.

--A haulage truck was totally destroyed when it was struck by a freight train. The train's locomotive weighed 200 tons and pushed the truck 2400 feet down the tracks. Stop signs were installed on both sides of the crossing, which was used exclusively by mining company trucks. The victim's truck would have traveled on the haulage road parallel with the railroad tracks until the road made a right turn to cross the tracks. This meant that as the truck turned to cross the tracks, the train would have been approaching on the driver's blind side.

Lessons to be learned: Mine operators need to be alert to potential problems at railroad crossings. In one case, it appears that the driver knew the train was coming and tried to beat it through the crossing. Drivers need to error on the side of caution since it's easy to misjudge a train's speed and distance. Furthermore, a train can take over half a mile to stop from a speed of 30 miles per hour. A factor in the other accident may have been the layout and alignment of the intersection. Making a right turn close to the crossing would have made it difficult for the driver to see the on-coming train.

Accidents Where the Edge of the Road Gave Way

--A front-end loader went off a roadway when the edge of the slope gave way. The embankment was constructed of fine-grained material that was highly erodible. The embankment material had not been well compacted and was near its angle of repose. The accident investigation revealed tension cracks and slides along the edge of the road. The berm along the accident area was only 18-inches high.

--A truck-driver trainee died when the edge of the roadway gave way under the truck's right front tire. There was no berm where the accident occurred. The 27-year old victim was driving with a trainer and was about two hours into his first shift when the accident occurred.

Lessons to be learned: One of the important purposes of a berm is to prevent the weight of heavy haulage equipment from getting too close to the edge of the slope. To serve this function, boulder berms need to be placed back from the edge and earthen berms need to have adequate base width. Earthen berms are sometimes cut to steepen their inside slope. If this is done, operators should ensure that a sufficient amount of material is initially placed so that the base width of the steepened berm will still be at least as large as the base width of an uncut mid-axle height berm at the material's normal angle of repose.

When a haulage road is founded on fill material, the fill must be adequately compacted and placed at a flat enough slope to ensure that it will support the weight of haulage equipment. This can be achieved by compacting the material in thin, horizontal lifts.

Accident Involving Intersection Layout

--The driver of a 1-ton service truck died when his vehicle was run over by a 190-ton-capacity haulage truck. The accident occurred at an intersection after dark. The service truck was ascending an 8 percent grade on the main haul road. The haulage truck was on a crossing road that intersected the main road at a 55-degree angle. The layout of the intersection is shown in Figure 1. As the service truck approached the intersection it was to the haulage truck driver's right. After stopping at a stop sign, the haulage truck driver pulled out and ran over the service truck.

The accident investigation revealed that the service truck's strobe light was on but the truck's headlights were not. To improve their ability to see down the main haulage road, haulage truck drivers normally turned their truck slightly to the right as they stopped. Area lighting was provided at the intersection. The sharp right angle at the intersection, the downgrade to the right, the left-hand traffic pattern, and the presence of trolley-assist equipment mounted on the haulage truck, all made it more difficult than normal for the haulage truck driver to see traffic coming up the main haul road.

Lessons to be learned: This accident illustrates that the right-side blind spot needs to be taken into account in laying out haulage road intersections. Roads intersecting at less than 90 degrees should be avoided as they exacerbate the problem drivers have in seeing whether traffic is approaching from the right. The extra work involved in re-aligning intersections should pay off in improved safety and efficiency at intersections. Where possible, the angle at which roads intersect should be made more than 90 degrees to improve the driver's ability to see the conditions to the right. When a haul road intersection cannot be modified to eliminate the right side blind spot, then truck-mounted cameras should be used to minimize the blind area.

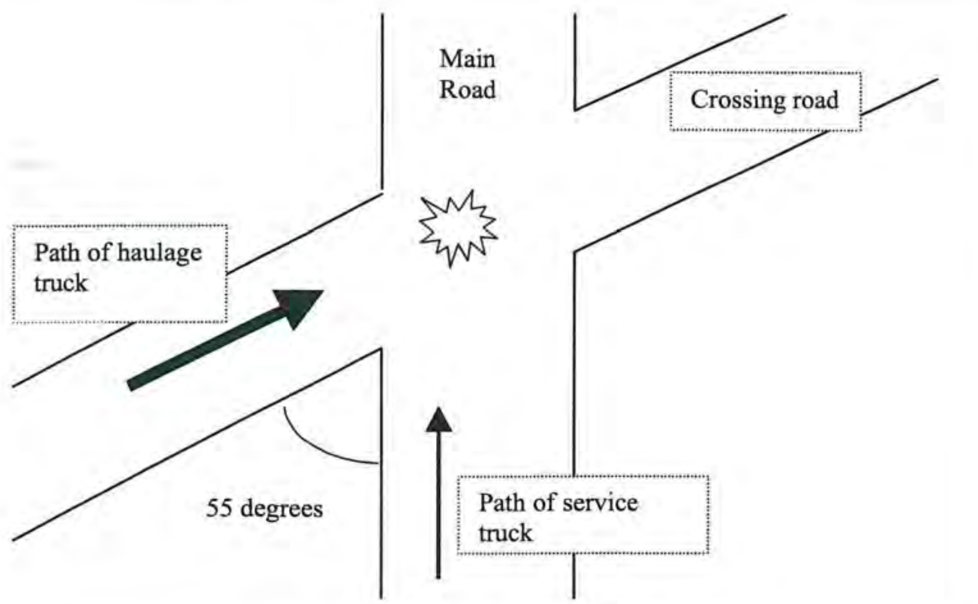

Figure 1. Intersection layout where haulage truck ran over service truck.

Accident Involving Right-of-Way Rules

--A miner who was leaving mine property in his personal pickup truck was killed when his truck collided with a haulage truck. The empty haulage truck was driving down the center of the crowned haulage road when the pickup came around a curve. The haulage truck driver steered to the left (i.e. drove on the left side of the road) anticipating that the pickup would pass on his right side. The pickup instead steered to the haulage truck's left (i.e. drove on the right side) and the vehicles skidded head-on into each other.

Lessons to be learned: Right-of-way and traffic-flow rules need to be established and posted. Training needs to be provided to ensure that all miners understand the right-of-way rules. Traffic signs need to be posted at frequent intervals to act as reminders for the miners and to help ensure that visitors on the property can safely negotiate the mine roads.

Accidents Involving Water Trucks

--A water truck driver lost control while descending a 1000-foot long section of 10-percent grade road. The driver was killed when the truck left the road and crashed into a ravine. The accident investigation revealed that the front brakes had been removed and the other brakes on the truck were defective.

--The driver of a water truck died when his truck jack-knifed as he attempted to make a sharp turn. The truck had a 5000-gallon capacity trailer-mounted water tank that was about three-quarters full at the time of the accident. The victim had 20 years of mining experience. The accident investigation revealed that the vehicle's tractor and trailer were not designed to be used together.

--A fatal accident occurred when a water-truck driver lost control of the vehicle while rounding a curve. The articulated truck turned on its side and the victim, who was not wearing the seat belt, was trapped underneath the vehicle. The accident occurred as the truck descended a dry roadway on a 7 to 9 percent grade. The 2500-gallon water tank was equipped with two baffles to restrict water movement from front to back. The tank was carrying about 1200 gallons of water at the time of the accident. The accident investigation revealed that the trailer brakes were defective. The victim, who normally worked in quality control, only occasionally operated the water truck.

--An eyewitness saw a water truck descending an 8 percent grade at a high rate of speed and sliding sideways as it rounded a curve. The 18-year old water truck driver died when the truck turned on its side, pinning him underneath it. The truck had been modified and equipped with a water tank. The tank measured 5 by 7 by 15 feet, and had eight compartments to lessen the effects of water movement. The accident investigation revealed a defect in the transmission that could allow the vehicle to jump out of gear. The victim had been wearing a lap-type seatbelt. He had three weeks of experience and had not been trained for the task of driving the water truck.

--A water truck driver was killed when he lost control of the vehicle, which went through an axle-height berm and tumbled down a hillside. The accident occurred while the 25-ton capacity water truck, which was about half-full at the time, was descending a grade that varied from 12 to 16 percent. The accident investigation found numerous mechanical problems with the truck including defective brakes.

Lessons to be learned: Water trucks need to be maintained to acceptable safety standards just like production equipment. Water trucks should be baffled front-to-back and side-to-side to minimize the effects of water movement in the tank. Water-truck drivers need to be trained to drive conservatively to allow for the forces that may be generated by water surges in the tank.

Accidents Involving a Visitor Driving on Mine Property

--A telephone company technician died when he lost control of his 1-ton service truck while leaving mine property. The vehicle overturned and the victim was thrown out. The accident occurred after dark as the vehicle was descending a 12 percent grade. This was the fourth time that the victim had been on this property over a nine-month period.

--A welding contractor was fatally injured when his truck skidded through a three-way intersection, went through a berm, and fell 160 feet over a highwall. The road was curved and on a 10-percent grade approaching the intersection. The skid marks leading up to the berm were 125 feet long. The accident occurred just after midnight. The accident investigation revealed that several defects would have compromised the performance of the vehicle's rear brakes.

Lesson to be learned: Visitors on mine property may not be used to the steep grades, or the unusual right-of-way rules on mine property. Traffic control signs, warning signs and lighting should be installed keeping in mind the perspective of visitors on mine property. Roadside reflectors, especially on sharp curves and at intersections, would be particularly helpful for nighttime driving for persons, such as vendors or outside maintenance personnel, who may only occasionally drive on mine roads. Haul road conditions should be covered in hazard training.

NON-FATAL INJURIES TO TRUCK DRIVERS WHILE OPERATING A TRUCK

For the four-year period from 1996 through 1999, the accident data submitted by mine operators was reviewed for injuries that occurred while operating a truck. Judgements are involved in evaluating this data, as the information is often limited. But the data does provide a perspective on where the accidents and injuries are occurring. The data revealed the following:

- Over the four-year period there were 990 injuries in degree 1 (fatality) through degree 5 (days of restricted work activity only) that occurred while operating a truck.

- Roughly 24 percent of the accidents fall into the category of "loss of control." The primary causes given for the loss of control were: defective brakes or drive lines; problems with shifting gears or being in the incorrect gear; sliding on the road surface; and driving too fast for conditions.

- In 42 cases, or about 4 percent of the total, the loss-of-control accidents occurred while the truck was climbing a grade and the truck ended up rolling back down the grade. In many of these cases the truck reportedly stalled when the driver attempted to shift gears on the grade.

