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Overview of Ground Control Research for Underground Coal Mines in The United States

CMark
National Institute for Occupational Safety and Health, Pittsburgh Research Laboratory, Pittsburgh, PA, USA

ABSTRACT: Underground coal mining continues to evolve in the U.S., and more reserves are being mined under deeper cover, with worse roof, or with interactions from previous workings. At the same time, the mining community is responding to higher safety standards and intense competitive pressures. The need for effective ground control design has never been greater. Ground control safety issues that have been addressed by recent the National Institute for Occupational Safety and Health (N10SH) research include: Improving roof support performance; Maintaining safe tailgate escapeways from longwalls; Optimizing pillar design for retreat mining; Controlling multiple seam interactions; Predicting roof conditions during extended cuts, and; Preventing massive pillar collapses. As funding from both government and the private sector has diminished, the emphasis in research has focused on providing the mining community with practical techniques for improved ground control design. Many projects have successfully employed empirical methods that emphasize the statistical analysis of case histories from underground mines. Other projects have employed numerical models and large-scale laboratory testing of roof support elements. Using these data, NIOSH has developed an entire toolbox of computer programs that have been effectively transferred to the mining community.

1 INTRODUCTION

Roof falls have been the single greatest hazard that underground coal miners face in the U.S. Throughout the 20 century, roof falls accounted for approximately half of all deaths underground. While overall safety in U.S. coal mines has improved dramatically in the last 50 years, fatality rates continue to exceed other major industrial sectors (Fig. 1). Fatalities due to ground falls still make up a significant portion of this rate.

Currently, underground coal production in the U.S. is split almost 50-50 between large longwall mines and smaller, room-and-pillar mines. Most longwalls operate at depths of cover in excess of 300 m. Room-and-pillar operations are still primarily at shallow depth, often working small, irregular deposits that were abandoned by earlier miners. Approximately 20% of the room-and-pillar coal is won on retreat faces (Mark et al> 1997a).

Today's underground coal industry faces intense competitive pressures from the $4/ton Powder River Basin strip mine coal and from the pace-setting million-ton-per-month longwalls. Ground failures can hardly be afforded in this climate, yet they continue to occur. Some examples:

Roof falls: In 1998, more than 1,800 unplanned roof falls occurred where the roof had already been supported. While few of these resulted in injuries, each one represented direct threat to Ufe and limb, and an indirect threat to ventilation, escape, and equipment. In each of these roof falls, the majority of which occurred in intersections, die roof bolt system failed to perform successfully.

Massive Collapses: In 1992, miners were splitting pillars at a southern West Virginia mine when the fenders in a 2.3 ha area suddenly collapsed. The miners were knocked to the floor by the resulting air blast, and 103 ventilation stoppings were destroyed. At least 12 similar events have occurred in recent years, miraculously without a fatality (Mark et al., 1997b).

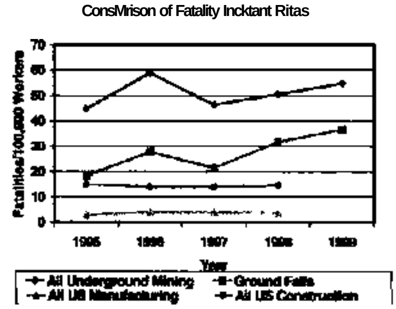


Figure 1. Fatality rates in mining and other U.S. industrial sectors.

Pillar Squeezes: At a Kentucky coal mine, pillars were being extracted in the main entries under 270 m of cover. The pillars began to crush in response to the vertical load, resulting in a roof fall that killed two miners. This incident is an extreme example of hazardous conditions that can be associated with slow pillar failure. Research has identified at least 45 recent instances of pillar squeezes in room-and-pillar mines (Mark and Chase, 1997).

Longwall Tailgate Blockages: In 1984,26 miners at the Wilberg Mine in Utah could not escape a deadly fire because of a tailgate roof fall. Similar blockages were common in the 1980's, and 50 cases have been documented (Mark, 1992).

Multiple Seam Interactions: Studies indicate that the majority of remaining room-and-pillar reserves, and 33% of longwalls, will be subject to multiple seam interactions. At one West Virginia mine where four seams had previously been extracted, a fatality occurred when the roof collapsed without warning beneath a barrier pillar.

The National Institute for Occupational Safety and Health (NIOSH) has the primary responsibility for conducting research to reduce mining hazards in the U.S. NIOSH continues the tradition of the U.S. Bureau of Mines, which was closed in 1995. Mining research is conducted at two Research Laboratories, one in Spokane and the other in Pittsburgh.

The past 20 years has seen a steady decline in the resources devoted to ground control research. The labor- and instrumentation-intensive field studies of past years are rarely feasible today. As a result, NIOSH scientists have had to develop new approaches to conducting ground control research.

2 THE COAL MINE ROOF RATING (CMRR)

One approach that has proven exceptionally successful for solving complex problems İs the empirical, or statistical, approach- It relies on the scientific interpretation of actual mining experience represented as case histories. For example, hundreds of longwall and room-and-pillar panels are mined each year, and each one is a full-scale test of a pillar design. Once data has been collected on enough of these case histories, statistical techniques can be used to determine those combinations of factors most likely to result in pillar failure. A key advantage is mat critical variables may be included even if they are difficult to measure directly, through the use of rating scales. A significant breakthrough was the development of a rock mass classification system specifically applicable to coal mine roof.

Coal Mine Roof Rating (CMRR) was proposed because neither traditional geologic reports nor laboratory strength tests on small rock samples adequately described me structural competence of mine roof. The CMRR combined 20 years of research on geologic hazards in mining with worldwide experience with rock mass classification systems (Molînda and Mark, 1994). Field data was collected from nearly 100 mines in every major U.S. coalfield (Figure 2).

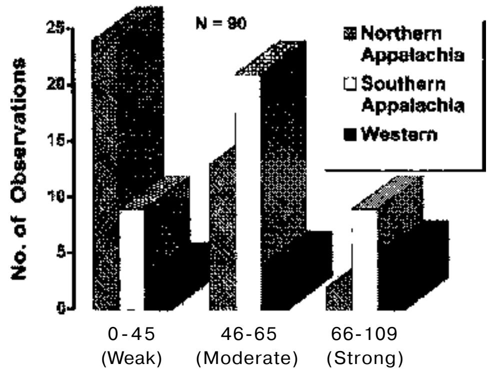


Figure 2 The Coal Mine Roof Rafing observed in various U S. coalfields.

The CMRR weighs the geotechnical factors that determine roof competence, and combines them into a single rating on a scale from 0 to 100. The underlying philosophy of the CMRR is that it is not the strength of the intact rock that determines the stability of a mine roof, but rather the defects or discontinuities which weaken or destroy the roof beam.

The CMRR makes four significant contributions:

- Focuses on the characteristics of bedding planes, slickensides, and other discontinuities that weaken the fabric of coal measure rock;
- Applies to all U.S. coalfields, and allows meaningful comparison even where lithologies are quite different;
- Concentrates on the ability of the immediate roof to form a stable structure, focusing on the characteristics of the strongest bed within the bolted interval, and;
- Provides a methodology for geotechnical data collection.

Originally, the data for the CMRR was collected at underground exposures like roof falls and overcasts. To make it more generally useful, procedures were developed for determining the CMRR from drill core (Mark and Molinda, 1996). The drill core procedures employ the Point Load Test to estimate the uniaxial compressive rock strength and the rock strength parallel to bedding. A new conversion factor from point load index of strength (Is(so)) to uniaxial compressive strength (UCS) has been determined from a large data base provided by a large U.S. coal company (Rusnak and Mark, 2000).

The CMRR has found many applications in ground control research and mining practice, as described in many of the examples below. It has also been successfully applied in Australia and South Africa (Colwell et al., 1999; Mark, 1998; Mark, 1999). A computer software package has recently been developed that makes the CMRR easier to use and to integrate into exploratory drilling programs.

3 DESIGN OF LONGWALL GATE ENTRY SYSTEMS

In the fifteen years after 1972 the number of U S. longwall feces grew from 32 to 118 (Barczak, 1992). The new technology created a host of operational and safety problems, including the maintenance of stable travelways on the tailgate side (Figure 3). Researchers initially viewed gate entry ground control primarily as a pillar design issue. The clear correlation between larger pillars and improved conditions that had been established by tnal-and-error at many mines supported this approach.

In comparing longwall pillars to traditional coal pillars, „the *most* obvious difference is the loading. Longwall pillars are subjected to complex and severe abutment loads arising from the retreat mining process. The loads are also changing throughout the pillafs service lives. The major contribution of the original Analysis of Longwall Pillar Stability (ALPS) was a formula for estimating the longwall pillar load, based on numerous underground measurements (Mark, 1990).

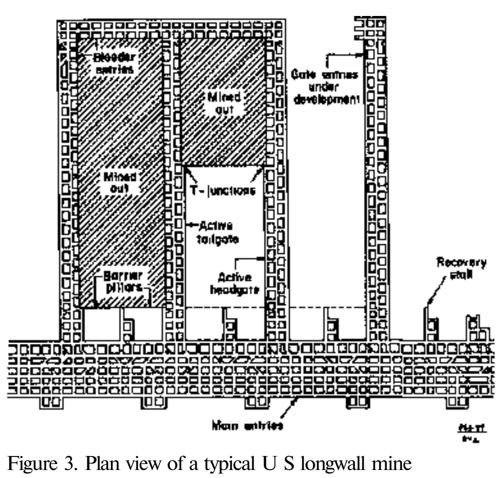


Figure 3. Plan view of a typical U S longwall mine

It became clear, however, that tailgate stability required more than good pillar design. Other factors, such as roof quality and artificial support, must be important. Data were collected from approximately 55% of all U S. longwall mines, selected to represent a geographic and geologic cross-section of the U.S. longwall experience. A total of 64 case histories were classified as "satisfactory" or "unsatisfactory." Unsatisfactory conditions almost always caused the mine to adjust their design m future panels. Satisfectory designs were used for at least three successive panels without significant ground control delays.

Each case history was described by several descriptive variables, including the ALPS stability factor (SF), the CMRR, entry width, and primary support rating. Multİ-variate statistical analysis showed that when the roof is strong, smaller pillars can safely be used (Mark et al., 1994). For example, when the CMRR is 75, the an ALPS stability factor (SF) of 0.7 is adequate. When the CMRR drops to 35, the ALPS SF must be increased to 1.3 (figure 4). Significant correlations were also found between the CMRR and both entry width and the level of primary support.

Since 1987, ALPS has become the most widely-used pillar design method in the U.S. The ALPS-CMRR method directly addresses gate entry performance, and makes U.S. longwall experience available to mine planners in a practical form. Tailgate blockages are far less common today than they were 10 years ago, and ALPS can surely claim some of the credit.

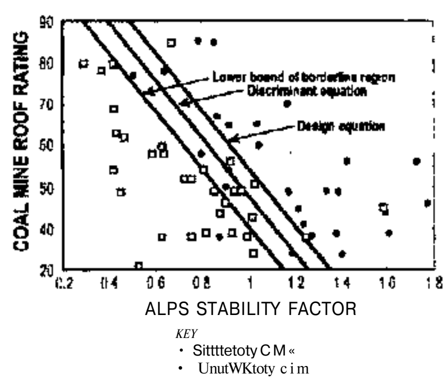
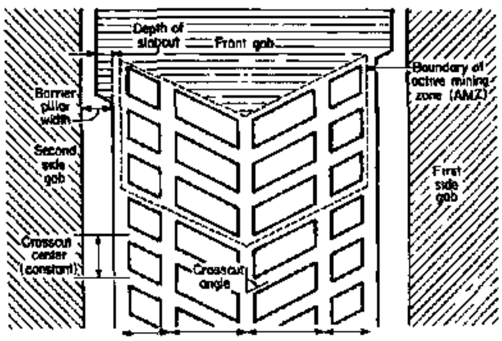


Figure 4. ALPS case history data base and design guidelines

4 PILLAR DESIGN FOR RETREAT MINING

The classical empirical pillar strength formulas were all developed for room and pillar mining However, none ever attempted to consider the abutment loads that occur during pillar recovery operations. The abutment load formulas used m ALPS provided a means to rectify that shortcoming.

The Analysis of Retreat Mining Pillar Stability (ARMPS) employs the same basic constructs as ALPS, adapted to more complex and varied mining geometries (Mark and Chase, 1997). The abutment load formulas have been adapted to three dimensions, to account for the presence of barrier pillars and previously-extracted panels. Features such as varied entry spacings, angled crosscuts, and stab cuts in the barrier can all be modeled (Figure 5).



Entry centos (variable]

Figure 5. Model of a room-and-pillar mining section used by the ARMPS program.

To evaluate me validity of ARMPS, more than 200 retreat mining case histories were obtained from field visits fliroughout the U.S. When the entire data set was evaluated, it was found that 77% of the outcomes could be correctly predicted simply by setting the ARMPS SF to 1.46 (Figure 6). When the data set was limited to cases where the depth of cover (H) was less than 200 m, die accuracy improved to 83%. The conclusion seems to be that ARMPS works quite well at shallow depth and moderate width-to-height (w/h) ratios (Mark, 1999). Research is currently underway to determine what other factors need to be included when designing squat pillars at great depth.

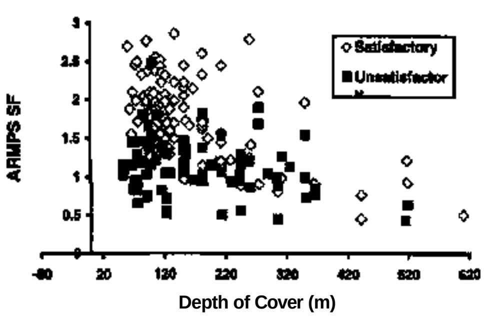


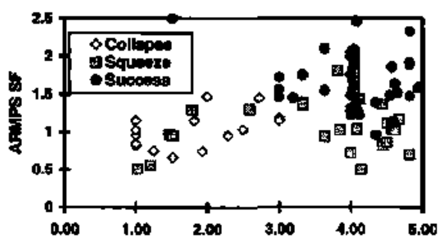
Figure 6. The ARMPS case history data base.

The study also answered some ancient questions regarding the value of laboratory tests to determine the UCS of coal specimens. The analyses clearly showed that UCS was of no value whatever in predicting the strength of coal pillars, thus confirming the results of an earlier study (Mark and Barton, 1996). It also found that the best results are achieved with ARMPS when the in situ coal strength is assumed to be 6.2 Mpa. The study concluded that while the in situ strength of U.S. coal seams is probably not uniform, laboratory tests do not measure the geologic features (like bedding planes and rock partings) which are most likely responsible for variations in seam strength.

S MASSIVE PILLAR COLLAPSES

Most of the pillar failures included in the ARMPS data base are "squeezes" in which me section converged over hours, days or even weeks. Another important subset are IS massive pillar collapses (Mark et al., 1997b). These occurred when undersized pillars failed and rapidly shed their load to adjacent pillars, which in turn tailed. The consequences of such chain reaction-like failures typically include a powerful, destructive, and hazardous airblast.

Dabi collected at 12 massive collapse sites revealed that the ARMPS SF was less that 1.5 in every case, and was less that 1.2 in 81% of the cases. What really distinguished the sudden collapses from the slow squeezes, however, was die pillar's w/h ratio (Figure 7). Every massive pillar collapse involved slender pillars whose w/h was less than 3. Laboratory tests have shown that slender pillars typically have littie residual strength, which means that tiiey shed almost their entire load when they fail. As the specimens become more squat, their residual strength increases, reducing the potential for a rapid domino failure. The mechanism of massive collapses has been replicated in a numerical model (Zipf, 1996).



Wtdth-To-Helght Ratio

Figure 7. Pillar collapse case histories in the U.S.

Two alternative strategies were proposed to prevent massive pillar collapses. Prevention requires increasing either the SF of the pillars, or their w/h ratio.

Containment is used if barrier pillars are used to separate compartments in which high extraction is practiced. The small pillars may collapse within a compartment, but because the compartment size is limited, the consequences are not great Design charts have been developed for each approach, considering the width of the panel, the seam thickness, and the depth of cover (Mark et al., 1997b).

6 LAMODEL: A NUMERICAL MODEL FOR MULTIPLE SEAM DESIGN

Multiple seam situations and other complex mining geometries do not lend themselves readily to simplistic empirical models like ALPS and ARMPS. Numerical methods are the alternative approach, but to be useful they must realistically portray the behavior of large volumes of rock. In addition, they must not require rock material properties that cannot be easily determined.

To address these concerns, NIOSH has developed the displacement-discontinuity model LAMODEL (Heasley and Salamon, 1996a). LAMODEL simulates the overburden as a stack of homogeneous isotropic layers, with fiictionless interfaces and with each layer having the identical elastic modulus, Poisson's ratio, and thickness. This "homogeneous stratification" formulation does not require specific material properties for each individual layer, and yet it still provides a realistic suppleness to the overburden that is not possible with the homogeneous overburden (Salamon, 1989; Heasley and Salamon, 1986b).

For practical pillar design using a DD model, the input coal strength is generally derived from empirical pillar strength formulas which are solidly based on observed pillar behavior, as opposed to laboratory tests (Mark and Barton, 1996). Similarly, the gob and overburden properties in the DD model are calibrated so that the resultant gob and abutment stresses closely match field measurements/observations such as the abutment load formulas in ALPS or ARMPS. This technique of combining empirical pillar strength and abutment load formulas with the analytical mechanics of a displacement-discontinuity model capitalizes on the strengths of both the empirical and analytical approaches to pillar design. Using this technique, a displacement-discontinuity model can be the most practical approach for stress analysis and pillar design in complex mining situations such as; multiple seams, random pillar layouts and/or variable topography (FigureS).

7 GUIDELINES FOR ROOF BOLT SELECTION

Despite more than half a century of experience with roof bolting, no design method has received wide acceptance. To begin to improve this situation, NIOSH evaluated the performance of roof bolt systems at 37 mines (Molinda et al., 2000). Success was measured in terms of the number of roof fells that occurred per 3,000 m of drivage with a particular roof bolt design when other geotechnical variables were held constant. A variety of statistical techniques were used to explore trends in the data.

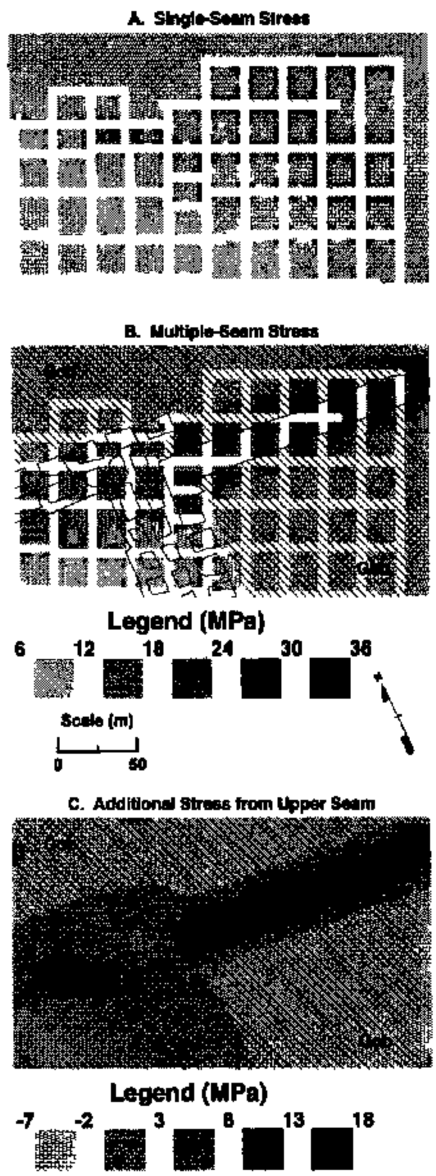


Figure 8. Stress analysis of multiple seam interaction using LAMODEt

The study evaluated five different roof bolt variables, including length, tension, grout length, capacity, and pattern Roof spans and the CMRR were also measured, Stress levels could not be measured directly, but the depth of cover was used as a surrogate.

As expected, the competence of the roof rock, represented by the CMRR, was the single best predictor of the roof fell rate. More surprising was the importance of depth. The higher horizontal stresses encountered in deeper mines apparently require greater levels of roof support (Figure 9). Important findings were also made regarding bolt length and intersection span. Unfortunately, the data was too sparse and too scattered to allow conclusions to be made regarding tension and other roof bolt variables.

The study's findings were used to develop guidelines for designing roof bolt systems (Mark, 2000). Building upon an equation initially proposed by Unal (1984), a formula for selecting bolt length was proposed:

$$L_B = 0.12(I_s)\log_{10}(3.25H)\left\{\frac{100 - CMRR}{100}\right\}$$

Where: L_B = Bolt Length (m)
Is = Intersection span (average of the sum-of-the-djagonals, m)
H= Depth of Cover (m)

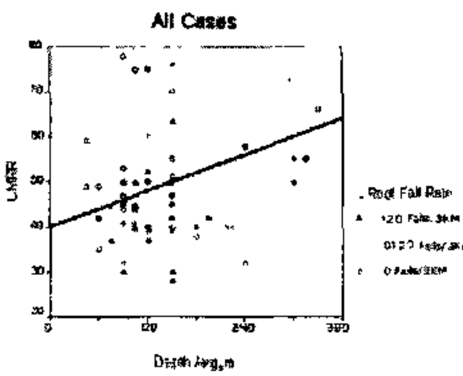


Figure 9. Roof bolt performance case histories

Once the bolt length has been determined, the bolt pattern and capacity is determined using the following equation:

$$PRSUP = \frac{L_B\ N_B\ C}{14.5\ (S_B\ W_e)}$$

Where: NB=Numberofboltsperrow
C=Capacity (kN)
Sß=Spacing between rows of bolts (tn)
W_e=Entry width (m)

The suggested value of PRSUP depends on the CMRR and the depth of cover, as expressed in the following equations:

PRSUP = 15.5 - 0.23 CMRR (low cover)

PRSUP = 17.8 - 0.23 CMRR (high & moderate cover)

Figure 10 shows these equations together with the field data from which they were derived The design equations are slightly more conservative than the discriminate equations that they are based oa The guidelines are currently being implemented into a computer program called Analysis of Roof Bolt Systems (ARBS).

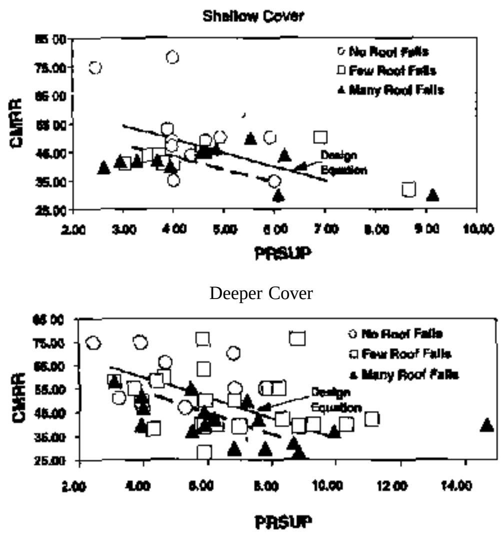


Figure 10. Design equations for roof bolts.

8 SUPPORT TECHNOLOGY OPTIMIZATION PROGRAM

The 1990's saw an unprecedented development of innovative supplemental roof support technologies for underground coal mines. Compared with the traditional wood posts and cribs, the new supports provide better roof control and material handling advantages. The new supports include both engineered wood products and novel concrete designs.

As new support systems are developed, they should be tested to determine their performance characteristics. NIOSH operates a world-class facility called the Safety Structures Testing Laboratory (Barczak, 2000a). During the past seven yep-s, over

1,000 tests have been conducted on various support systems. As a result of this effort, 18 new support systems have been introduced to the mining community.

To facilitate the use of diese new supports, NIOSH developed the Support Technology Optimization Program (STOP). STOP includes a complete database of the support characteristics and loading profiles obtained from die testing (Barczak 2000b). Using criteria introduced by the user, STOP can determine the support pattern that will carry the required load and provide convergence control. Comparisons among the various support technologies are easily made. STOP can also estimate material handling requirements and installation costs. Figure 11 shows a typical screen from the STOP program.

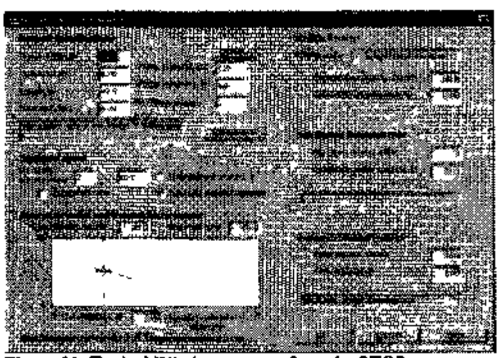


Figure 11. Typical Windows screen from the STOP program.

9 CONCLUSIONS

The NIOSH ground control program has focused on providing die mining community with practical tools for improving the safety of U.S. underground coal miners. Using these techniques, mine planners can optimize pillar design and support selection for a variety of mining techniques.

Transferring these tools to the industry is an integral part of the program. Traditional techniques, such as conference presentations and NIOSH publications, are employed extensively. But innovative meühods are also employed to bring die research results directly to die end users. Open Industry Briefings are regularly held in numerous coalfield locations, to allow researchers direct access to their customers. Software packages are made available free of charge, and hundreds are distributed at meetings or in response to requests. Most recenüy, all the ground control software has been posted on the NIOSH mining website for easy access.

The technology transfer efforts have paid off in many ways. Large segments of the mining community uses NIOSH software routinely for many aspects of mine design. Mine operators and safety regulators both consider NIOSH as the central source for information ground control. While it is hard to measure directly, there is every reason to believe that our efforts have helped make underground coal mines safer places to work.

REFERENCES

Barczak, T. M The History and Future of Longwall Mining in the United Stales. BuMines IC 9316,1992,26 pp.

Barczak, T. M. NIOSH Safety Performance Protocols for Standing Roof Supports and Longwall Shields. Proceedings: New Technology for Coal Mine Roof Support, NIOSH IC 9453, 2000a, pp. 207-222.

Barczak, T. M. Optimizing Secondary Support with the NIOSH Support Technology Optimization Program (STOP). Proceedings: New Technology for Coal Mine Roof Support, NIOSH IC 9453,2000b, pp. 151-164.

Colwell, M., Frim, R. and Mark, C. Calibration of the Analysis of Longwall Pillar Stability (ALPS) for Australian Conditions. Paper in the Proceedings of the 18th International Conference on Ground Control in Mining, Morgantown, WV, 1999, pp. 282-290.

Heasley, Keith A., and Miklos D. G. Salamon, 1996a, A New Laminated Displacement-Discontinuity Program, Proceedings of the Second North American Rock Mechanics Symposium, Montreal, Quebec, Canada, June 19-21,p. 1879-1886.

Heasley, Keith A., and Miklos D. G. Salamon, 1996b, A New Laminated Displacement-Discontinuity Program: Fundamental Behavior, Proceedings of the Fifteenth International Conference on Ground Control in Mining, August 13-15. p. 111-125.

Mark, C. Analysis of Longwall Pillar Stability: An Update. Paper in the Proceedings of the Workshop on Coal Pillar Mechanics andDesign.USBMIC9315,1992, pp. 238-249.

Mark, C. Pillar Design Methods for Longwall Mining. USBM IC 9247,1990,53 pp.

Mark, C. Design of Roof Bolt Systems. Paper İn Proceedings of the New Technology in Roof Support, NTOSH IC 9453, 2000, pp. 111-132.

Mark, C. Empirical methods for coal pillar design. Paper in Proceedings of the Second International Conference on Coal Pillar Mechanics and Design, Vail, CO, NIOSH IC 9448, pp. 145-154.

Mark C. Comparison of Ground Conditions and Ground Control Practices in the USA and Australia. Proceedings, 17th International Conference on Ground Control in Mining Morgantown, WV, 1998, pp.

Mark C. Comparison of Ground Conditions and Ground Control Practices in the USA and South Africa. Proceedings, 18th International Conference on Ground Control in Mining. Morgantown, WV, 1999, pp. 63-71.

Mark C, and T. M. Barton. The Uniaxial Compressive Strength of Coal: Should it be Used to Design Pillars? Proceedings, 15th International Conference on Ground Control in Mining. Golden, CO, August 13-15,1996, pp. 61-78.

Mark, C, and F. Chase. Analysis of Retreat Mining Pillar Stability. Paper in New Technology for Ground Control in Retreat Mining, NIOSH IC 9446,1997,pp. 17-34.

Mark, C, F. E. McCall, and D. M. Pappas. A Statistical Overview of Retreai Mining of Coal Pillars in the United States. Paper in New Technology for Ground Contrai İn Retreat Mining. NIOSH IC 9446,1997a, pp. 2-16.

Mark, C, F. Chase, and R. K. Zipf, Jr. Preventing Massive Pillar Collapses in Coal Mines. Paper in New Technology for

Ground Control in Retreat Mining, NIOSH IC 9446, 1997b, pp. 35-48.

Mark, C, F. E. Chase, and G. M. Molinda. Design of Longwall Gate Entry Systems Using Roof Classification. Paper in New Technology for Longwall Ground Control: Proceedings of the USBM Technology Transfer Seminar, USBM SP 94-01,1994, pp. S-18.

Mark, C. and G. M. Molinda. Rating Coal Mine Roof Strength from Exploratory Drill Core. Proceedings, 15th International Conference on Ground Control in Mining. Golden, CO, August 13-15,1996, pp. 415-428.

Molinda, G. M., C. Mark, and D. R. Dolinar. Assessing coal mine roof stability through roof fell analysis. Proceedings: New Technology for Coal Mine Roof Support, NIOSH IC 9453, 2000, pp. 53-72.

Molinda, G. and C. Mark. The Coal Mine Roof Rating (CMRR>~ A Practical Rock Mass Classification for Coal Mines. USBM IC 9387,1994,83 pp.

Rusnak J., and Mark, C. Using the Point Load Test to Determine the Uniaxial Compressive Strength of Coal Measure Rock. In Peng SS, Mark C, eds. Proceedings of the 19* international Conference on Ground Control in Mining, Morgantown, WV, 2000, pp 362-371.

Salamon, M. D. G 1989b Some Applications of the Fnaionless Laminated Model Paper in Proceedings of the 30th U S Symposium of Rock Mechanics. A. A. Balkema, Rotterdam. 1989, pp. 891-898.

Unal, E. Empincal approach to calculate rock loads m coal mine roadways Proc. 5 Intl. Conf On Ground Control in Mining, Morgantown, WV, 1986, pp. 232-246.

Zipf, R. K. Jr Simulation of cascading pillar failure in room-and-pillar mines using the boundary element method. Paper m Proceedings of the 2nd North American Rock Mechanics Symposium, Balkema, 19%, pp. 1887-1892.

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Use of the Internet and Information Technology in Mining

T.S.Golosinski
Department of Mining Engineering, University ofMissouri-Rolla, Rolla, MO 65409-0450, USA

ABSTRACT: Rapid advances of information technology and widespread use of the Internet revolutionizes business in general and the mining industry in particular. The paper summarizes the related developments and their impact on the mining industry.

1 THE INTERNET

"The internet is the biggest thing since I've been in business" said Jack Welch, Chairman and CEO of General Electric. As a result of Internet applications his company, GE, plans to cut 15% from its cost base of $100 billion (one hundred thousand millions) in both 2001 and 2002, five times its typical annual growth in productivity that varies from 3 to 4 % (Anon. 1999).

Jack Welch' view İs widely shared by others in the business community in general and in the mining industry in particular. As a result the Internet is driving a global transformation of both businesses and personal lives. It redefines all aspects of how companies work, both internally and externally. The pace of this change has been rapidly accelerating, facilitated by rapid advances in enabling technologies, decreasing cost of computing power, and significant advances in data and knowledge management.

The term Internet is used in this paper to describe îhe whole cluster of technologies mat depend upon and enhance it. Its first widespread application to business was B2C: Business-To-Consumers, by now firmly established business practice. Initially this Internet application appeared to be relatively simple, the activities limited to selection of a vendor, selection of an item to be acquired, processing of the credit and shipment. As *a* result Internet shopping has become a commonplace in lives of many people worldwide. Only later the intricacies of this business became apparent. These include pricing strategies, security of transactions and data, prevention of disruption and loss of data, and the need to change attitudes of employees in companies involved İn B2C.

The next to come was B2B, Business-To-Business applications of the Internet. These are far more complex and difficult than B2C. Purchases are made under long-term contracts, products have to conform with strict specifications, purchasing decisions are made by teams, and there may be a significant time difference between the time of making the decision and completing the related transaction. However, they are believed to offer an opportunity to revolutionize the way in which business gets things done.

Responding to the challenge a number of companies İn the process or consider transforming themselves from process-focused, internal organizations into external ones by engaging suppliers, customers and partners through an electronic communications infrastructure. The emerging business model is summarized in Table 1 below.