- About 21 percent of the injuries were attributed to the driver being jarred or jostled in the cab because of poor road-surface conditions. Reasons given for these injuries included driving through a rut, hitting a hole or a rock, and bogging down in a soft spot in the road. In 1998 and 1999, this category of injury accounted for 103 injuries. Even though many of these injuries resulted in no lost-work days, the overall average was over 15 lost-work days per injury.

- About 11 percent of the injuries occurred while the driver was dumping. Accidents included: trucks going over the edge at dump points; drivers being jarred when the bed came down hard; drivers being injured when the front of the truck lifted then dropped during the dumping process; and trucks tipping on their side as the bed was raised to dump. Trucks tipped on their side because material hung up on one side of the bed, such as from being frozen, and/or because the truck was not sitting level (from side to side) when the bed was raised.

- Almost 10 percent of the driver injuries occurred while the truck was being loaded. The most common occurrence was for the driver to be jarred when either a large boulder was dropped into the bed of the truck, or the bucket of the loader struck the truck.

- In 18 cases over the four-year period a truck driver was injured when they left the bed up after dumping and the bed later struck an overhead object.

- In 13 cases the accident was attributed to the operator having fallen asleep while driving.

- In 36 accidents, or almost 4 percent of the total, a water truck was involved.

- In metal and non-metal mining, 9.5 percent of the accidents occurred to contract truck drivers, while in coal mining, nearly 36 percent of the accidents involved contractor drivers.

HAUL ROAD DESIGN FEATURES TO IMPROVE SURFACE HAULAGE SAFETY

Based on the surface haulage accidents that have occurred over the past five years, the following recommendations are made to mine operators to improve haulage roads and surface haulage safety.

- **To the extent practical, minimize the steepness of haul roads.** Steep grades can reduce or eliminate the operator's margin for error, especially if there is a lapse in equipment maintenance or a truck is overloaded.

- **On steep grades provide measures, such as escape ramps, to control runaway vehicles.** As indicated by several of the accidents described above, runaways can occur from factors such as defective brakes, overloading, driving too fast for conditions, or operator error or inexperience. Provisions like escape ramps or straddle berms provide a way to rescue a runaway vehicle and prevent a serious injury or fatality. Though not documented, there is anecdotal evidence of the value of ramps and straddle berms in coming to the rescue of runaway trucks on mine properties. Considering the high number of hauls that are made, runaway control measures can provide a relatively inexpensive way to provide an ever present insurance against a serious accident.

- **Install signs indicating the actual steepness of grades, in percent** and provide retarder-performance-chart decals in truck cabs. Most new haulage trucks display information in the cab on the speed that should not be exceeded, and the proper gear to use, while descending various grades. The speed is limited so that the truck's retarder capacity is not exceeded. By indicating how steep the grade actually is, a mine operator provides the drivers with the information they need to drive the trucks more safely. It should be noted that the grade and speed information provided by truck manufacturers is based on the premise that the truck is not overloaded. Overloading compromises the braking and retarder performance and can force the equipment operator into an unsafe operating mode.

- **Construct larger than mid-axle-height earthen berms, especially in more critical areas,** such as near bodies of water, at curves on downgrades, etc. Berms constructed to mid-axle height should be considered a <u>minimum</u> criteria for berm height. Experience has shown that berms of this height will not necessarily keep a vehicle from leaving the roadway. One Canadian province requires that berms be at least three-fourths of the tire height. A study by the U.S. Bureau of Mines concluded that to restrain a vehicle, berms may need to be 3 to 4 times the axle-height depending on the vehicle's speed and angle of impact. When determining the size of berm to use at various points along a roadway, mine operators should consider the conditions both with respect to the likelihood of a vehicle going out of control (e.g. grade of road, sharpness of curve, etc.), and the consequences if a vehicle does leaves the roadway (e.g. presence of a body of water).

- **Maintain adequate roadway width.** A common safety recommendation is that haul roads should be wide enough to allow clearance of at least half a truck width on each side of a truck. That is, the width of a two-lane road should be at least 3.5 times the truck width. As larger capacity trucks are purchased, haul road width needs to be increased to compensate for their greater width. Narrow roads require the driver to get close to the edge, which can lead to accidents where the edge gives way under the heavy truck weight. Roads that are too narrow also increase the chances for collisions, or for drivers to inadvertently hit the berm or get into the ditch, and, in general, provide the drivers with little margin for error.

- **Ensure adequate sight distance at intersections and RR crossings.** In laying out intersections consider the difficulty that drivers have in seeing to their right side. The installation of large, convex mirrors may be useful at some situations where sight lines cannot otherwise be improved. Avoid situations, such as road intersections near railroad crossings, where trucks crossing the tracks could get backed up and either overhang or be trapped on the tracks. Keep vegetation trimmed back where it restricts sight distance.

- **Install roadside reflectors** to assist drivers, especially those with limited experience on the mine's roads, with seeing the haul road layout at night. Reflectors are especially useful at curves and intersections. As one ages, one's ability to see at night diminishes. Roadside reflectors are a common feature on interstate highways and can serve the same safety function on mine roads.

- **Re-examine road maintenance procedures.** The number of injuries from hitting holes, ruts and rocks on haul roads points out the need for diligence in road maintenance. Keeping roads well-graded not only benefits safety but also improves haulage efficiency from the standpoint of tire maintenance and fuel use. Overloading haulage trucks can compounds road maintenance problems due to the increased likelihood of spillage and the higher tire-to-ground contact pressures.

- **Check on the condition of the water truck** and the training provided to water-truck drivers. The number of fatal accidents involving water trucks should raise a red flag for mine operators. Accident investigations reveal a tendency for water trucks to be poorly maintained and for water-truck drivers to lack adequate training.

- **Consider roads from the perspective of a visitor on the property.** Roads that may be fine for miners who use them everyday and have become used to them, may present a problem for visitors or for newly-hired workers. For everyone's safety, mine operators should attempt to view the roads through the eyes of a visitor. This is especially important when an unusual traffic pattern, such as driving on the left side, is used. Consider, for example, whether a vendor or outside maintenance person, after spending a few hours in the shop, would remember to "drive left" upon leaving the shop area. Ensure that signs are provided at frequent intervals and not just where persons enter the property.

- **Re-evaluate surface-haulage training.** In many surface haulage accidents the equipment operator had limited experience and training. In 29 percent of the surface haulage fatal accidents that occurred between 1994 and 1998, the victim had less than one year of mining experience. In 40 percent of the accidents the victim had less than one year of experience at the particular mine where the accident occurred. Training needs to ensure that truck drivers are familiar with the operator's manual for their truck, particularly with the truck's braking /retarding controls and capabilities. Loader operators need to understand the importance of not overloading the trucks. Training in these areas should not be limited to formal classroom efforts but should be regularly included in pre-shift assignment meetings, tailgate safety meetings, and during general work discussions.

CONCLUSIONS

With the high number of hauls made every day in the mining industry, the layout and condition of haulage roads plays a significant role in surface-haulage safety. This paper has reviewed some of the accidents that have occurred over the last five years that have involved haul road features. A scenario common to many of these accidents was a combination of steep haul road grades and either defective truck brakes and/or inexperienced or inadequately trained truck drivers. Fatal accidents involving water trucks appeared to be unusually common. This paper includes a number of recommendations for measures to improve haulage road safety. A safe surface-haulage program requires a combination of good haul road layout and maintenance; reliable equipment maintenance; and effective training for equipment operators. Deficiencies in any of these three areas will increase the chances for accidents.

A goal of MSHA's Office of Technical Support is to help identify accident remedies and share safety ideas with all elements of the mining industry. For additional information on accident remedies, and other information pertaining to mining safety and health issues, see MSHA's web page at www.msha.gov.

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JOLTING AND JARRING INJURIES IN SURFACE MINE HAUL TRUCKS

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ABSTRACT

Powered haulage has been, and continues to be, a major source of severe accidents, injuries, and fatalities at metal/nonmetal surface mines. Between 1986 and 1997, truck drivers accounted for 63% of the lost-time injuries in surface haulage. This project was undertaken to reduce the number and severity of lost-time injuries among operators of these trucks. Shock accelerations were measured on trucks at a western surface mine during representative work cycles to determine the shock environment of the operator. Acceleration data was collected from the floor and seat of two types of haulage trucks. Shock tests were also run at Caterpillar, Inc.'s, proving grounds in Green Valley, AZ, to determine the magnitude of shocks resulting from a rough road and from occasional loading events. A controlled rock drop onto the bed of a haulage truck and a side impact were measured. A system that ties acceleration data with Global Positioning System (GPS) data was developed to aid in identifying haul road problems. A biomechanical investigation revealed that a side impact will cause the erector spinae muscle groups on the side opposite to contract first. Then the near side muscles react approximately 20 ms later, which could increase the likelihood of injury from a side impact.

INTRODUCTION

Powered haulage has been, and continues to be, a major source of severe accidents, injuries, and fatalities at metal/nonmetal surface mines. Between 1986 and 1997, injuries to truck drivers accounted for 63% of the lost-time injuries. Analysis of MSHA accident data from 1991-1997 in metal/nonmetal surface mine haul truck operator back injuries indicate that out of four hundred events haul truck jarring accounted for 27%; being hit by loader 11%; slips and falls 10%; loading (rock jar) 7%; and vehicle road jarring 8%. In surface haulage between 1986-95 truck drivers accounted for 64% of lost-time injuries and operating a haul truck accounted for 60% of the back injuries. Back injuries are the leading cause of lost time and the most costly class of non-fatal injuries.

The objective of this research is to reduce jolting and jarring injuries among operators of heavy mining equipment, particularly haulage truck drivers. Characterization of the magnitude and frequency of jolts and jars will lead to a better understanding of their causes and enable researchers to evaluate different types of engineering controls that could reduce trauma to operators and lower the incidence of back injuries. The research is part of a project called "Engineering Controls for Reducing Jolting/ Jarring Injuries in Surface Mines" at the Spokane Research Laboratory (SRL) of the National

Institute for Occupational Safety and Health (NIOSH). In this project, researchers are investigating the causal factors of jolting and jarring injuries and the effects of long-term exposure to jolting and jarring.

FIELD TESTS

Data were collected under actual field conditions during representative work cycles at a western surface mine interested in reducing lost-time injuries among its haulage truck operators. These data were obtained from two types of haulage trucks (truck A and truck B) manufactured by different companies.

Jolts and jars were measured using a Dallas Instruments Saver mounted to the pedestal of the driver's seat with a strong magnet at the point where the seat is bolted to the cab floor. This instrument package contains an internal triaxial set of piezoelectric accelerometers, a charge amplifier, and a data logger with 8 megabytes of memory. A Bruel and Kjaer type 4322 seat pad, which also contains a triaxial set of piezoelectric accelerometers, was attached to the driver's seat cushion to measure those accelerations. The three orthogonal directions (x, y, z) were oriented according to ISO 2631 [1]. From the driver's perspective, x is positive forward, y is positive to the driver's left, and z is positive upward.

For truck A, the Saver monitored jolting and jarring for 11 hr, 37 min, between 2:45 p.m. on June 8 and 7:08 a.m. on June 9, 1999. For truck B, the Saver monitored jolting and jarring for 18 hr, 1 min, between 12:47 p.m. on June 9 and 6:48 a.m. on June 10, 1999. The threshold for triggering data collection was determined empirically at 1.5 g's on the z channel of the seat cushion. Seven events above 1.5 g's were recorded on truck A, and five events were recorded on truck B. Other set-up parameters were filter frequency, 200 Hz; range, ±50 g's; samples per second, 512; recording time, 8 sec; and samples per event, 4096. The 12 events were converted from the Saver file format to ASCII and imported into a software program called DADiSP, a product of DSP Development Corp.,[1] for further analysis.

The acceleration shocks for truck A are shown in figure 1 and for truck B in figure 2. Comparing the shocks on truck A with the shocks shown in figure 3 and operators' written logs, the authors determined that shock events A1, A3, A5, and A6 were caused by loading. The other three shock events on truck A (A2, A4, and A7) and all shock events on truck B were caused by rough ground, as determined by comparing figure 4 and operators' written logs. The average peak frequency of the shocks on the seat cushion in truck A was 35 Hz, and the average frequency of the "rough ground" shocks was 1.4 Hz. Interestingly, the average peak frequency of all truck B shocks was also 1.4 Hz.

FIELD TESTS AT CATERPILLAR PROVING GROUNDS

Caterpillar, Inc. offered us the use of their proving grounds in Green Valley, AZ. We accepted their generous offer and devised three sets of experiments to investigate the possible relation of jolting/jarring to operator injuries. The instruments used in the three tests were the same as those used to collect data at the mine. The set-up parameters were threshold level, 1 g; filter frequency, 100 Hz; range, ±20 g's; samples per second, 512; recording time, 4 sec; and samples per event, 2048.

In the first test, a large rock was dropped into the bed of a Caterpillar truck from over 10 ft above the bed surface. The drop produced a distinctive curve (figure 3) in which a significant jolt was registered in the z-direction when the rock struck the truck bed. The jolt due to the rock drop was the most severe jolt that we measured at the test track.

[1]The mention of specific products or manufacturers does not imply endorsement by the National Institute for Occupational Safety and Health.

In the second test, the truck was hit from the side by a loading shovel (figure 4). The side impact was regarded as rather significant. An observer riding in the truck noted that the side roll and side impact were the most uncomfortable part of his ride. Consequently, side impacts were selected for further investigation.

A third experiment was designed to investigate the effect of different loading conditions of the truck. A comparison of a truck running a bump course empty with a truck running the course fully loaded was used in that test. The truck was driven along a test course (dirt road) with a series of bumps positioned randomly along the course.

The truck was driven along the course at 5 mph. Figure 5 shows an empty truck going over a bump, while figure 6 shows the same truck going over a bump loaded. The truck suspension, when properly maintained, compensates for the different loads rather well.

A

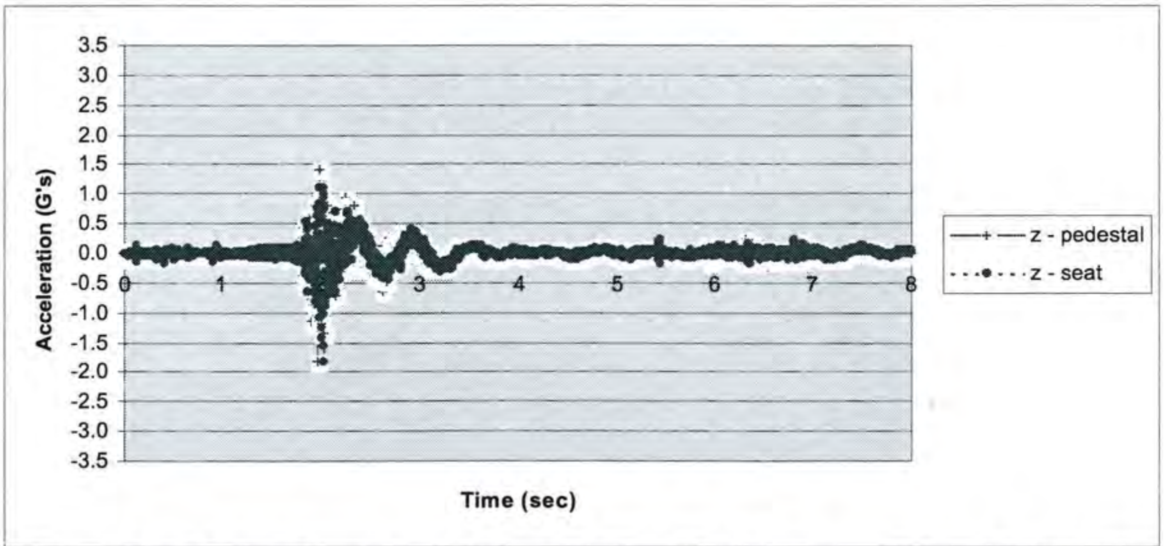


Figure 1.—Truck A shock events (continued). *A*, Loading event, A1; *B*, rough ground event, A2; *C*, loading event, A3; *D*, rough ground event, A4; *E*, loading event, A5; *F*, loading event, A6; *G*, rough ground event, A7.

B

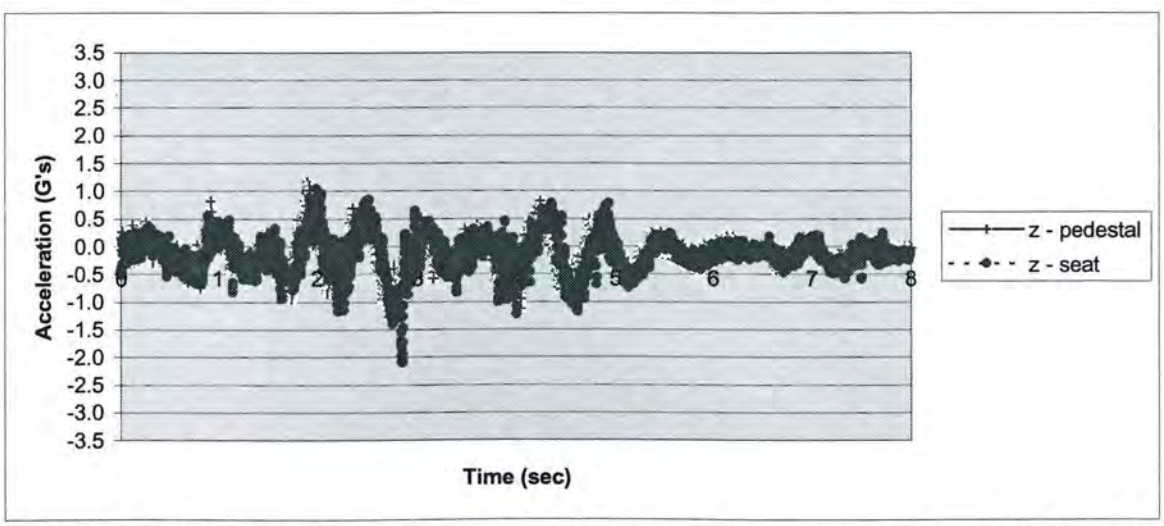
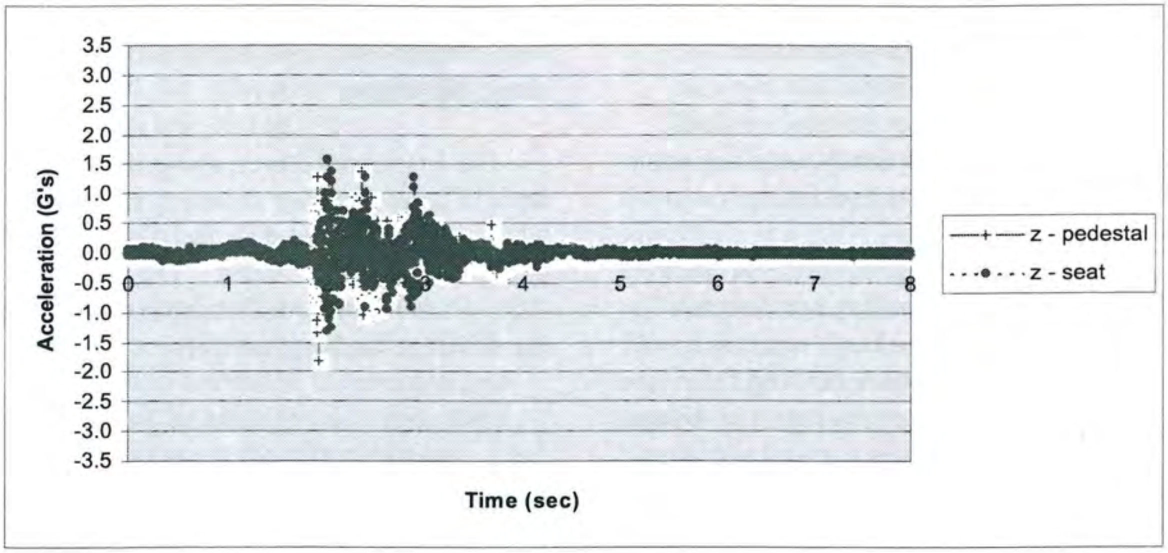
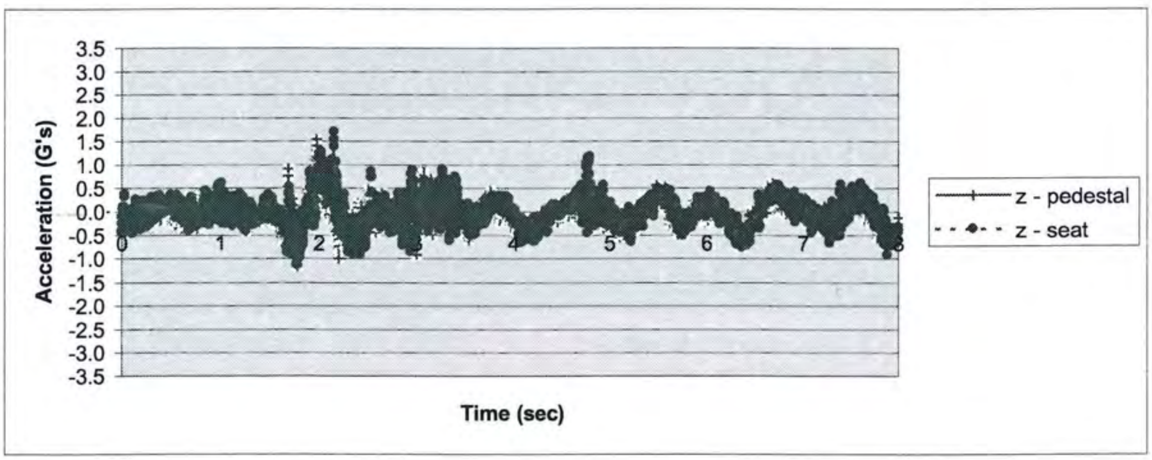


Figure 1.—Truck A shock events (continued). *A*, Loading event, A1; *B*, rough ground event, A2; *C*, loading event, A3; *D*, rough ground event, A4; *E*, loading event, A5; *F*, loading event, A6; *G*, rough ground event, A7.