Table 1 The emerging business model (after The Economist, 20QO)

	Industrial Age	*Digital Age*
Companies	Inwardly focused	Extended enterprises
Customers	Limited access to manufacturer	Direct access to manufacturer
Suppliers	Arms length relationships	Electronic relationships
Intermediaries	Stand-alone entities / separate processes	Extended enterprise links / shared processes
Employees	Hierarchical and functionally managed	Empowered teams, cross-functionally managed

Majority of today's companies, including most of the mining companies are re-evaluating their operations and strategies in order to take full use of the Internet. These evaluations usually include:

1. Identification of these aspects of their operations and strategies that are affected by the Internet, and where it offers an opportunity to develop a competitive advantage

2. Identification of the capabilities needed to take full advantage of the opportunities offered by the Internet, especially in areas of customer and supplier liaison, and

3. Provision of the capabilities defined in 2. above İn expeditious and timely manner.

Specific business models used to implement the new strategies and adapt the company to the digital age differ. These are usually site-specific modifications of two principal approaches. The first calls for grouping all e-business activities together and their separation from the mainstream activities of the company. Typical example is the approach used by Western Mining Corporation to create e-WMC business, discussed in some detail below (Western Mining Corporation, 2001). This separate business entity deals with Internet-based parts of WMC cobalt / nickel / copper selling activities.

In a modification of this approach numerous mining companies, including WMC, have formed independently operating joint ventures that are providing a wide range of e-business services. Typical example here is the GlobalCoal (1998-2001), discussed briefly below. By getting involved in e-business these mining companies expect to leverage some oftheir existing knowledge, expertise and core competencies to generate additional value. On the other hand they look for an opportunity to test various e-commerce models and approaches for their possible introduction into the core operations at a later date (Clarry et al, 2000).

The second approach calls for integration of e-business into mainstream company activities at all stages of its operation. No successful example of this approach to a mining business exists so far. However, several companies are giving it a serious consideration.

2 THE INTERNET AND MINING

The applications of the Internet in mining date back several years. The first successful applications were those for monitoring and control of equipment performance, often involving a variety of dispatch systems, and later extended to equipment condition and performance monitoring. The first full-fledged attempts to use e-business in mining took place less then two years ago, when several miners began offering their products for sale via Internet. At the same time a variety of third-party e-marketplaces were created to facilitate e-commerce in products, services, equipment and supplies. Some of these were created either by mining company consortia or with active participation of mining and related companies. Most recently use of the Internet for management and control of all aspects of mining operations is proposed.

Several selected, typical examples of various Internet applications in mining follow. The first deals with an early form of mining e-business, e-sales. The second is e-marketplace application. Following that the Internet based outsourcing schemes are discussed (Application Service Providers, or APS). Finally an example of an emerging e-management is given.

2.1 *E-sales*

One of the fist mining-related e-sales ventures was established by WMC, Western Mining Corporation of Melbourne, Australia. It was developed over the last two years, first to sell cobalt produced by this company. Later on nickel sales were added, followed by copper sales. On line supplies tendering was added more recently followed by a full fledged electronic e-marketplace, Quadrem, that provides a procurement solutions to the mining, minerals and metals industry.

The first two activities, cobalt and nickel sales, are relatively simple. E-WMC posts the availability and premiums for defined quantities of its cobalt and nickel on its website, by location, for review and information of potential clients. This makes it easy for WMC to participate in the spot and near-term cobalt and nicket market and expands its client base, as the site in question is freely accessible.

The copper sales business differs, the difference reflecting die fact that most of WMC copper is sold through long-term contracts to well established customers. The prime purpose of the site is to enhance WMC relations with these customers. Thus access to the site is limited, and it is designed to "enable customers direct access to real time data affecting their copper deliveries, 24 hours a day and 7 days a week". The site also lists uncommitted spot copper quantities.

The other two sites represent more recent approach to e-business and are similar to the GlobalCoal business. The latter (GlobalCoal, 2001) is being established by several mining companies with coal interests, namely Rio Tinto, Anglo American, Billiton and Glencore International. This site involves several newer features of e-sales that are: customized trades, tender / RFQ (request-for-proposals) feature and extensive coal-related news section.

Differences between the different parts of the e-WMC business illustrate the development process of mining related e-business. They also illustrate the differences in approach to e-business as determined by various types and quantities of the traded commodity.

It was recently reported that e-WMC business is a successful one. It allowed WMC to lower the cost of sales transactions by over 50%, resulted in better premiums on its products and allowed me company significantly improved access to the customers (Voss, 2000).

2.2 *E-marketplaces*

More recently a number of mining related e-marketplaces has been established, quite often with active participation of various mining companies. The purpose of e-marketplaces is to facilitate not only sales of products, but also purchases and sales of supplies, services, training and education. Most of e-marketplaces include a variety of features that facilitate price negotiations. The fees charged on transactions conducted through the e-marketplace support its development and maintenance.

One of the most advanced e-marketplaces related *to* mining is MyPlant (2001). Originally established by Honeywell International, the company that sells its products through extensive network of intermediaries, the site was intended to capitalize on the intermediary expertise. More recently the Honeywell ownership was diluted to encourage wider participation of other, independent companies. Industries served by MyPlant include mining, minerals and metals, petroleum, pulp and paper, chemicals and pharmaceuticals and power generation. Overall MyPlant membership is claimed to be in excess of 14,000 distributed over 95 countries. The main task of MyPlant is to provide plant personnel with "interactive, cost effective solutions that address current process manufacturing challenges". As such MyPlant brings together plant professionals that seek information, answers and resources with solution providers who have the relevant technology and expertise.

In addition to plant expertise and services, MyPlant offer several other services. These are:

• MyExpert, a network of experts who can answer specific plant-related questions

• MyJobs 'that allow users to reach, recruit and manage jobs and job seekers world-wide

• MySkills, a place for professionals to exchange views, as well as to gain, share and manage knowledge

• MyEquipment that provides on-line access to the equipment that a mine professional needs

• MyCommunties, which offers the ability to share the solutions, views and resources with the peers.

MyPlant benefits members and solution providers through reduced search and transaction costs, and more efficient match of solution providers and plant personnel.

2.3 *Vendor managed assets*

The mining industry leads the others in wide use of Application Service Providers (APS). The best example here is involvement of vendors in management of mining company assets and in particular of its principal mining equipment.

Any electronic data related to production, equipment performance, equipment and plant condition, and the like, can be easily transmitted over the Internet and made available to off-site vendors in real time. Caterpillar's VIMS Wireless, Vital Information Management System (Caterpillar, 2001), is a typical example of such technology. Competing technologies are available from Komatsu (Modular Mining Systems, 2001), Euclid-Hitachi and others.

Thus an equipment supplier, or a third party, can monitor mining equipment performance and condition, and provide its diagnostics, without the need to be *physically* present at the minesite. With this data at hand the vendor can order a service, a maintenance job or a repair, order me required parts and secure appropriate maintenance expertise. Considering high technical sophistication of today's mining equipment, and frequent lack of advanced expertise at remote minesites, asset management by third parties offers an opportunity to significantly improve asset performance.

As this technology is refined and its reliability is assured, the mining industry will be able to manage its equipment in a manner that İs uniform thought various equipment brands and mine locations. Furthermore it will be feasible to assure that the equipment complies with pre-set production objectives in an optimum way. Thus the mines will be able to concentrate on its core competencies: resource discovery, its exploitation and ore processing, while outsourcing the non-core equipment management to specialized application service providers.

2.4 *E-Management*

Several non-mining companies, in cooperation with vanous mining companies, are now developing the mine information management systems, the end-to-end systems, that will allow all engineering, purchasing and sales activities in a mine to be handled by a one • seamless system The first embryonic system of this type was Caterpillar's

METS system, described in an earlier presentation to this Congress (Golosinski and Ataman, 1999),

During the last three years the Caterpillar offering has significantly expanded and is marketed under the name MineStar (Caterpillar, 2001). In addition to METS system in now includes wireless VIMS (Vital Signs Motoring System), dispatch systems, and other components. Similar system is offered by Komatsu (Modular Mining Systems, 2001), known as Intellimine. Both the systems are being developed by mining equipment manufactures and intended primarily to optimize performance of the rnining equipment.

More recently several software companies are involved in development of end-to-end business packages. The most advanced appears to be the offering of SAP Company, known as SAP Mining (SAP, 2001). The package consists of seven "layers" of software, each layer dealing with different aspect of a mining operation. These are:

1. Enterprise Management that deals with strategic planning, accounting, data warehousing, and the like
2. Customer Relationship Management that deals with customer service and liaison, acquisition of resources and sales management
3. Acquire and Develop that deals with exploration and development of the mining prospects
4. Mining and Primary Processing that deals with mining, primary processing and related logistics
5. Secondary Processing that deals with downstream processing of ore
6. Sales and Distribution that deals with product sales, their delivery and invoicing, and
7. Business Support that deals with human resource management, procurement, fixed asset management, environment, safety and other support operation.

As an example of layer content, the mining part of the Mining & Primary Processing includes individual packages that deal with long and short term mine planning, production scheduling, unit operations (drilling & blasting, ground support, load & haul, hoisting and handling), material supply, maintenance of equipment and plant, and product and activity costing.

To author's knowledge no fully implemented, complete system of this type is operational at the time of writing this paper. The main challenge appears to be development of seamless and reliable interfaces between various subcomponents that often were developed by independent, third-party vendors.

In another development, several developers of general mine planning software, most notably Gemcom, have started work on developing interfaces between their software and other business systems used in mines. The latter include interfaces with dispatch systems, with geotechnical modeling software, and the like.

3 DISCUSSION

In addition to applications described above, a variety of other e-business providers, not named above, are active in the field. Some of these were listed by Annon. (2000), Annon. (2001), Gibbs (2001), Hudson (2000), Ludeman (2001), Voss (2000) and others. Additional provides are entering business each day. This proliferation indicates that a consolidation of the e-business industry is bound to take place at some time İn the future. Few of the current providers are likely to survive in a long term. As indicated by experience with B2C business, the most likely to survive are the providers who enjoy strong financial position, are supported by established mining companies, and who provide the service in the most effective and efficient way.

Many predict rapid spread of business uses of the Internet. E-commerce is expected to reach several trillion US dollars in 2004 (Gibbs, 2001), with the annual growth rates of 125% (Luderman, 2001). These predictions may be somewhat optimistic. The recent survey conducted in the USA by Forester Research and the National Association of Purchasing Management reveals that over 80% of surveyed companies are advanced no more than 20% on the road towards using the Internet for purchasing, and that other business uses of the Internet are even less advanced *(USA Today,* January 23, 2001).

Unlike many other traditional industries, the muling industry is in a unique position to take full advantage of the Internet. Characterized by often remote and desolate mine locations, concentration of expertise and supplies in areas away from mine locations, fragmentation of the industry itself and the multinational character of many companies the mining industry stands to gain most from use of the Internet The many benefits that can be gained include (Clarry et al, 2000):

- Decoupling availability of skills from the location of workforce. This is facilitated by easy access to expertise and technology providers, by ability to remotely control operations and monitor equipment performance, by ability to communicate performance, condition and diagnostic data, in real time, to service providers, and by ability to manage the knowledge needed for effective and efficient conduct of operations.
- Reducing isolation of remote operations through easy access to expertise, third-party information, and by facilitation of targeted professional development and education of personnel.

- Centralizing and/or outsourcing of infrastructure and non-core activities.
- Improving efficiency of the supply chain by streamlining and simplifying its logistics, by improving efficiency of purchasing transactions, by ability to monitor supplies İn real-time, by improving links between similar mines or mines in the same locations, and by pooling resources available to these mines.
- Improved efficiency of product sales through ability to address wider range of potential clients and providing these clients with superior service in terms of product quality, timeliness of supply and competitive pricing.

Not surprisingly the Australian mining industry appears to be most advanced in implementation of business applications of the Internet. This industry İs strong and modem, located in a remote area of the world, and able to take advantage of sophisticated communication and computing infrastructure, as well as widespread computer literacy of the mining personnel. It was between the first to use e-business and e-marketplaces and by now has developed a body of related expertise.

4 CONCLUSIONS

The effectiveness- and efficiency of the mining industry can be significantly improved by incorporation of the Internet in various aspects of mining operations. Internet based sales of mining products, purchasing of supplies and expertise, and remote monitoring of equipment performance and condition, all are proven feasible and beneficial. The relevant business models and solutions are commercially available.

Full realization of the opportunities offered by Internet and related 62B businesses require re-engineering of the traditional mining companies. The most effective re-engineering patterns and models are emerging from between the wide variety of those used currently. The wide interest shown in use of the Internet by mining companies allows to predict that new, Internet empowered mining business models will be soon adopted by most mining companies. This in turn will lead to more efficient mining operations in the future.

REFERENCES

Anon. 1999. Competing in the digital age: How the Internet will transform business. The Economist Intelligence Unit (http://w w w. ebus i ness forum. corot

Anon. 2000. Mining the Internet. Mining Magazine, vol. 183, No. 5. November, pp. 204-208.

Anon. 2001. Internet enabled business solutions offered for mining companies. Mining Engineering, vol. 53, No. 1, January, p. 12.

Caterpillar, Inc. 2001. httn:/.'ww w .cat .com

Clarry, D., RufTo, K and Mottola, L. 2000. E-business, mining and strategy. CIM Bulletin, vol. 93, no.1042, pp. 105-109.

Gibbs, B. 2001. E-commerce: the mining journey begins. Mining Engineering, vol. 53, No. I.January, pp. 17-22.

Globalcoal. 1998-2001. http://www.plobalcoal.com

Golosinski, T. S. and Ataman, K. 1999, Total mining systems: a dream or reality? Proceedings, lô"[1] Mining Congress and Exhibition of Turkey, pp. 89-92.

Hudson, M. 2000. E-Commerce finally finds the coal industry. Coal Age, vol.105, No.12. December, pp. 28-30.

Luderman, F. 2001. Mining's B2B e-marketplaces - get ready. Mining Engineering, vol.53, No.l. January, pp. 23-28.

Metalsite. 2001. htm://www.metalsite.net'MctalSite Is/

Modular Mining Systems, 2001. http://www.mmsi.com

MyPlant. 2001. http;//wwy, .mvptant.com

SAP. 2001. SAP Mining Solutions, http:// www.mvSAP.cum

The Economist, 2000. A Survey of E-Management. November 11, 2000 (also available at http://www.ebusinessforum.com>

Voss, B.2000. Plug it in. World Mining Equipment, vol. 24, No. 9. November, pp. 66-67.

Western Mining Corporation. 2001. http://www.wnic ,com/ewmc

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Development and Challenges in the Canadian Mining Industry

F.Hassanİ
Department of Mining and Metallurgical Engineering, McGill University, Montreal, Canada

ABSTRACT: Canada is one of the leaders in the world for developing and employing highly efficient and advanced mining technologies to supply the world with various minerals, oil sand and coal The Canadian mining industry's goal for future is to continue to remain one of the world leaders by increasing the value of mining products for customers. The industry will employ processes and produce products at lower costs, reduce energy consumption. At the same time will continue to minimize the adverse environmental, health and safety effects associated with mining and mining products. Achieving this goal will provide enormous benefits to the Canadian prospering population, their quality of life and the environment. This paper is to give a brief overview of the Canadian Mining industry and then discusses some of the development of the Canadian mining industry in the past decade. Attempt is made to highlight related challenges, which will be the basis for the necessary future research and development program.

1 INTRODUCTION

1.1 *The value of mining to Canada*

Mining products are the building blocks of our society. They are still the primary materials in our buildings, roads, and machines. Mining products are the basis of many products that have become increasingly critical to everyday life in the twenty-first century-advanced materials. Canadian mining products are metallic minerals, industrial minerals, precious stones, as well as, energy resources such as oil sand and coal.

The economic benefits of mining are far reaching; it is not an exaggeration to say that mining helped build this country. It opened Canadian frontiers. Glace Bay, Rouyn-Noranda, Val d'Or, Chibougamau, Sept-Iles and Labrador City, Sudbury, Timmins, Kirkland Lake, Cobalt, Flin-Flon, Thompson, Fort McMurray, Trail, Kimberley, and Dawson City all started as mining towns. At present mining takes place in almost every Canadian province. However it is more important in Ontario, Quebec, Alberta, British Columbia and Saskatchewan. Hundreds of mines operate in Canada today, providing materials for the manufacturing, construction, automotive and chemical industries, as well as the energy resources upon which we are so dependent. Hence, this industry affects the Canadians socially and economically. The mining industry continues to helped Canada's rural and developed regions by creating many jobs and providing the opportunity of growth in the areas. It is estimated for every job created within the mining industry, another job is created to an indirect source of this industry.

Mining provided direct employment for over 368,000 people in 1998. Mining operations are often the leading employers in the communities where they operate. Mining employees eam some of the highest wages of all Canadian industries. Contrary to popular perception, the mining industry has a low rate of occupational injury, lower than retailers, hospitals, and hotel. The mining and mineral processing industries contributed approximately $28.0 billion to the Canadian economy. If, energy resources contribution to the economy is added to the above, the total will be approximately 50 billion Dollars. It exports 80 percent of mining production, which comprises 15% of the Canadian total export. However, the mining industry's contribution to the GDP does not, by definition, take into account the total contribution of the mining industry.

A recent study demonstrated that for every $1 billion in output created by the mining, smelting and refining sectors direct demand of goods and services increase by $615 million. Hence, a strong Canadian mining industry is an engine of growth of Canada's small and medium companies in the areas of services, consulting engineering and equipment industry. At present, one in seven of all Canadian export dollars comes from mining. Mining has helped make Canada a major world trader. Toronto is considered capital of the world and is home not only to the Toronto stock exchange, but also to more than 400 mining and exploration company offices. On the other hand, Vancouver is the world's exploration centre with more than 850 mining and exploration company offices. Furthermore, Montreal is home to a number of mining companies and is an important location for mining research and development and education. McGill University and Ecole Polytechnique jointly offer the only bilingual mining engineering CO-OP program in Canada. Globalization of the mining industry has brought other developments. Canadian financial institutions have become world leaders in equity financing for global exploration and mine development. More than '/i of the world's mining companies are listed in Canada. At the beginning of January 2000, approximately 1500 companies were listed on Canadian exchanges, compared to 38 on the London metal exchange, 35 on the New York stock exchange, 66 on the Johannesburg stock exchange and 342 on the Australian stock exchange. This is why it is not a very big surprise to find more than half of the mining analysts of the world in Toronto.

Today, Canada is perceived as being the world largest and most sophisticated mining finance in the world and it is a major contributor to the Canadian finances. Canada's mining success has been based on good geology, continued support for research and development, applied new technologies, and progressive mining and environmental regulations. This combination has made Canada one of the most preferred targets for exploration capital in the world as a destination for international mineral exploration capital. In 1996 Canada ranked second, after Australia. Canada also ranks among the top five producers of 17 major minerals and metals It headquarters some of the world's largest and most successful multinational mining companies such as Inco, Barrick Gold, Alcan, Cominco, Falconbridge, Placer Dome and Noranda.

2 DEVELOPMENTS

The Canadian mining industry in the past two decade has overcome many of its operational difficulties as well as increasing the efficiency of its operations to compete with outside competitors by investing heavily in research and development Canada is a world leader in application of mining technology. Mining companies spend over $100 million annually in Canada on research and development. Over 85% of the work force use advanced technology, including electronics, advance materials, expert systems, and telecommunications.

New bulk mining methods has dominated the raining operations in Canada in the last decade. The applied rock mechanics research and development at depth especially in the rockburst-prone ground, together with innovation in support system, and mine backfill., rock characterization techniques as well as the deployment of the semi automated/ automated excavating , drilling and bolting machinery has contributed to the present success. Brief overviews of some of these developments are highlighted herein.

2 1 Applied geophysics

The mining industry has developed and employs sophisticated techniques to explore and characterize mineral resources. Minimizing the need for extensive capital and advanced work. This reduces the costs of, and the environmental disruption that can be associated with, finding economic resources. Ground disturbance associated with exploration and development is minimized and the accuracy of the measurement of resource volumes and quality has been improved. Reserve maps are more accurate and mine plans are designed make mining more productive. Improved efficiency in mineral extraction reduces dilution and therefore reduces mineral processing costs and minimizes wastes.

These achievements have been possible with application of advance geophysical techniques, such as Ground Penetrating Radar, Radio tomography, and *3D* seismic as well as laser stope profiling systems and MSR (Miniature Seismic Reflection.). Further development^nd challenges in this area, will be in better interpretation of the signals as well as combining and fusing data from different techniques to produce a more detailed and accurate images of geo-structures.

2.2 Mine backfill

From the tremendous activity in backfill technology in the past 20 years, it is clear that mine backfilling continues to be highly significant to our industry from both support and environmental point of views The large number of research projects awarded by the Federal Government and Provincial Government together with the mining industries addressing different issues concerning backfill operation has contributed greatly İn improving mine backfill operation as well as keeping Canada in the forefront of mine backfill technology in the world.

In recent years, through the increasing awareness of backfill's benefits to mining operations, as well as the technological developments in the application of an energy and economically-efficient system, backfill has gained a place as an integral part of the whole mine design and operation. Processes such as mixing, dewatering, transportation and placement as well as geotechnical considerations have improved substantially and have been the subject of much discussion. Due to the complexity of backfill systems, applications of backfilling to different mining environments require extensive experience. Such experiences, documented as case studies, are very useful especially when the time comes to apply new technology, such as expert systems, artificial intelligence and an object-oriented program to backfill design.

The computerization of backfill design and implementation has already begun but there are still significant areas of the backfill operation that require technological and theoretical advancement. Hydraulic transportation, although relatively well understood in practical terms, continues to vex scientists and engineers alike, although advances in plug flow understanding have contributed to an increasing interest in total tailings backfill.

The placement of backfill has often caused problems in practice but has rarely received the attention it deserves in terms of research. Tight filling of all types of fill and reduction in coning and segregation for rockfill all require research if the advances required are to be made. Total tailings placement is always cited as a 'grey' area although very little research seems to have been done and cost effective solutions found. As stricter environmental standards are adopted then we will be forced to look more closely at these issues and the benefits will certainly outweigh the investments required to solve these problems.

Mining with backfill requires reliable and energy efficient pipeline systems for transporting waste material underground. Such systems offer advantages in terms of convenience, low operation and maintenance cost, İn addition to being environmentally "correct".

The technical constraints imposed on such systems may be summarized as follows:

- If possible, the system should operate with full utilization of its potential energy.
- The material transported should be carefully engineered through appropriate mix design to ensure a stable and flowable fill mixture with maximum solids capacity.
- Flow velocity should be carefully selected and maintained to avoid unsteady flow and excessive wear of the pipeline.

Although the conveying of solid-liquid mixtures in pipelines has been a subject of extensive research and development for various commodities over the last forty years (e.g. coal and phosphate over long distances), its application to backfilling with the above-mentioned constraints is relatively recent.

This type of backfill practice has been developed simultaneously in different parts of the world. A brief summary of the use of backfill in Canada is given below:

1933 Use of furnace slag and pyrrhotite tailings at the Home Mine, Noranda, Quebec.

1935 Use of standard gravel at Falconbridge Mine in Ontario.

1948 Use of sand/tailings at Frood Mine, INCO, Sudbury, Ontario.

1959 Use of cemented tailings at Falconbridge.

1960 Use of cement to stabilize sandfill at [NCO.

1960 Use of cement to stabilise rockfill at Noranda, Geco Mine.

1962 Use of cemented tailings at Frood Mine, INCO.

1962 Hydraulic cemented mine backfill became standard.

1970 Development and optimisation of hydraulic fill and rock fill throughout the industry, worldwide

1976 Original developments of paste fill in Germany at Preiessage, Bad Grund Mine with the use of putzmeister pumps.

1978 Paste backfill development in South Africa by Chamber of Mine and CAMERO.

1981 Development of pastefill in Helca, Luck Friday Mine in U.S.A.

1983 First pastefill operation in Canada. Development of paste fill with tail spinner, Dome Mine of Placer Dome in Timmins, Ontario.

1983 Placer Dome and McGill University. Research and development of paste fill at McGill University.

1984 Research and development of paste fill at INCO.

1994 Paste fill production at Garson Mine of INCO.

1995 System development for standard paste fill by various groups in Canada

1998 In situ behaviour of past fill in stopes, Cambior lnc and McGill University.

In bulk mining with backfill, inter-chamber pillar recovery operations are closely related to stability of surrounding fill, which is governed by slope size and mechanical properties of die host rock. Results of practical studies show that cement or binder content İn a fill and its slurry density are essential factors affecting fill stability and the economy of backfilling. The uncertainty in fill design, based solely on theoretical or numerical modelling techniques, without understanding of the behaviour of fill insitu or practical input, may result in fill block failure or excessive consumption of cementing material. In cyclic backfilling, the filled stope is utilised mainly as a work platform. As such, the cement content required of this platform is generally higher than that necessary in delayed backfill because of the short curing times available and the need for rapid deployment of heavy mining equipment in the stopes

In order to quantify the factors fill stability and to optimize economic effects, it is essential to consider a rational and practical design approach upon which operators can effectively manage backfill technology. Basically, backfill design should consist of determining fill composition and the water requirements needed for fill preparation to produce an acceptable mix having certain physical and mechanical properties i.e. strength and backfill cost estimation.

Binding agents, such as Portland cement or flyash, are applied in backfill technology to improve the mechanical properties of the fill, i.e. strength. Chemical additives such as flocculants, accelerators, and retarders are employed to improve the fill permeability, flowability of the slurry and the consolidation properties of the fill

These fill materials can be used either as 'full plant tailings' or deslimed by using hydro cyclones to meet desired percolation requirements. Their size compositions largely depend on ore processing and deshming technology. The fraction of 10 urn particles in classified tailings is usually less than 20% of total mass by Weight. Sand from surface alluvial basins is widely used in mine backfill and its particle size is generally less than 2 mm. Waste rock from mine development or quarries must be crushed to meet transport requirements. The maximum size for pipeline transportation is less than 1/4 the diameter of the pipe; in the case of hydraulic transportation, this means aggregate sizes less than about 60 mm whilst aggregates up to 30 cm can be transported by truck or conveyor.

The most recent development in mine backfilling is the introduction of the paste fill.

Paste fill has a higher pulp density, between 75 and 85% by weight, depending on the grain size distribution. They utilise total tailings and may often incorporate sand or waste rock. The material can be transported from surface and does not require İn-situ dewatering. Cement may be added at sites of preparation or immediately prior to placement.

Paste fill represents state-of-the-art fill technology and holds tremendous long-term potential in mining. The application of paste fill could significantly reduce the cyclical nature of mining, improve ground conditions, speed up production and greatly reduce environmental costs.

Although a great deal of experimental data has been gathered, there remains the challenging task of making the best use of it. One way of achieving this is to encourage more basic research and development in the mechanics of flow of highly concentrated suspensions. The outcome of such research could be in the form of semi-theoretical or mechamstic models, the validity of which may be checked by comparison with experimental data. Such models would then serve for scaling up to different pipe diameters and flow conditions and for prediction of the effect of a particular variable on the overall performance of the system.

In recent years in Canada, the behaviour and support capacity of mine backfill at narrow vein as well as bulk mining operations has been studied by the author at McGill University with collaboration with mines of Cambior Inc. Some of the data are presented at this conference.

The mine backfill (1998) book by F. Hassani and J. Archibald covers İn detail all aspects of mine backfill design and use.

2.3 Support system

Canada experiencing the effects of declining grades as the shallower to medium depth richer deposits are exploited. Canadian mines are heading literally downwards to greater depths. This in itself means higher mining costs and greater operational problems related to support loads, stability of the openings, ventilation and other stress-related problems. Crucial to the mining process, whether at depth or on the surface, is the in place support system. If the opening we are concerned with is important for short term or long-term mining activity, it needs to be supported for the projected life of the opening, at the least.

It's impossible to cover *all* aspects of this topic in a few pages, so this keynote address will briefly address the panorama of support systems presently used İn Canadian mines and the many contributions made to its development by mining practitioners and Canadian research establishments. An attempt is made to concentrate on the major elements of support systems used in Canadian mines without unduly neglecting any related issues.

The design of a mine support system, first involves the identification of the potential failure mode of the rockmass, followed by the design of the support system that will prevent the identified failure mode for the projected life of the rockmass and the constraints of the project economics. As long as raining activity in Canada did not face long term problems of depleting reserves, environmental concerns and difficult economic choices, the rockbolt was the ubiquitous element of support used in underground mines. Initially, they were mass produced in specific lengths, which had nothing to do with the depth of the rockmass to be supported, nor its nature. The rockbolt today has its important use in rockmass support and will no doubt continue to serve its purpose. As is often the case with development and progress, necessity being the mother of invention has caused modifications in rockbolt design and even newer support systems to be developed for mining at depth in Canadian mines. The support systems which are used in Canadian mines such as Polyurethane (Spray-on) Support, Shotcrete, Rock bolting, Cable Bolting will be addressed:

In the deep hard rock mines of Canada, high in-situ as well as mining induced stresses due to total ore extraction can lead to seismic events and rockburst. These unpredictable phenomena usually result in rock fells and the weakening of otherwise solid rock masses to the point of requiring major support, if mining is to continue. As mining in Canada gets deeper there has been an increase in the occurrence of seismicity and rock bursts. The support of rockmass in burst-prone ground is therefore a major concern for the mines experiencing this phenomenon and for research establishments in Canada. The Geomechanics Research Centre of Laurentian University in Sudbury was involved in the design of support appropriate for use in burst-prone ground. Tannant et al. (1996), İn their overview have identified three main functions of a support system as follows:

- To reinforce the rockmass, thus enabling it to support itself (Hoek and Brown, 1980);
- To retain or prevent broken rock from spalling and falling, thus possibly resulting in progressive failure and unravelling of the rockmass
- To securely hold or tie back the retaining elements, thus preventing gravity-driven rock falls.

In their research, they characterized support elements into six categories and gave examples of their support roles. They further grouped the load-displacement behaviour of support elements into six general characteristics as stiff versus soft, strong versus weak, and brittle versus ductile and produced a guidelines n form a table showing load-displacement parameters of various support elements from their research.

23.1 Polyurethane lining

Polyurethane lining is a spray-on lining, which has application properties that make it suitable for automated support installation. It also has potential where fast development and short ground support cycle times are desired. It is used in various underground support applications including prevention of small rock falls and rock unravelling, control of rock fracturing and bulking in rock that is failing progressively, and, prevention of small-scale strain bursts near the face of tunnels (Archibald et al., 1997). Polyurethane lining has been used successfully by Inco Ltd. to support ground in narrow-vein mining stopes and to protect mesh and bolts on the backs of top sill drifts exposed to blast damage (Espley, 1998). In the latter application, polyurethane effectively replaces shotcrete

Tannant (1998) describes a testing system designed to evaluate the performance of membrane liners in situations where the membrane is employed to support jointed or fractured rock. The test also measured the capacity of the membrane to resist loads from small-scale wedge failures and provide comparative assessments with mesh. Spraying polyurethane over an arrangement of inter-locked concrete blocks constituted the test panels. They were then loaded with a 300-mm pull plate and the loads and displacements were measured. The testing is claimed to demonstrate the importance of creating a near-continuous membrane in order to provide effective support for the blocks. When the limng failed to bridge the gaps between blocks, the resulting support capacity was severely compromised.

Archibald et al. (1997) presented a report on the field and laboratory support response of spray-on Mineguard Polyurethane liners. They described the brand name "Mineguard" as an innovative spray-on rock lining material developed, manufactured and tested primarily for use in underground hard rock mine and other geotechnical sites in order to provide rapidly-depioyable area support coverage. They state that it is intended to replace or work in conjunction with either shotcrete or the screen component of bolt-and-screen support systems in a variety of support roles. Mineguard can easily and rapidly be deployed on rock surfaces, and has the ability to achieve over 90% cure within seconds of being applied, thus offering considerable benefits for mining or excavation operations in which rapid rates of advance and a high degree of automation are essential. Mineguard support is claimed to be similar in cost to bolt-and-screen support methods and cheaper than non-reinforced shotcrete support. Additionally, the material handling requirements are low.

The authors carried out six-year, laboratory and in-situ mine assessment trials to determine support and other physical response capabilities of Mineguard for various mining and geotechnical applications. In their paper under reference, they review practical considerations, usage and results of several field case histories, which illustrate various mining applications of Mineguard. Other conclusions from their study are the following:

Its bright colouring improves lighting conditions in underground environments. This property obviously improves worker safety.

Mineguard polyurethane liner material has been shown to be one of the most effective sprayable barriers to radon gas diffusion yet measured. Its potential application in uranium mines and other mines where such gases may be present are obvious.

2.3.2 Cable bolting

Cable bolting was introduced into die Canadian mining industry over 20 years ago and it has since become, with relatively good success, one of the most important support systems in large underground openings. A conventional cable bolt is a flexible tendon comprised of a number of steel wires wound into a strand, which is grouted into a borehole. Cable bolt boreholes are usually drilled İn a grid pattern, to provide reinforcement and support for the walls, back and floor of underground or surface opening.

The cable can usually bend around fairly tight radii, thus making cable bolting a versatile form of support, especially in cases of long boles and tight working environments. The capacity of the cable bolt is transferred to the rockmass through the grout, which is usually made of Portland cement and water. Thus, they are used in underground hard rock mines to provide a safe working environment, increase rockmass stability, and control dilution of ore from slope boundaries. Cable bolts find greatest use in large spans of underground mines such as major intersections, large underground chambers or in active mining stopes, since larger spans in general result İn greater potential for large free blocks or broken rock falls. Cable bolts can be installed deep into the rockmass thus providing reinforcement and preventing separation along planes of weakness such as joints. By maintaining a continuum nature in the rockmass, cable bolts help to mobilize the inherent strength of the rockmass, thereby improving overall stability.

By far, cable bolting has received the greatest attention in research over the past several years. Research work in Canada in both the laboratory and İn the field by McGill, Queens and Laurentian University, together with mining companies such as Inco and Noranda has identified three principal factors that contribute to the poor performance of plain 7-wire strand cable bolts :

- Poor quality grout,
- Poor quality rockmass providing low radial stiffness at die borehole walls, and
- Mine-induced stress changes which further reduce the rockmass quality.