C



D



E

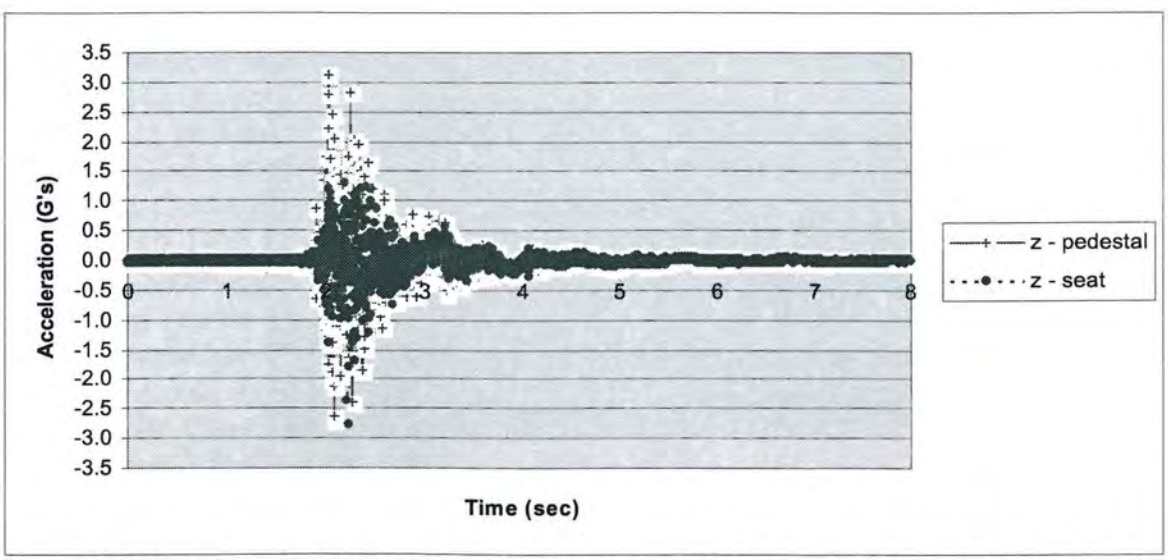


Figure 1.—Truck A shock events (continued). *A*, Loading event, A1; *B*, rough ground event, A2; *C*, loading event, A3; *D*, rough ground event, A4; *E*, loading event, A5; *F*, loading event, A6; *G*, rough ground event, A7

F

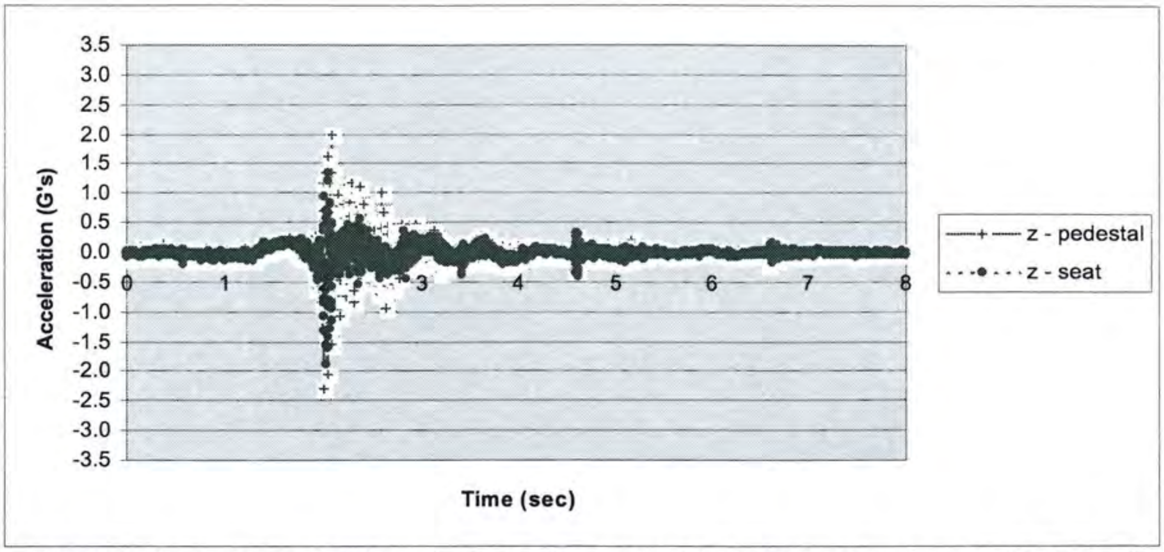


Figure 1.—Truck A shock events (continued). *A*, Loading event, A1; *B*, rough ground event, A2; *C*, loading event, A3; *D*, rough ground event, A4; *E*, loading event, A5; *F*, loading event, A6; *G*, rough ground event, A7.

G

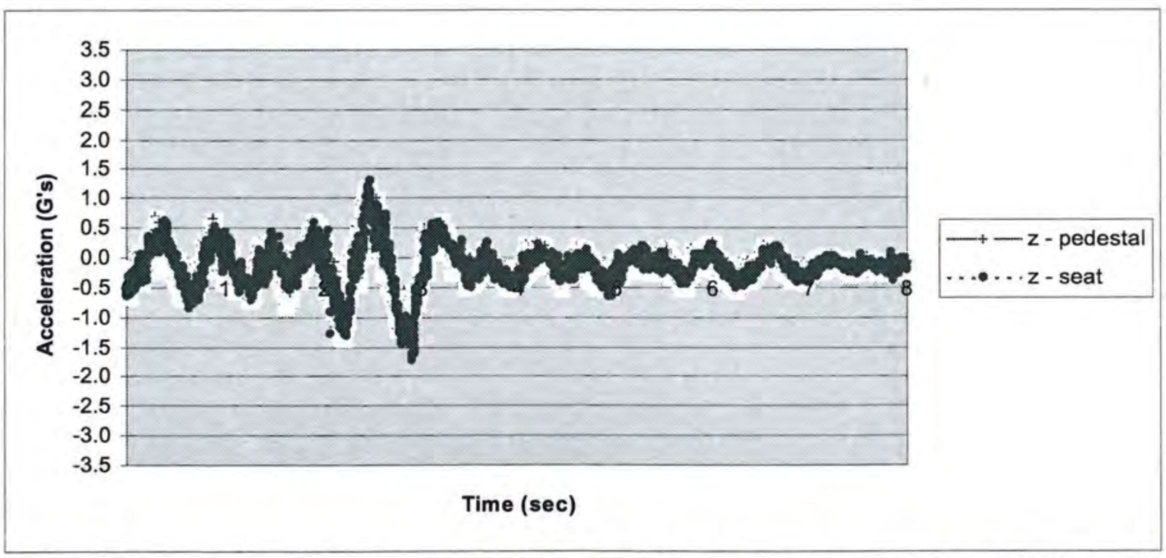


Figure 1.—Truck A shock events (continued). *A*, Loading event, A1; *B*, rough ground event, A2; *C*, loading event, A3; *D*, rough ground event, A4; *E*, loading event, A5; *F*, loading event, A6; *G*, rough ground event, A7.

A

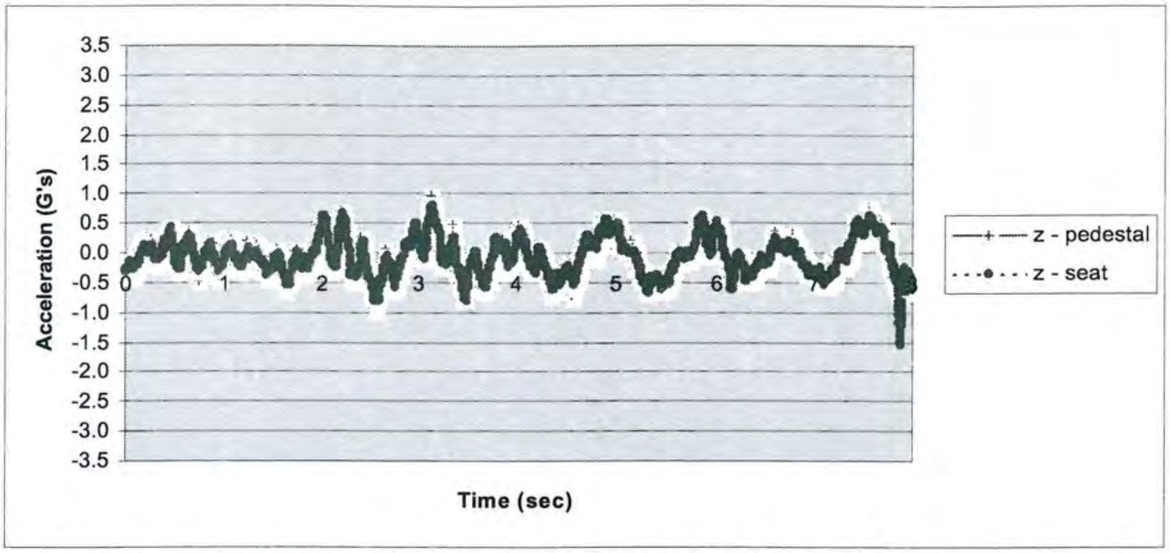


Figure 2.—Truck B shock events (continued). A, Event 1; B, event 2; C, event 3; D, event 4; E, event 5.