These three problems have been the subject of detailed research and progress can be reported in solving them. As an example, the introduction of the Garford bulb and the bulge cable offered a practical solution to the three problems while the use of modified geometry cable bolts has been shown to improve bond strength (Hyett et al., 1995).

Most research, however, has dealt mainly with short embedment length pull tests whereas design issue usually involve long cable bolts. Research in this area has led'to the development of computer software nanied CABLE (Computer Aided Bolt Load Estimation), which attempts to extrapolate between thi?- too» \using a numerical simulation (Bawden et al^"l995). Bawden, Möosavi and Hyett (1996) furthejc address the theoretical research of load distribution problem alo$g fully grouted bolts, together with some parametric studies on the effect of grout water: cement ratio, rockmass modulus and face plates on cable bolt performance.

Advances in the theory, application and numerical simulations of cable bolts have been made in several institutions and research centres İn Canada. In the field of laboratory and theoretical studies relating to cable bolts, Hassani et al. (1992, 1995) performed experimental and numerical studies of grouted cable bolt support systems. Khan and Hassani (1993) also examined the application of rigid composite tendons in ground support in mines. Major factors affecting the use of cable bolting have been studied both in the laboratory and in the field. Hassani and Rajaie (1991.) discussed their investigation of the optimization of a particular shotcrete cablebolt support system. They demonstrated the unique yielding behaviour of the particular type of cable bolt, together with its high peak and residual load bearing capacity over conventional cable support systems. Advances in validating the theory and numerical simulations of cable bolts were also made with the development of an instrument called SMART (Stretch measurement to Assess Reinforcement Tension, Hyett et al. 1998). In tests carried out thus far, it has been shown that SMART does not interfere with the bonding process.

Although the primary purpose of cable bolting is in ground support, its use directly affects other important aspects of mining such as dilution control can have a very direct and large influence on the cost of mining. The cost of dilution is high: waste rock is mucked, trimmed, crushed, skipped, milled and impounded in a tailings disposal area at great and unnecessary cost. Anderson and Grebenc (1995) provide a useful illustrative case history of dilution control through the understanding of the causes of failure in one stope and the effective design of support for the adjacent stope by use of cable bolting and backfill.

They discussed the factors to be considered in assessing the cost of dilution. The required information - % dilution, % recovery and % overbreak - is collected from a laser survey of each stope after mining is complete.

There are now in Canada, several grouting systems which are able to mix and pump thick (<0.40 water: cement (w: c) ratio) and even super thick (<0.30 w: c ratio) grouts. The importance of the Portland cement grout in determining the cable bolt capacity has been demonstrated both in the laboratory and in the field, (Reichart et al., 1992, Hyett et al. 1992, Hassani et al., 19).

Cablebolts can be used to support, reinforce or contain rockmass around most types of excavation found in underground mines, including drifts and intersections, open *stope* backs and walls, cut-and-fill stopes, draw points and permanent openings. The following examples are typical of cable bolt application and layout.

It is rare and expensive for a mine to provide access and drifts solely for installing cable bolts. Cable bolting patterns are therefore usually designed depending on the stope and access configurations. The particular borehole pattern selected depends on the intended function of the cable bolts and the available access to the site.

Proper installation İs very crucial to the successful attainment of the objective of using cable bolts. After installation, follow-up and careful observation of the effects of installation are important so that changes, if found necessary, can be effected in the next round of installation. Hutchinson and Diederich, 1996, showed a Cable bolting Cycle, which provides a comprehensive overview of the steps involved in cable bolting operation. It is a cyclical, iterative process, which should be worked through a number of times as mining progresses to ensure that the cable bolting operation is well tuned.

Previous research in the use of cable bolts established that failure most commonly occurred by slip at the cable-grout interface and that the peak strength is related to frictional rather than adhesional resistance. Poor quality grout is one identifiable factor that has contributed to the low performance of the plain 7-wire strand cable bolt. Hyett et al. (1992), carried out a comprehensive investigation to determine the physical and mechanical properties for cement grouts with water/cement ratio varying between 0.70 and 0.25. They pointed out that the factors which affect the physical and mechanical properties of grout (which is essentially Portland cement and water) - namely, the type of cement, its treatment before use, the water/cement ratio and the pumping system - may contribute to the ultimate capacity of die cable bolt.

They examined in detail the factors relating to the major components of cement grouts, namely, the Anhydrous Portland PortlandjCement (APC), Fresh Cement Paste (FCP), and the Hydrated Cement Paste (HCP). For each factor, they made recommendations ranging from storage of the Portland cement both on the surface and in underground environment, to mixing and pumping of the grout by using MINPRO or MAI pumps. These are some of their important findings:

On Anhydrous Portland Portland_Cement (APC), they found that two different cement specifications are used for cable bolting in Canada: normal (type 10) and high-early (type 30). They found that the finer grained type 30 required careful and proper storage in preserving its shelf life, it was more expensive than type 10, and its strength was less than that of type 10 after 2 weeks. Among their recommendations were that operators should use type 10 Portland cement unless the cement paste was for short-term support use of less than 10 days, and that the cement bags arriving at the mine site should be checked for "hardness". Any hard bag detected should cause the whole batch to be sent back to the supplier for a fresh supply.

On Fresh Cement Paste (FCP), they noted that some water is necessary in the paste to make it easy to mix and pump and that for practical cable bolting, ensuring that the specified grout is actually pumped up the cable bolt holes is the most single important quality control issue. They found that for water, cement ratio between 0.70 and 0.35, the bulk density of the paste increases from 1.6 to 2.10 g/cm^3. This property can therefore be used to estimate the water/cement ratio for pastes with values lying within this range.

On thick FPC, they recommend the use of MINEPRO or MAI pump although their pumping efficiency decreases with water/cement ratios < 0.35. They also recommended batch mixing rather than continuous mixing of the grout. On Hydrated Cement Paste (HCP), they carried out laboratory tests to determine the physical and mechanical properties of cement paste with water/cement ratio varying between 0.70 and 0.25. Samples were mixed using an MAI pumping system and left for 28 days to cure at a relative humidity of 95%. They obtained the following results:

The UCS, tensile strength and Young's Modulus increase for 0.70 < water/cement ratio < 0.35, with the Poisson's ratio remaining nearly constant at 0.18. Only the Young's modulus continues to rise for water/cement ratios < 0.35. The 28-day dry density can be used to estimate the water/cement ratio.

Hyett et al. (1992) earned out a laboratory and field research programme in their investigation of the major factors that influence the bond capacity of grouted cable bolts. All tests were conducted on standard 5/8" (15.9 mm) 7-strand cable grouted using type 10 Portland cement pastes. The results indicate that cable bolt capacity most critically depends on the following factors:

- The cement properties, which are primarily controlled by water/cement ratio.
- The embedded length, and,
- The radial confinement acting on the outer surface of the cement anmîus.

They found that the properties of the cement paste varied with the water/cement ratio of the mix, and that a low ratio (< 0.40 by weight) can increase the peak cable bolt capacities by 50 - 75%. According to the authors, this is attributable to then-high uniaxial compressive strengths and their high Young's moduli. They also report that the effect is maximized under conditions of high radial confinement. On the other hand, the use of super-thick pastes (0.30 and less) may be both impractical and undesirable, first because of their limited pumpability and second because of their inconsistency in strength.

Bawden et al. (1995) present a numerical formulation for determining the axial load along a cable bolt for a prescribed distribution of rockmass displacement- Their formulation is based on research findings (Fuller and Cox, 1975; Goris, 1990; Hyett et al., 1992) that the bond strength of a fully grouted cable bolt is frictional rather than adhesional in nature, and that during the process of debonding, a progressive increase in the mismatch between the cable and the grout first splits the surrounding grout annulus, and thereafter pushes the grout wedges aside. Depending on the stiffness of the borehole wall, a reaction pressure develops that controls the normal stress acting at the cable grout interface where slip is occurring and hence the bond strength of the bolt.

In another paper by the three authors, Bawden et al. (1996), they present an explanation for the observation that fully grouted reinforcement is more effective in hard rock that behaves as a discontinum than in soft rock. They present analytical solutions for displacement and load distribution along an untensioned fully grouted elastic bolt of specific bond stiffness, which is activated during excavation either by a continuous or discontinuous distribution of rock displacement. They report that results indicate that significantly higher axial loads are developed for the discontinuous case. They also carried out parametric studies, using a finite difference formulation combined with a non-linear model for the bond behaviour of a cement grout of a seven-wire strand cable bolt, to show that lower loads are developed in soft rock.

An excellent book, on all aspects of cable bolting, has been written by Hutchinson and Diederich (1996) with the sponsoring of the Canadian Mining Industry.

2.3.3 Shotcrete

The use of shotcrete İn Canadian hardrock mines has experienced rapid growth over the past 10 years. Field and laboratory evaluation of its performance as a ground support system has been investigated by several researchers. The work of the Geomechanics Research Centre of Laurentian University, Ontario is typical of the kind of research done in this area. They confirm that in some cases, the introduction of shotcrete has helped to extend the mine life or has reduced rehabilitation costs and production delays. While shotcrete is now accepted as a viable support option, there is increasing evidence that better guidelines are needed to ensure appropriate (cost-effective) selection and application of different types of shotcrete in widely differing underground environments.

The Geomechanics Research Centre has been involved in numerous field and laboratory investigations of shotcrete performance and is working towards establishing guidelines to assist the mining industry in selecting the most appropriate type of shotcrete or support system for a given application.

A zone of failed rock usually develops around excavations at depth or in highly stressed ground. This failing rock dilates and bulks in volume as it fails. Support systems designed to control the failure process and to maintain stability and safety in the excavation must be able to accommodate these deformations. Hence, much of GRC's testing has focused on the performance of shotcrete under large imposed displacements.

Loading of the shotcrete may occur very rapidly m rockburst situations or gradually over time when progressive failure processes dominate the rockmass response near the excavation. From numerous tests on shotcrete with various loading rates (pull tests, impact tests, and explosive loading) much needed data about the capacities of shotcrete under field and large-scale testing conditions has been generated. Some of the findings include:

Contrary to results from tests on small-scale shotcrete beams, GRC has found that mesh-reinforced shotcrete offers more toughness and higher load carrying capacity than steel-fiber reinforced shotcrete at very large imposed displacements. Preventing excessive tangential stresses in the shotcrete is necessary for optimal shotcrete performance m excavations that will experience large convergence or closure after me shotcrete is applied.

Shotcrete can retain its functionality near large production blasts. Shotcrete (plain, steel fiber, or mesh reinforced) can survive peak particle velocities in the order of 1 to 2 m/s as long as the underlying rock is not forcibly ejected into the excavation and the shotcrete İs applied as panels that are not highly stressed.

The addition of shotcrete to mesh greatly improves the mesh's load carrying capacity and results in a retaining component that has superior energy absorption properties.

GRC plans further testing to evaluate shotcrete performance (Tannant, 1998). Other areas of interest include the abrasion and impact resistance of shotcrete for use in ore passes and storage bins, comparative evaluation of shotcrete, Mineguard and new types of fiber reinforced shotcrete, and the development of procedures for designing shotcrete-based support systems for different applications. Our research will also focus on developing better tools for predicting the demand (load, displacement, and energy) that may be placed on shotcrete for specific excavation geometries, stress levels, and rockmass conditions.

One of the major problems in the use and application of shotcrete is quality control as well as ensuring the thickness of the shotcrete lining. This issue is currently being addressed by the author at McGill University and within the next six months special non-destructive testing equipment will be available directly give the above information on site (Hassani et al., 2001).

3 AUTOMATION

As the mining industry moves towards the twenty-first century, the opportunity to apply emerging technologies to enhance production and resource performance and provide new products are critical to the industry's ability to serve the nation and achieve profitability. Once these technologies are developed and in place, they will allow the industry to use its energy, land, capital and labour resources even more efficiently during all stages of the mining cycle. This will in tum, create a safer, less environmentally disruptive industry with higher quality output at lower cost. Satellite communications systems and information processing technologies are already reducing costs and minimizing environmental disruption associated with reserve characterization and production. Automated machines reduce worker exposure to hazards while in situ processes contain the disruption associated with extraction and processing.

Canada is considered to be amongst the leaders in technology in the mining industry. There are many companies considering Automation because of new discoveries of complex deposits. Canadian companies have invested greatly İn Robotic mining technology and techniques that offer a number of positive benefits and some unique engineering challenges. These Canadian companies believe that on-line information about geology, (geophysical, geomechanical and geochemical), production rates and quality will provide a significant advance in mine engineering planning and logistics.

The automation research in mining was initiated by (Canadian Centre for Automation and Robotics in Mining (CCARM) at McGill University in Montreal with support of mining companies such as Inco, Noranda and Falconbridge. In recent years, a five years project with partners Sandvick, Tamrock, Mining Equipment Inc, DynoNoble and Inco was initiated to create a teleoperated mine. This Mine Automation project Or MAP started in 1996,and has had tremendous achievements.

Remote mining in combination with some simple automation offers some solutions to some technical issues. Robotic mining or automation of equipment requires communication systems, positioning and navigation and process engineering, momtonng and control systems. These processes are easier to develop for surface mines in Canada however the challenge relies İn underground environments. Currently most mining operations are searching for voice communication and some limited computer control and video surveillance. Inco was able to develop a commercial communication system, which is now marketed by Automated Mining Systems.

Communication advances are the main reasons for the use of robotic mining. Inco contracted with IBM in 1988 to develop an advanced communication system for underground mining based on CATV and radio transmission technology. This system has been installed, tested and deployed in mining operations such a Stobie Mine.

Since robotic mining is gaining support, Canadian companies are now considering an enhancement of the communication system components.

Canadian automation is now to a point where engineering systems provide online information directly to machines for set-up of drills and the provision of map co-ordinates. It is also possible to use Drift drilling usingjumbos by teleoperation from surface. These machines will, in turn, provide feedback to blasting systems and rock classification systems. Furthermore, explosives loading equipment can feed emulsion-based variable energy explosives into each hole placing a detonator that can be fired over the network electronically from surface. Finally, Robotic mucking will picked-up the fragmented rock (ore or waste) and move it to the appropriate dumping point.

This new technology allows minimal ground support, as the opening will be smaller since there will be no people in the workings. If ground support is required it will be done only where needed to ensure equipment survival. Similarly, ventilation will be reduced, as no personnel will be needed in the operation. Road conditioning can be accomplished by equipment such as the Road Router TM to ensure that the high-speed reliable machines can work effectively in this robotic environment. Furthermore, Constant feedback of production, engineering and maintenance information will be transmitted to ensure each mining mission is accomplished.

4 ENVIRONMENT

Increasingly stringent environmental policies in Canada will put upward pressures on production, processing and product costs at the same time that international competition and alternatives to mining products will require that costs remain competitive. Environmental costs can be significant. For example, the cost of environmental compliance in Canada for metal mining, processing and fabrication was about 10 percent of total costs.

It is very important briefly address mine waste in the context of the new environmental era. The disposal of tailings is a major environmental problem. It becomes more serious with the increasing exploration for metals and working of lower-grade deposits. It is estimated that the Canadian mining industry produces in excess of 500 million metric tonnes of solid waste each year - a 1988 estimate that includes metal, uranium, coal and industrial minerals. Wastewater associated with the mining and milling operations constitutes an equal amount of liquid waste (Intergovernmental, 1988). Quebec alone İs estimated to have produced 102.1 million metric tonnes of tailings in 1990 (Rallon, 1989). This waste has to be disposed of in the most environmentally and economically acceptable manner.

Traditionally, three techniques are available to the mining industry to dispose of its waste: (1) underwater (lake) disposal; (2) underground disposal (backfilling) and (3) surface disposal. Underwater disposal lacks public acceptance and has limited application, this method requires a deep body of water. Tailings must be chemically harmless, and piped to deep water to avoid the most biologically productive areas near shore zones (Vick, 1981). Some believe that underground disposal represents the future of waste disposal in the mining industry. New backfill methods like total tailings fill can utilise all size fraction tailings and recent advances suggest that İt may significantly reduce surface disposal, which has traditionally been the most common method of waste disposal. The high percentage of fines within current tailings as well as the presence of sulphide ores pose some difficulties with tailings pond management.

The increasingly strong environmental lobby, in its search for scapegoats, are now turning to the law in their pursuit of the directors of mining companies. They can be held personally responsible for pollution attributed by their company

It becomes clear that as the mining industry moves in to new millennium will have to clean up its act. Otherwise the North American mining industry will go the way of the spotted owl and the desert tortoise and become the latest addition to the endangered list. Over die years the regulatory conditions have "snowballed" and before they strangle the industry it must change and keep ahead by utilising new technology. Such technology exists and as we start the 21st century the importance of an appropriate total mine solid waste disposal can only increase.

5 CHALLENGES AND VISION FOR THE FUTURE

As the Canadian mining industry moves into the twenty-first century, it is required to continue produce products with lower costs and superior qualities, while minimizing environmental disruptions and maximizing safety of the labour in order to stay competitive. The mining industry must continue to practice responsible stewardship of national resources by developing and applying advanced mining and environmental management technologies in Canada.

Advanced mining techniques and technologies are the key to increase productivity and permit exploration, extraction and processing to occur with maximum efficiency and minimum environmental impact. The development of new technologies and the transfer of appropriate technologies developed for other applications. Transfer of these technologies and development of new technologies will enable the industry to reach its goals and provide Canada with materials critical to economic, environmental and energy sustainability. Paradoxically, successful development and application of new technologies that incorporate advanced technologies such as computers, communications and robotics together with new mining techniques will be the key to the mining industry's improved profitability.

The future success of the mining industry will also depend on its relationship with the public. Public opinion is one factor that will determine whether young men and women become mining engineers rather man engage in opportunities in other industries such as high technology sector. Another factor is the lack of academics available to teach mining related" subjects. The public needs to be further informed of the value of mining. So to insure that they recognize the value of the goods derived from mining and their integral role in our everyday life.

The lack of highly trained mining engineers for the industry is going to be one of the major challenges facing the industry. The decline in number of students choosing mining discipline in the Canadian universities is the indicative of the future shortage.

It is vital that in the schools, teachers and students to have accurate information about mining and the importance of minerals to their lives. The mining industry and the education community needs to provide accurate, interesting, and informative teaching materials and delivery systems that reach the majority of administrators, teachers, and students.

A strong massage should be sent out stating that the mining industry provides professional opportunities that attract the best and the brightest. University mining programs should employ interdisciplinary faculty and should offer programs that enable students to learn all of the skills applicable to an advanced mining industry, including environmental sciences, chemical engineering, computers and robotics, advanced communications, mineral economics and international relations as well as language training.

The industry needs to work together with the public to ensure that environmentally sound minerals resources that are required to sustain a high quality of life in Canada and throughout the world.

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REFERENCES AND BIBLIOGRAPHY

Adams, D.J., Gurtunca, R.G., Jager, A.J., and Gay, N.C., ((989). Assessment of a new mine layout incorporating concrete pillars as regional support. In Innovations in mining backfill technology, Proc. of the 4th International symposium on mining backfill, Montreal, 2-5 October, 1989, Hassani, Scoble and Yu Editors, Rotterdam : A.A. Balkema, 1989, pp. 199-208.

Anderson, B., and Grebenc, B., (199S). Controlling dilution at the Golden Giant Mine. CIM Mine Operators Conference, Timmins, Canada, paper no. 4, 14p.

Archibald, J.F., Espley, S.J., & Lausch, P. (1997). Field and laboratory Support Response of Spray-on Mineguard Polyurethane Liners. Proc. Int. Symp. On Rock Support: Applied Solutions for Underground Structures. Lillehammer, Norway, pp. 475 - 490, June 1997.

Aref, K. and Hassani, F.P. (1988), Strength Characterization of Paste Backfill. Proceedings of the International Conference of Applied Rock Engineering. Newcastle upon Tyne, pp. 4-8.

Bawden, W.F., Moosavi, M. & Hyett, A.J., (1995). Computer Aided Bolt Load Estimation (CABLE). 97* A.G.M. of d.M. Rock Mechanics and Strata Control Session, Halifax, Nova Scotia, may 14-18, 1995.

Clarke, I.H. (1988), Evaluation of Cemented Tailings Backfill in South African Mines. SAIMM Special Publication No. SP2, pp. 77-89.

Clarke, LH. (1988), The Properties of Hydraulically Placed Backfill. Proceedings of Backfill in South African Mines, Johannesburg SAIMM, pp. 15-33.

Corson, D.R. (1971), Field Evaluation of Hydraulic Backfill Compaction at the Lucky Friday Mine, Mullan, Idaho, U.S. Bureau of Mines, RI7546.

Corson, D.R. and Wayment, W.R. (1967), Land Displacement Measurement in a Backfill Stope of a Deep Vein, U.S. Bureau of Mines, RI 7038.

Epsley, S. (1998). Implementation of Minguard spray on liners in the hard rock mining industry. M.Sc. Thesis, Laurentian University.

Gurtunca R.G., Adams D.J., (1991). Determination of the in situ modulus of the rockmass by the use of backfill measurements. J. S. Afr. Inst. Min. Metall., vol.91, no.3. March 1991, p.81-88.

Gurtunca R.G., Jager A.J., Adams D.J., Gonlag M., (1989). The in situ behaviour of backfill materials and the In: Innovations in mining backfill technology. Proceedings of the 4th international surrounding rock mass in South African gold mines.

Hassani, F.P., and Rajaie, H., (1990). Investigation into the optimization of shotcrete cable bolt support system. Proc. of 14lh Congress of the Council of Mining and Metallurgical Institution, Edinburgh, Scotland, pp. 119-130.

Hassani, F.P., Mitri, H.S., Khan, U.H., & Rajaie, H. (1992). Experimental and numerical studies of the cable bolt support systems. Rock Support in Mining and Underground Construction, Kaiser an McCreath (eds), 1992 Balkema, Rotterdam

Hassani, F.P., Khan U.H. & Mortazavi, A. (1995). Laboratory testing of fully resin grouted weldgrip fiberglass rockbolts. Rock Mechanics Daemon & Schultz (eds), 1995 Balkema, Rotterdam.

Hassani F. P. and Archibald, J.F.(1998) Mine backfill , published by Canadian Institute of Mining and Metallurgical Engineering Montreal Canada, CD Format.

Hassani, F.P., Fotoohi, K., and Doucet, C, (1998). Paste backfill performance in a narrow vein gold mine. 3rd North American Rock Mechanics Symposium, Cancun, Mexico.

Hassani, F.P., Momayez, M., and Guevremonl, P., *Odanold D.* (2001). Insitu non-destructive thickness measurement of shotcrete lining. Presented at annual general meeting of CIM , Quebec city, 12pp.

Hassani, F.P., Momayez, M., and Guevremont, P., (1998). Shotcrete measurement tests : Preliminary investigation at INCO's Stobie Mine using MSR Impact-echo and GPR. SubSurface Sensing Laboratory, McGill University, Montreal, Canada, report no. SSSL-98-003,34 p.

Henderson, A., Newman, P., Landriault, D. and Antoniazzi, P., (1997). The Cost Advantage of Using Paste as a Backfill. 99lh CIM Annual General Meeting, Vancouver, British Columbia, April 27-May 1,1997. CD-ROM.

Hoek, E., and Brown, E.T., (1980). Underground Excavation in Rock. Inst, of Mining and Metallurgy, London, England, 527 p.

Hutchinson, D.J., and Diedericks, M.S., (1996). Cablebolting in underground mines. BiTech Publishers Inc., Vancouver, Canada, 406 p.

Hyett, A.J., Bawden, W.F. & Coulson, A.I., (1992). Physical and mechanical properties of normal Portland cement pertaining to fully grouted cable bolts. Rock Support in

Mining and Underground Construction, Kaiser an McCreath (eds), 1992 Balkema, Rotterdam

Hyett, A.J., Bawden, W.F. *Sc* Reichert, R.D., (1992). The Effect of Rock Mass Confinement on the Bond Strength of Fully Grouted Cable Bolts. Int. J. Rock Mech. Min. Sei. & Geomech. Abstr- Vol. 29, No. 5. pp. 503 - 524,1992

Hyett, A.J., Bawden, W.F., Maccsporran, G.R. & Moosavi. M.. (1995). A constitutive law for bond failure of fully grouted cable bolts using a modified Hoek Cell. Int. J. Rock Mech. Min. Sei. & Geomech. Abstr. Vol. 32, No. 1, pp. 11-36,

Hyett, A.J., Bawden, W.F., Hendrick, N. and Blackall, J-, (1995). A laboratory evaluation of the 25 mm Garford bulb anchor for cable bolt reinforcement. CIM Bulletin, Vol. 88, No. 992,

Hyett, A.J., Moosavi, M. and Bawden, W.F.. (1996). Load Distribution Along Fully Grouted Bote, With Emphasis on Cable Bolt Reinforcement. International Journal for Numerical and Analytical Methods in Geomechanics, Vol. 20,517-544,

Khan, U.H., and Hassani, F.P. (1993). Analysis of new flexible and rigid composite tendons for mining. *Innovative Mine Design for th 21" Century,* Proceedings of the International Congress on Mine Design, Kingston, Ontario, Canada, 23-26 August 1993.

Landriault, D., Welch, D. and Morrison, D., (1996). Mine Tailings Disposai as a Paste Backfill for Underground Mine Backfill and Surface Waste Deposition. Prepared for the Society.for Mining, Metallurgy, and Exploration, Inc. Short Course, March 9 and 10.

Tannant, D.D., (1998). Personal communication. Geotechnical Research Center (GRC), Laurentian University, Sudbury, Canada.

Tannant, D.D., McCreath, D.R., and Kaiser, P., (1996). Impact test on shotcrete and implications for design for dynamic loads. 2nd North American Rock Mechanics Symposium Montreal, Vol. 1, pp. 367-373.

Thibodeau, D. (1989), Insİtu Determination of High Density Alleviate Sandfill. 4th International Symposium İn Mine Backfill. Innovation in Mining Backfill Technology, 267 pp.. Editor F.P. Hassani et al., Balkema Publishers.

Intergovernmental Working Group on the Mineral Industry. 1988. "Report on the economic and policy aspects of acid discharge". By the Federal/Provincial/Sub-Committee on Mine Water. August 29,1988,64p.

Rallon, H., 1989. "L'industrie Minérale du Quebec". Le Service de la Statistique et de L'économie minérale de la Direction de L'analyse Economique et du Développement Minier. Ministère de L'Energie et des Ressources, 1989, 89 P-

Scoble ,M. (1994) Competitive at depth: Re-engineering and hard rock mining process. Rock Mechanics.Nelso&Laubach(eds),BaIkema .Rotterdam ,p3646,

Vagenas, N. Scoble, (1998) Issues İn underground hard rock mining automation, Mine planning and equipment selection Edited Singal ,Balkema,Rotterdam,p469-474,

Vick, S.G., 1981. "Sitting and design of tailings impoundments". Mininp Engineering. Vol. 33. No. 6. June 1981. pp. 653-657.

Udd, j. (1989) Backfill Research in Canadian Mines. Innovation in mining backfill technology .3rd Int. mine backfill Symposium, Montreal .Edi. F Hassani et al. Balkema pubL pp3- İ 3

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Recent Developments and Outlook for Clean Energy from Coal without Combustion

T.M.Yegülalp
Henry Krumb School of Mines, Columbia University, New York, NY 10027, USA

ABSTRACT: Development of new technologies and improvements on existing technologies are providing opportunities for coal *to* maintain its strong standing as a major competitive energy source for many centuries into the future. These technologies provide means for development of sustainable methods of production of hydrogen from coal without combustion and subsequent production of clean energy using fuel cell or turbine technology. In this paper, a review of past and current studies in this area is presented, concepts leading to design of zero emission power production are presented, and the impact of these developments on future energy and mining industries, contribution to improving environmental quality are examined. Finally, a preliminary estimate of power cost is provided.

1 INTRODUCTION

The release of greenhouse gases, primarily CO2 due to combustion of fossil fuels is a major global concern. Although, coal is not the only fossil fuel, which by combustion contributes to the continuously increasing CO2 content of the atmosphere, it is the major contributor to İt per unit of energy produced. In the United States 84% of energy consumed, (81.557 10^{15} Btu, or 23.89 10^{12} kWh) comes from various fossil fuels. Specifically, coal, natural gas, and petroleum represent 23, 23, and 39% of energy consumption {DOE EIA 2000a) and contribute 549.3, 649.7, and 311.8 million metric tons/year of carbon emissions respectively (DOE EIA 2000b).

It is desirable to develop processes, which will allow the use of vast coal resources without the environmental consequences due to emission of greenhouse gases, SO2, NOx, and particulate matter. The traditional method of producing power by coal combustion and steam generation, in spite of many improvements, is losing ground simply because of the fact that conventional combustion itself leads to inefficient energy production and release of environmentally undesirable byproducts. In the following, we will examine the problems associated with production of energy by coal combustion, proposed methods for removing *CO2* from the atmosphere, and review newly emerging technologies, which may provide economical and sustainable answers to these problems.

2 $C0_2$ DISPOSAL PROBLEM

Depending on the availability of local resources each country has a different mix of fossil fuel and renewable energy to meet its demand. Regardless of its impact on the global warming, CO2 produced by combustion of fossil fuels constitutes a serious problem in the long run. Since the beginning of the 19^{th} century CO2 content of the atmosphere has risen by about 30%, from 280 ppm to 360 ppm (Siegenthaler & Oescher 1987; Keeling et al. 1995). The increase recorded during the last 40 years (from 315 ppm to 360 ppm) accounts for more than 50% of the total increase during the last two centuries. Considering the current and projected future fossil carbon consumption and the available fossil carbon resources, it is conceivable that in the distant future CO2 levels would reach intolerable levels (Yegulalp et al. 2000).

Safe and permanent disposal of $C0_2$ resulting from the combustion process is the key to sustainability of energy production from fossil fuels. Since coal is the most carbon intensive fossil fuel, this problem impacts the coal combustion more than other fossil fuels. In order to prevent CO2 accumulations in the atmosphere we need to collect and dispose of the combustion products. Unfortunately, as we burn coal, we produce a mixture of gases including CO, CO2, SO2, and NO_x along with remaining oxygen and nitrogen as well as particulate matter composed of carbon and ash. Separation of CO2 from this mixture for disposal is a formidable task.

which makes this option technically difficult and economically undesirable (Yegulalp et al. 2000).

Here, we are not concerned with the approach of avoiding CO2 production by using other forms of energy. Although these methods can and will contribute to the reduction of CO2 emissions but energy conservation and energy efficiency will not be sufficient to meet the ever growing global demand. Alternative forms of energy that don't produce CO_2 are still far too expensive to compete. Without a major and unexpected technological breakthrough for an economically viable and sustainable carbon free energy resource, it is not realistic to expect a growing world energy market without a major contribution from coal.

Sequestration technologies can prevent the accumulation of CO2 in the air without limiting the use of fossil fuels. Sequestration can be accomplished İn a variety of ways. CO2 can be collected at the point of combustion or later taken from the air. CO2 can be stored İn gaseous form or could be chemically transformed before it is disposed as waste. It has been suggested that some of it can be recycled back into the economy.

2.5 Biomass Sequestration

Biomass generation has been considered as a method of sequestration. This however is a means of collecting energy, which ultimately will be wasted. It İs not feasible to store the perpetual accumulation of carbon as biomass. A mature forest will lose about as much biomass as it generates. Since biomass collection rates are very small (Ranney & Cushman 1992), one needs to dedicate unrealistic amounts of land or ocean to use this option as the sole means of sequestration of CO2 (Lackner et al. 1998; Sedjo & Solomon 1989). It İs also shown that the annual collection of carbon on an acre (0.4 ha) of land at best compensates for a couple of minutes worth of CO2 released from a one GW coal-fired power plant (Ranney & Cushman 1992).

2.6 Underground Injection

Another option could be to inject CO2 into some suitable geological formation for permanent storage. This idea is already being practiced at a limited extent for various purposes such as to dispose of the CO2 stripped from natural gas in Norway. Because of high carbon tax in Norway (\$55/t CO_2) CO_2 it is feasible to inject CO2 stripped from natural gas into an aquifer 1000 m under the sea floor in the North Sea (Kaarstad & Audus 1997). In crude oil and natural gas production, CO_2 is injected to increase production rates. A recent study (Akihiro et al. 2000) forecasts that separation, liquefaction and injection of CO2 from a coal-fired power plant in Alberta Canada would cost from C\$56 to C\$64 per ton of CO2 disposed into an aquifer. Some CO_2 can be injected to recover methane from deep coal seams (Gunter et al. 1997) as a combinatîoéisf disposal and methane recovery system. " ' v ^ '

2.7 Ocean disposal

There are various forms of ocean disposal, which differ in how and where CO; İs introduced into the ocean (Herzog et al. 1997). CO2 can be transported in an undersea pipeline from the shore, or it can be introduced from a ship that carries it to a deep part of the ocean. It can be introduced as a compressed gas at great depth, injected as a water clathrate, it can be introduced as dry ice or bubbled into intermediate depth where it dissolves in the water. Very deep storage has the advantage that the CO_2 becomes denser than water and forms a layer on the bottom of the ocean, which only gradually dissolves into the surrounding water (Herzog et al. 2000). However, although ocean circulation guarantees that over time the highly soluble CO2 is mixed into the ocean the allowable change in pH will limit how much can be stored İn the ocean. Approximately 1000-Gt of carbon added as bicarbonate ions to the ocean would change the overall pH by 0.3 (Yegulalp et al. 2000).