B

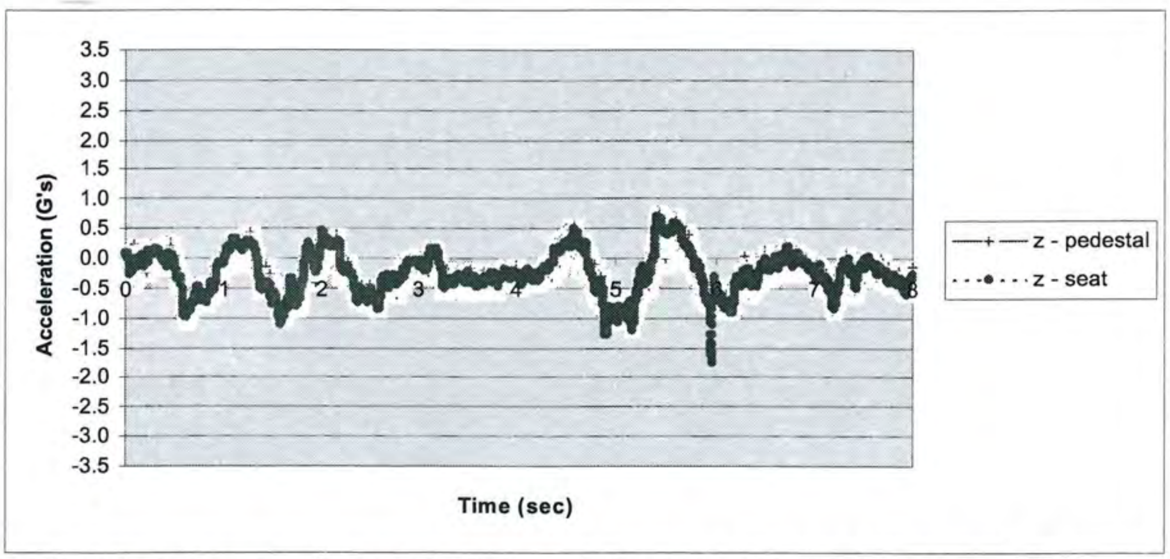


Figure 2.—Truck B shock events (continued). A, Event 1; B, event 2; C, event 3; D, event 4; E, event 5.

C

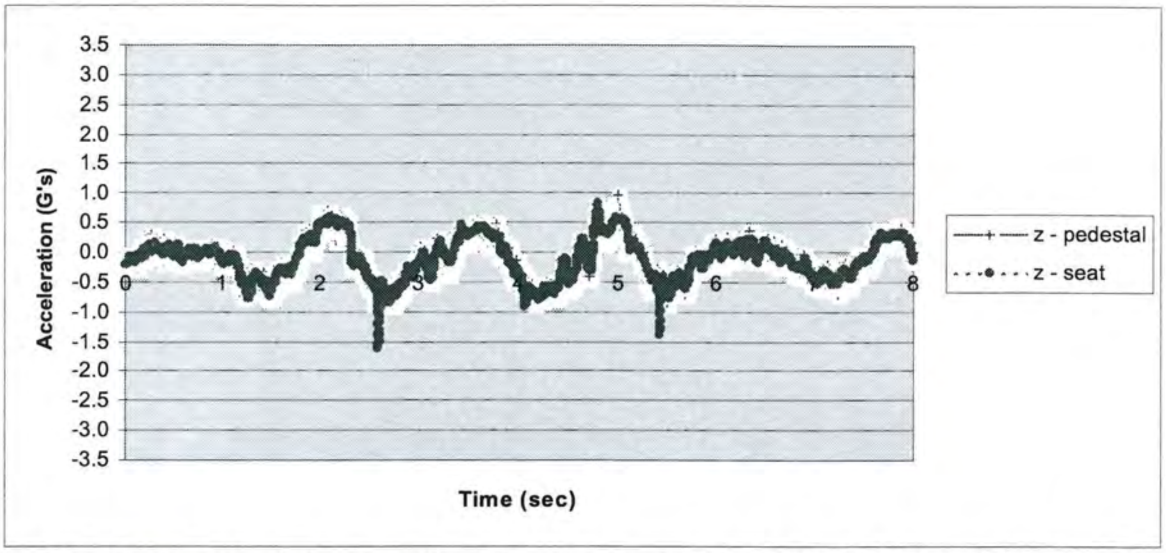


Figure 2.—Truck B shock events (continued). A, Event 1; B, event 2; C, event 3; D, event 4; E, event 5.

D

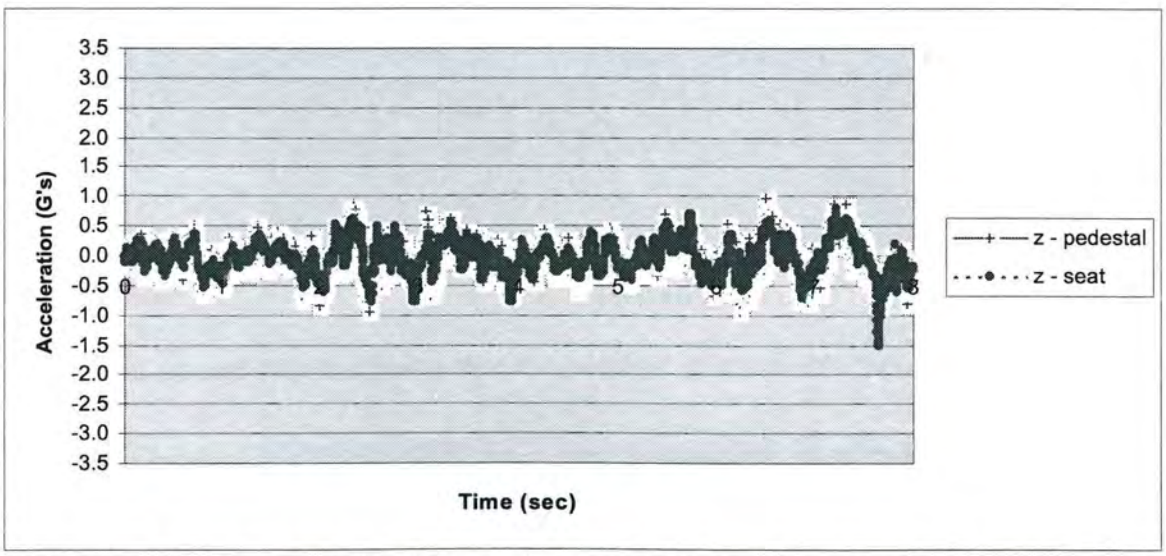


Figure 2.—Truck B shock events (continued). A, Event 1; B, event 2; C, event 3; D, event 4; E, event 5.

E

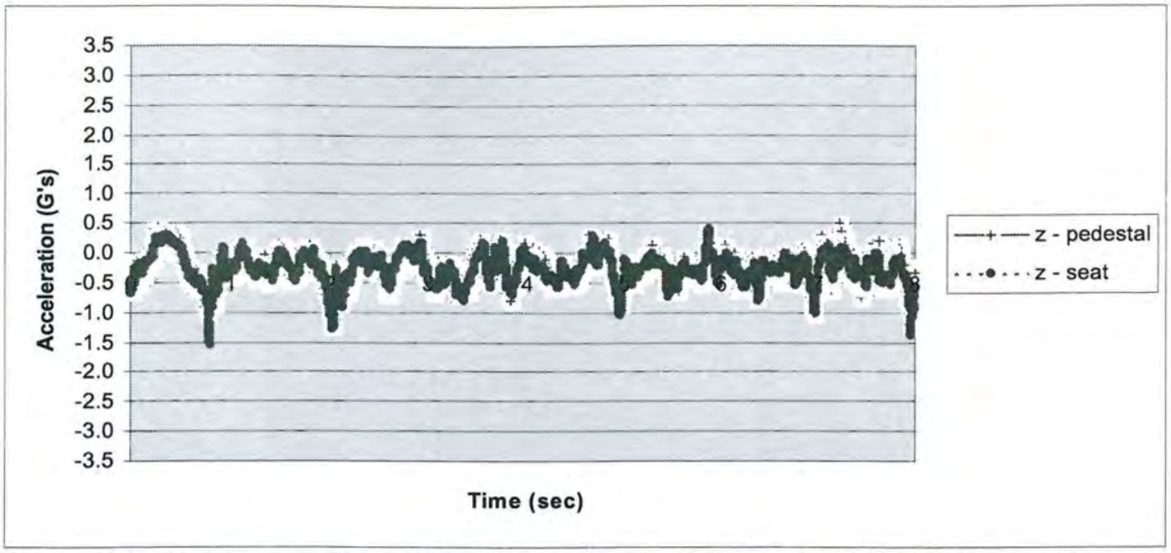


Figure 2.—Truck B shock events (continued). A, Event 1; B, event 2; C, event 3; D, event 4; E, event 5.

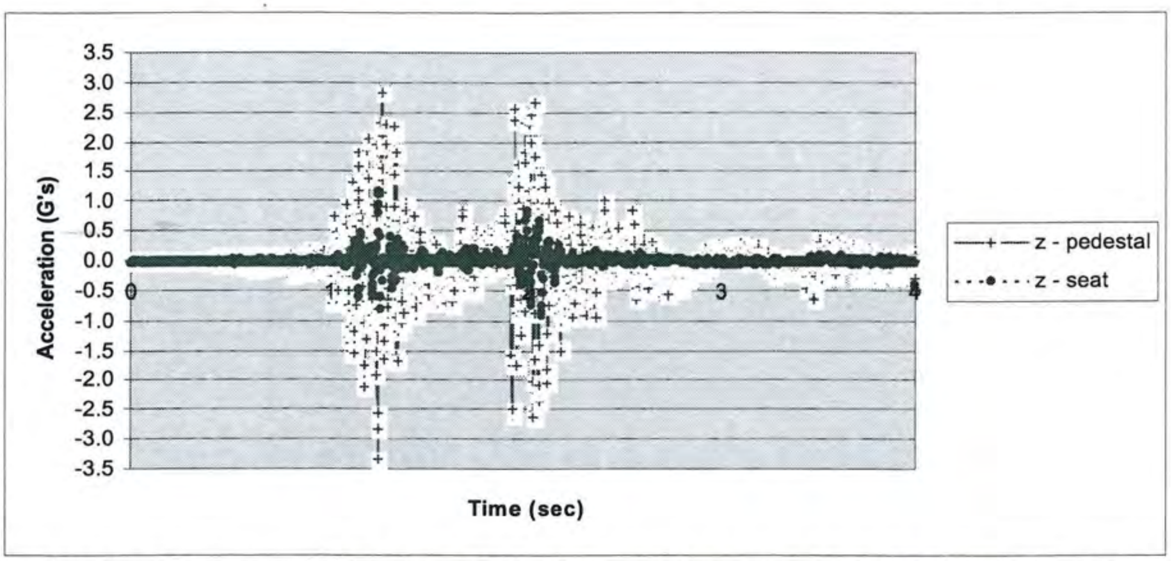


Figure 3.—Acceleration measured after dropping 1-1/2-ton rock from a height of 10 ft onto the bed of a haulage truck.

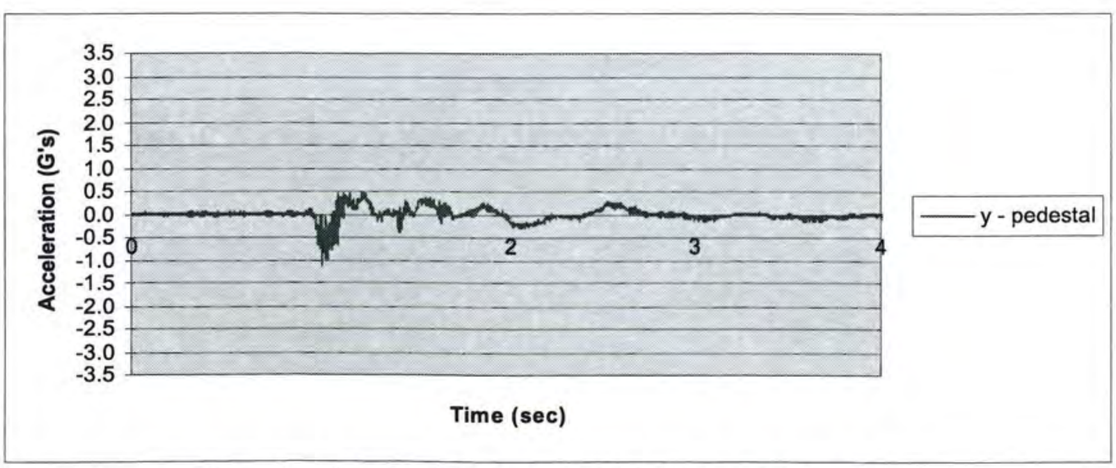


Figure 4.—Acceleration when truck hit in the side by a shovel.

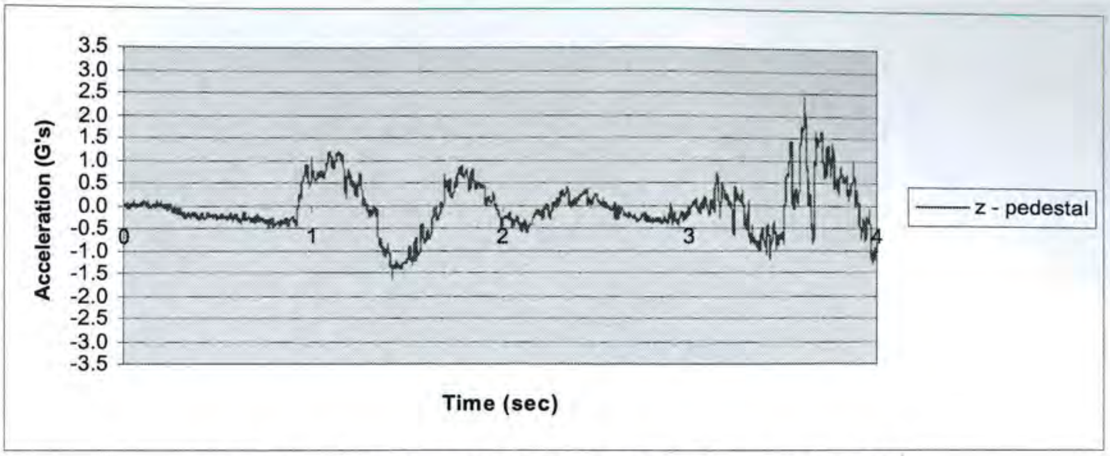


Figure 5.—Accelerations on truck going over course empty.

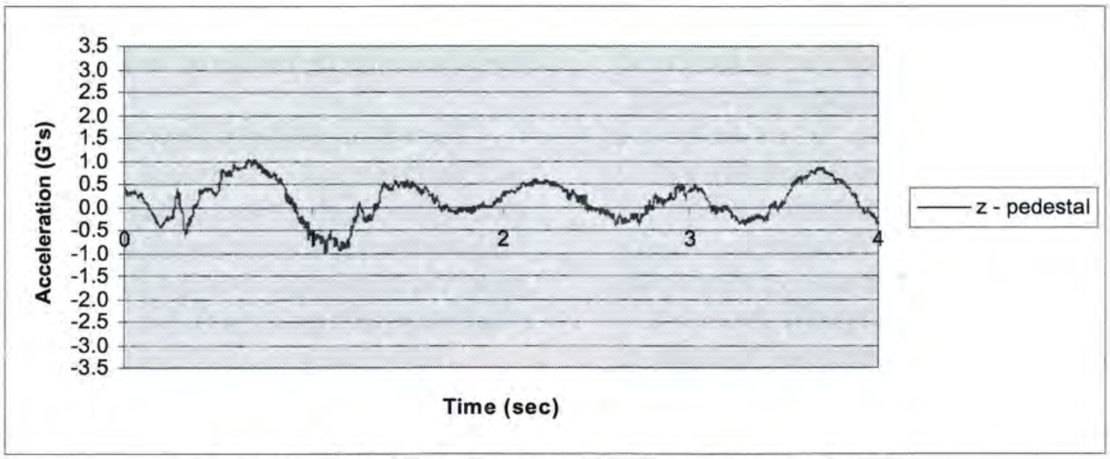


Figure 6.—Accelerations on truck going over course loaded.

GPS-ACCELERATION STUDIES

A typical epidemiological study involves recording when and where people react to a hazard and plotting the results on a map. To determine the frequency and causes of jolting and jarring, it is necessary to determine when and where these shocks occur, so establishing a relationship between jolt occurrence and location is important. A system that ties acceleration data with Global Positioning System (GPS) data was developed, assembled, and tested to provide an imprint of the jolts on a mine map (figure 7). This information will be of value in providing feedback to truck operators about how their driving affects jolting and jarring and identifying where haulage problems are.

Figure 7.—GPS-acceleration data system

Many verbal reports are available from truck drivers concerning where and how they received injuries while driving, but no way has been available until recently to establish where these injuries occurred. However, recent experiments with GPS by SRL researchers [2] indicate it is feasible to couple an accelerometer on the frame of a truck with GPS. The accelerometer would send signals to GPS hardware. When the truck is jolted or jarred, the shock would appear on the mine dispatcher's screen in real time. When a cluster of jolts in a particular location is displayed on the screen, corrective actions could be taken. One such action might be to provide information to a shovel operator that he or she is loading the trucks in a manner that jolts the operator. A rough spot on a haulage road could be resurfaced. The GPS could lead to refining information about what conditions most frequently cause or contribute to jolting and jarring. Investigations continue on what software components could allow the display of jolts on a computerized mine map in real time.

HUMAN RESPONSE TO SEATED JOLT AND VIBRATION ENVIRONMENTS

The human response to impact is closely related to vibration, but repeated impacts are fundamentally different in that there acceleration signatures are transient in nature. Vibration is typically present for long periods of time; a worker may be exposed for hours at a time. Impact, on the other hand, refers to a short burst of energy lasting only a fraction of a second. A large body of literature exists on the response to vibration, and in particular, vertical sinusoidal vibration. However, not much is known about impact response and the potential risks associated with repeated jolts experienced while operating many heavy off-road vehicles, trains, and similar industrial equipment.

The effects of impact on the body are not as well characterized as vibration. In fact, very little is known about the seated impact or sudden load environments in terms of the effect on the spine, the back musculature, or any number of other body systems.(2)

Dr. David Wilder (2), director of the Vibration and Seating Laboratory, Lower Spine Research Center, University of Iowa, has provided extensive data to this project. Understanding the human response to seated jolt, impact, and vibration environments holds significant promise for alleviating back problems in the workplace. The act of sitting can impose significant mechanical risk to the posterior aspect of the lumbar intervertebral disc. Repetitively imposed additional stress and strains from impact, vibration, and unexpected loads can put the posterior disc at increased risk. During a sudden load event, the erector spinae muscle groups in the lumbar region overcompensate for the sudden stimulus, potentially contracting with inappropriately high tension levels, thereby producing high loads on the intervertebral discs in an asymmetric manner. The muscle of the side opposite the impact contracts before the muscle of the near side, approximately a 20 ms lead [3]. That could enhance the effect of the side impact and contribute to an injury.

Many vehicle operators develop low back problems which may relate to exposure to vibration or repeated impact environments. Although poor muscle tone can be a contributing factor in back injuries, stiffness is not necessarily beneficial. In fact there are some circumstances, such as a driver twisting to look over his shoulder, which would increase the mechanical impedance of the spine and make it more susceptible to injury from a vertical jolt. This investigation has concentrated on factors which appear to be important and changes which could be constructive. Side impact has not yet been taken into account in truck seats as it has in the seat design for race cars.

CONCLUSION

In the sitting position, the posterior aspect of the lumbar intervertebral disc is placed at greater mechanical risk. Repetitively imposed additional stresses and strains from impact, vibration, and unexpected loads can put the posterior disc at increased risk. Multiple impact exposure challenges the seated individual's ability to prepare and cope. By documenting the body's response to these stresses we can then look at cockpit and design isolation systems and other engineering controls to reduce operator injuries.

1. Driving over rough ground and loading were the primary causes of the jolting and jarring events recorded at the surface mine. Jolting and jarring caused by dumping did not appear in the events recorded.

2. The double-strike hypothesis, where the first jolt sets up the driver for injury by a second jolt, was investigated, but no evidence of the double-strike was seen in the mine data. However, the possibility of its occurrence is evident in the rock drop experiment.

3. GPS can be used as an epidemiological tool for studying and characterizing jolting and jarring.

4. A side impact could cause a back injury.

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[1] International Standards Organization. 1997. Mechanical Vibration and Shock. Evaluation of Human Exposure to Whole-Body Vibration, Part 1: General Requirements, ISO 2631-1:1997(E).