2.8 Carbonate Disposal

With the exception of biomass generation, all sequestration methods propose disposing of CO2 in gas form. A new technology originally suggested by Seifritz (1990) suggests that it is possible to dispose of CO; in the form of carbonates (Lackner, et al. 1998). The technology is based on the well-known reaction of CO2 with common mineral oxides to form carbonates like magnesite or calcite. The resulting product is an environmentally safe carbonate and it is thermodynamically stable.

In nature, however, calcium and magnesium are rarely available as binary oxides. They are found typically as calcium and magnesium silicates. The carbonation reaction is exothermic for common calcium and magnesium bearing minerals. As an example, consider the following carbonation reactions of forsterite and serpentine;

$$\frac{1}{2}Mg_2SiO_4 + CO_2 \rightarrow MgCO_3 + \frac{1}{2}SiO_2 + 95kJ/mole \qquad (1)$$

$$\frac{1}{3}Mg_3Si_2O_5(OH)_4 + CO_2 \rightarrow MgCO_3 + \frac{2}{3}SiO_2 + \frac{2}{3}H_2O + 64kJ/mole \qquad (2)$$

Both of these reactions are favored at low temperatures. Technologies being developed involve the use of abundant natural silicates such as serpentine and accelerate the CO2 acceptance process at an industrial scale (Yegulalp et al. 2000; Lackner et ai. 1997). At present, this process is still at an early research phase. However, recent reports indicate that significant progress has been made during the last two years (O'Connor et al. 1999; O'Connor et al. 2000).

3 HYDROGEN FROM COAL

One approach to use coal as a clean energy source is to convert carbon İn the coal to hydrogen as the first step to zero emission energy production. This is based on the well-known CO2 acceptor process, which has been pioneered by Consolidation Coal Company earlier (McCoy et al. 1976; Fink et al. 1977). The basic idea İs to assist the reforming shift reactions, which make hydrogen from water by turning carbon into CO2 and use CaO to remove the CO2 from the reaction products.

Expanding on this idea, Ziock and collaborators are developing an anaerobic process for hydrogen production from coal (Lackner et al. 1999b; Yegulalp et al. 2000) in which coal, water and lime are used to form hydrogen and limestone as an intermediary. The process of producing hydrogen consists of the following four steps:

Step 1: Carbon in coal is reacted with hydrogen to produce methane, i.e.

$$C + 2H_2 \rightarrow CH_4 + 75\,kJ \qquad (3)$$

Step 2: Methane is reacted with water and lime to produce hydrogen and CaCOî, i.e.

$$CH_4 + CaO + 2H_2O \rightarrow CaCO_3 + 4H_2 - 75\,kJ \qquad (4)$$

Step 3: CaO is recovered by calcining CaCOî and producing a pure stream of $C0_2$, i.e.

$$CaCO_3 + 178\,8\,kJ \rightarrow CaO + CO_2 \qquad (5)$$

Step 4: Electrical power is generated using a fuel cell or heat from combustion by combining hydrogen and oxygen, i.e.

$$2H_2 + O_2 \rightarrow 2H_2O + 571\,7kJ \qquad (6)$$

The energy released m the fourth step is partly used for the heat needed in the third step for calcining $CaC0_3$. The remaining energy (571.7 - 178.8 = 392.9 kJ) is approximately the same amount of energy released by combustion of carbon m the coal.

$$C + O_2 \rightarrow CO_2 + 393\,5\,kJ \qquad (7)$$

The hydrogen needed in step 1 İs produced in step 2 by recycling half of the hydrogen output in step 2. The purpose of producing intermediate methane in the first step is to keep the lime used for carbonation isolated from the impurities present in the coal. For example, the sulfur is separated between steps one and two so that methane entering into reaction with CaO is sulfur free. The net result of this four-step process is clean energy from coal with easily manageable pure waste products such as gypsum from sulfur, ash separated in the first step, and pure CO2 ready for an efficient sequestration process. Several detailed discussions of this process are available elsewhere (Lackner et al. 1999; Saunders, 2000; Yegulalp et al. 2000).

A preliminary cost estimation has been carried out as a basis for more detailed and design-based cost study (Saunders 2000). This study concluded that the cost of producing electric power using solid oxide fuel cell technology and hydrogen from coal based on the methodology outlined above could be as low as $0.05/kWh without CO2 sequestration.

4 POWER GENERATION

Hydrogen, a clean carrier of energy, can be used in a variety of ways to generate power Conventional technologies based on combustion and subsequent steam generation as a basis for power generation are inefficient and are giving way to more efficient turbine technology or fuel cell technology to produce electric power at efficiencies in the order of 60 - 70%. It is clear that new 'energy generation complexes will have the opportunity and economic incentive to employ cleaner generation systems with higher efficiencies.

4.1 Hydrogen fueled combustion turbines

The US DOE Office of Fossil Fuels and Office of Industrial Technology are partners in a program called Advanced Turbine Systems. The program has invested in research to produce high efficiency, low cost, low emission gas turbine systems that will be commercially available in 2002. The technical goals for utility systems are: efficiency greater than 60%, emissions less than 9 ppm NO_x and 20 ppm CO2 and unburned hydrocarbons, and cost 10% below environmentally equivalent 1992 turbine systems (Layne & Zeh 1999). General Electric and Westinghouse are developing systems rated at 400MW and 420MW respectively. Hydrogen fueled turbines are considered part of the long-term plan of the *next* generation systems with anticipated employment in

2010 (Layne 1999). Utilization of hydrogen combustion turbines is associated with implementation of the Vision 21 Program (Ruth 2000) containing research and development plans for 2000 through 2008. This technology will enable Vision 21 to reach the goals of 60-70% electrical efficiency and zero emissions while producing market rate electricity by 2015. Hydrogen combustion technology is also being developed through Japanese government funded research.

The New Energy and Industrial Technology Development Organization through its World Energy Network Program is developing hydrogen energy systems. This program consists of multiple phases. Phase I (1993-1998) included the development of an optimum hydrogen combustion turbine system. The goal of the program is to reach greater than 70.9% thermal efficiency (LHV) without CO_2, NO_x, and SO* emissions. The system must also be as reliable, available, and maintainable as current natural gas combined cycles (Bannister et al. 1997). Three corporations are involved: Westinghouse, Mitsubishi, and Toshiba.

Both Westinghouse and Mitsubishi Heavy Industries analyzed hydrogen combustion turbine systems employing current components or easily modified existing technology. The near term Westinghouse plant model had a net efficiency of 65.2% (LHV) while the optimal Mitsubishi plant design had 72.8% (LHV). However, the Mitsubishi data is from an interim report while the Westinghouse data is from the final report and refinements may change the outcome. The Westinghouse near term model plant efficiency was reported to be as high as 73.5% (LHV) İn an interim report (Bannister et al. 1997). Both systems will be considered, it appears Mitsubishi Heavy Industries has been selected for researching hydrogen fuel utilization in Phase II (1999-2003) of the WE-NET program (Saunders 2000).

4.2 Fuel cell technology

Fuel cells generate electricity and heat electrochemically like batteries. A fuel at the anode (natural gas or hydrogen or CO) and an oxidant such as oxygen or just air at the cathode is supplied. Depending on the electrolyte used we consider three basic types of fuel cells: Phosphoric acid, molten carbonate (lithium or potassium), and solid oxide (stabilized zirconia). Most of the recent research and development effort has been focused on the solid Fuel Cell technology. Currently in the U.S. research is being funded by the Department of Energy (DOE), the Electric Power Research Institute, and the Gas Research Institute. Under the DOE Vision 21 Program conceptual plant designs that would provide market rate electricity from fossil fuels with zero emissions are being developed with the target of commercialization by 2015. The goal is to produce fuel cell/gas turbine hybrids with 60% efficiency by 2003 and 70% efficiency by 2010. Additionally, 2P' century fuel cells using natural gas or coal-dertved fuels are targeted for 70% efficiency in 2010 and 80% efficiency in 2015 (Ruth 2000). The first two groups of projects were selected for Vision 21 in March and August 2000. Central to the success of meeting zero emissions criteria is the development of fuel cells.

5 A COMPLETE SYSTEM

A complete system for near-zero emission power production using coal as its raw material can be implemented in the near future as new technologies for coal processing, hydrogen production, power generation, and CO_2 sequestration become technically and economically feasible. Such a system will consist of the following components (Figure 1): *(i)* one or more coal mines, *(ii)* a coal processing and hydrogen production plant, *(Hi)* a power plant using hydrogen as fuel, *(iv)* ä CO2 processing plant producing MgCOî, *(v)* a surface or underground mine to supply raw material for CO2 processing and as a waste disposal site

J. / *Coal mines*

There is essentially little difference from the conventional coal mining operation. The only significant difference is that coal quality would have minimal restrictions. Since there will not be a conventional coal combustion in the subsequent stages, sulfur is not a quality factor. The amount of sulfur would however affect the consumption of limestone at the subsequent stages. It is foreseen that essentially all sulfur is converted into marketable gypsum. The ash content of ROM coal would affect the efficiency and material throughput at the hydrogen plant. Therefore, coal washing at the mine site will be the same as before.

Transportation of coal from mine site to the hydrogen plant could be in various ways. Since it is desirable to have fine coal (e.g. <-l mm) in the hydrogen plant, it would be feasible to crush all coal and transport it as slurry by a pipeline. Otherwise crushing will take place at the hydrogen plant.

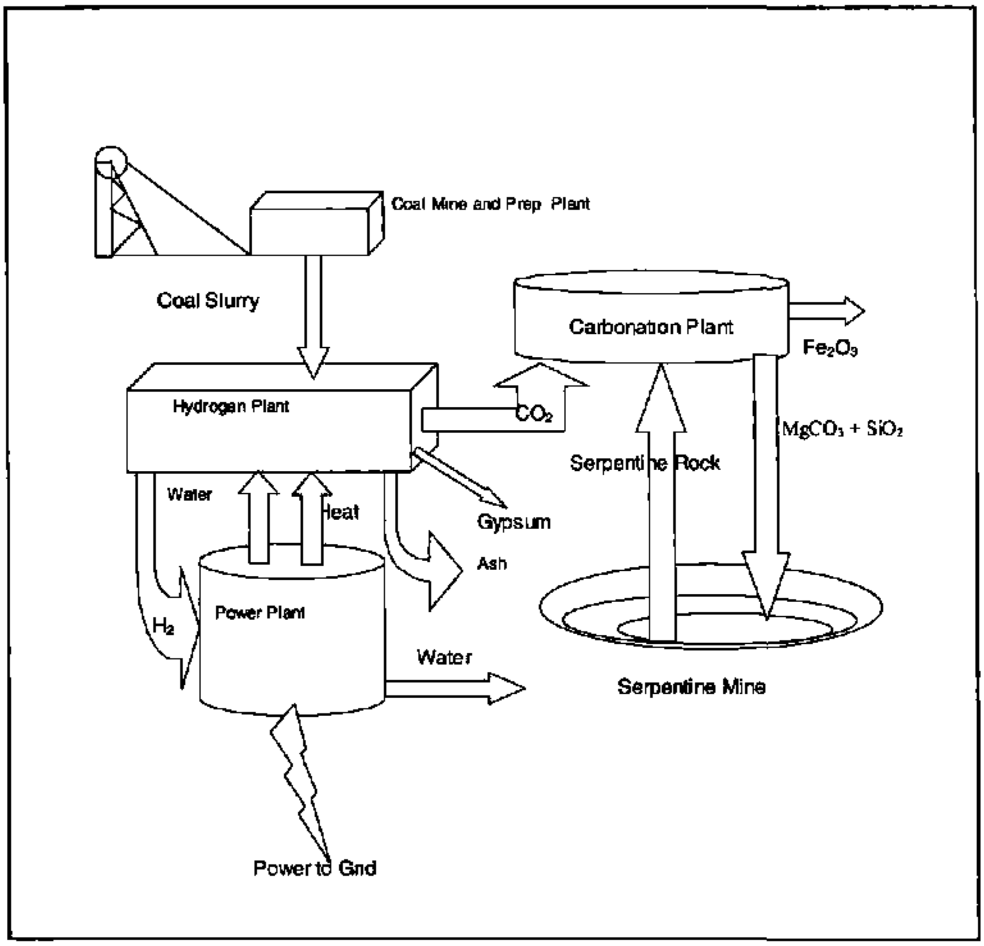

Figure 1. An example of a complete power system with CO: sequestration.

5.2 *Hydrogen Plant*

The plant will consist of three serially connected fluidized bed reactors (Figure 2).

5.2.1. Reactor One (Gasifier)

In the first reactor, hydrogen and carbon are reacted to form methane as described in equation (3). Since this reaction is exothermic, it is expected that some water will be added to control the temperature and to cany the excess heat energy to the second reactor where it is needed. This water could automatically be introduced since coal input stream would be moist and in the form of slurry. The primary product of this part would be methane. There will be also steam, sulfur compounds, ash, minimal CO and CO2, since there is no air introduced into the input stream. Solids are separated from gases and gases are processed through a scrubber to separate sulfur from the gas stream.

5.2 2. Reactor Two (decarbonizer)

The second reactor will receive a gaseous input stream primarily methane and steam. Additional water or steam is introduced to complete the hydrogen balance (equations 3 and 4). CaO regenerated in the reactor three is introduced here to react with methane and steam. The resulting products will be hydrogen and CaCOj. Half the hydrogen generated is recycled back to reactor one, and the remaining is sent to the power plant after a final scrubbing and filtering process. CaCOî is sent to reactor three for regeneration of CaO by calcining and reuse.

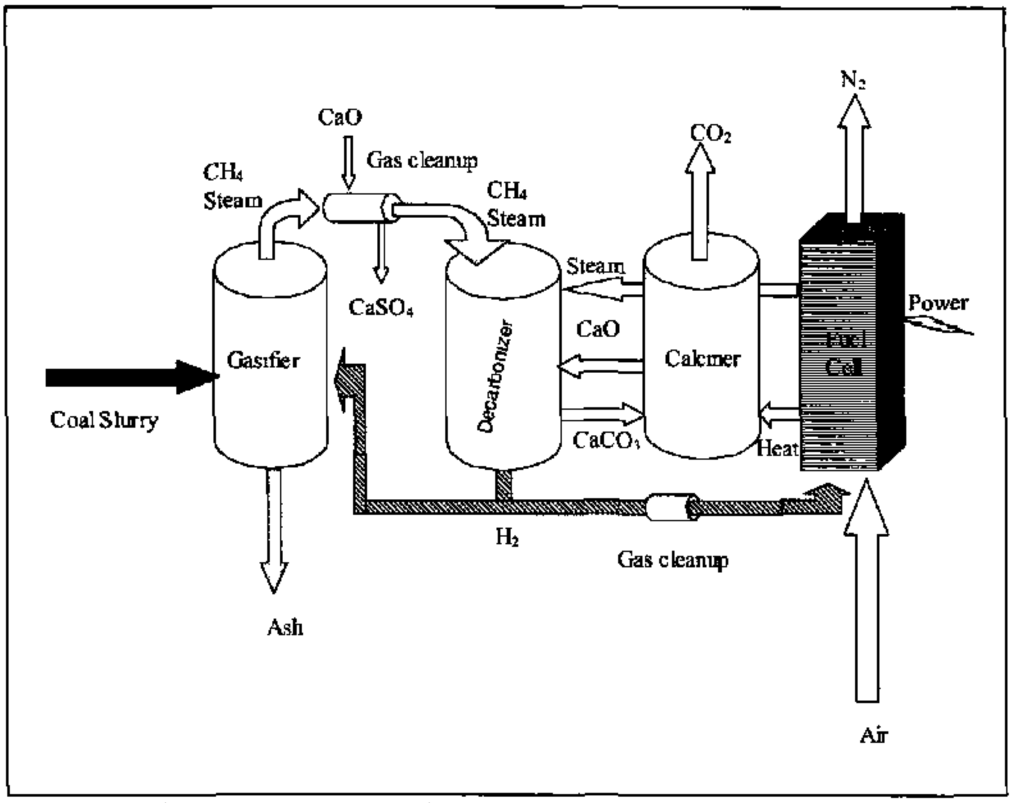

Figure 2 Concept of hydrogen and power production from coal

5.2.3. Reactor Three (Calciner)

The third reactor is a typical calcining system. The necessary heat (178 8 kj/mole) is supplied from the power plant. The product CaO is sent to reactor two, and CO2 is sent to the sequestration plant for disposal. Since CO_2 generated at this stage is pure and concentrated, the subsequent sequestration process will be much more efficient than that of separating and processing of conventional combustion gases.

5.3 *Power Plant*

Any one of the systems descnbed above (Section 4) can be utilized in the power plant. The success in the development of efficient and high capacity fuel cells would determine the near and long-term configuration of the power plant It can be also feasible to transport the hydrogen produced to another site where it is converted to electrical power. However, the heat needed in the third reactor would necessitate at least some of this hydrogen to be used locally.

5.4 *Serpentine Mine*

In this configuration a surface or underground mine is used to provide the necessary raw material (e.g. serpentine, forsterite) to be used in the sequestration process. It is economically advantageous to place the hydrogen plant where the raw material for this process is located. This will prevent large quantities of rock from being transported to the source of CO2 or large quantities of COi from being transported to the mine site. Furthermore, the final product of the carbonation process ($MgCO_3$), along with sihca and other solid waste could be economically disposed off at the mine site.

A typical bituminous coal has about 70% carbon. Complete combustion of a ton of coal will yield 2.6 tons of CO2. A typical serpentine would have approximately 40 % MgO (Goff & Lackner 1998) content, which can be used to accept CO_2 Thus for every ton of coal processed one would need approximately 6 tons of serpentine to be mined and processed.

Mining method and m..hods to be used for disposal of sequestration products would strongly depend on the local geology and structure. However, because of quantities involved, costs associated with mining and crushing serpentine would be comparable to those of large-scale bulk mining costs.

5.5 *Carbonation Plant*

The complete system will have a carbonation plant at the site. The plant will utilize one of the two promising processes for carbonation of serpentine. The first process involves chlorination leading to production of magnesium hydrochloride, which in turn will react with CO2 to yield $MgC0_3$. This process is recently put to use to produce MgCb for electro winning of Mg metal from asbestos waste in Canada (Watson et al. 2000). The second process involves direct reaction of *CO2* with serpentine and water at favorable temperatures and pressures to produce $MgC0_3$ (O'Connor et al. 1999; O'Connor et al. 2000). Both findings are promising and are expected to lead to design and implementation of plants that can be employed for sequestering large quantities of concentrated CO2.

6 ECONOMICS

A preliminary cost estimation was carried out as a basis for more detailed and design-based cost study (Saunders 2000). This study concluded that the cost of producing electric power using solid oxide fuel cell technology and hydrogen from coal could be as low as $0.05/kWh excluding CO2 sequestration costs. Another study was carried out to assess the product markets, the technology market and the effects of clean coal technology on the related industries (Knight 2000). This study focused on $S0_2$, NOx, and CO2 abatement markets and status and future projections of power needs and costs. It was concluded that in the long-term, if for sustainable development all anthropogenic carbon emissions must equal anthropogenic carbon sequestration, a competitive environment for carbon sequestration technology will exist, and carbon sequestration technologies will become a part of the global energy infrastructure.

Economics of zero emission power generation should not be assessed solely on the basis of power generation at a plant but should also take into account external costs and benefits to the society as a whole.

On the cost side, we need to include the cost of coal mining and transportation to the power plants, hydrogen generation, power production with a fuel cell or turbine system, waste disposal at the power plant, and CO2 sequestration. Except for the CO? sequestration, these costs can be estimated with some extrapolation or interpolation of current plant construction and operating costs. Sequestration cost will depend on the method employed (see Section 2). In case of sequestration by carbonation, mining cost would be of the same order of magnitude of current large-scale mining costs. Processing of serpentine rock for CO2 removal would depend highly on the results of ongoing research and development efforts.

On the benefit side, economic and societal benefits of production of power without pollution, elimination of penalties or limitations (SO2, NO x, particulate matter, fly ash disposal) for coal burning power plants, as well as elimination of CO2 emissions need to be accounted for. In addition, in case of sequestration by carbonation, products and byproducts of this process would yield marketable quantities of $MgC0_3$. SiOi, and iron oxides, and in case of certain serpentine types, some quantities of Cr could also be produced. For example chemical analyses of various peridotites show (Goff & Lackner 1998) that while MgO content varies from 42 to 50%, these rocks also contain 39 to 44% $Si0_2$ 7 to 9% FeO and FeA,, 0.1 to 0.3 % NiO. 0.1 to 0.17% MnO, and 0.3 to 1.2 % Cr_30_3 by weight.

Ultramafic rocks contain many mineral resources. Chrome, platinum group metais, nickel, cobalt, and diamonds come from various ultramafic rocks and their eroded products, whereas manganese copper, mercury, and other metals are sometimes obtained from within the bodies from enclosing rocks (Maddock 1964). Determination of the exact quantities or an average of the quantities of mineral byproducts that would be produced in the US or globally is beyond the scope of this paper. However, following an example of the Del Puerto, CA ultramafic rock formation discussed in literature (Goff & Lackner 1998), and considering the chemical composition of major elements found in a serpentinized peridotite sample from Del Puerto ultramafic body shown in Table], we can estimate that for each ton of CO2 sequestered 2.1 tons of rock needs to be processed. Table *2* shows the corresponding amount of products and byproducts of this process for each ton of sequestered $C0_2$

Table I. Chemical composition of Serpent inized Peridotite in Del Puerto Ultramafic Body".

	% Weight
SiO.	38.80
MgO	42.50
FeO	4.74
Fe_2Oi	3.23
NiO	0.27
MnO	0.13
CÏÏOT	0.60
Other	9.73
Total	I0O.00

"Source Goff& lackner 1998

Table 2. Yield Per Ton of CO; Sequestered.

	Yield (fa)
SiO_3	815
MgCOî	1867
Fe	126
Ni	4
Mn	2
Cr	9
Other	204

Del Puerto rock body is reported to contain 33.6 Gt ultramafic rock with varying properties (Goff & Lackner 1998). With the above properties, it would sequester 16 Gt $C0_2$. and yield 2 Gt iron, 70 Mt nickel, 28 Mt manganese, and 140 Mt chromium. It is clear that economic recovery of these byproducts will depend on the process utilized to treat serpentine for CO2 acception and separation of MgCOs from Fe, Cr, Ni and Mn compounds.

To set a scale for the future impact of such a development on mining and related industries, we may just look at the current coal usage for energy generation in the United States. In 1999, The United States consumed ($21.698x10^{15}$ Btu) 6359 GWh energy generated at coal-fired power plants (DOE EIA 2000a). This in tum generated 549 Mt $C0_2$. If all of this were sequestered by mineral carbonation method using serpentine bodies similar to Del Puerto rock body as raw materia! sources, the annual stone production for this purpose would be 1153 Mt. This is in the same order of magnitude of the US crushed stone industry's annual output of 1560 Mt (USGS 2000). '

7 CONCLUSIONS

In this paper, we summarized the recent significant developments towards creation of sustainable clean energy supply systems based on coal. As these new technologies introduced at industrial scales new power generation capacities could be created using new and clean technology replacing the inefficient and polluting power plants. This is particularly significant for the developing countries, where serious power shortages need to be overcome without contributing to the production of greenhouse gases. Because of improved efficiencies on the order of 60 - 70% instead of 33% at coal-fired power plants, coal mining needs would be reduced to almost half of what is needed today to meet the current energy output based on coal. This may be bad news for coal miners, but generation of a huge job and investment opportunities for mining industry at the sequestration side would easily counterbalance the anticipated loss. From resource conservation point of view, this development would extend the life of existing resources by mining less for the same energy needs, and by allowing high-sulfur reserves to be included among the minable reserves. Implementation of hydrogen generation and hydrogen-based power plants can come to be a reality before the sequestration systems are in place. This is an interim solution for areas where atmospheric pollution from coal-fired power plants needs to be addressed urgently.

REFERENCES

Akihiro, H. & Frimpong S. 2000. $C0_2$ emission forecast and economic modeling for Alberta under the Kyoto Agreement on global wanning. *Proceedings of 6''' lint. Conf. on Environmental Issues and Management of Waste in Energy and Mineral Production.* Calgary Alberta, Canada, Balkema, Rotterdam, pp. 673 -681.

Bannister, R.L., Newby, R.A., Yang, W. 1997. Development of a Hydrogen Fueled Combustion Turbine Cycle for Power Generation, ASME Paper 97-GT-14.

DOE EIA, 2000a. *Annual Energy Review 1999.*. Energy Information Administration, Office of Energy Markets and End Use, USDOE Report DOE/EIA-0384(99), July 2000Washington, DC.

DOE EIA, 2000b. Emissions *of Greenhouse Gases in ike United States 1999,* Appendix B, Repon E1A/DOE-0573 (99). October 31,2000

Fink, C. E- Sudbury J.D., & Curran, G. 1977. The $C0_2$ acceptor coal gasification process, *Energy Technology Handhook,* D.M. Considine, ed., McGraw-Hill Book Company, pp 1-219-1-222.

Goff, F. & Lackner, K.S. 1998. Carbon dioxide sequestering using ultramafic rocks. *Environmental Geosciences.* 5 (3) 89-101.

Gunter, W. D., Gentzis, T., Rottenfüsser, B. A. & Richardson, R. J. H. 1997. Deep coalbed methane in Alberta, Canada: a fuel resource with the potential of zero greenhouse gas emissions. *Energy and Conservation Management,* 38 Suppl,S217-S222.

Herzog, H., Eliasson, B. & Kaarstad, O. 2000. Capturing greenhouse gases. *Scientific American.* Feb. 2000. New York.

Intergovernmental Panel on Climate Change (1PCC), 19%. Technologies, policies, and measures for mitigating climate change, *JPCC Technical Paper I:* pg 39. November 1996.

Kaarstad, O. & Audus, H. 1997. Hydrogen and electricity from decarbonised fossil fuels. *Energy and Conservation Management.* 38 Suppl, S431-S436.

Keeling, C.D., Whorf, T. P. Wahlen, M. &.. van der Plient, *İ.* 1995. Interannual extremes in the rate of rise of atmospheric carbon dioxide since !980. Nature, 375,666-670.

Knight, S., 2000. Clean coal technology: Assessment of product markets, technology market, and effects on related industries. Unpublished M.S. thesis report, HKSM, Columbia University.

Lackner, K.S., Butt, D.P., & Wendt, C.H. 1997. Magnesite disposal of carbon dioxide. *The Proceedings of the 22 nd International Technical Conference on Coal Utilization & Fuel Svstems,* March 16-19. 1997, Clearwater, Florida. U.S.A., 419-430.

Lackner, K. S.. Bun. D. P. & Wendt, C. H. 1998. The need for carbon dioxide disposal: A threat and an opportunity. *Proceedings of the 23 rd International Conference on Coal Utilization & Fuel Svstems,* Clearwater Florida, March 1998,569-582.

Lackner, K.S. Ziock, H.J., Harrison, D. 1999b. Calcium oxide assisied hydrogen production from fossil fuels & water. Los Alamos Report, LA-UR-994291, (1999).

Layne, Abbie W. 1999. Developing the Next Generation Gas Turbine Systems — A Nationwide Partnership Presented at the 1999 Power Gen International New Orleans, LA - December 2,1999, US DOE FETC

Layne, Abbie W. and Zeh, Charles M. 1999. The U.S. Department of Energy's Advanced Turbine Systems Program, 1999, US DOE, FETC.

Maddock, M.E. 1964. *Geologic map and sections of the Mount Boardman Quadrangle, Santa Clara and Stanislas counties, California.* Sacremenlo: California Division of Mines and Geology, Map Sheet 3,1 -.62,500 (color).

McCoy, D,, Curran, G., & Sudbury, J.D. 1976. CO; acceptor pilot plant , Proceedings of the 8[lK] Syntetic Pipeline gas symposium, Chicago, IL, Oct. 18-20, 1976. American Gas Association Catalogue no. L51176.

O'Connor, W.K., Dahlin, D C , Turner, P.C., & Walters, R. 1999. Carbon dioxide sequestration by ex-situ mineral carbonation. Second Dixy Lee Ray Memorial Symposium: Utilization of Fossil-Fuel Generated Carbon Dioxide in Agriculture and Industry, August 31 - September 2, 1999, Washington, D.C.

O'Connor, W.K., Dahlin, D.C, Nilsen, D.N., Walters, R.P., & Turner, P.C. 2000. Carbon dioxide sequestration by direct mineral carbonation with carbonic acid. In Proceedings of the 25 International technical Conference on Coal Utilization & Fuel systems, Coal Technology Asociation, march 6-9,2000 Clearwater, Florida.

Ranney, J. W. & Cushman, J. H. 1992. Energy from biomass. In R. Howes & A. Feinberg (eds.), *The Energy Sourcebook:* 299-311. American Institute of Physics, New York.

Ruth, L.A. 2000. DOE's Vision 21 Program- Clean energy plants for the 2 P' century. Presented at SME Annual meeting, Feb. 28 - mar. 1, Salt Lake City, UT.

Saunders, C. 2000. Three month study of mass flows, heat balance, and costs for a coal based hydrogen production and electricity generation plant with near zero emissions and a pure $C0_2$ product stream for sequestration. Unpublished M.S. thesis report, HKSM, Columbia University.

Sedjo, R. A., & Solomon, A. M. 1989. Climate and forests. In N. Rosenberg, W. E. Easterling 111, P. R. Crosson & J. Darmstater (eds.). *Greenhouse Warming: Abatement and Adaptation.* : 135-129. Proceedings of a workshop held in Washington DC, June 14-15, 1988 Resources for the Future, Washington DC.

Seifritz, W. 1990. $C0_3$ disposal by means of silicates. *Nature,* 345,486.

Siegenthaler, U. & Oeschger, H. 1987. Biospheric $C0_3$ emissions during the past 200 years reconstructed by deconvolution of ice core data. *Tetlus,* 398,140-154.

USGS, 2000. *Mineral Commodity Summaries.* USGS, US Dept. of the Interior, Washington, DC.

Watson, K., Ficara, P., Charron, M., Peacey, J., Chin, E., & Bishop, G., 2000. The Magnola demonstration plant: A valuable investment in technology development and improvement. *Magnesium Technology 2000.* Proceedings of the Symposium sponsored by the Light metals Division of The Minerals, Metals & Materials Society (TMS), Kaplan, H.I., Hryn, J.N., & Clow, B.B. (eds.) TMS Annual meeting, Nashville, TN March 12-16,2000, pp. 27-30.

Yegulalp, TM., Lackner, K.S., & Ziock, H.J., 2000. A review of emerging technologies for sustainable use of coal for power generation. *Proceedings of 6' Jint. Conf. on Environmental Issues and Management of Waste in Energy and Mineral Production.* Calgary Alberta, Canada, Balkema, Rotterdam, pp. 375 - 382.

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Modern Mining Technology and its Application within RAG Coal International AG

K.F. Jakob
Vice Chairman, RAG Coal Internationa! AG, Essen, Germany

ABSTRACT; Today's structure of the RAG group of companies represents years of change in converting from purely a coal mining company to an energy and technology corporate group. Coal mining within RAG is separately organized, in Germany with Deutsche Steinkohle AG (DSK) and internationally with RAG Coal International AG, which was founded in 1999 after a larger US acquisition. Declining production in Germany gives much more importance to the international coal trade. Access to international coal resources is important to staying successful İn business. As a new concept under RAG Coal International AG, we have organized international coal production, coal trade and sales, and the mining equipment manufacturing side, trying to use synergies to the greatest possible extent. In principle, all coal mining producers receive benefits from extremely operationally proven and market-orientated equipment availability. Recent developments have largely focused on improving longwall equipment. The trend of improvement is in principle towards:

- much more resistant and stronger AFCs,
- 2-m-wide roof support shields,
- electrohydraultc system control and integrated computerized operation.

In particular, system integration is in constant focus through Deutsche BergbauTechnik (DBT), and recent developments have shown great success in international application. The productivity records of longwalls have always been set with DBT equipment over the past decade. There are a variety of applications worldwide where DBT longwall equipment has shown its reliability, the extended material life of its individual components, and functional and operational advantages through system integration.

1 INTRODUCTION

RAG Coal International's role in the world of international coal mining, modem mining technology, and within our own company is very significant.

In this paper, firstly, a brief introduction to the corporate structure of the RAG group of companies will be given. Afterwards, the worldwide mining and trading activities of RAG Coal International will be briefly introduced. Finally, the latest developments in German mining technology and their international application will be presented.

2 STRUCTURE OF RAG GROUP OF COMPANIES

Today's structure of the RAG group of companies represents years of change. From its origin in 1969 as a single coal company tied to the Ruhr-Mining Region in the west of Germany, RAG has developed into an energy and technology group operating worldwide.

RAG Holding concentrates on the strategic leadership of its six divisions, of which two are coal-related.

Deutsche Steinkohle AG (DSK), founded in 1998, is responsible for all domestic (German) hard coal mines and coking plants. DSK is the only hard coal producer in Germany. The company employs some 60,000 people (2000).

RAG's largest unit is a portfolio of energy and technology holdings:

Our company RAG Coal International AG is within that portfolio. We are focused on overseas coal production, coal trading, coal sales and mining equipment manufacturing.

The other divisions are:

- RAG Immobilien AG, which is RAG's real estate, wholesale trading and distribution company;
- STEAG, with business activities in power generation, international IPP projects and electronic systems;

- the RUTGERS group of companies, a manufacturer of chemicals and plastics;
- the Saarberg Group, with diversified activities in the field of oil products, trading, rubber products manufacturing, environmental technology and energy.

In addition, three RAG-owned service companies provide the corporation with vocational training, information technology systems and insurance.

3 RAG'S COAL ACTIVITIES

3.1 *German coal mining*

As mentioned above, DSK. is die only hard coal producer in Germany. The company operates twelve deep mines and two coking plants. In 2000, DSK produced 34 Mt (metric) of coal.

In Germany, the production of hard coal is subsidized. These subsidies are based on an agreement between the hard coal industry, DSK, and the German Federal Government.

The agreement, reached in March 1997, defines a financial framework for the years from 1998 to 2005. It provides the German hard coal industry with a perspective to retain a substantial size and, therefore, guarantees long-term access to national coal resources. Nevertheless, the coal subsidies will be almost halved over the 1998-2005 timespan.

Only nine or ten mines will remain in operation in the long term. The adjustment process will result in the closure or merger of mines. Correspondingly, coal production will decline from 40 Mt (1999) to 26 Mt (2005). The number of employees at DSK is to be reduced, from 72,000 in 1999 to 36,000 in 2005. Last year, the adjustment process at DSK was extremely painful. The number of personnel was reduced by almost 12,000 and production has already declined by 6 mt.

3.2 *International coal mining*

Analogous to the trend of global concentration of larger and competent coal producers, RAG acquired the Cyprus Amax Coal Company in the USA in 1999. As a result of this acquisition, RAG Coal International was founded in April 1999 in order to concentrate RAG's international coal business under one company holding.

RAG Coal International AG is an international coal producer and is also involved in international coal trading, German coal sales, and the manufacturing of mining equipment. This amalgamation of different businesses under one umbrella is part of RAG's new concept.

In 2000, RAG Coal International's turnover reached 4 bn. USD. The corporation employs some 6,000 people and maintains sales, service, and mining locations in all five continents.

Our international coal mining subsidiaries operate in the US under RAG American Coal Holding, in Australia under RAG Australia Coal Pty., and in Venezuela under Carbones del Guasare.

The main emphasis of RAG Coal International's mining activities lies in the US. From 14 mines in six US states, RAG American Coal Holding expects production of around 60 Mt in 2000. With mines operating in the Powder River Basin, Colorado, the Midwest, Pennsylvania and West Virginia, RAG American Coal Holding is one of the few US domestic coal producers which are able to ship coal of varied quality in terms of heat content, ash, moisture and sulfur levels, allowing customers to meet their diverse quality needs.

Looking at annual coal production, RAG American Coal Holding ranks 5^{th} amongst the 10 big coal producers in the United States. Above it are Peabody, Arch Coal, Kennecott and Consol.

In Australia, RAG Australia Coal's operations include the German Creek, Burton and North Goonyella mines, all located in Queensland. In total, our Australian coal production will reach some 6 Mt.

In Venezuela, we own a minority interest in the Paso Diablo Mine through Carbones del Guasare; the mine produces about 7 Mt of steam and PCI coal annually.

Our trading activities are growing. Last year, we traded some 24 Mt internationally, which places us third of the major international coal traders, behind Glencore and American Metals & Coal International Inc. (AMCI). About 10 Mt of coal was imported into Germany last year in order to compensate for declining German production and to enable us to serve our customers with the coal quality they are used to.

The internationally based DBT Deutsche Bergbau-Technik engineers and manufactures underground mining equipment and is the fourth column of RAG Coal International. The company is the world leader in underground raining technology. As a specialist in longwall, transport and system control technology, DBT sells systems solutions that have been tested and proven under the extreme conditions of German mining.

DBT's manufacturing and sales centers are located in all the important underground coal mining regions and markets of the world.

DBT's longwall equipment is known for its reliability and efficient use in nearly every coal mining region İn the world. The main focus of this paper is to explain why this is a fact and how it has developed.

4 MINING TECHNOLOGY AND RECENT DEVELOPMENTS

Not only in Germany, but also in the US and other advanced coal mining nations, there have been major influences on the development of longwall equipment. For example, the US market is extremely competitive and only those underground longwall mines which produce coal at the same or lower cost than surface mines or room and pillar operations remain. The general trend has been towards the production of more and more coal from ever fewer longwalls in operation. Similar developments are reported from Australia. The international coal mining community certainly watches new developments, especially İn these countries, which are known for their high technical standards and are also referred to as trend setters with regard to coal mining.

If we take, for example, a longwall today and its different components, we find a variety of possibilities to improve individual items. However, it *is* the whole set of equipment which has to operate under sometimes very hard mining conditions and which has to display good operational and productivity performances.

In order to get a better picture of what the state of the art is and where developments are going, we should look more deeply into the individual components of a longwall, which are basically:

- the transport technology with armoured face conveyor (AFC),
- the drive technology,
- the roof support technology with electrohydraulic shield supports,
- the system of automation and electrohydraulic controls;

4.1 *Armoured face conveyor and drive technology systems*

The basic principle of the armoured face conveyor, the AFC, has remained essentially the same since its inception. The AFC not only conveys coal, but also acts as a track for the mining machine - the shearer or the plow - and serves as a reference rail for the shield supports. Modern AFCs are up to 1,342 mm wide, with an installed carrying capacity of more than 5,000 metric tons per hour. The operating voltages are up to 3,300 volts (50 Hertz) or 4,160 volts (60 Hz).

Over the years, the pan width together with the deck-plate thickness have increased significantly, and this has been the case particularly during the past 10 years. This has also been the case with the thickness of the profiles and the breaking strength of the pan connectors.

Contrary to common thought in the early 1990s, the hardest material is not the best for wear resistance. Instead, high-strength manganese-based steel shows minimum wear, especially as the production rate increases over time. This material's surface hardens as more and more coal is conveyed. As an example of maximum pan life in good conditions, some AFCs have conveyed more than 20 million tons and are still operating with the originally supplied 40-mm deck plate.

New AFC developments were initiated by coal operators who were mining in difficult conditions and conveying more rock, which is especially the case in Germany, but can also be found internationally in cases when the longwall is operated close to fault zones and/or faces with larger stone beds in the coal seam. A new development is the DBT PF 5 pan model. Compared to the older PF4 pan model, the profiles are larger and the typical deck-plate thickness has increased to 50 mm. The dog bones each have a breaking strength of more than 4,500 kN, compared to 4,000 kN for the PF4.

DBT's face conveyor systems in operation have horsepower installations of up to 3,200 kW (each drive frame is capable of 2 x 800 kW) with high AFC chain speeds of up to 1.8 m/s. The AFC system is designed for maximum carrying capacity and/or maximum face length.

An important component in maximizing the performance of high-horsepower AFCs is an intelligent drive system for soft start, load sharing, and overload protection. The Controlled Start Transmission (CST) drive system, developed especially for application with a chain conveyor, provides a user-friendly drive control unit.

4.2 *Roof support technology*

DBT's shield support is available for seam heights from 0.6 to 6 m, with setting and yield loads tailored to the operator's requirements and the geological conditions. Most of these shields today are of two-leg design. The support capacity, meanwhile, exceeds 1,0001 if required. The original 1.5-m shield width has grown to 1.75 m. The advantages are obvious. Fewer shields are required for the same face length, which reduces the total number of shield units and therefore the costs. Furthermore, longwall move times can be shortened using fewer shields. Recent developments indicate that shield width may grow up to 2 m; the application of this new type of shield support will certainly depend on individual condition allowances. A 2-m-wide prototype shield was exhibited by DBT at MinExpo in Las Vegas in October 2000.

Leg diameters today have grown up to 400 mm, and they are typically double-telescopic cylinders, maximizing the support density at a given open to closed height ratio. The large diameters improve the hydraulic flow characteristics with the leg and yield quick minimum shield lowering and setting times. The maximum operating pressure may be greater than 350 bar.

4.3 *Automation and electrohydraulic controls*

The PM 4 electrohydraulic control system moves the operator one step closer to full automation of longwall systems. The basic concept is "in-shield intelligence". Through individual power groups, there is little or no limitation in the flexibility to run the mining sequence efficiently. A central computer can be located at either the headgate or surface (or both) for maintenance data acquisition and face monitoring.

All shield features activated by the PM 4 electrohydraulic shield control system are programmable. This provides a safe environment for longwall personnel remote, from the moving shields with all longwall mining methods.

With increasing levels of automation and the need to make installations more user-friendly, DBT has moved a PC-based system with a standard Windows platform and Pentium processor that can be used both underground and on the surface.

4.4 *Mining technology development*

In summary, the recent developments of modern mining technology have been largely focused on longwall equipment. Individual components have been upgraded in terms of reliability, material life, functional aspects and system integration. The trend is in principle towards:

- much more resistant and stronger AFCs with enlarged pans and profiles,
- 2-m-wide roof support shields,
- electrohydraulic system control and integrated computerized operation.

The main reason for this development is me permanent pressure to cut costs on the operators' side. Furthermore, there is a demand for equipment which can also be used economically under deteriorated mining conditions which, as we all know, can occur anywhere and at any time.

Under RAG's group of companies, we are running the concept of the integrated development of mining equipment in Germany. Thus, there are in principle three main advantages (among others) with mining equipment of German origin:

One is that the equipment is designed to be resistant under the unique German mining conditions, which are harder than conditions anywhere else in the world. The second is that the same equipment is applied in mines all over the world, with a lot more success and more economically than other equipment due to the integrated equipment components approach of DBT. The third advantage is that adequate maintenance through DBT's after-sales service package ensures:

- maximum system availability and high longwall productivity,
- critical spare parts management to minimize system downtimes,
- significantly longer operational use (life) of the system.

5 INTERNATIONAL APPLICATION

The chart shows the development of longwall productivity world records which were achieved with DBT equipment in the past. We achieved 850,000 tons production in one month at Twentymile mine in Colorado, US. At other mines, e.g., West Elk from Arch, world records were also set.

Longwall performance at the Twentymile mine has been nothing short of amazing. During 1989, the mine began full-scale longwall production, using DBT's AFCs and supports, and a Long-Airdox shearer. What is commonly referred to as the "superwall" started operation in April 1996. The 256-m-wide longwall face is 2.6 m thick and the panel length is up to 5,400 m. The face equipment consists of 148 1.75-m-wide DBT 2-leg shields with a capacity of 840 tons. The PM 4 electrohydraulic shield control system is used for full-face automation.

The I,332-mm-wide PF 4 AFC runs a 42-mm twin inboard chain at a speed of 1.81 m/s. It is powered by three 740 kW drives with the intelligent CST drive system. The guaranteed carrying capacity is more than 4,500 tons per hour. The excellent load sharing allows Twentymile to easily exceed this carrying capacity and as much as 6,000 tons per hour have been recorded over the belt.

Twentymile has consistently set and broken production records, starting with 435,000 clean tons for the month of October in 1994 with old DBT longwall mining equipment. The current monthly record from a single longwall (850,000 clean tons) was set in June 1998. Longwall availability was 99 % in that month.

US Steel Mining ordered a completely new plow longwall face from DBT with the latest technology in 1999. The plow operates with 2 x 400 kW and AFC with a CST 30 drive system with 1,040 kW of installed power. With this set-up, the mine was able to extend the face length up to 319 m and to increase productivity shortly after starting the new longwall operation As a result of one week's full operation, a new world record for thin seams (13m) was set with 22,710 clean tons per day

In Australia, at BHP's Cnnum mine and at Mini's Oaky Creek No.l mine, new records could be achieved by using a 1,132-mm-wide PF4 AFC with CST drive from DBT (Cnnum 43,715, Oaky Creek 38,880 mtpd)

A lot of other examples can be given to show the excellent performance of DBT longwall equipment in underground coal mining all over the world - in China, Poland, the CIS, South Africa and, of course, in Germany

The key to the future enhancement ot longwall mining productivity is not only enhancement in all system components, but also an overall increased system performance DBT puts all its efforts into engineering and manufacturing the best equipment with the target of increasing productivity and giving our customers the best value

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Research and Innovations for Continuous Miner's Cutting Head, for Efficient Cutting Process of Rock/Coal

A.W.Khair
Department of Mining Engineering, West Virginia University, Morgantown, U.S A.

ABSTRACT: This paper examines the fragmentation process of rock/coal by cutting head, mainly cutting tool and cutting drum. It deals with: 1) mechanics of fragmentation from quasi-static to dynamic conditions, 2) the effect of bit geometry on fragmentatiQn process and multiple bit interaction, 3) optimization of bit geometry and cutting parameters for efficiency, 4) reduction of fine products and noise generation during fragmentation process, and 5) improvement of cutting efficiency.

1 INTRODUCTION

The impact of bit-coal/rock interaction during the cutting process in underground mines is a great concern to the mining community of the world. Rock/coal cutting bears directly on rock/coal dust generation, which causes "black lung/silicosis" in miners. Furthermore, rock cutting generates radiance of sparks causing face ignitions and loss of millions of dollars m productivity, safety and economy. These face ignitions and the consequent loss of millions of dollars in productivity and compensation for respirable rock/coal dust related diseases are attributed to the cutting action of continuous miners. *

Since 1970, the U.S. Federal Government has paid over $11.7 billion to more than 470,000 miners with coal workers* pneumoconiosis and to their survivors (Newmeyer, 1981). A world report by NIOSH on work related lung disease investigations shows that a total of 13,744 deaths (see Figure 1 for further details) occurred due to silicosis related diseases during 1968-1990 (NIOSH, 1994).

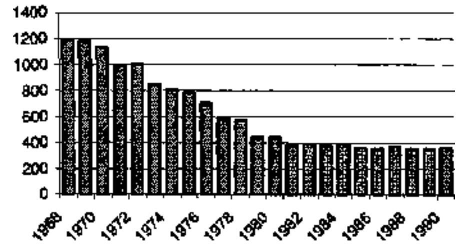


Figure 1. Number of deaths in the US due to Silicosis over 1968-1990

The high demand for coal production has increased the need for mechanized coal cutting and roof support in underground coal mines. On the other hand our coal reserve is shrinking, forcing operators to mine thin coal seams and subsequently to cut roof/floor rocks in order to mamtain sufficient clearance for equipment. At the present time approximately 65 longwall faces (Coal Age, February 1998, pp.22-27) and more than 2000 continuous miners are in operation in U.S. Enormous miles of entries are developed by these continuous miners for longwall operation as well as room and pillar mining. The amount of silica and respirable dust generated by excavating coal and cutting roofs with continuous miners is the major concern for the industry. An U.S. government printing office stressed that every year more than 250 workers in the U.S. will die from silicosis and more than 1 million U.S. workers are exposed to crystalline silica (NIOSH, 1997). Unfortunately, coal mine operations contribute significantly to these statistics.

The continuous mining machines which were introduced in the 1950's, now account for more than half the production of coal from underground mines. Unfortunately, these continuous miners, which were designed for increasing productivity, have also increased the concentration of repirable dust in the mines. Improving the fragmentation process by understanding the mechanisms of coal/rock breakage will not only reduce respirable dust at the face, but it will also decrease the amount of respirable dust that is liberated during the secondary handling such as loading and transportation etc. The fragmentation process in coal/rock is affected by the following parameters: (1) machine operating parameters (primary factor), (2) in-situ condition, and (3) physical and mechanical properties of coal/rock (secondary factors). This paper mainly deals with the major primary factors, machine operating parameters.

2 BACKGROUND

Since die introduction of continuous miners in I950's, not much change has been made in the cutting drum design for efficient excavation and reduced respirable dust generation. The problem in continuous miner head/drum is mainly associated with bit/tool and drum geometry. The bits'/tools' tips and the bodies are not designed properly, resulting in inefficient performance of machine and tools, producing high levels of noise and fine particles and generating enormous amounts of respirable dust. In a typical continuous miners drum, the bits cut face randomly and their cutting abilities mainly depend on the geometry of the bits.

In the past, enormous research has been carried out to select the design parameters for cutting tools on a trial and error basis (Organiscak, et al, I995). There are two shapes of cutting bits commonly utilized, namely, wedge type and point attack type. Although point attack type bits are used most frequently in the US, research indicates point attack bits suffer a lot of bit tip wear and damage. This is largely due to their inefficient rubbing contact with the wall of the cut groove (ridges/lands) (Reddy, 1998).

Bit wear can be defined as the removal of material from the surface as a result of mechanical action. The mechanism of bit wear can be adhesion, abrasion, oxidation, or diffusion depending on cutting conditions. A study was carried out to study the principles of bit wear and dust generation (Khair, et al, 1992). In this study four types of point attack/conical used bits were obtained from different underground coal mines. The analysis showed that many bits did not rotate properly during cutting. The intention of using conical bits in the United States coal mining was to keep bit tip sharp, it should wear symmetrically, as it rotates during the cutting process. As rock and coal debris plunge into the spacing between the bit blocks and bits, lock in of the bit into the bit block results, The same study showed that worn bits with 15% weight loss generated about 26% more dust man the new bits. Researchers at USBM (Roepke, et al, 1976) indicated that the rate and form of bit wear highly depend on bit temperature. Diffusive wear becomes the dominant form when bit temperature is higher man the critical temperature. Bit velocity is the main parameter to influence the bit temperature. The wear rate of steel, stellate and carbide tools is reported to be independent of bit velocity when the bit velocity is below a critical value of 165 to 220 ft/min. Wear was observed to increase very rapidly above the critical velocities (Roepke, et al, 1976). Since bit velocity increases the temperature of the bit, it is necessary to insure that bit velocity is below the critical value. However, low bit velocity will reduce production.

The major problems in the cutting action of the rotary cutting drum, which excavates the cutting face, are the following: (1) non-uniformity of the cutting depth for each individual bit along the cutting path, (2) generating secondary dust, which may be much more than the primary dust generation due to cutting action, (3) excavating material in a confined state/solid face without pre-cut free faces/slots. In a rotary cutting action the shape of the groove along the path of an individual bit resembles a crescent moon. Each bit on the drum starts the cutting face from zero depth of cut and as the bit penetrates further into the face, the depth of cut increases to a maximum at the center line of the path of each cutting bit, then the depth of cut decreases to zero when the bit exits the cutting face. Researchers in USBM developed a linear cutting drum (Roepke, et al, 1995). A comparison of laboratory experiment utilizing drums of the same size indicated that, when both drums have reached 75% of the maximum cut depth, the rotary drum has removed approximately 33% of the total volume, while the linear drum has taken only 15% in the shallow cutting region (Roepke, et al, 1995). However, under variable seam thickness, which requires both sumping and shearing, me difference in total dust generation may not be as significant, comparing rotary cutting and linear cutting of this particular design. One of the important aspects of this design is to modify the regrinding process of the typical rotary cutting drum. The linear cutting drum did not get out of the laboratory because of two major reasons: a) the concept was totally unfamiliar to the mining industry; b) the drum required a very high torque gear box to be practically utilized.

A laboratory study was carried out at WVU to study dust generation due to regrinding (Khair, et al, 1991). The assessments of dust generation, in this study, indicated that dust generation by regrinding depends on the size of the particles being cut during primary excavation (i.e., cut by the first line of bits). Higher dust concentration coefficients were obtained by regrinding finer particles. Increasing depth of cut creates less fine particles and reduces dust generation by regrinding. Among the parameters considered in this study the depth of sump has the most significant effect on dust generation by regrinding. Dust generation also significantly depends on hard grove grindability index. The coal with higher grindability index has higher dust concentration coefficients. Higher velocity of cutting head causes higher dust concentration. Dust concentration by regrinding Is linearly proportional to the amount of coal left for regrinding (Khair, et al, 1991). This study recommends that loading the entire coal removed/excavated in each cutting cycle will help to reduce regrinding. It has been said that "using blunt, high speed bits, {continuous mining machines) probably are the best machines for forming dust that could be invented, except for a grinding stone" (Roepke, et al, 1995). This concern has been substantiated. In a study by USBM (Roepke, et al, 1995) indicated that a continuous miner produces 70% of total dust while sumping, and only 20% while shearing. The remaining 10% is attributed to gathering and loading.

3 RESEARCH ON ROCK/COAL CUTTING

Efficient rock/coal cutting is a result of the optimum use of available resources in a continuous mining system. Research has demonstrated that specific energy and specific respirable dust must be kept at minimum to produce the optimum parameters of the rock/coal breakage process.

Mechanisms of Rock/Coal Fragmentation;

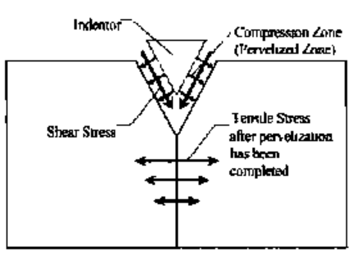


Figure 2 Shows stress develop during indentation

The fracturing process is governed by quasi-static and dynamic forces. In a recent study by Khair, et al. (2000a) (2000b) the fracturing process in Tennessee sandstone specimen, subjected to a wedge indentor is exhibited holographically. From the failure process point of view, it's obvious that initial wedge-rock interaction results in high stress concentration (compressive and shear, Figure 2) at the area of the contact zone, causing micro-failure of material in the vicinity of the contact zone (between wedge and rock, Figure 3). As the stress exceeds the strength of the material, it results in pulverization of the interface zone and stress redistribution in the specimen (see Figure 4). As the loading continues, the wedge penetrates further and pulverizes the contact zone (see Figures 5-6). The process continues and is reflected in Figure 7 until the extension of the pulverization zone stops where sufficient tensile stress (splitting stress) develops to initiate failure, (see Figures 8-9), Figure 10 shows the results of different experiments, using special holography. Displacements at the beginning of loading process are very high and diminish as the specimen reaches failure. This reduction of displacements results in pulverization of the interface and stress distnbution occurs at the area of contacts between wedges and rocks At that instance tensile stress sufficiently develops and indicates that lateral displacement perpendicular to the direction of wedge face develops very high prior to the specimen failure for three wedge angles. Failure of the specimen occurs (see Figure 10). The thickness of the pulverization zone in the wedge-rock interface mainly depends on two factors. (!) material characteristics such as brittle, ductile behavior of material in particular cracks, discontinuities, porosity and flaws existing in the material, which are more susceptible to become pulverized and allow wedge to penetrate deeper into the material. To extrapolate this fact further, ductile/soft material requires deeper wedge penetration prior to splitting/fragmentation than brittle/harder material; (2) wedge angle, the higher the wedge angle less wedge penetration and less crashed material produced. However, higher wedge angle subjects more wear (see Figure I la-b)

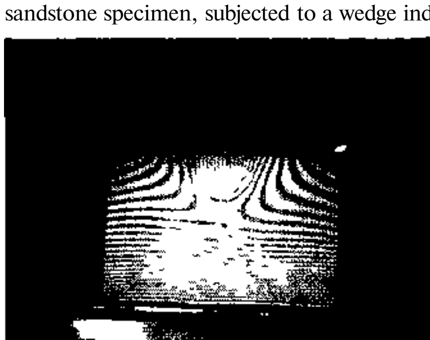


Figure 3 Shows high displacements in the area of contact wedge-rock due to high stress concentration, perpendicular to the specimen face.

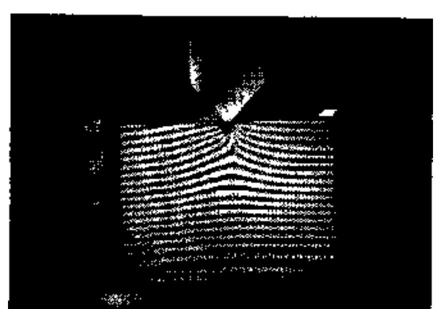


Figure 4 Shows crushed and pulverized contact zone between wedge-rock

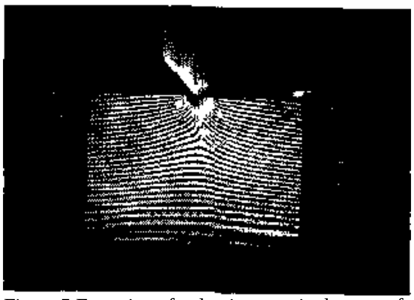


Figure 5 Extension of pulverize zone in the area of contact zone between wedge and rock

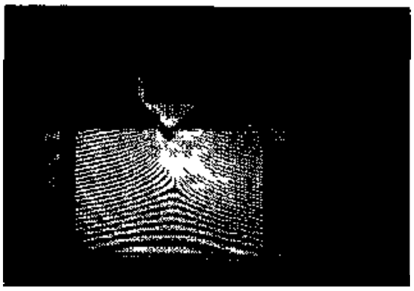


Figure 6 Shows continuous process of wedge penetration and displacements/pulverization development in the rock

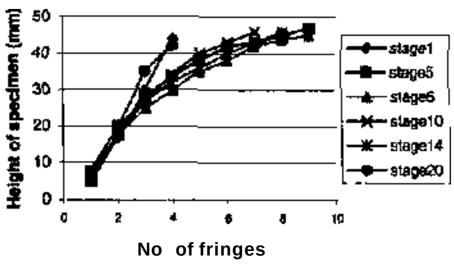


Figure 7 Displacements using 90° indentor in unconfined condition, indicates variation of displacement curve due to stress redistribution

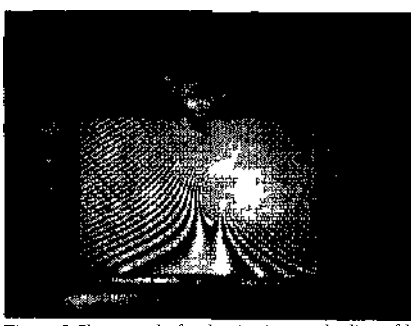


Figure 8 Shows end of pulverization on the line of loading action and development of tensile stresses

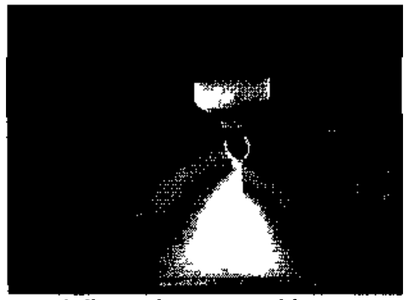


Figure 9 Shows pulverize zone and fracture extension in the specimen

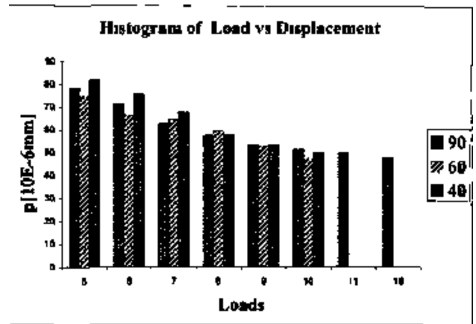
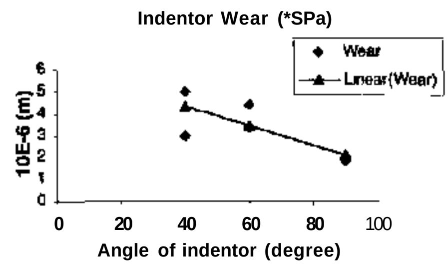
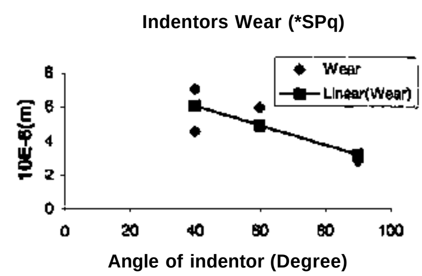


Figure 10 Histogram of Load and Displacements for three indentor angles in lateral direction



'Unfiltered arithmetic mean of the departures of the surface from the mean value of the fractured surface

Figure 11 a Correlation between angle of indentor and bit wear



Unfiltered RMS parameter corresponding to Spa

Figure 11 b Correlation between angle of indentor and SPq for indentors

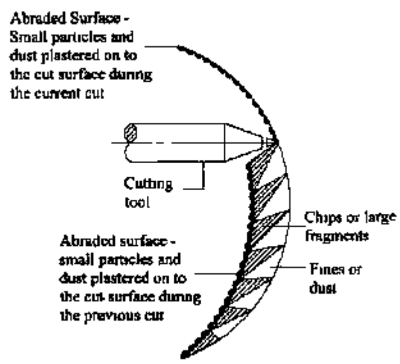


Figure 12 Simplified diagram of crushing and chip formation when cutting coal with continuous miner

Fracture process in coal/rock by rotary cutting showed that the dynamic forces causes fracture formation and fracture extension, while the quasi-static forces are responsible for grading the fracture surface. Observations during the tests indicated that after the cutting head induces certain fracture (different intensities, magnitudes, and lengths), its rotational velocity slows down. Analyzing the cutting action, when the bit enters the coal it indents and compresses the coal under it and shears off the fragments. This process yields coal fragments, coarse and fine, and dusts particles (see Figure 11V The key to analyzing and understanding the source of dust generation is to identify the different phases of the cutting process and to correlate them with the particles. Previous investigators have observed that during the indentation of a cutting bit into a brittle material, two phases occur (Paul, et al. 1965) (Miller, et al. 1968). These are called the crushing phase and chipping phase. This was later extended to linear cutting by observing the cutting action under very low speed (Warner, 1970). In this study it was theorized that when the bit first contact inelastic subsurface cracking takes place. Such an action leads to crushing due to the coalescence of the cracks. Crushing and subsurface cracking will produce fine fragments and dust. It was observed that the cutting and thrust forces build up and increase during the crushing/pulverizing phase of the material and drop when a major crack is generated, resulting in chip formation.

Cutting Parameters:

The important parameters of rock/coal cutting are 1) attack angles, 2) bit geometry, 3) depdi of cut, 4) bit spacing, 5) water-jet assisted pressure.

I) Bit Attack Angle:

Research has been carried out by Khair, et al (1989) utilizing four different attack angles, 15°, 30°, 45°, and 60°, were used in this study (Fig. 12). The most ideal condition for force transmission by the bit to the coal will be a 30°-45° attack angle, where the area of contact between the bit and coal is at a minimum and causes a high stress concentration in the coal block. As a result, less force is required to break the coal. The smaller attack angle will not only require a larger normal force (thrust) to penetrate the coal, consequently resulting in more friction heat, especially during the quasi-static loading condition (grading). The 60° attack angle is similar to the 15°; however, in this case the front portion of the bit will have a larger surface area of contact with the coal. Its influence on fragmentation will be during the dynamic loading cycle and the larger area of contact will make the bits behave as if they were blunt.

Figure 13. Shows attack angle in the experiment

2) Bit Geometry:

Conical bits are commonly used in the U.S. mining industry and discussions are focused on this type of bit. In regards to bit geometry, there are two elements associated with bit geometry, 1) bit tip and 2) bit body. Research has been carried out by many investigators to characterize bit geometry i.e. bit tip angle, and size, bit body geometry and stream linear of bit tip and body for efficient fragmentation, durability, ignition, respirable dust generation, specific energy consumption, noise generation, breakout angle and multi bit interaction (Roepke, et al. 1976, Khair, et al. 2000a, Khair, et al. 2000b, Roepke, et al. 1983, Khair, et al. 1989, Srikanth, 2000, Khair, 1996, Khair, 2001).

3) Depth of Cut:

Past research indicated that as the depth of cut increases the specific respirable dust is reduced. Deeper cutting enable interaction between adjacent cuts and help produce larger chips of material. Roepke and Hanson (1983) found that the average cutting force increases with increase in depth of cut while the specific respirable dust and specific energy decreases with depth of cut.

It is known that a proper depth of cut to bit spacing ratio reduces specific dust generation. This ratio depends on machine cutting parameters and physical and mechanical properties of rock (Achanti, 1998). Research work was conducted by a number of people on rotary cutting bits. Research at USBM (Roepke, et ai. 1976) demonstrated that the specific energy and airborne dust (ARD) decrease significantly as the cutting depth increases and the optimum tool spacing to cutting depdi ratio ranges from 2 to 3. Further study by the USBM researchers (Roepke, et al. 1983) concluded that different bits do not affect the ARD as significantly as cutting depth or specific energy, but various bits have different forces and energy requirements necessary to maintain a prescribed cutting depth.

Barker (1964) and Pomeroy and Brown (1968) reported that optimum spacing depends on the depth of cut. For a cut spacing at which neighboring grooves interact, the cutting forces decrease after reaching a maximum. The maximum normally corresponds to the condition of high product volume, low specific energy and low dust generation. Research also indicated that specific energy decreases with depth and spacing (Srikanth 2000, Khair, 2000).