[2] Miller, R., et al. 1999. Tying Acceleration and GPS Location Data To Create a Mine Management Tool. Presentation SME Annual Meeting, Denver, CO, Preprint 99-118.

[3] Fethke, N. B. Erector Spinae Response to Low Amplitude Single-Strike Lateral Impact. Masters Thesis, Biomedical Engineering, University of Iowa, May 2000

MSHA COURTESY TRUCK INSPECTIONS

James E. Beha

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The idea for penalty-free truck inspections arose during a series of 11 Mine Safety and Health Administration (MSHA) sponsored Independent Contractor Seminars held around the country during the fall of 1998. These seminars were in response to the increasingly high incidence of contractor fatalities. The seminars were successful in bringing together all segments of the mining industry to discuss steps that can be taken to reduce contractor accidents and fatalities.

While participation in the seminars was strong among other segments of the mining industry, truckers found it difficult to miss a day's haul to attend. It was at that point that J. Davitt McAteer, Assistant Secretary of Labor, Mine Safety and Health Administration, determined that MSHA needed to work harder to reach out to this group of contractors, especially since coal truck drivers had experienced the highest percentage of contractor fatalities.

Since 1984, there have been 48 coal mining fatalities in Virginia, West Virginia, and Kentucky, involving trucks. Independent Contractors were involved in 30 of those fatalities. All but 2 of the 19 trucker fatalities that have occurred since 1991 were independent contractors (fig. 1).

The majority of these fatalities resulted from the driver's inability to maintain control of the vehicle due to its unsafe operating condition, i.e. improperly maintained brakes, retarders, air systems, steering linkages and drivelines. Clearly, efforts to prevent these types of accidents would have to focus on the driver's knowledge of how to perform a meaningful truck examination, his desire or incentive to do so, and the need to repair defective components.

The coal trucking business is now dominated by independent contractors instead of hourly employees hired by coal companies to work as drivers and maintenance crew. Generally, contract coal haulers are paid by the load and responsible for their own equipment maintenance. Some do a fine job of inspecting and maintaining the trucks' critical safety related components. Others fall victim to the "it can't happen to me" syndrome and do little inspection and no maintenance. In other cases the desire to operate a safe truck is there, but the dollars to replace or repair are absent.

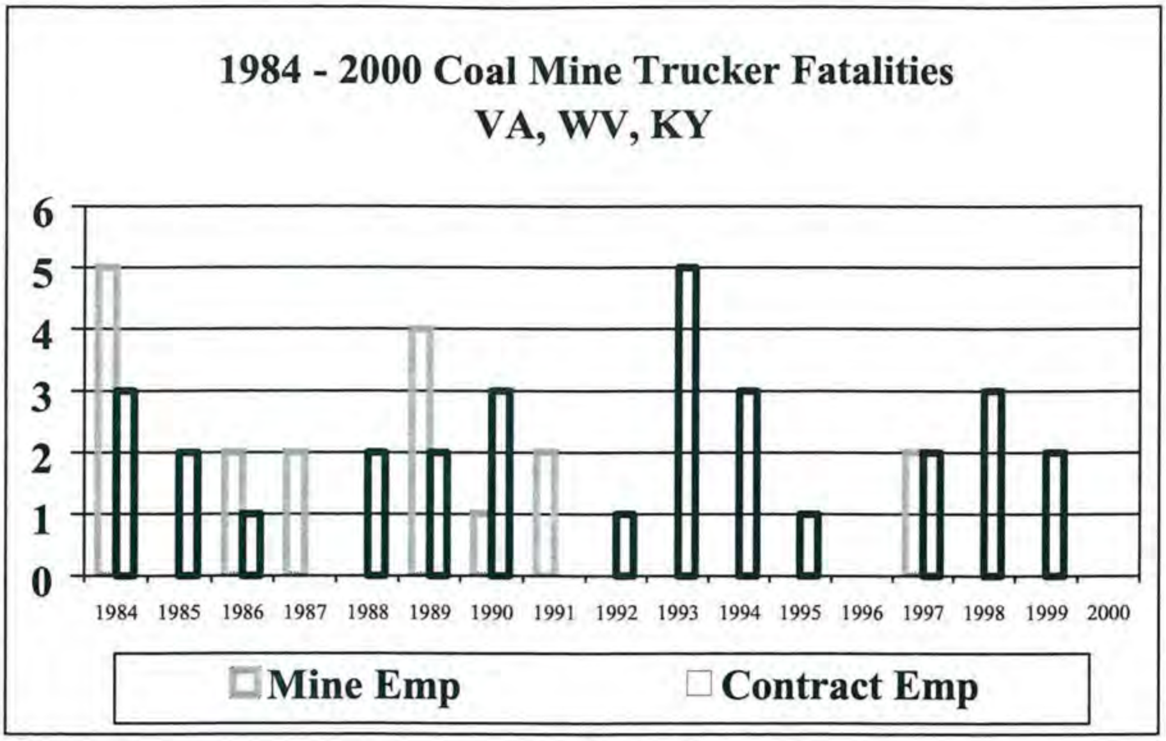


Figure 1

The paid by the load arrangement, while making economic sense in some situations, sets in motion forces that too often result in 70,000 gross vehicle weight rated trucks traveling on up to 22% grades at 150,000 actual GVW, and some with steering and braking deficiencies.

A coal truck with only two of six wheels braking when the pedal is applied, carrying twice its manufacture rated load, traversing a curvy mountainous haul road that would challenge a mountain goat, is not the prescription for a "nice or safe day on the job" (fig. 2).

Figure 2

Truck owner/driver absence from the national seminar series is easily understood. Parked trucks mean lost income. Owner/operators already hauling on low margins simply cannot afford to take a half day off to attend. Few of us would have been there if parking our truck to attend meant money out of our pocket. It became clear that improved trucker safety rested on our ability to establish better lines of communication with that important segment of the industry. In early 1999, MSHA laid the ground work for a series of voluntary, penalty free truck inspections in recognition that " if they won't or can't come to us, we'll go to them."

Courtesy truck inspections were designed to create a forum in which MSHA could:

- demonstrate that we are interested in the driver's health and safety
- heighten driver awareness of the causes of previous trucking accidents and prevention of similar ones
- ensure that each driver knows how to conduct a meaningful pre operational inspection of his vehicle
- review and clarify owner/driver responsibilities for compliance with mandatory standards
- demonstrate that MSHA is willing to work with the industry to try new methods to solve old problems
- establish a rapport with drivers on which future interactions could be built

It was determined that the initiative's chances of being successful were greatly increased to the degree the following parameters could be met:

1. Advertised - truckers had to be aware courtesy inspections were available (fig. 3,4)
2. Convenient - conducted at sites along regular haulage routes
3. Voluntary - conducted off mine property
4. Penalty free - no violations or fines
5. Expeditious - done in short amount of time so that truck downtime kept at minimum
6. Informative - provide information to drivers that they feel to be useful (fig. 5,6)
7. Substantive - conduct high quality inspection of truck components (fig. 7,8)
8. Personable - drivers must feel comfortable, not threatened or intimidated by inspectors (fig. 9)

Figure 3

MSHA offers free inspections of coal trucks

Goal is to cut traffic deaths, non-fatal wrecks

Figure 4

Figure 5

Figure 6

Figure 7

Figure 8

Figure 9

Figure 10

Beginning in the spring of 1999, courtesy truck inspections were conducted in West Virginia, Virginia, and Kentucky, by inspection teams from MSHA District's 4, 5, 6, and 7 respectively. Inspection teams performed roadside courtesy truck inspections at mobile stations, off mine property, near routes commonly traveled by coal haulage trucks. During the inspections, drivers joined inspectors in a thorough 54 point examination of their trucks (fig 10). Inspectors explained what they were examining and how to check the various components and systems. Drivers frequently asked questions about inspection procedures, regulations, and the safety implications of various safety defects. Drivers were provided instruction in the proper way to perform effective pre-operational checks, given a short videotape that summarizes the procedure, and safety material packets that specifically address safe truck operation and maintenance (fig. 11).

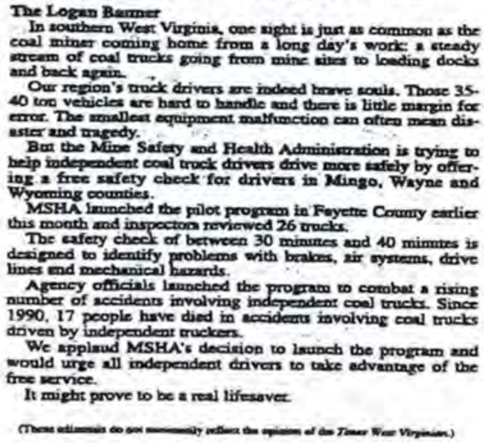
Figure 11

Drivers of 488 trucks voluntarily pulled their trucks into one of 21 inspection sites established in West Virginia, Virginia, and Kentucky. Over 113 Independent Contractor Trucking Companies participated. No defects were found on 132 of the trucks inspected. Problems were found on 292 trucks that would have resulted in a violation during a regular MSHA inspection. The most serious type of problems, those meeting the North American Uniform Out-of Service Criteria for Commercial Vehicles presenting a danger to the trucker and potentially anyone else in the vicinity of the truck while it is operated, were found on 50 of the trucks. These trucks were voluntarily taken out of service and repaired.

Driver response was very positive. Many drivers indicated that these were the most thorough and informative inspections they had ever received. Drivers seemed genuinely appreciative when our inspection team pointed out problems with the trucks that could potentially cause an accident if not repaired.

The complimentary inspections resulted in several positive outcomes. Among them are:

- better working relationship with the trucking segment of the industry
- drivers who are more aware of the importance of regular truck inspection and maintenance
- drivers who know how to conduct thorough pre-op inspections of their trucks
- drivers who understand MSHA's truck inspection procedure
- drivers who know the causes of recent trucking fatalities
- general acceptance by the industry, contractors, and the public at large that these inspections are a "good thing to do" (fig. 12)



MSHA's decision applauded

The Logan Banner

In southern West Virginia, one sight is just as common as the coal miner coming home from a long day's work: a steady stream of coal trucks going from mine sites to loading docks and back again.

Our region's truck drivers are indeed brave souls. Those 35-40 ton vehicles are hard to handle and there is little margin for error. The smallest equipment malfunction can often mean disaster and tragedy.

But the Mine Safety and Health Administration is trying to help independent coal truck drivers drive more safely by offering a free safety check for drivers in Mingo, Wayne and Wyoming counties.