Many studies have addressed the influences on respirable dust generation during coal cutting process. Research was conducted at WVU (Reddy, 1998) utilizing a series of single and multiple bit experiments on coal using a laboratory scale cutting machine in order to investigate the sources of respirable dust generation both at macro and micro levels, Khair, et al (1989) documented several issues in rock cutting process that need to be addressed and the concern for respirable dust in the report submitted to USBM. The Bureau of Mines conducted a series of experiments using four different coal types to determine the effect of attack angle and asymmetric bit wear on airborne respirable dust (ARD) generated by point attack bits and in energy consumption (Roepke, et al. 1983). They established that the depth of cut had significant effect on the respirable dust and specific energy

Research conducted at WVU indicates that specific respirable dust increased with increasing bit spacing in rotary cutting. As the bit spacing increases the grooves made by the bits do not interact and hence the ridges do not break. Instead of the formation of major chips, regrinding occurs in die grooves producing significant amount of fine dust. As the cutting depth increases the amount of respirable dust generated reduced as deeper cuts enable me interaction of adjacent cuts and help in production of major chips (Achanti, 1998). A series of preliminary laboratory experiments were carried out at the Department of Mining Engineering at WVU (Khair, et al. 1989). Figure 14, shows experimental set up. Figure 15, shows test coal block, Figure 16a-c shows tested coal blocks in different cleat directions and bit spacing. In this study a series of experiments were run with a 7.62cm (3in) spacing. Three bits were mounted in an echelon pattern and a 6.35-7.62cm (2.5-3 in) deep cut was made without breaking the boundary walls/ledges between the bit paths (see Figure 16a). This series of tests were carried out for the 15°, 30°, 45°, and 60' attack angles. With 3.80cm (1.5in) spacing and five bits mounted m an echelon pattern, the side walls/ridges of the bit path were broken when the coal blocks were tested İn both face and but cleat direction (see Figures 16b and 16c). In these experiments the fracture surface was more regular when tested against butt cleat, walls/ridges between the cutting paths broke only partially and irregularly (see figure 16c). A total breakage of the walls/ridges created a free face (see figure \6b), thus reducing the required resultant forces to cut the coal (Khair, et al. 1989). The concept of relationship between depth of cut and bit spacing in order to remove lands/ridges between the cutting paths is illustrated by Figure 17. As it was indicated earlier that depth of cut not only reduces primary and secondary dust generation, but also reduces required specific energy, depending on bit geometry. A series of preliminary experiments were carried out by Khair (1996) Figure 18 shows typical expenmental setup and Figure 19 shows tested rocks utilizing bits of different geometry. Among the tested bits, US2 performed very well. This high performance of the US2 type bit was due to two geometric parameters, namely high clearance angle and prism shape of the cutting face of the bit. which further reduced the surface contact area of the bit during the cutting process. These two factors reduced the specific energy consumption for the bit, in particular, under deeper cutting condition (i.e., at 3mm depth of cut, specific energy consumed by the bit is 18.4 MJ/m^3, and at an 18mm depth of cut, the consumed specific energy was reduced to 24 I MJ/m^3 with a corresponding mean nominal force to mean cutting force ratio of 0.91 and 0.53, respectively). Results also indicate that specific energy consumed by the bit decreases with depth of cut. The damaged surfaces of the rock corresponding to different depth of cut are present in Figures 20 and 21. In deeper cutting most of the energy was consumed in the fragmentation process rather than grinding material, hence resulting in a larger product size and fewer fine particles (see Figure 22-24)

Figure 14 Expenmental setup earned out in 1989.

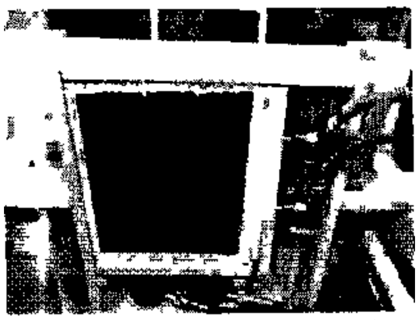


Figure 15 Typica! Specimen located in the confining chamber and ready for experiment

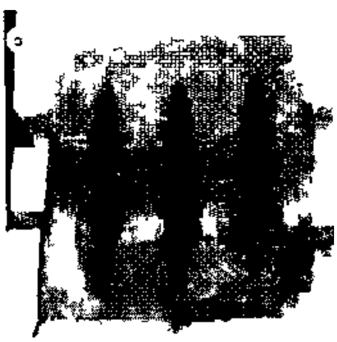


Figure Ì6a Coal blocks cut with 3 in bit spacing ta^e cleat

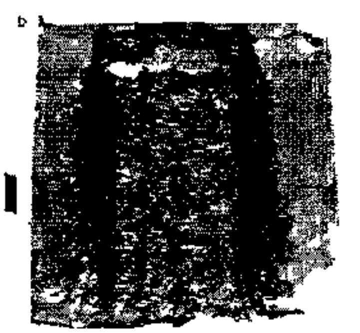


Figure 16b Coal blocks cut with 1 5 in bit spacing face cleat

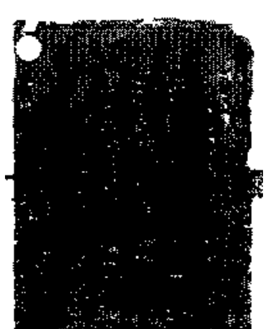


Figure 16c Coal blocks cut with I 5 in bit spacing but cleat

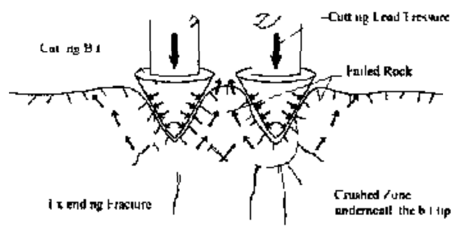


Figure 17 Relationship between depth of cul and bit spacing

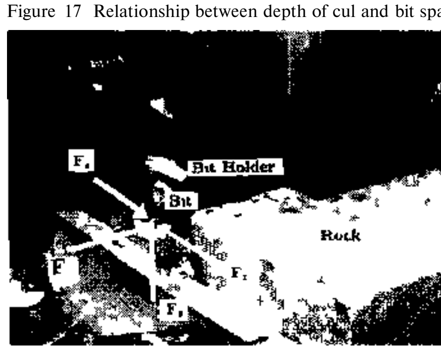
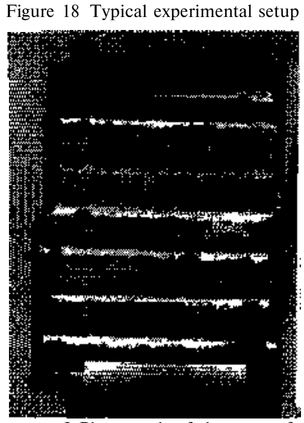


Figure 18 Typical experimental setup



I Photograph of the cut surface tested Godula sandstone block

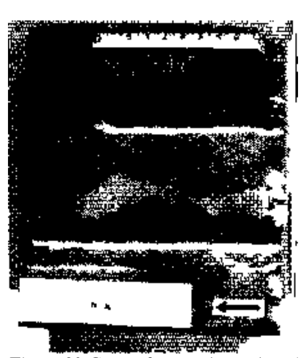
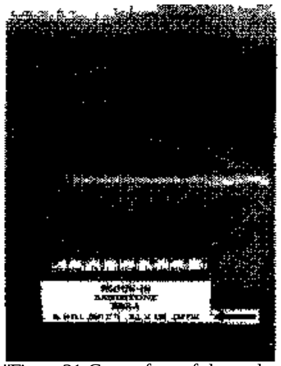


Figure 20 Cut surface ot the rock at 12 and 9 mm depth of cut utilizing US2 bit



"Figure 21 Cut surface of the rock at 18mm depth of cut, utilizing US2 bit

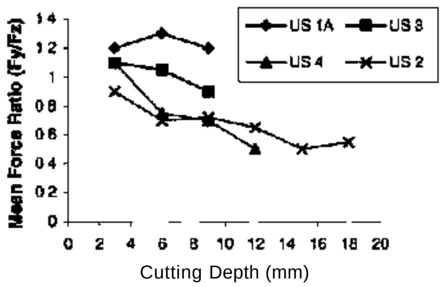


Figure 22 Variation of mean normal force/mean cutting force with increasing depth of cut

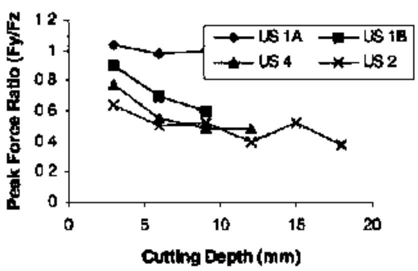


Figure 23 Variation of mean peak norma! force/mean peak cutting force with increasing depth of cut

4) Bit Spacing

Linear cutting has been earned out in order to study the influence of bit spacing on the energy consumption and amount of noise produced during cutting (Khair, 2001) The results indicate that the noise levels usually mcrease with an increase in the cut spacing to depth of cut ratio for individual bits There is an increase in the noise level and energy consumption of 4 5% and 24 1% respectively when the bit spacing to cut depth ratio was increased from 1 to 2 Optimum bit spacing reduces confinement and provides free space, which results m less energy consumption and reduces dust generation Furthermore the linear cutting indicated that the noise levels mcrease as the bit tip size increases The noise levels increased by about 5 dB with a larger body and tip size Therefore by an optimum bit spacing to depth of cut ratio and optimum bit tip size a reduction up to 10 dB noise resulted during the cutting process It should be emphasized that the bit tip size is not the only parameter that effects bit performance, but also the geometry of bit tip and bit body, a stream-lined shape is important Figure 25 shows the types of bits used Figure 26 shows the linear cutting experimental setup, and Figure 27 shows the cutting process (Khair, 2001) Experiments conducted at WVU, utilizing rotary cutting machine, indicates energy consumption and specific respirable dust reduces as the bit spacing to depth of cut ratio decreases from 2 to 0 3 (Snkanth, 2000) m another expenment the amount of respirable dust produced increases as the bit tip angle increases from 60° to 75° and it reduces from 75° to 90" (Achanti, 1998)

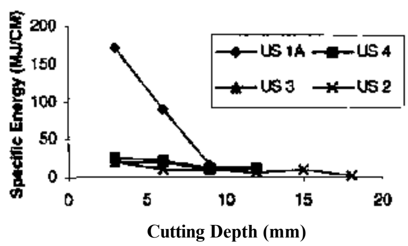


Figure 24 Variation of specific energy with depth of cut

Figure 25 Shows the types of bits used in experiments

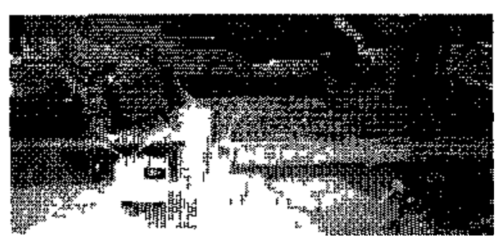


Figure 26 Shows the linear cutting experimental selup

Figure 27 Shows the cutting process

5) Water Jet Assist Cutting:

The external water spray system is the typical technology associated with continuous miners in underground coal/rock cutting process. In this technique a canteen of water is sprayed above the cutting head in order to suppress the respirable dust. However, the internal water spray jet system involves a water-jet directed towards the rock bit interaction point. This is known as "wet drum" and is similar to a shear drum. This results in keeping the bit cool, suppressing dust, reducing cutability index of the rock and increasing deep cutting performance ofthe cutting drum which results in reduced specific dust generation in underground and inhibit sparks to retard ignition. Studies carried out at WVU (Khair, et al. 1998a, Khair, et al. 1998b) in regards to cutting rock under three experimental set-ups, a) dry cut, b) external water spray cut (see Figure 28), and c) internal water spray jet cut (see Figure 29). The results were astonishing in terms of bit wear, machine penetration/cutting force, depth of cut and respirable dust generation. A high content of quartz with quartz cement found in Tennessee sandstone caused excessive removal of material from the bit body, in dry and external water spay cutting, and damaged tip of bit, while no substantial damage was observed when water spay/jet method (see Figure 30a, 30b, and 30c). Figure 30a shows wear of the bits with respect to the new one and Figure 30b and 30c show the variation of cumulative weight and height loss of the bits respectively in the above experiments. Even though, the dust was not measured in a wet set-up since the wear on the bit is due to the abrasion and impact of die bit on rock, therefore, their action was minimum, water spray/jet method application and it certainly has changed the mechanical property of rock in comparison to the dry and external spray system. The results of these studies (Khair, et al. 1998a, Khair, et al.!998b) showed that wear rate on the cutting bit is a controllable factor, and the potential for other problems such as fractional ignition and respirable dust could be reduced. In a communication with a JOY Mining Machinery Engineer it was indicated that their wet drum continuous miner in a Utah mine, reduces respirable dust up to 80%, dust samples reduced from 2.0 mg/cu.m to 0.4 mg/cu.m with water flow rate of 26.5 g/m.

Figure 28 and Figure 29 Shows external water spray cut and internal water spray jet cut respectively

Figure 30a. Shows wear of the bits with respect lo the new one.

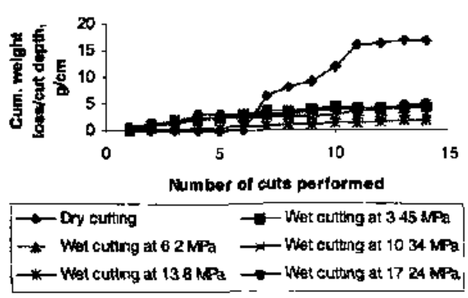


Figure 30b. Cumulative weight losses in a series of 14 cuts performed on Tennessee sandstone

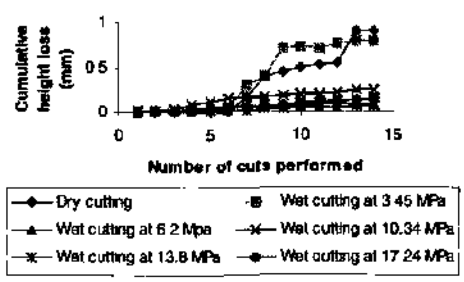


Figure 30c Cumulative height losses in a series of 14 cuts performed on Tennessee sandstone

4 IMPROVEMENT

In the past, technological innovation in mine equipment and machinery, resulting from research, had little chance of implementation in field application. However, this trend has changed in the last few years, because of safety concerns and high productivity demands. Continuous miners of today are more highly advanced in hydraulic, electrical, electronic, and mechanical technology than the ones built a decade ago. Unfortunately the cutting head of continuous miners have received the least attention to improve coal/rock fragmentation, reduce dust and increase efficiency of the machine. However, some important research in the area of cutting drum has been carried out which will enhance the performance of the cutting drum. There are many types of cutting tools with different bit geometry shape, size, and tool tip available to cut materials of different strength and abrasivity. Polycrystalline diamond compact (PDC) bits could be used for hard and abrasive materials, however, the use of such a bit in the field is restricted by cost. Unfortunately, the geometries of these cutting tools are not optimized to reduce respirable dust and specific energy. The important elements for cutting tools are deep penetration with least wear and energy consumption. In a research work by Khair (1996), recommendations in regards to optimum bit geometry were presented. Following these recommendations, a series of new cutting tools were developed by two major tool-manufacturing companies. Research on optimization of cutting tool, for cutting different geological materials, is underway by the author.

During sumping process, where most of the regrinding takes place, scrolls, similar to the longwall shearer machine, will help to transport material from the sump to the gathering arms. The scrolls on the cutting head of some continuous miners, used for trona mine are implemented and the results are highly favorable. The productivity of the continuous miner has increased significantly by increasing depth of cut. However, if the depth of cut to bit spacing is not optimized it results in excessive amount of respirable dust generation and high energy consumption. If the ridges between the bits were not broken/fragmented during bit penetration there is crashing of these lands/ridges by the bit blocks. The use of water jet assisted cutting has been implemented in a number of continuous miners in relatively dusty coal mines. Of course water jet assisted mining not only suppresses dust generation, it also helps retard ignition, facilitates rotation of bit İn bit block, and increases efficiency of the cutting tool and cutting head. Perhaps the most inefficient cutting of continuous miner drum is lack of free face. The geometry of the drum is not modified to cut material toward free the face. Research in this area is underway by the author.

5 CONCLUSIONS

Efficient utilization of continuous miner cutting head requires optimization of cutting tool, and drum geometry, understanding of fracture mechanisms associated with cutting material and constructing cutting tools ideal for the type of material to be cut. It İs essential to design tools for reduction of dust generation and cutting efficiency. Implementation of scrolls, internal water spray system, with deep cutting and optimum depth of cut to bit spacing, certainly increase productivity, efficiency, reduce respirable dust generation, retards ignition and increase the useful life of the cutting tools.

REFERENCES

Achanti.V.B., 1998, Parametric Studyof Dust Generation with Ridge Breakage Analysis Using a Simulated Continuous Miner, Ph.D. dissertation, Department of Mining Engineering, WVU, Morgantown, WV. pp.83.

Barker, J. S., 1964, "A laboratory Investigation of Rock Cutting Using Large Picks," International Journal of Rock Mechanics and Mining Sciences, Vol. I, pp.519-534.

Coal Age, U.S. Longwall Census, February 1998, pp. 22-27.

Khair. A. W., "The effect of bit geometry and size on cutting parameters in linear cutting. 2001, SME-AIME Annual Meeting and Exhibition, Denver, CO, February 26-28, 2001, preprint Number 01 -86, 12pp.

Khair, A. W., "The effects of bit geometry on rock cutting efficiency". Applied Occupational and Environmental Hygiene, Vol. 11, Number 7. July 1996, pp. 695-700.

Khair, A. W., "The effects of cutting tool design on respirable dust generation", Proceedings of International Scientific and Technical Conference on Respirable Dust Hazard Control in the World Mining Industry, Szczyrk, Poland, Sept.17-19,1996, pp. 79-87.

Khair, A. W., and Achanti, V. B., "Laboratory Investigation of Bit Wear in Water Jet Assisted Cutting of a Continuous Miner", Proceedings of 3rd North American Rock Mechanics Symposium (NARM '98), Cancun, Mexico, Jan 3-5,1998b.

Khair. A. W., and Achanti, V. B., "Experimental Investigation of Rock Fragmentation Efficiency of a Simulated Wet-Drum Continuous Miner", Proceedings of Pacific Rim International Conference on Water Jet Technology, New Delhi, India, Feb. 3-5. 1998a, pp. 460-*66.

Khair, A. W., Addala, S., and Pezowicz, C, "Effect of Cutting Tool Angle on Failure Mechanism of Rock in Rock-Tool Interaction", Proceedings of the Ninth International Symposium on Mine Planning and Equipment Selection, Athens, Greece, November 6-9,2000b, pp. 601 -606.

Khair, A. W., Addala, S., Pezowicz, C, and Bernakiewicz, M., "Fracture Process in Rock by Wedge Indentors utilizing Holographic Interferometry, Process of the Fourth North American Rock Mechanics symposium NARMS 2000, Seattle, W A, USA, July 31-August 3.2000a, 6pp.

Khair, A. W.. Reddy, N. P., and Quinn, M. K., "Mechanisms of Coal Fragmentation by a Continuous Miner", Journal of Mining Science and Technology, March 1989, pp. 189-214.

Newmeyer, G.E., "Cost of black lung program". Mİn. Congress Journal, 67(11) 1981, pp. 74-75.
Khair, A. W., Xu, D., and Ahmad, M., "Principles of Bit Wear and Dust Generation", Proceedings of New Technology in Mine Health and Safety. SME, Littleton, CO, February 1992, pp.175-183.
Rhair, A. W., Xu. D., "Laboratory approach to study dust generation due to regrinding", Proceedings of 5th U.S. Mine Ventilation Symposium, SME Publisher, Littleton, CO. 1991, West Virginia University, Morgantown, WV, June 3-S, 1991,pp.215-224.
Miller, M. H., and sikarskie, D. C, "On the penetration of rock by three-dimensional indentors", Int. J. Rock Mech. Mİn. Sei., 1968, Vol 5, pp.375-398.
Organiscak, J.A., Khair, A. W., and Ahmad, M., Studies of Bit Wear and Respirable Dust Generation, SME-AIME Annual meeting and exhibit, Denver, CO, March 6-9. 1995, Preprini No. 95-214. Also in SME Mining Transactions/Annual Bound, Vol. 298, 1995, pp. 1874-1879.
Pomery, C. D., and Brown. J- H., 1968, "Laboratory Investigation of Cutting Process Applied to Coal Winning Machines", Journal of Strain Analysis, Vol. 3, No. 3, pp. 232-243.
Paul, B., and Sikarskie, D. L„ "A preliminary theory on static penetration by a ridge wedge into brittle rock material", Transaction of SME-AIME, Dec. 1965, pp.372-383.
Reddy, N. P., 1988, "Characterization of Coal Breakage by Continuous Miners", Ph.D. dissertation, Department of Mining Engineering, WVU, Morgantown, WV, 184pp.
Roepke, W. W., et a!., (1995), Improved Performance of Linear Coal Cutting Compared with Rotary Cutting, USBM, Rl 9536, pp.34. .
Roepke, W. W., and Hanson, B. D., (1983). "Effect of Asymmetric Wear of Point Attack Bits on Coal Cutting Parameters and Primary Dust Generation", USBM, RI 8761, 16pp.
Roepke. W. W., Lindroth, D. P., and Myren, T. A., (1976), "Reduction of dust and Energy During Coal Cutting Using Point Attack Bits With Rotary Drum tested and produced the same or less, -.32cm (-1/8 in) product.
Srikanth Addala, 2000, Relationship between Cutting Parameters and Bit Geometry in Rotary Cutting, Master Thesis in Progress, Department of Mining Engineering, WVU, Morgantown, WV.
U.S. Department of Health and Human Services, Division of Respiratory Disease Studies, National Institute of Occupational Safety and Health, Work related lung disease surveillance report. August 1994.
U.S. Department of Labor, National Institute of Occupational Safety and Health, A Guide to working safety with Silica, 1997, pp.551-664.
Warner, E. M., 1970. Machine and cutting element design, Mine Conference J. August, pp35-43.

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Thin Spray-on Liners for Underground Rock Support

D.D.Tannant
Department of Civil & Environmental Engineering, University of Alberta, Edmonton, Canada

ABSTRACT: Rapid setting, thin, spray-on polymeric liner materials for underground rock support are being tested in Canada. Thin polymer liners have performance characteristics that lie between those of shotcrete and mesh. They are a welcome addition to the 'tool box' of support types and have a role to play where rapid application rates and areal support of rock are needed. A continuous liner that is firmly adhered to the rock creates effective rock support. Various approaches are used to examine the load capacity of a liner. Interpretation of the available test data and introduction of simple support models show that the two likely failure modes are adhesion loss and tensile qr shear rupture of the membrane. Different failure modes occur depending on the relative tensile and adhesive strengths of the liner and the anticipated magnitude of the rock displacements.

1 INTRODUCTION

Thin spray-on liners are a new form of rock support that is receiving increasing attention by various mines around the world. Various liner materials are currently being developed and tested in Canadian mines. They can all be generically classified as multi-component polymeric materials. Thin polymer liners have applications in hard-rock mines as a replacement for either wire mesh or shotcrete. They function well as the areal support component in a support system that also incorporates rock bolts.

This paper gives a historical overview of the development and testing *of* spray-on liner materials and discusses various mechanisms by which thin liners function to support and stabilize underground excavations in rock. Important liner properties include the tensile strength and the adhesion to the rock, which control the liner's load capacity, and the ultimate strain, which controls the liner's displacement capacity.

Various factors to consider when using thin spray-on liners, including advantages and disadvantages when compared to welded-wire mesh or shotcrete are reviewed. Simple models that illustrate various support functions and the important design parameters for thin liners operating under various conditions are presented. Finally, a review is presented of current thin liner use in Canadian mines, outlining successes and future challenges.

2 DEVELOPMENT OF THIN LINER SUPPORT

The installation of conventional rock bolts and wire mesh is labour intensive and time consuming. In addition, underground personnel are frequently injured while installing rock support. While the installation of rock bolts or other tendon support elements can be mechanized, mesh installation still requires manual labour. One method for overcoming some shortcomings of mesh is the use of shotcrete, in particular, steel fibre reinforced shotcrete. While the use of shotcrete rapidly gained acceptance within many Canadian mines in the 1990's there are still problems associated with the logistics of trans porting large quantities of shotcrete *materials to* active headings far underground. In addition, it was noticed that deeper drifts in many Canadian mines underwent substantial deformations. These deformations exceeded the displacement capacity of the shotcrete rendering the shotcrete itself a hazard.

As an alternative to rock bolts and mesh or shotcrete, MIROC (Mining Industry Research Organization of Canada) began an investigation of rapid setting, thin, spray-on liner materials for ground support. The first tests on thin spray-on liner rock support technology were initiated in Canada in the late 1980's (Archibald et al. I992). Initial research lead to the development of a polyurethane based product call Mineguard™. Modifications to the chemical formulation of Mineguard and extensive laboratory testing continued throughout the I990's (Archibald et al. 1997, Archibald & Lausch 1999).

In the 1990's, Canada's largest nickel mining company, INCO Ltd., embarked on a strategy to move toward robotic mining methods. The use of a thin spray-on liner for underground rock support offered INCO numerous advantages in terms of speed of application and minimizing transportation of materials. INCO thus became a key advocate of thin liner research and testing and sponsored numerous laboratory and field tests using Mineguard (Figure 1) throughout the 1990's (Espley et al. 1995, 1996, Espley-Boudreau 1999, Tannant 1997, Tannant et al. 1999). In the late 1990's, these tests also included a new product based on hybrid polyurethane/polyurea mixture called Rockguard.

Figure 1 Manual applrcalion of a thin polyurethane (Mineguard) liner to rock.

Meanwhile in 1996, researchers from South Africa began exploring the use of another thin liner product that was latex-based. This product was known as Everbond (Wojno & Kuijpers 1997) and has since evolved into another product call Evermine. Researchers in Australia have also been exploring the use of thin liners for rock support and have conducted field tests in Western Australia.

By the mid to late 1990's news of spray-on liners being used m Canada's hard-rock underground mines reached many other interested manufacturers and vendors of a wide variety of spray-on products. Falconbridge Ltd. (anoüıer large Canadian mining company) began its own research effort to find appropriate liner materials While many products were tested, it was found that most did not possess adequate physical or chemical properties. One product call TekFlex (a water-based, polymer modified cementitous material) was found to show promise and field trials of this material were initiated in cut and fill stopes (Pritchard et al. 1999, 2001).

A variety of new products are in the development and testing stages. These include a polyurea-based product called RockWeb and a methacrylate-based product called Masterseal 840R01 or Superskin (Spearing & Champa 2000) As of 2000, there are about six different manufacturers of spray-on materials for thin liners that are competing in the market for underground rock support in Canada.

There are currently about 55 mines around the world that are considering the use of thin liners for rock support. The greatest interest is in North America, Australia, and South Africa. Given that thin liner technology is still in its infancy, it is likely that other products will come forward for testing and evaluation. Good liner materials adhere tenaciously to the rock surface, cure quickly, and have high tensile strength. This paper does not focus on comparing one product versus another. Refer to the publications listed in the references for properties of specific liner materials. Instead, the remainder of the paper examines design issues related to thin liner support technologies.

3 ROCK SUPPORT PROVIDED BY THIN LINERS

A principal objective of support is to assist the rock mass in supporting itself. It is difficult for a support system to hold up the dead weight of rock once the rock mass has loosened (Hoek & Brown 1980). This İs particularly true when usmg thin liners, because they have a limited load capacity. In jointed or fractured rock masses, a thin liner prevents the rock mass from dilating, loosening and unraveling, thus forcing fragments of the rock mass to interact with each other creating a stable beam or arch of rock. To be effective at helping establish a stable zone of rock, a Imer must be able to limit the kinematic movement of individual rock blocks. If conditions allow the rock mass to loosen excessively, then the liner's function can switch to retaining the loose rock in place between rock bolts.

Conventional support in the hard-rock mining industry makes use of rock bolts or other tendon support to hold large key-blocks in place while wire mesh is used to retain the small rock pieces between the tendons. In some cases, shotcrete is used in a dual role for supporting both larger key-blocks as well as smaller pieces of loose rock.

Most support design focuses on the load capacity of the support. It is equally important to consider the support's displacement capacity, especially in situations where large ground convergence and significant relative displacements or shear displacements between adjacent rock blocks are expected. Only through knowledge of the displacement capacity of various support types can proper design and selection of support be made for a given application.

3.1 *Displacement capacities ofareal support*

Shotcrete, polymer liners, and steel mesh mobilize support resistance at different displacements. Materials that are sprayed onto the rock such as shotcrete or liners are able to generate support resistance at small rock deformations (in the order of millimetres). Mesh it a truly passive support and requires substantial displacement (in the order of 100's of millimeters) before it offers a support resistance (Tannant 1995, Tannant et al. 1997). Mesh is effective at catching and hold small falls of rock, but it provides minimal resistance to the initiation of the rockfall itself. Sprayed materials operate differently because they are able to offer support resistance at small displacements Therefore, they can prevent rockfalls from happening in the first place.

Shotcrete, especially reinforced shotcrete, can generate much higher support resistance than thin polymer liners. However, in situations where large ground convergence occurs, the more flexible thin liners may provide superior support over the full range of rock deformations. For example, ınsitu pull tests using a 250mm diameter plate pulled through 70 to 100mm thick steel fibre-reinforced shotcrete showed that the shotcrete could only sustain 5 to 10mm of relative displacement before the shotcrete ruptured and failed (O'Donnell & Tannant 1998).

Laboratory pull tests on 1.5m square panels made from concrete blocks coated with 50 to 60mm of steel-fibre reinforced shotcrete also showed a limited displacement capacity (Tannant & Kaiser 1997, Kaiser & Tannant 1997). The shotcrete panels attained peak strength and fractured after relative displacements of 5 to 10mm. In comparison, concrete blocks coated by a polyurethane membrane tested in similar conditions did not reach peak load until 40 to 50mm of displacement and the load was maintained for up to 100mm of displacement (Tannant 1997).

Figure 2 shows schematically the different load-displacement performance for various areal support types. Liners are expected to have performance characteristics that lie between mesh and shotcrete.

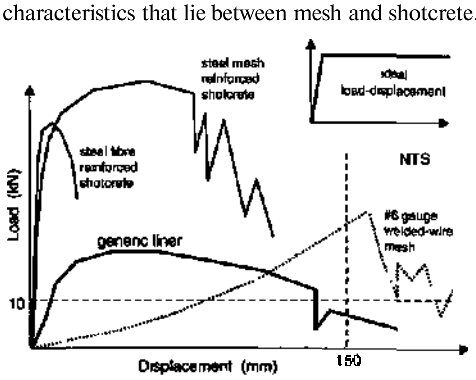


Figure 2 Load versus displacement capacities from pull tests on various area) support types.

3.2 *Support of small loose rocks - Glue action*

A thin liner may simply 'glue' or bond loose pieces of rock to adjacent competent rock No other conventional support type is designed to act like glue between adjacent rocks One exception may be rock masses that are grouted before excavation occurs.

The mass of loose rock that can be safely held m place depends on the liner adhesion to the rock and the polymer's tensile or shear strength Another parameter, which is difficult to determine, İs the adhesive bond width. The effective bond width dictates the area over which the membrane acts while carrying a tensile load. The estimated properties for a polyurethane-based liner and a liner made from polymer-modified cement are listed in Table 1. Note these values are rough approximations at best.

Table 1 Typical liner material properties assuming a 4mm thickness

Polyurethane	
Tensile strength, σ_t	8MPa
Adhesive strength, σ_a	1MPa
Bond width, w_b	5mm
Polymer cement (after 4 hrs)	
Tensile strength, σ_t	1MPa
Adhesive strength, σ_a	1MPa
Bond width, w_b	8mm

Two support functions and their related liner failure modes are shown in Figure 3. Both cases rely on penetration of polymer material into gaps between loose rock blocks. Although penetration into real fractures or joints can occur in the field, it should be negligible. If the liner design relies on shear rupture though polymer material infilling open fractures or joints Figure 3a or adhesion between suspended loose rock and competent rock Figure 3b, then it is likely that poor site preparation practices (scaling) have been used. Field evidence suggests that careful site preparation is critical to the success of a thin liner and loose rock should be scaled down before application of the liner. Nevertheless, the simple models shown in Figure 3 can be used to evaluate, in a general manner, the holding capacity of a spray-on material that acts like glue.

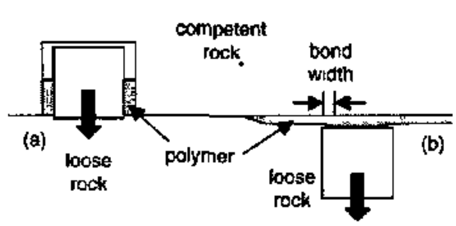


Figure 3 Possible support functions involving small loose rocks with an assumed shape of a 100mm cube.

In situations where open joints or fractures exist it is possible for the sprayed polymer to penetrate into the loose rock. Figure 3a show a simple model of a loose cube-shaped rock bonded to competent rock by a polymer around its four sides. The force required to remove the block from the competent rock depends on tbe depth that polymer penetrates the gap and the shear strength of the polymer. Based on laboratory conditions where sprayed polyurethane was used to coat concrete blocks separated by gaps of 1 to 2mm, the depth of consistent penetration is limited to less than about 20 to 50mm. A penetration, *dp* of 20mm will be assumed here. Material testing for polymer membranes is typically performed to measure tensile properties; therefore the shear strengths of these materials are largely unknown. The shear strength of a polyurethane membrane is likely to be similar to its tensile strength, assumed to be about 8MPa. The force, *F* needed to pull the loose bonded rock (with perimeter length *L)* away from the competent rock is:

$$F = L \cdot d_p \cdot \sigma_t = 0.4\text{m} * 0.020\text{m} * 8\text{MPa} = 64\text{kN} \qquad (1)$$

Even if the polymer penetrated on only one side of the loose block the force required to remove the rock (16kN) would be much larger the weight of the rock. For example, if the rock were a 100mm cube, it would weigh only about 26N. Hence, the support capacity is nearly three orders of magnitude higher than the block's weight.

By assuming another simplistic model of a loose rock glued to competent rock over a 100mm by 100mm bonded area, one can determine the effectiveness of a polymer material that has penetrated a gap between a loose rock and competent rock (Figure 3b). In this scenario, the polymer adhesion is the weak link. Although the polymer is assumed to fully cover the contact between the rocks, the adhesive strength is not mobilized over the whole area because progressive failure of the adhesive bond would occur. If it is assumed that the whole perimeter of die rock, over an effective bond width of 5mm, carries load before adhesive failure mitiates, then the force needed to dislodge the rock, *F* is given by:

$$F = L \cdot w_b \cdot \sigma_a = 0.4\text{m} * 0.005\text{m} * 1\text{MPa} = 2\text{kN} \qquad (2)$$

This is more than sufficient force to hold in place the weight of a small rock. Even if eccentric loading acts on the rock such that only one side of the rock is loaded in tension, the force holding the rock in place would be 0.5kN, which is much greater than the weight of the small rock.

The two simple models show that a thin polymer is quite effective at holding small rocks in place if sufficient polymer material is able to fill the gap between the loose and competent rock. This ability is clearly evident in laboratory tests when loose rocks are bonded together by any of the polymer materials. Once bonded together the individual rocks are virtually impossible to tear apart by hand.

However, for actual liner design purposes in a drift with careful scaling it may be best to ignore any possible penetration of the polymer into fractures or joints. Any penetration that does occur will likely improve the support capacity of the liner. But it is important to remember that the overall objective 's to have tight, sound rock present before the liner is sprayed. Failure to do so means that a sufficient portion of the rock mass's self-support capability has already been compromised before the liner is applied.

3.3 *Support for drifts - Membrane action*

At scales larger than that depicted in Figure 3, i.e., for general rock support across the back of a drift, the liner performs a support role by resisting relative movement between individual blocks of rock and possibly acting as a suspended membrane in tension carrying rock loads (Espley et al. 1999). A thin liner applied to the excavation surface, especially at the locations of fractures and discontinuities, is effective at resisting relative movement between individual blocks of rock. The liner performs this function through a combination of a gluing action as described earlier and a membrane action. The membrane action becomes more important if the liner is forced to experience larger deformations.

A liner is most effective when applied to the rock before significant movement takes place. In some cases, the liner can 'lock' the blocks together keeping relative block displacements small (<lmm) and thus function to stabilize the rock mass around the excavation. In other cases, when rock mass conditions, stress levels, and the excavation geometry combine to generate larger rock deformations or convergence, a thin liner may not be able to suppress relative displacements and a zone of unstable rock will develop. Under these conditions, the liner'typically acts like a deformable membrane to retain and hold the rock in place.

The support function that a thin liner may play depends on the amount of the rock that İs involved and the magnitude of relative displacements between adjacent rock blocks. It is important to recognize that large convergence may not be a problem so long as the rock moves inward in a uniform manner.

3.4 *Potential hner support failure modes*

The model shown in Figure 4 can be used to analyze the support capacity of a liner with thickness / holding a loose rock block that undergoes either small or large displacements. The surface area of the block coated by the liner is assumed square in shape with width *s*. The block is assumed to move vertically downward a distance *d* thus inducing stress in the liner. The first check is to determine whether the liner ruptures at small displacements due to either shear or diagonal tensile stresses around the perimeter of the block (Figure 5).

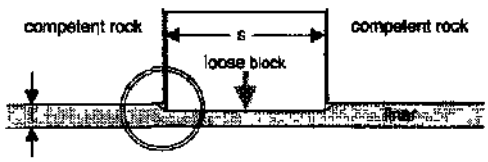
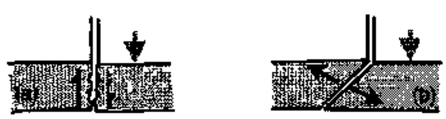


Figure 4 Model for rock support by a liner assuming a square block moving vertically downward.

3.5 *Small deformations (<lmm relative rock movement)*

Thin liners can resist shear displacements of up to a few millimetres. They achieve this by using a combination of shear, adhesive, and tensile strength. Relatively high liner stiffness is probably beneficial in this case. At small rock displacements the liner functions to prevent unraveling of small rock fragments, lock small rock blocks or wedges in place (key blocks), prevent loosening of the rock mass, mobilize interactions between rock blocks, and establish a stable arch of self-supporting rock. At small block displacements a thin liner acts like shotcrete in an active manner.



direct shear through liner diagonal tensile rupture of liner

Figure 5 Liner failure modes at small block displacements caused by either shear rupture or diagonal tensile rupture.

The liner can fail in two modes Figure 5. It is assumed that failure of the adhesive bond does not occur. Given that a typical liner is only a few millimetres thick, direct shear failure or diagonal rupture of the membrane must occur within the first few millimetres of relative rock displacement. These two failure modes are most likely when the liner adhesive strength is similar to the tensile strength. Note that this İs the situation for unreinforced shotcrete, which is why shear or diagonal tensile failure modes occur in shotcrete.

For the failure modes shown in Figure 5 the support capacity (expressed here as force per unit length around the block perimeter) is a simple function of the liner thickness and either the shear or tensile strength of the liner. As before, given lack of test data, the shear strength will be assumed equal to the tensile strength.

$$F = t \cdot \sigma_t \quad \text{per metre} \tag{3}$$

Assuming a hner thickness of 4mm and tensile strengths of either 1 or 8MPa (Table 1), Equation 3 yields a support capacity in range of 4 to 32kN/m. If the block size was Im by lm, with a density of 2600kg/m^3, then the liner could theoretically hold the weight of a block that was 0.6 to 5m high. Note, it is overly optimistic to expect a 4mm thick liner to hold up the weight of 5m of rock. In a real excavation the loading conditions would irregular and would greatly reduce the membrane's support capacity.

From the geometry shown in Figure 5b, one could argue that for diagonal tensile rupture, the true thickness of the liner carrying stress is greater than the liner thickness by roughly l/sin45°. However, given the uncertainty in the parameters, this effect is ignored.

While the approach presented here is illustrative, • rigorous liner design is nearly impossible given the complicated geometry of the interacting rock blocks and the unknown nature of all the forces acting through the arch of stabilize rock mass. Clearly, sprayed polymer materials are capable of holding in place small rocks as demonstrated earlier. But support design for a whole drift should be a philosophy or approach that dictates the need to maintain the inherent rock mass strength. The application of a liner is just one of many activities that can be used in this regard. For example, careful blasting practices are important too. In blocky rock masses, the use of a polymer liner may aid in the development a stable Voussoir beam.

Observations gathered from field and laboratory pull tests indicate that neither of the two failure modes depicted in Figure 5 are common for poiyurethane liners.

3.6 *Large deformations (=lmm relative rock movement)*

When conducting large pull tests on liners, the block displacements observed at the peak load are typically much greater than the thickness of the membrane. This demonstrates that the membrane is able to deform and stretch before it fails. In order for significant stretching to occur, some adhesion loss must also occur, providing a debonded length of membrane for stretching. Therefore, adhesion loss followed by tensile rupture is an important process from a design point of view.

These observations are consistent with the physical properties and liner thicknesses in use today (Table 1). Using the data for a polyurethane liner, the force needed to shear through a liner is about the same as the force needed to rupture the liner in tension (Equation 3) and is 32kN/m for a 4mm thick liner. The force required to initiate adhesive debonding (G„*vty,) is 5kN/m Therefore, when the adhesive strength to the rock İs significantly less than the liner tensile strength and the effective bond width is roughly the same as the liner thickness, adhesive failure around the displaced rock must occur first.

When the adhesive strength is less than the tensile strength the liner adhesive bond may progressively fail around the displacing block. By debonding, a section of liner rotates and begins to act in tension to resiot the weight of the moving block as shown in Figure 6. Under these conditions, the liner can tolerate relatively large block displacements Force equilibrium can be achieved when the vertical component of the tensile forces acting in the liner equals the weight of the block (assuming no fnctional resistance along the sides of the block)

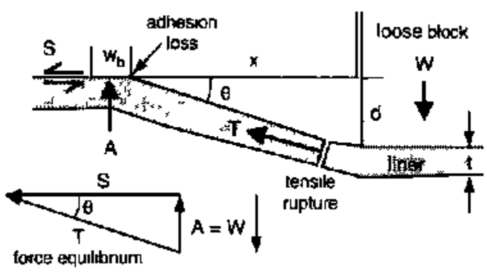


Figure 6 Interaclion between hner adhesion and tensile strength lo support the weigh! of a displaced block (only half of the model is shown).

The model first looks at the adhesive capacity of the membrane If the block movement causes progressive adhesive failure, the debonding will progress away from the edge of the block (Figure 6). In doing so, the area over which the adhesion acts grows because the perimeter length increases. It is assumed that the area eventually becomes large enough to create an adhesive force A that satisfies force equilibrium with the weight of the block. The width of the debonded zone x at equilibrium or when tensile rupture occurs is calculated from:

$$A = 4\sigma_a (s + 2x) w_b = W \quad (4)$$

where W is the weight of the block, a_a is the average adhesive strength of the membrane acting over the effective bond width w/„ and s ıs the width of the block. Equation 4 can be used to determine the width of the debonded area.

Adhesive support from the liner has now been fully mobilized so attention can now turn to the tensile strength of the liner. It is reasonable to assume that the tensile rupture will occur near the perimeter of the block, in which case the maximum tensile force T that can be carried in the plane of the membrane is:

$$T = 4s\ \sigma_t \cdot t \quad (5)$$

The vertical component of the tensile force must equal the block weight at equilibrium There is a geometric relationship between the block's weight and the tensile force in the liner By estimating the block weight and knowing the maximum allowable tensile force in the liner, the minimum angle 6 can be determined

e = arcsin(»'/7˙) (6)

This angle will define the minimum vertical block displacement needed to ensure that the vertical component of T ıs equal to the block weight W. The vertical block displacement at equilibrium is:

$$d = x \tan\theta \quad (7)$$

Based on the model shown in Figure 6, at the moment of tensile rupture the following relationship must hold true.

$$\sigma_t \cdot s \cdot t\ \sin\theta = \sigma_a (s + 2x) w_b \quad (S)$$

It is useful to note that the greater the angle 9 or for larger displacements and liner elongation, the greater the capacity. However, there is a limit to the allowable displacement that is governed by the elongation capacity of the liner. In this model, the following relation must not be violated.

$$\sqrt{x^2 + d^2} < (1 + e)x \quad (9)$$

Where e is the elongation at peak strength for a given linef product determined from laboratory tests. A typical value for e might be 0.2.

3.7 *Discussion*

The two models shown in Figure 6 and Figure 7 illustrate the interaction between adhesive and tensile properties of a liner. Equations 3 to 9 can be used to predict the block height that can be supported by a liner before it ruptures İn tension at a given block displacement In Figure 7, the block height and width (square cross-section) are plotted against the vertical block displacement at equilibrium. The assumed polyurethane liner properties in Table 1 were used

For all data plotted in Figure 7, the maximum liner elongation was less than 10%. Based on this simple model and assumed liner strengths it appears that elongation capability greater than roughly 10% may not be needed. However, if die liner's tensile strength is close to the adhesive strength, then it is advantageous to have a higher elongation capacity.

The curves plotted in Figure 7 implicitly assume a perfectly plastic matenal response for the liner at a tensile stress equal to the liner's tensile strength. Larger block displacements at equilibrium may occur if the liner has a strain-hardening response.

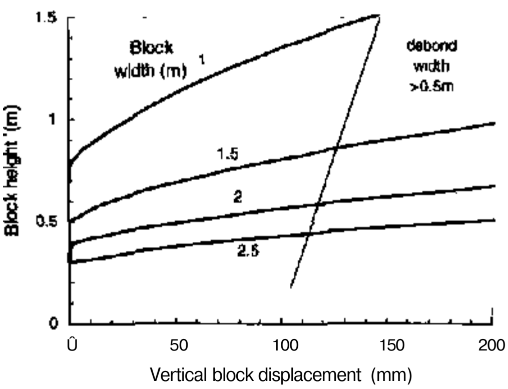


Figure 7 Block height vs. vertical block, displacement at equilibrium fora thin liner with o„=lMPa, w_b=5mm, Oi=8MPa, t=4mm. rock density=2.6g/cc. Points plotting beyond the üîn line indicate situations were the debonded width exceeds 0.5m.

The block heights shown in Figure 7 at block displacements of zero refer to blocks that do not cause progressive adhesive debonding and hence the block is stabilized at very small displacements by the shear and/or diagonal tensile rupture support mechanisms shown in Figure 5. For example, the force required to initiate adhesive failure İs 5kN/m, which for a Im by Im block is equivalent to a 0.78m block height. At block heights below 0.78m, the model shown in Figure 5 applies and liner failure will be unlikely if the tensile/shear strength is higher than the adhesive strength. For heights greater than 0.78m the model shown in Figure 6 applies, but only up to a point. Because once debonding initiates, the debonded width can quickly exceed 0.5m with further increase in block height or weight. The suspension support function illustrated in Figure 6 is quite sensitive to the debonded width. In practice, effective support from a liner is probably lost once the debonded width exceeds about 0.5m.

It is the combination of adhesive and tensile strengths the fundamentally controls the load the liner can support. Higher adhesive strength shifts the liner support function toward failure modes of shear or diagonal tensile rupture (Figure 5) and hence results in smaller displacements at failure. The adhesive strength to the rock must be about the same as the liner's tensile strength to most effectively utilize the tensile strength of a liner.

The tensile stresses in a liner can only counteract gravitational forces in loose rock blocks when the liner is either applied or is deformed such that a component of the tensile forces act vertically. The liner in the model presented in Figure 6 starts in a horizontal orientation and hence has no support capacity until vertical block displacement changes the liner orientation around the perimeter of the block. In a real excavation, perfectly flat backs are unusual and hence non-horizontal sections of liner may be oriented to more effectively hold loose rock.

The issue of liner failure modes and the corresponding displacements has important implications for support design and further studies should be conducted to verify the actual failure modes under field conditions. For now, based on pull tests and limited field observations, it appears that liner failure begins as adhesive bond loss near the displaced rock. As displacements continue, the zone of adhesion failure propagates away from the block and tensile stress builds in the liner. Ultimate liner failure occurs as tensile rupture and/or adhesion loss at larger block displacements.

3.8 *Adhesion and effective bond width*

Increasing the adhesive strength and/or the effective bond width increases the load capacity of the membrane. At present, the effective adhesive bond width for most liner materials is unknown although it may be back calculated from laboratory tests. For example, two punching tests (Archibald et al. I993) performed on three concrete blocks coated with Mineguard may be used to give a rough estimate of the effective bond width for a polyurethane liner. The setup for the testing was similar to the conditions shown in Figure 4. The force required to displace the centre block relative to the two side blocks was 1.73 and 1.1OkN. The centre block was 180mm long and adhesion was mobilized on each side of the block. Therefore the effective bond width w_h assuming an adhesive strength of 0.9MPa is found from:

$$w_b = \frac{load}{\sigma_a \cdot L} = \frac{load}{0.9\text{MPa} \cdot 2(0.18\text{m})} \tag{10}$$

The estimated bond widths are 5.3 and 3.4mm. It is quite likely that the effective bond width varies depending on the polymer type, applied liner thickness, and substrate conditions. Thicker and stiffer membranes probably have larger bond widths. For comparison, work by Fernandez-Delgado et al. (1979) and Hahn and Holmgren (1979) suggest that the effective bond width between rock and shotcrete for good adhesion is in the order of 50mm.

The load capacity calculated on the basis of adhesion is probably also a function of liner thickness because thickness probably affects the effective bond width. However, the lack of tests precludes assessment of this effect. Simple laboratory testing techniques similar to those presented by Tannant et al. (1999) are needed to better quantify liner material properties.

Plated rock bolts installed after a liner is applied can function to increase the effective bond width or the adhesion and hence help mobilize the full tensile capacity of the liner at smaller block displacements.

4 ECCENTRIC AND CANTILEVER LOADING

The models presented in the previous sections assume simple uniform loading conditions on the membrane. There are situations that violate this assumption. Tannant et al. (1999) described two case histories where liner failure occurred as a result of progressive tearing caused by large slabs of rock that rotated and cantilevered from the back.

In one case, a Mineguard liner was used to support a narrow drift (2m span) in highly stressed rock. A problem occurred near the advancing face where loose fractured rock caused the liner to sag between two bolts; this material was easily knocked down by a scoop bucket. A key factor contributing to the problem was the fact that the liner was not continuous to the drift face, i.e., the back at a distance of one round from the face was only supported on three sides by the liner because the newly excavated round had not been coated yet. The lack of a liner allowed roof displacements (sagging) to initiate near the edge of the blast-damaged liner and propagate away from the drift face- This created a cantilever effect in terms of the loads imposed on the liner. One positive aspect was that the liner gave ample visual warning that excessive displacements had occurred.

The other case history involved application of Mineguard to two rounds in a 3.0 to 4.3 m span drift that was driven along a swarm of sub-parallel, steeply dipping veins. The drift was excavated as part of a drift-and-fill mining method for the narrow veins and it was the third cut in a bottom-up mining sequence. While washing and scaling the roof after blasting the second round it was evident that stress-induced fracturing was occurring from the sound of "rock noises" in the roof and shoulders of the drift. The stability of the roof decreased over time due to the progressive nature of ,the creation of stress-induced fractures. The rock fracturing lead to a fall of a large slab of rock located İn the roof of the second round, which had not yet been totally coated with the liner (Figure 8). In total, roughly two tonnes of rock fell from the roof. The fall of ground also peeled some of the liner from the back.

The stress fractured slabs were observed to extend over both rounds because, coincident with the fall of ground, the roof above the first round suddenly moved downward about 50mm. However, the presence of rock bolts prevented the slabs from ultimately falling to the floor İn the firsi round. This example shows that the addition of rock bolts may be required in many situations where a spray-on liner is be used. In particular, rock bolts are probably need in addition a thin liner where the drift span exceeds 3 to 4m or where the rockmass quality is less than "good". When installing rock bolts the use of an automated rock bolt machine is recommended.

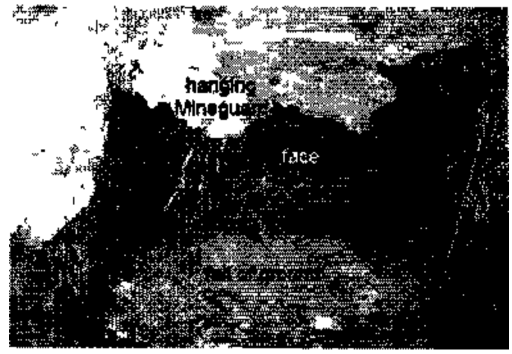


Figure 8 View towards face of a drift showing a fall of ground related to stress-induced fracturing and incomplete coverage of the roof with the liner

The two documented falls of ground occurred where the liner was not present up to the face of the drift because a new round had been previously excavated. In both cases, slabs of rock m the roof were able to move downward near the face because they were unsupported thus forming a cantilevered slab. Ultimately the slabs of rock tore through the liner at some distance from the face. In these cases the loading on the liner was concentrated at the liner's edge. High tensile stresses caused by the rotational displacement of the slab progressively ripped the liner until the slab was able to fall.

Full areal coverage by the liner is needed to minimize local straining and progressive tearing mechanisms. A continuous membrane that is firmly adhered to the rock creates effective support. Adequate rock scaling and cleaning are essential for good adhesion. Smooth liner overlap between rounds and adequate liner thickness (to bridge all rock fractures/joints) are essential for creating a continuous membrane. The use of a robotic spray arm removes the risk of ground falls while building up a continuous liner over the roof and upper parts of the walls.

5 OTHER CONSIDERATIONS

Thin polymer liners have a number of attributes that warrant special attention when evaluating or designing a liner for a mining application.

5.1 *Timing of liner application*

To gain maximum benefit from thin liner support it is important to apply the liner as soon as possible in newly blasted headings. The objective is to minimize rock mass loosening in a proactive manner. Thin liners are not very effective at 'tightening up' a rock mass that has been allowed to loosen.

The ability to rapidly apply a thin spray-on liner near the face of a newly blasted heading permits installation of support before the rock mass has time to loosen. In fact, a thin liner can be sprayed on the back of a fresh round before the blasted muck pile is removed. The rapid application rates achievable with thin liners means this type of support can be applied sooner to the rock than any other support type currently in use.

5.2 *Rock-support interaction*

Thin liners or shotcrete are superior to wire mesh in terms of their ability of have intimate contact with the rock and to mobilize rock interactions at small deformations. Mesh is largely a passive support that often only carries the dead weight of small rocks that have fallen between rock bolts. While a thin liner can perform a similar holding function to mesh, it is better suited to mobilizing rock-support interactions at small displacements (millimetres to centimetres). In contrast to some forms of shotcrete, the compliant nature of liners allows them to continue to function over a wider displacement range. One interesting application İs the use of a combination of mesh and sprayed-on liner. Although this support is fairly expensive, it offers superior strength and deformation properties.

5.3 *Continuous coverage versus web*

Experience to date suggests that creation of an unbroken thin membrane over the entire rock surface provides the best support. The liner follows the contour of the rock surface although the liner thickness is usually greater near open fractures or sharp concave depressions İn the surface.

Intelligent spraying equipment may be developed to identify the location of joints and fractures and then only spray the liner material in these locations thus creating a liner that functions like a spider's web. This could significantly reduce polymer material consumption and decrease application times. A web of liner material bridging across the joints and fractures would help prevent relative rock displacements. However, if ground conditions become so severe that the polymer web cannot prevent these displacements it is likely that a liner consisting of a continuous membrane would perform better. A continuous liner is simply more robust and has higher load carrying capacity than a web of polymer material. Nevertheless, where ground conditions are appropriate, a web of polymer material is likely to very effective and very economical.

5.4 *Performance near blasts*

Field observations have shown that a thin liner performs well in close proximity to blasts. When mesh and bolts are used within a metre of an advancing heading it is usual to see extensive damage to the mesh after each blast. The mesh is torn by the fly rock. A typical drift round will damage the mesh over a distance of roughly 3 to 5m from the blasthole collars. When supporting the back in preparation for the next round, some of the damaged mesh must be removed and new mesh installed. This is a time consuming process that exposes personnel to hazards from small falling rocks as well as cuts from torn mesh.

Thin sprayed-on liners can be used up to the face, and on the face itself if needed. After a round is blasted, portions of the liner further than one metre from the face typically sustain only minor damage such as small nicks, cuts and abrasions. The damage is most pronounced where the supported surface protrudes into the drift or on surfaces that face the blast. As expected, the worst damage occurs immediately adjacent to the face. At these locations, the liner can be peeled back about 0.3 to 0.5m from the face by the blast. Further from the face, the liner typically experiences only small nicks and cuts.

Near the face, where the liner has been torn from the rock there will normally be flaps of liner material adhering to the rock. These flaps of matenal must be cut away in preparation for the next application of the liner. In small headings, the flaps of liner material pose a problem. For example, when mucking out a drift, a scoop can accidentally catch a flap of liner and pull off quite a large section of the still good liner. Further equipment development is needed to simplify the process of trimming away flaps of liner matenal created by the blast. One operational procedure that has minimized this problem is to taper the thickness of the liner toward the face. The thinner liner near the face is more likely to tear without peeling off the rock. This leaves a narrow zone where the liner is destroyed by the blast but there is a clean transition to essentially intact liner.

5.5 *Long-term performance*

Polymeric liners have not been used for more than a few years in routine mining applications. Therefore, little operational evidence exists concerning their longevity in a mining environment. Initial research suggests that most materials in used today have very good resistance to acids and bases (Archibald & DeGagne 2000). Some polyurethane liner materials alter their colour and appear to degrade when exposed to sunlight. This should not be a concern in the underground environment.

One concern with polymer liners is their creep characteristics. Simple tests have demonstrated that most liner materials will creep and rupture at stresses much less than the values quoted for their tensile strengths. The impact of creep on the load capacity of a liner m conditions where a liner is supporting the gravitation load from loose broken rock İs unknown. Further research is needed to evaluate the performance of polymer liners under sustained loading conditions. The Canada Centre for Mineral and Energy Technology CANMET is working with Falconbridge to address this issue. Fortunately most liner materials can sustain large strains prior to rupturing. This allows for visual identification of areas experiencing problems and allows remedial actions to be taken before a fall of ground occurs.

5.6 *Safely*

Mine accident statistics demonstrate that the activities associated with the installation of mesh are relatively hazardous. Mesh installation is a labour intensive and manual operation and personnel are exposed to small rock falls, cuts, slips and strains in the process. In contrast, liner spraying is amenable to robotic application, which essentially eliminates these hazards.

5.7 *Rock visibility*

Thin liners can be sprayed on the face of an advancing drift to provide support against hazards such as small rockbursts. One advantage of thin liners compared to shotcrete is the ability to still see major rock structure and bootlegs after application of the support. The bootlegs from previous blastholes can be easily identified such that the new blastholes are not collared near the bootlegs.

It is also easy to identify features such as joints and rock type where the rock 'roughness' varies from one type to another.

Liner materials that are white in colour provide a major improvement in the general lighting conditions İn an underground environment.

5.8 *Dirty or weak, crumbly rock*

Thin liners have not been used successfully on weak, crumbly rock. Where the rock is weak or covered in dust it is impossible to create good adhesion between the liner and the rock. Without good adhesion a liner does not work. There have been cases where small pockets of high-grade sulphide ore have coated by liners. Eligh-grade sulphide ore can have a sugary, crumbly texture and little tensile strength. As expected the liner did not adhere to this rock type. The rock itself must possess sufficient tensile strength,

5.9 *Contamination of ore*

When mining through supported areas, the ground support becomes mixed with the blasted ore. Some studies have indicated that the presence of shotcrete in ore may cause detrimental effects in the milling and mineral recovery process. It İs not known if this is an issue with the various types of liner materials. Fortunately, the quantity of liner material needed to support a given area will be substantially less than for shotcrete.

5.10 *Application rates*

INCO completed costing and time studies for the activities needed to install various support types (Espley-Boudreau 1999). The studies were based on a 4.9m by 4.9m drift with a 3.7m drilled round achieving 3m advances. It was assumed that bolts and mesh were installed using a scissor-Hft truck with hand-held stoper and jack-leg drills. The shotcrete was applied manually with dry-mix equipment, and the polymer liner was sprayed with a hand-operated spray gun. The application rates were found to be 0.11 to 0.15m^2/min for 1.8m long mechanical rock bolts and welded-wire mesh; 0.1 to 0.33m^2/min for manual application of 50mm thick fibre-reinforced shotcrete (no rock bolts); and 1.8 to 2.3m^2/min for polymer liners (no rock bolts). When the labour component is included, the study found that polymer liners can be applied at a rate of about 60m"/man-shift versus 20m or 40m^2 per man-shift for bolts and mesh or shotcrete respectively.

The application rate for shotcrete can be increased by an order of magnitude by adopting the wet-mix method and using remote semi-automated equipment. Similar productivity improvements are expected for thin spray-on liners once they become more widely used and specialized spray equipment is developed.

5.11 *Costs*

The material costs for some polymer liner materials are presently quite high, ranging between Cdn \$25/m^2 and \$50/m^2 for an assumed application thickness of 4mm. These costs on a per metre basis are similar to 50mm of steel fibre reinforced shotcrete. The material costs for rock bolts and mesh cost the least, at about \$10 to \$13/m^2. However, it must be recognized that the installation of conventional mesh and bolts is both time consuming and labour intensive. In all cases, the total support costs involving labour and equipment were much larger than the material cost. It is important to remember that the material costs do not control the overall economics of the support selection.

The economic benefits from using min sprayed polymer liners are realized by the higher productivity created by reduction of the time needed for support installation. Further gains are possible when material transportation and handling cost are considered. Compared to shotcrete, a lot less material needs to be moved underground to the working face when using thin polymer liners.

Studies by INCO indicate that the total support cost using polyurethane liners (roughly $125/nr) İs similar to bolts and mesh and cheaper than mesh-reinforced shotcrete (Espley-Boudreau 1999). This cost does not account for substantial productivity improvements that are forecast to occur once semi-automated spraying of thin liners is implemented.

6 AN EXAMPLE OF CURRENT THIN LINER USE IN CANADIAN MINES

Falconbridge Ltd. is making routine use of TekFlex as mesh replacement in cut and fill stopes and permanent development drifts at the Fraser Mine {Figure 9). Thin liners combined with systematic mechanized rock bolting are being used to help support the back of 80 to 85% of all headings. Mesh is still used in the areas that experience stress-driven fracturing resulting in the generation of slabby rock (Pntchard per. coram. 2001) or in areas where the surface roughness of the rock is high thus requiring excessive quantities of liner per metre of drift (Pntchard et al. 2001).

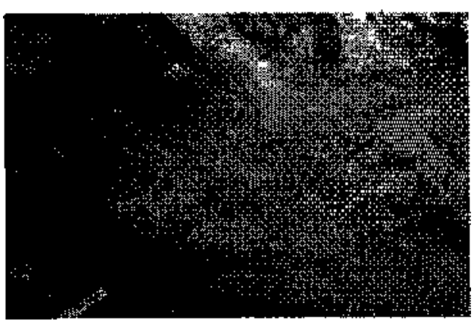


Figure 9 TekFlex spraying at Fraser Mine

Three key factors contributing to successful implementation of thin liners at Fraser Mine were (1) adopting stricter blasting and site preparations procedures, (2) better use of mechanized bolters, and (3) improved materials handling capabilities (Pntchard et al. 1999). One challenge that was overcome was the implementation of strict quality control procedures for site preparation, which included washing and scaling the back. When operators failed to follow the site preparation procedures, poor adhesion of the liner to the back resulted.

Perimeter control blasting techniques involving improved attention to blasthole location and alignment were found necessary to produce good quality ground conditions for a liner application. Once the operators realized the importance of a smooth, sound, clean rock surface, better care was devoted to the drilling of blastholes.

The support installation procedure is as follows. Once a heading is mucked out, die area is mechanically scaled to bring down large loose material. The round is then bolted with a mechanical bolter. Before spraying begins high pressure water scaling is performed. Once scaling is completed, the liner materials are mixed and readied for application. A calculated quantity of the liner material is sprayed onto the surface. A total of 200 litres is sprayed in a 4.6m wide by 3.5m high drift (4.2m advance), and down the walls to a height of 2.5m from the floor. About 300 litres is sprayed in a cut and fill environment where the breasting width is 11m, with 4.2m advance. TekFlex is sprayed with a mobile boom arm to cover the back to a thickness of about 4mm, and the walls to a thickness of 3mm.

One benefit from adopting a thin liner was increased awareness of the importance of high quality work. This resulted İn improved perimeter control, drill hole alignment, and a reduction in bootlegs. Furthermore, rock bolts are now being placed where they are needed most to support the rock rather than restricted to specific locations in order to hold sheets of mesh in place. When mesh İs used, the bolting pattern is eight bolts per (1.8m x 3.4m) sheet of welded-wire mesh on a 3-2-3 pattern resulting in a bolt density of about 1.6 bolts/m^2, accounting for mesh overlap. Use of mesh often results İn the installation of more bolts than are required for the ground conditions. Elimination of the mesh has allowed a wider bolt spacing (1 bolts/m^2) thus nearly doubling bolter productivity. The use of a thin liner enables the bolting density to match the ground conditions.

7 CONCLUSIONS

Thin liner support is an emerging technology that is applicable to underground support of blocky rock masses. A variety of different liner materials are currently being investigated and some are now being used in routine support applications within Canadian mines.

Thin polymer liners have performance characteristics that lie between those of shotcrete and mesh. They are a welcome addition to the 'tool box' of support types and have a role to play where rapid application rates and areai support of rock are needed.

The liner must adhere well to the rock and hence, the use of thin spray-on liners is not recommended where the rock surfaces are dirty or can not be cleaned or where the rock has a crumbly texture. A continuous liner that is firmly adhered to the rock creates effective rock support.

Various approaches were used to examine the load capacity of a liner. Interpretation of the available test data and introduction of simple support models show that die two likely failure modes are adhesion loss and tensile or shear rupture of the membrane. Different failure modes occur depending on the relative tensile and adhesive strengths of the liner and the anticipated magnitude of the rock displacements.

Virtual all liner materials are very effective at holding in-place small pieces of rock. When a liner is used to support a larger area it can do so through a combination of adhesion and shear strength at small (<lmm) relative block displacements. If excavation conditions generate larger block displacements, the liner acts like a supporting membrane and a combination of adhesive strength, tensile strength, and liner elongation serve to eventually create force equilibrium in a displaced loose block.

More research is needed to determine design values for the tensile, shear, and adhesive strengths of different liner materials. It is equally important to gain a better understanding of the effective bond widths that carry adhesive stress during progressive debonding of a liner from the substrate material. Field trials are useful for identifying liner performance and potential failure modes and aid with the development of reasonable models for liner design.

There are a wide variety of factors that must be considered before adopting wide spread use of thin sprayed on liners. Fortunately, it appears in some cases that liners offer increased safety, better productivity, and lower overall mining costs compared to conventional bolt and mesh or shotcrete.

ACKNOWLEDGEMENTS

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REFERENCES

Archibald J.F. & DeCagne D.O. 2000. Recent Canadian advances in the application of spray-on polymeric linings. Presentation 10 the Mining Health and Safety Conference, Sudbury, 20p.

Archibald J.F. & Lausch P. 1999. Thin spray-on linings for rock failure stabitizatioa *Proc. 37th US Rock Mechanics Symp.,* Vail. *Rock Mechanics for Industry,* Balkema, Rotterdam, 617-624.

Archibald J.F. Espley S.J. *St.* Lausch P. 1997. Field and laboratory response of Mineguard spray-on polyurethane liners. *Proc. Int. Symp. on Rock Support,* Lillehammer, 475-490.