MSHA launched the pilot program in Fayette County earlier this month and inspectors reviewed 26 trucks.

The safety check of between 30 minutes and 40 minutes is designed to identify problems with brakes, air systems, drive lines and mechanical hazards.

Agency officials launched the program to combat a rising number of accidents involving independent coal trucks. Since 1990, 17 people have died in accidents involving coal trucks driven by independent truckers.

We applaud MSHA's decision to launch the program and would urge all independent drivers to take advantage of the free service.

It might prove to be a real lifesaver.

Figure 12

Drivers/Owners and Contract Coal Haulers continue to show an interest in having MSHA inspectors come to their shops, offices, and work areas to train drivers on the importance of quality equipment inspections.

MSHA

COURTESY
INSPECTION

TECHNICAL SESSION IV

TRAINING WORKSHOP

DEVELOPING AND USING STREAMING MEDIA FOR DISTANCE LEARNING AND TRAINING

Jason Lockhart, Director

Media Development and Emerging Technologies
Virginia Tech University

Editor's note: This manuscript is based on a PowerPoint presentation made at the 31st Annual Institute on Mining Health, Safety and Research.

Primary Services

The Multimedia Lab provides a number of primary services to the engineering departments at Virginia Tech. Among the primary services we provide are media and informational technology consulting, training, interactive web page development, interactive CD development, an open lab, and research services.

Our consulting services consist of content creation, content conversion, interactive development, user interface design, systems integration, and network delivery.

The training we provide includes two hour short courses in both fall and spring FDI's, as well as three day intensive track FDI's during the summer. We do remote site training in one, two, and three day sessions, and we also lead one-on-one application specific training sessions.

We encourage web development at a number of different levels by providing base level HTML design services, custom DHTML and Javascript interfaces, custom Flash or director Shockwave Components, and maintenance and support.

Our CD development services include full end-to-end design and delivery, custom interface design, content conversion and creation, and customized delivery modes.

Another important feature of our lab is that it is an open lab. This means 24 hour access, seven days a week; first come, first served utilization; 40 hour a week support; a full range of content creation and conversion tools; state of the art equipment; and loaner hardware.

The research component of our mission includes research on emerging technologies, effects of these technologies on the teaching and learning process, and best practice research.

Technology Overview for Streaming Media Development

There are four main issues one needs to be concerned with when developing streaming media applications for training purposes: hardware, software, standards and streaming formats, and delivery issues.

Technology

The first piece field production hardware that you should consider is a video camera. If you are on a tight budget, you can get by with a single chip miniDV camera for about $1,200. If your budget permits, I recommend a 3 chip miniDV Sony DCR-TRV900 which costs about $1,800.

For audio recording, I recommend the Sony MZ-R70, a portable minidisc recorder that costs about $300. You must then decide between wired or wireless microphones. You can get a reliable wired microphone for about $30, whereas high quality wireless microphones such as the SHURE UHF Presenter run around $350.

You will also need a photographic light kit—about $1,500—and a photographic tripod, which will cost around $120.

OK, now you've got your field production hardware, but you also need post-production hardware: a PowerMacintosh or a High-End Pentium III Workstation. The minimum expense you should expect here is $2000. You will need a single processor 400 MHz or better, 256 MB RAM, 36 GB SCSI HD in addition to the internal drive (about $800 with a controller), a FireWire (IEEE 1394) interface card (about $100), and an analog to DV media converter (about $350). Again, these are the minimum requirements for your hardware. I recommend spending around $3,500 for a dual processor 500 MHz or better, 512 MB RAM, two 50 GB SCSI HD's (RAID 0) (about $2,500 with a controller and RAID SW), a tape backup system (about $2,000), and a CD-R/RW Drive (about $300).

Software

I recommend the following software for creating high tech training materials. For image editing and compositing, consider Adobe Photoshop (about $600). Abode Premier, about $500, will suffice for video editing and post-production. Macromedia Freehand, about $350, provides you with a vector graphics drawing tool. And finally, Macromedia Director 8 Studio, about $900, will give you an animation and interactive development tool.

For technology, then, you are looking at a minimum cost of $9,100, but I recommend investing in equipment that will cost about $14,750. Unfortunately, there are still other costs to consider: media, and support and maintenance.

Standards

When it comes to video standards, you have the choice of analog or digital. I will discuss digital first, but when it comes to digital video and standards I am reminded of the saying, "the bleeding edge is a fun place to play, but I wouldn't want to live there."

Digital content production consists of the following steps: creating your source, capturing (putting it in the computer), editing, compressing, and distributing.

Another quotation that digital video reminds me of comes from C. P. Scott: "Television? The word is half Greek and half Latin. No good will come of it." Let me explain by providing you with a bit of history on video. The system we use is NTSC (National Television Standards Committee). The current format began in 1935, standards were adopted in 1941, color was added in 1953, and technology has remained virtually unchanged since then. The picture is 30 frames/second, which is actually 60 interlaced half frames (fields)/second, which have a maximum resolution of 720x486.

Color is simply an afterthought in this situation. The black and white portion of the signal contains most of the detail, and color based on RGB additive primary colors is simply painted on with a broad brush.

Now, let's do the math here. With a maximum resolution of 720x486, you've got 349,920 pixels. Multiply this by 30, the number of frames per second, and you've got 10,497,600 pixels per second. 10.5 million approximate bits pixels/sec x 24 bits = 253 million bits/sec, and that's without audio. Thus, broadcast production format is 270 mb/sec. That's like sucking the ocean through a soda straw!

Now let's look at Internet delivery bandwidth. The common modem is 56K (bits/sec). Network and VA Videoconferencing uses 1 M. Compressed NTSC, at 60:1, takes 4.5 M, so for 8 bit NTSC you need 115M. Now remember, broadcast TV production uses 270M.

In the best case scenario, a 56K modem has less than 5% of the bandwidth of a 1xCD ROM. A 10-BaseT Ethernet connection equals approximately an 8xCD ROM. However, rapid advancements in compression technology have made low bandwidth delivery possible. At the same time, advances in network technology such as 56K modems, DSL, cable modems, and LAN have increased the available bandwidth. Nonetheless, not only is this still like trying to suck the ocean through a soda straw, in this case we don't actually own the entire soda since others may be sharing it (or hogging it)!

Another extremely important consideration in Internet delivery is the quality of service; that is, the ability to guarantee timely delivery of data. Video needs a relatively large bandwidth and cannot tolerate delay. Data loss affects audio and video and there is no time to resend data, so you need priority routing.

Let me now compare HTTP and RTSP. HTTP stands for HyperText Transfer Protocol and is the most common web transfer protocol. It will attempt to transfer data as fast as possible, and can use a "generic" web server. HTTP's progressive download allows users to watch while downloading; this is sometimes called HTTP streaming. HTTP may not work well with long programs or live programs, and it may not get priority routing in current (or near future) network designs.

RTSP—Real Time Streaming Protocol—is a true streaming media protocol. It attempts to deliver media to the user as needed in a smooth, consistent manner. RTSP generally needs special server architecture. It is the best choice (and possibly the only choice) for long programs and live shows, and it will eventually get priority routing through many networks.

Internet Delivery Formats

Real Networks' SureStream™ allows for simultaneous creation and distribution of more than 6 data rates in a single file. Its wide market penetration means many people have the player already loaded, and it allows you unlimited webcast viewers.

Microsoft Media Technologies provides another delivery format choice and it is FREE!! It uses MPEG 4, has a multiple bit rate capability, and a wide range of bandwidths. Did I mention it was FREE?! This format has strong potential for continued growth and the player comes installed on most Windows machines.

The next delivery format option I want to talk about is Quicktime. The MPEG 4 used by Microsoft is in fact based on Quicktime. Quicktime has a wide range of bandwidths and the player comes preinstalled on most Apple machines. It has good quality at Internet bandwidths and it is also FREE, but you need a hefty server running OS X.

Cisco IPTV will provide you with high bandwidth video and is a good integrated content manager. It is a well engineered system designed for intranet use and tests show it has a high reliability. Cisco IPTV is currently in use at Virginia Tech for a specific project and is not generally available for public use at this time.

To see a demo on Internet delivery formats visit the web page at the following address: www.vbs.vt.edu/dv/dvpage.html.

Digital Content Production

Content production is composed of five considerations: source, capture (put it into the computer), editing, compression, and distribution. You want to start your production with as high a quality source as possible. Lighting is critical here. Steady camera work is also very important so use a tripod whenever possible; otherwise, move the camera as little as possible. Finally, TEST YOUR SOURCE MATERIAL EARLY!

For the capture phase of production, a wide variety of capture devices are available. Capture at the highest rate and quality possible, which may require many Mb/sec, and if possible capture uncompressed signal. Archive your source material and test it.

When it comes to editing, choices abound. Just a few of the utilities available for editing include Media 100, Adobe Premier, Final Cut, Avid Cinema, Strata Videoshop, Movie Player, or I-Movie (included with current Macs), After Effects, and Quicktime Pro. Edit in as high a quality as possible. And, as always, test, and then archive the finished product.

Compression is perhaps the most crucial step in the production process. Determine your target audience and distribution system to help you determine the final data rate(s). Factors to consider in compression include frame size/resolution, frame rate, and audio quality. For Internet delivery, a good rule of thumb is to compress to about 2/3 of the target bandwidth (i.e. 56K modem = 40 Kbps). Then test.

Distribution can be done with CD ROM, the Internet (either HTTP or RTSP), and DVD. Once again, test. I know this is getting redundant, but it is hard to overemphasize the importance of testing.

Tips

So you're asking yourself, "I have all this hardware…now what?" Well, I have a friend who used to say, "Just because you have the ultimate driving machine, doesn't make you the ultimate driver." So, tip number one: Hire someone who knows what they are doing! Simply having the hardware and software doesn't mean you can produce effective training materials. Tip number two: hire someone who knows what they are doing! Third, look into Train-the-Trainers workshops. And finally, hire someone who knows what they are doing!

Uses in Training

Distance learning and training is most effective for process instruction. It is also particularly suited for training individuals in a just-in-time mode, as well as for training people in remote sites from a central location without any travel involved.

Process examples of on-line training include 14 Bus Short, Acceptable Use Policies, and Clean Room Construction Protocols. Examples of just-in-time training include any situation in which an individual employee needs training. The training is always available when needed, right here, right now. No scheduling of facilities or time off is necessary, and all materials are available for review after the training is completed. Other benefits of on-line training include the ability to update your training instantaneously, and to train anyone, anywhere in the world.

Thank you. At this point, I would like to take any questions that you might have.