Archibald J.F, Mercer RA. & Lausch P. 1992, The evaluation of thin polyurethane surface coatings as an effective means of ground control. *Proc. Int. Symp on Rock Support - Rock Support in Mining and Underground Construction,* Balkema, Rotterdam, 105-115.

Archibald J.F. Mercer RA. & Lausch P. 1993. Innovative ground support using rapid deployment of spray-on liners *Canadian Institute of Mining and Metallurgy Annual General Meeting,* Alberta.

Espley S.J. Langille C. & McCreath D.R. 1995. Innovative liner support trials in underground hardrock mines at INCO Limited. *Canadian Institute of Mining and Metallurgy Bulletin* 88(992^,66-74.

Espley S.J. O'Donnell J.D.P. Thibodeau D. & Paradis-Sokoloski P. 1996. Investigation into the replacement of conventional support with spray-on liners. *Canadian Institute of Mining and Metallurgy Bulletin,* 89(1001), 135-143.

Espley S.J. Tannant D.D. Baiden G. & Kaiser P.K. 1999. Design criteria for thin spray-on membrane support for underground hardrock mining. *Canadian Institute of Mining and Metallurgy Annual General Meeting,* Calgary, published on CD-ROM, 7 p

Espley-Boudreau S.J. 1999. Thin Spray-on Membrane Support and Implementation in the Hardrock Mining Industry. M.A.Sc. thesis, Laurentian University, Sudbury, Ontario, 31 lp.

Femandez-Delgado G., Cording E.J., Mahar J.W., & Van Sint Jan M.L.. 1979. Thin shotcrete linings in loosening rock. *Proc. 1979 Rapid Excavation and Tunneling Conference.* American Institute of Mining, Metallurgical, & Petroleum Engineers, 790-813.

Hahn T. & Holmgren İ. 1979. Adhesion of shoıcrete to various types of rock surfaces. *Proc. 4th Congress of the Int. Society for Rock Mechanics.* Balkema, 431-439.

Hoek E. & Brown E.T. 1980. *Underground Excavations in Rock.* Institute on Mining and Metallurgy, London, England.

Kaiser P.K. & Tannant D.D. 1997. Use of shotcrete to control rockmass failure. *Proc. Symp. on Rock Support,* Lillehammer, 580-595.

O'Donnell J.D.P. & Tannant D.D. 1998. Field pull tests to measure insitu capacity of shotcrete. *Canadian Institute of Mining and Metallurgy Annual General Meeting,* Montreal, published on CD-ROM.

Pritchard C. Swan G. Henderson A. Tannant D. & Degville D. 1999. TekFlex as a spray-on screen replacement in an underground hard rock mine. *Canadian Institute of Mining and Metallurg}- Annual General Meeting,* Calgary, published on CD-ROM, 6 p.

PritchardC. Swan G. Hill R.&AmickM. 2001. TekFlexasa spray-on screen replacement in an underground hard rock mine. *Mining Engineering, SME* in press, 4p..

Spearing A.J.S. & Champa J. 2000. The design, testing and application of ground support membranes for use in underground mines. *Proc. MASSMIN2000,* AuslMM.

Tannant D.D. 1995. Load capacity and stiffness of welded-wire mesh *Proc. 48th Canadian Geotechnical Conference,* Vancouver. 729-736.

Tannant D.D. 1997. Large-Scale Pull Tests on Mineguard™ Spray-on Liner. Report to INCO Limited, Copper Cliff.

Tannanı D.D. & Kaiser P.K. 1997. Evaluation of shotcrete and mesh behaviour under large imposed deformations. *Proc. Symp. on Rock Support,* Norway, 782-792.

Tannant D.D., Kaiser P.K. & Maloney, S. 1997. Load-displacement properties or welded-wire, chain-link, and expanded metal mesh. *Proc. Symp. on Rock Support,* Norway, 651-659.

Tannant D.D. Espley S. Barclay R. & Diederichs M. I999. Field trials of a thin sprayed-on membrane for drift support. *Proc.* P"" *ISRM Congress on Rock Mechanics,* Balkema, 1471-1474.

Tannant D.D. Swan G. Espley S. & Graham C. 1999. Laboratory test procedures for validating the use of thin sprayed-on liners for mesh replacement. *Canadian Institute of Mining and Metallurgy Annual General Meeting,* Calgary, published on CD-ROM, 8 p.

Wojno L. & Kuijper J.S. 1997. Spray-on, user friendly and flexible support for mine excavations. *Proc. Int. Symp. an Rock Support,* Lillehammer, 671-683.

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A Case Study on Safety Factor and Failure Probability of Rock Slopes

M.U.Ozbay
Colorado School of Mines, Mining Engineering Department, Golden, Colorado, USA

ABSTRACT: Performances of two different statistical methods in determining failure probability of rock slopes are evaluated using data from insitu plane failure cases. The first method, called "direct method", defines cohesion and friction angle as normal distributions and calculates a failure probability from the safety factor distribution. The second method, called "maximum likelihood method" estimates unique values for cohesion and friction angle by maximizing the safety factor distribution. The analyses using data from 48 plane failure cases show that both methods predict similar mean values for cohesion and friction angle, and thus, for safety factor. However, the maximum likelihood method predicts less scatter in the safety factor distribution, thus lower failure probabilities than the direct method for a given slope geometry.

1 INTRODUCTION

This paper describes the applications of the two different statistical methods to estimate probability of failure of slopes excavated in rock masses. Both methods use data from failed cases and determine failure probability from safety factor distributions. The "direct method" is based on the assumption that the safety factor S of a slope at failure is equal to 1.0 and back-calculates the strength parameters, namely the cohesion and friction angle, using the geometrical data obtained from failed slope cases {Hoek and Bray 1981). Depending on the level of inaccuracies involved in the geometrical measurements, the strength parameters obtained from the failed cases are most likely to differ from each other. These strength parameters, namely cohesion and friction angle, can be set as statistical distributions and used in the safety factor formula to determine a safety factor distribution. This safety factor distribution can then be used to obtain failure probabilities of slopes with different geometries. The "maximum likelihood method", also accepting that the determination of the true values of the failure geometries is practically impossible, assigns a particular statistical distribution to the safety factors from the failed cases and back-calculates the values of cohesion and friction angle that by maximizing this distribution around the central value S=1 (Salamon (1999). This process produces a standard deviation for the maximized distribution and unique values for cohesion and friction angle. The values of cohesion and friction angle now can be used to design a new slope and the failure probability of this slope can then be calculated from the maximized safety factor probability distribution.

In die following, these two methods are described İn further detail and evaluated using a data set established from 48 plane failure cases that occurred in the benches of a large open pit mine.

2 IN-SITUDATA

The data used is from 48 dry plane failure cases that occurred along a major joint set İn the benches of a large open pit mine (Calderon, 2000). It contains only those collected from clearly defined and well-exposed plane failures. Each case İs described in detail and includes the parameters relating to geometry of the failed blocks and joint inclination angle. Table 1 gives the range of the parameters measured during the surveys.

The shear strength of the joint planes is assumed to be governed by the Mohr-Coulomb shear failure criterion given as

$$\tau = c + \sigma_n \tan\phi \qquad (D$$

Table i Range of geometrical parameters measured tor the failed cases.

Dip of slip plane (°)	37-66
Dip of the slope face (°)	57-85
Height of the sliding block (m)	10 8-32 0
Weight of the sliding block (kN)	390 0-11100

where c = cohesion, σ_n = normal stress acting on the joint surface, and ϕ = joint's friction angle. The failure is assumed to occur according to the limiting equilibrium condition along dry discontinuity planes with no tension crack, which is expressed as

$$S = \frac{cA + W\cos\psi_p \tan\phi}{W\sin\psi_p} \tag{2}$$

where S = safety factor, W = weight of the sliding block, and ψ_p = the inclination angle of the failure plane.

3 DIRECT METHOD

In the direct method, it is assumed that the failure occurs when S=1, which, when substituted in (2), gives

$$c = \frac{W\sin\psi_p - W\cos\psi_p \tan\phi}{A} \tag{3}$$

If the geometry and the rock mass density from any two failed cases are known, Eq. (3) can be set for these cases and solved simultaneously to calculate the values of cohesion and friction angle. In reality, true measurement of the blocks and rock mass density is practically impossible. However, if there are several cases of failures and their geometries are measured with humanly possible accuracy, the statistical distributions of the cohesion and friction values can be established from n(n-1)/2 solutions; n being the number of failed cases.

Figure 1 shows the cohesion and friction angle values obtained from 48 plane failure cases. The solutions resulted in 948 points of intersection in the positive quadrant. The mean and standard deviation of these cohesion and friction angle values corresponding to the points of intersections are given in Table 2.

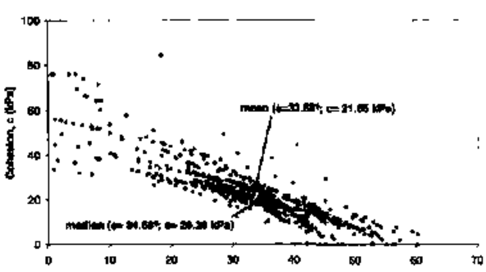


Figure 1. Cohesion and friction angle values calculated from 48 failed cases.

Table 2, Statistical parameters for cohesion and friction angle obtained from 48 failed cases using the direct method.

	Cohesion (kPa)	Friction Angle (°)
Mean	21.7	33.9
Standard deviation	12.50	9.84
Median	20.3	34.7

Based on the mean and standard deviation values given Table 2, a normal distribution of 10000 cohesion and friction angle values generated using Monte-Carlo technique is shown in Figure 2. Both data sets have large standard deviations, which result in negative values in die data sets, which are truncated to exclude negative values but retaining similar statistical parameters. The statistical parameters for the truncated normally distributed data sets are given in Table 3, which shows that the mean and standard deviation of the new data sets are sufficiently close to the original data given in Table 2.

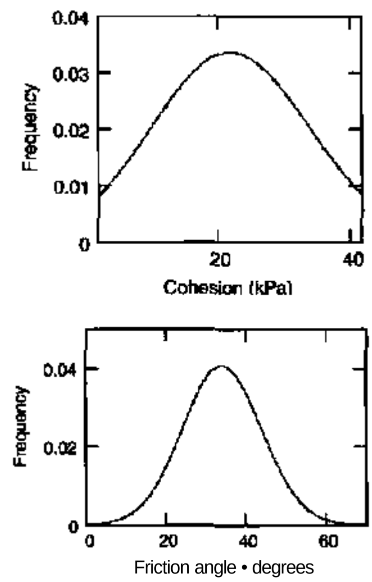


Figure 2. The normal distributions for cohesion (top) and friction angle (bottom) generated using the statistical parameters given in Table 2.

Table 3. Statistical parameters of cohesion and friction angle data from generated and truncated normal distributions.

	Mean	Median	St. dev.
Cohesion (Figure2) kPa	21.7	21.8	11.9
Friction angle (Figure 2) ϕ (°)	33.9	34.0	9.8
Cohesion (truncated) c (kPa)	21.9	21.8	11.5
Friction angle (truncated) ϕ	33.8	34.0	9.8

The truncated frequency distributions of cohesion and friction angle are used for determining the safety factor distributions. For the slope geometry parameters, the mean values from the 48 cases are used, and these are given İn Table 4.

Table 4 The geometrical parameters used for determining the safety factor distribution.

	Mean	St dev
Failure plane angle Vp (")	50.9	7.4
Block base areas A (m^2)	25.9	7.3
Block height H (m)	16 4	4.6
Slope face angle Hfr Ç)	67	51

The safety factors calculated using the strength parameters in form truncated distributions in Table 3 and the geometrical parameters given in Table 4 are shown as a histogram in Figure 3. The mean and standard deviations of the safety factor population are 1.001 and 0.324, respectively, and this, as a normal distribution, is also given in this figure.

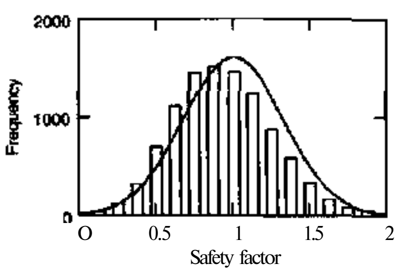


Figure 3. The safety factor distribution as obtained from the direct method

The validity of the mean values of the cohesion and friction values given in Table 3 can be assessed by plotting the joint's shear strength against the acting shear stress for the 48 failed cases, as shown in Figure 4. The straight line in this plot marks the location ofshear stress = shear strength, i.e. S=1. Most values of calculated safety factors lie close to this line, indicating that the method produced results that fit the insitu data reasonably well.

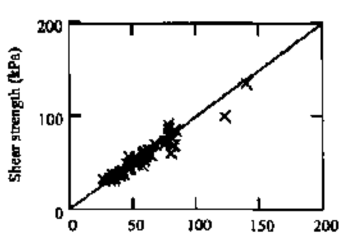


Figure 4. The safety factors for the 48 failed cases calculated using the cohesion and friction angle values estimated from the direct method.

4 MAXUIMUM LIKELIHOOD METHOD

In this method, the safety factor distribution function is maximized, with the objective of having the cohesion and friction values to result in a safety factor population concentrated around the central value of S=1. This can be achieved by multiplying the ordinates of the frequency distribution function corresponding to each failed case, that is maximizing the function

$$M(c,\phi,\sigma) = \prod_{i=1}^{n} f[\ln S(c,\phi,\sigma)] \tag{4}$$

where a is the standard deviation S, is the safety factor formula as given in (2) above and thus f(.) represents lognormal distribution function for assumed for the safety factors from the 48 failed cases.

The cohesion and friction values calculated from the maximum likelihood method, using the geometrical parameters given in Table 4, are given in Table 5. The standard deviation of 0.101, resulting from the safety factor population, is much smaller than that 0.324, obtained by using the direct method.

Table 5 The strength parameters obtained from the maximum likelihood method

	Mean	Median	Standard Dev
Cohesion c (kPa)	19.5		
Friction angle $ O	35 4		
Safety factor	0.9948	0.9946	0.101

Based on the values given in Table 5, the frequency histogram and the distribution function are given in Figure 5. To evaluate the validity of the cohesion and friction values calculated from the maximum likelihood method, Figure 4 is re-plotted in Figure 6. The plot is almost identical to that of the direct method, which is expected since the mean values of cohesion and friction determined by *the* use of the either method are similar.

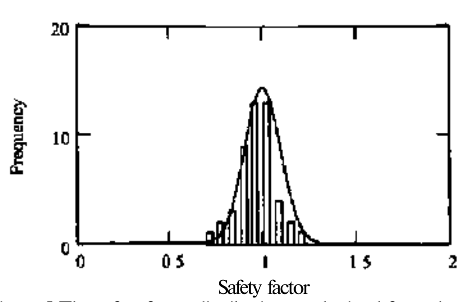


Figure 5 The safety factor distribution as obtained from the maximum likelihood method

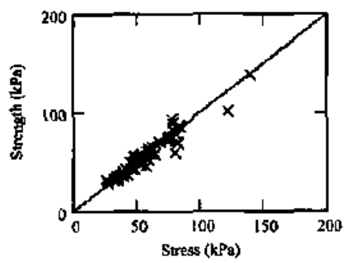
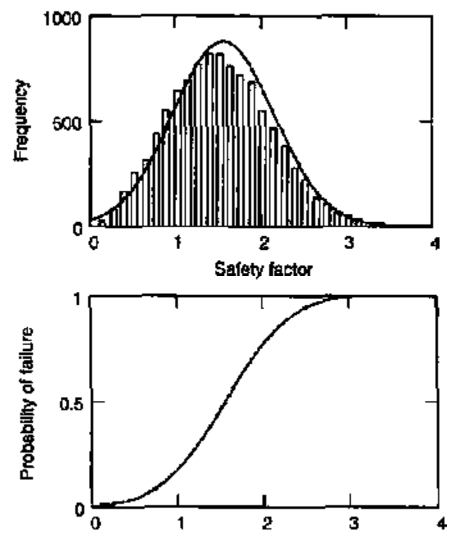


Figure 6. The safety factors for the 48 failed cases calculated using the cohesion and friction angle values estimated from the direct method.

5 FAILURE PROBABILITY ESTIMATIONS

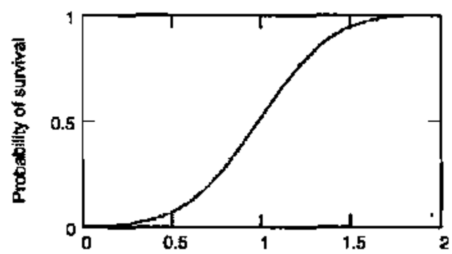
The failure probability estimations are made from the cumulative density functions originated from the direct and maximum likelihood methods described above. Two different approaches are considered in estimating failure probabilities. In the *probability of failure* approach, the probability density and cumulative density functions are re-built for a particular safety factor being considered. The proportion of the cases with S<1, determined from either of these functions, is then taken as the probability of failure. As an example, consider reducing the slope angle from the original 67° to 57°. The safety factor frequency distribution, obtained by using 57° for the direct method, is given in Figure 7(top). The deterministic safety factor in this case is S=1.54, which is about the same as the mean of the distribution in this figure. The proportion of the number of cases of S<1 gives the failure probability for this particular safety factor. Alternatively, the failure probability can be read direcdy from the ordinate of the cumulative density function shown in Figure 7(bottom), which for this example happens to be 17%.

In the *probability of survival* approach, always the frequency distribution obtained from the failed cases (e.g. Figure 3 or Figure 5), or their cumulative density function (Figure 8), is used regardless of the value of the safety factor being considered. Continuing with the example, the safety factor resulting from reducing the slope to 57° is 1.54. The proportion of the cases with S>1.54 makes up about 4.7% of all cases. That is, the probability that a slope with S=1.54 being part of all the failed cases is 4.7%. In the cumulative distribution curve, this corresponds to l-F(S), which is called *probability of survival,* (Salamon, 1999), with F(S) being the cumulative distribution function. In the maximum likelihood method, since cohesion and friction are determined as constants, probability of failure approach described above becomes inapplicable.



Safety factor

Figure 7: Frequency distribution and histogram of the safety factor distributions (top), and the cumulative density function (bottom) obtained from the direct method for 57" slope angle (S=1.54).



Safety factor

Figure 8: Cumulative density function of the probability fonction in Figure 3.

Table 6. Failure probabilities based on the *"probability of failure"* and *"probability ofsurvivaF* approaches calculated from the direct and maximum likelihood methods.

		Failure probability %		
		Direct method		Maximum likelihood
Slope angle	Safety factor	Probability offailure	Probability ofsurvivat	Probability ofsurvival
67	0.97	50.0	53.4	62.6
62	I.I3	34.3	34.3	12.3
57	1.54	17.0	4.7	0.0012
52	5.70	5.2	0	0

Failure probabilities calculated using the *probability offailure* and *probability of survival* approaches are compared in. As seen, the two approaches, as well as the two methods of determining safety factor distributions give significantly different results, especially at increased safety factor values. At S=5.7, the probability of survival predicts almost 0% failure probability, while the estimation with probability of failure approach is 5.2%, which is probably unrealistic. When comparing the methods, it can be seen that the maximum likelihood method predicts lower failure probabilities for S> 1 than the direct method.

Based on the discussions above, the following comments are noteworthy:

1. The reason that maximum likelihood method gives less probability of failure is due to lower standard deviations of the safety factor distributions originally calculated with this method.
2. Use of lognormal distribution with the maximum likelihood method, by definition, does not allow negative cohesion and friction angle values, thus probably more realistic than using normal distribution with the direct method.
3. Maximum likelihood method is less affected by the values of the geometrical parameters m the safety factor formula.
4. The maximum likelihood method appears to be a viable statistical tool and the potentials offered by this method are well worthy of further trials and research.

6 CONCLUSIONS

- For the failed cases analyzed here, the safety factor values calculated from the direct method and the maximum likelihood methods are similar, indicating that the methods are equally applicable in deterministic design.
- The larger standard deviations in the direct method require truncation of the distribution functions to avoid negative cohesion and friction angle values in the data sets. The negative strength parameters do not result in the maximum likelihood.
- The larger standard deviations in the direct method affect the failure probability calculations. Increasing die safety factor in this method İs less effective in reducing the probability of failure when compared to maximum likelihood method. This observation is significant as it points out that tie degree of improvement in failure probability by increasing safety factor İs dependent on the method used to determine the probability density function.
- The probability estimation using the concept of probability of survival appears to be a viable concept and deserve further research and trials.

REFERENCES

Salamoıı, M D G (1999) Strength of coal pillars from back-calculation. Proceedings of die *11^* US Rock Mechanics Symposium, pp 29-36 Vail, Colorado, USA

Calderon, A. R (2000) The application of back-analysis and numerical modeling to design of a large pushback in a deep open pit raine MS thesis. Department of Mining Engineering, Colorado School of Mines, Golden, Colorado, USA.

Calderon, A. R. & Ozbay, M U. (2001) Shear strength determination by back-analysis of slope failures using maximum likelihood method ARMA *ii*[st] US Rock Mechanics Symposium, Washington DC, USA.

Hoek, E. & Bray, J.W. (1981) Rock slope engineering. London; IMM.

17" International Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-39S-417-4

Application of Remotely Sensed Data and GIS in Assessing the Impact of Mining Activities on the Environment

T.Woldai
International Institute for Aerospace Survey & Earth Sciences (ITC). Enschede, The Netherlands

ABSTRACT: Primary industries such as mining form the backbone of developing economies throughout much of the world. A century of production driven, environmentally insensitive policies however, are leading to massive soil degradation and contamination, toxic vegetation, groundwater {surface and subsurface) pollution, mine dump disposal and landscape defacement around the mining areas. A systematic and multi-disciplinary approach of mapping, monitoring and controlling the impact caused by the mining activities is necessary so as to understand the character and magnitude of these hazardous events in an area. This paper first addresses the issues concerning mining and its impact on the environment and on goes to assess the various remotely sensed geo-İnformatioo tools available nowadays for capturing up-to-date and detailed earth observation data, processing and interpretation in mining induced environmental problems.

1 INTRODUCTION

Any actual mine site is just one point in a long line of activity before and after the digging starts. Unlike the days when mining was a low-tech 'pick and shovel' enterprise, with perhaps a bit of dynamite thrown in, today's mining and mineral extraction industry is high-tech and often chemically intensive.

It is a transitory land use that takes many forms; e.g., dredging, placer area mountaintop removal and contour operations. It requires an extensive network of industrial infrastructure that can disrupt the surrounding land and stretch far beyond the comparatively small footprint of the mine or exploration site. In spite of its great economic benefit therefore, *it* creates many environmental problems İn which the chemical and physical properties of the spoil change to create a hostile environment (Poulin & Sinding, 1993). While it is easy to believe they are not there, in many countries of the world, mining activities is resulting in disastrous impacts on the basic life-support systems such as clean air and water, productive soil and the Earth's rich biotic diversity by:

- Destroying or diminishing the utility of land for commercial, industrial, residential, recreational, agricultural, and forestry purposes,
- Causing erosion, sedimentation, subsidence and landslides,
- Contributing to floods,
- Pollutant loading to groundwater and surface water from acid mine drainage,
- Process solution leaks, spills, and surface subsidence,
- Destroying or altering fish and wildlife habitats,
- Impairing natural beauty,
- Damaging the property of citizens,
- Creating hazards dangerous to life and property
- Degrading the quality of life in local communities, and
- Counteracting governmental programs and efforts to conserve soil, water, and other.

Precautionary measures must be taken to stabilize these areas and/or proper planning be taken prior to mining so as to avoid serious pollution and degradation of the environment. Remote sensing from aircraft and satellites, is a powerful tool used in Earth resources mapping which can be adapted for environmental monitoring of mining induced activities. The large number of satellites orbiting the Earth allows for frequent downloading of images of any particular area. Combining such data with other ancillary data types in a Geographic Information System (GIS) allows decision-makers and planners with adequate and understandable information to assessing the impact within a relatively short period of time.

2 MINING AND THE ENVIRONMENT

2.1 *Mining Issues*

Mining involves several stages; namely, site development, mining, mineral extraction/ benefication/processing and mine closure. Each of diese stages of the mining process has the potential for different impacts of various degrees depending on the sensitivity of local terrain, the type of technology employed the skill and knowledge of the company and its ability to monitor and enforce compliance with environmental regulations.

Site development starts with the discovery of a potential mineral deposit. The next problem is devising a way to get all the necessary equipment and manpower to the site in order to begin the extraction process. Quite often, mine sites are situated in out of the way areas that require the construction of a road network (roads/ railways/air access routes), making deleterious road impacts one of this phase's biggest environmental problems. New access roads and townships often initiate unplanned development and squatting. The construction of road access to the area concerned causes disturbance of the surface, and has negative effects on vegetation, wildlife, deforestation and residents İn me area.

Mining involves three categories: surface mining, underground mining and in situ mining. Initial preparation of surface mine involves removing the vegetation, topsoil, blasting, gross sizing of ores and extensive modification of the topography of the entire area (see Fig.1). Such activity brings local air quality problems because of the amount of dust raised from the exposed earth. The surrounding area is littered with waste rock dumps, tailings and slime dams, creating a zone of residues. These are mostly earthy in nature and hazardous to health if the mineral extracted is dangerous (such as uranium and/or thorium bearing ore or asbestos), or if it has been washed with toxic or harsh substances (e.g., cyanide, mercury and sulphuric acid) to separate the valuable materials from waste products. The waste rock and soil dug up during ore extraction, called overburden, are mostly piled in dumps, which take up good land and pollute soil and water. Surface waters draining the dumps of waste rock carry pollutants to agricultural land and drinking water supplies. In most cases, the combination of the physical and chemical characteristic often renders steep, barren and unproductive land - a sterile landscape indeed. In many omer countries, the residues during the drier seasons often cvumble to a fine dust from which even a wind of moderate force can produce a miniature sandstorm. When it rains, the sodden dumps revert to amorphous slurry, which spreads to the nearby-cultivated land and water supplies.

All subsurface mining results in the transfer of material from beneath the surface to the surface itself. Abstraction of groundwater is often required. Like surface mining, underground operation cause surface disturbance by requiring land for waste rock disposal, storage of ore and low-grade material and siting of ancillary facilities. Removal of rock and subsurface water frequentiy results in surface subsidence and collapse. Subsurface mining creates the potential for mine effluent - acid mine drainage and heavy metal concentrations. These effluents can wreak havoc on watersheds and aquatic ecosystems. This acid mine waste quite often leaches into ground water, streams, and lakes causing pollution to entire streams and rivers. It can lead to fish kills and sterilisation of biologically rich areasr

Figure 1 Fiion Norte open pit mine in Tharsis, southwest Spain.

Finally, in situ leaching is probably the least known mining method and the process basically involves drilling smaller holes in the site, and using water based chemical solvents to flush out the desired mineral in a solvent state. At first glance, the rninimalization of waste represents one of the most environmentally beneficial aspects of this method. On the other hand, the injection of toxic substances directly into the earth both in and near the water table represents one of the least environmentally beneficial aspects of the method.

Mineral Processing. Once extracted, the mineral in question needs to be processed or cleansed of its impurities. This stage, also known as benefication commonly involves chemically intensive activities. Benefication/ processing may include mechanical, gravitational, magnetic, chemical, electrochemical, and/or thermal methods of separating target minerals from wastes and concentrating values prior to sale. Contaminants build up in the soils near smelters, some finding their way into food crops. Air pollution from smelters might release poisonous particles and noxious gases into the atmosphere. This, when not controlled, fine their way into the water, soil and vegetation usually causing severe health hazard to the miners and the community close to the mine site.

Airborne pollution from rubble dumps, slimes dams, open pits and overburden heaps in the form of dust İs potentially a serious hazard.

Waste disposal methods vary from site to site. They can be left İn piles next to the site, or backhoed into the mine, in either case, the issue of acid mine drainage remains a potential environmental problem. Waste is also stored in containing ponds or dams to minimise the acid mine drainage. These however, are not always stable. The chemically intensive nature of the benefication stage increases the possibility of an accidental spill, and human nature being imperfect means that spills are a common occurrence.

Mine closure. The closure of a mine often involves severe and devastating environmental and socio-economic impacts. The dramatic loss of employment and the faltering of many small enterprises in the area are a major blow to die population involved. With surface and underground mines, the main hazard revolves around erosion, weathering, seepage, impairing of natural beauty, damage to property of citizens and total failure of steep pit walls and waste pile slopes. Acid pH-values and hazardous mineral content may hamper revegetation, while dust from wind erosion of tailings, ponds and waste rock piles may cause air pollution. The sort of hazard and risk evolving from those abandoned mines may vary. There are those that we can see and those, which we don't. The problems that uranium, radium and other radioactive mines deliver are there to stay for centuries to come. Besides, abandoned and unguarded excavations including prospect pits, open pits, shafts and audits of which there are more than hundreds are hazardous to the reckless, the unwary, or the uninitiated. Closed plants, workshops, houses and infrastructure (roads, railways, etc.) are unsightly and may become safety hazards.

Site restoration projects are always problematic in many countries. Legal requirements for site restoration do not exist and hence, the adverse impact of such neglected mines is evident. First there is the issue of money. Nearly all have enacted various degrees of less than stringent environmental regulations in the hope ofdrawing İn foreign capital. Site restoration requirements as a result, get lost in the shuffle always. In the mean time, environmentally hazard continues at an alarmingly consistent rate.

2.2 *Mining Stages*

The types of mines vivid in many countries fall into:

Abandoned mines. Those, which were exploited in the past but, are in some way or other discontinued and not functioning anymore. Some of mem are pre-historical in nature or were reactivated at different periods and times in their history. In most countries, the active mines of today might have been the abandoned mines of the past. Whatever they are, the unregulated mines of the past typically dispose mine wastes without any environmental controls or constraints. In many settings, this old mine wastes remain vulnerable to wind and water erosion. They still release pollutants many years after the mines or enterprises have shut down. Where restoration laws are intact, their implementation can be highly problematic. Cleaning these mines and industrial sites nowadays is extremely costly and non-affordable by any government standard. Furthermore, cleaning up the sites may not result İn appreciable improvements in human health or the environment. Consider the case of environmental restoration of uranium mines in Namibia. Firstly, the volumes of the accumulated radioactive waste are far too high to be removed at a reasonable cost. Secondly, safer alternative disposal sites are either not available or else are impractical. Given resource constraints, what decision rules should guide activities for remediation? Which sites should be addressed first? These issues are hard to answer.

Reactivated and active mines. These have also their stories to tell. During their operating periods, many have not kept any environmental standards and have maintained no sparkling compliance records. Environmental studies even if they exist, are often incomplete, widespread or are in the form of uncontrolled mosaics and reconnaissance sketches.

The impact of mining is often not a W:al phenomenon. On the contrary, the effect that this hazard has might involve air, water and soil pollution, which might have a national, regional, and international character. To effectively map and apply environmental control methods, the potential hazards of each mining phase and their impact on the environment have to be understood.

Generally, each mining has its own local features, impacts and unique settings. Although it is always possible and easy to describe a generalised situation, each mine (whatever its status) has variable impact on the environment that requires differing mapping and monitonng approaches. Studying me impact of radioactive mines in general is not restricted to local situation alone. It can sometimes inflict larger area and health-wise anyone living in its surroundings (even when not using water, soil, etc.) contrary to sulphide mine which can sometimes involve local phenomena and where its impact on health is felt upon using the contaminant which is associated with it. Again, in most abandoned mines, original scenarios (before the mining starts) are hard to establish. For one thing there is no literature or maps available and no environmental impact assessment (EIA) was done prior to the activity. What went on İs therefore hard to establish.

In another situation, a reactivated mine might trigger the removal of old waste to another locality or pile İt İn the same area resulting in toxic chemicals polluting not only the rivers but also the soil. The landscape defacement might have gone unnoticed. Besides, the origins of the waste dumps created are hard to establish making it impossible to analyse the pollutant involved. The old contamination and those deposits outside the known mining sites are especially interesting since they might have already been covered by soil and vegetation and occasionally difficult to map.

3 REMOTE SENSING AND GIS TOOLS

Rational management of the natural resources and of the environment depends not only upon the wisdom of decision-makers but particularly upon the availability to them of necessary information. Traditional methods of storing information about resources and the environment and of retrieving it depend upon the use of maps. With most mining sites (abandoned, reactivated or active) İn the world: baseline datasets regarding the mine, mine operation, impact on soil, water, vegetation, flora and fauna, etc., is either missing or totally absent. When available are rather incomplete, inadequate or inaccurate for this purpose. Any study regarding past mining activities, extent and their current impact on the environment therefore has to start from zero. Remote sensing (e.g., airborne or satellite borne) is one of the best tool we have nowadays which can provide a matching spatial coverage. Nearly all are obtained in digital format. Even if they are not (e.g., standard aerial photographs) they can easily be scanned and incorporated as a tool into the interpretation processes. Modern image processing techniques of digitally formatted data are available m market nowadays, which can be used to enhance, manipulate, classify and subsequently interpret the data.

3.1 *Optical Remote Sensing*

Aerial photographs has been used since the early 20th century to provide spatial data for a wide range of applications. It is the oldest, yet most commonly applied remote sensing technique. *Nowadays,* almost all topographic maps are based on aerial photographs. The latter (taken either İn black & white or colour), because of its high resolution, also provide the accurate data required for many cadastral surveys, Earth resource and geo environmental surveys. Figure 2 represent the aerial photo of the Tharsis sulphide mine in Southwest Spain. Woldai & Fabbri (1998), using such photographs taken from different periods and at different scales were able to map the various land use, land cover, soil, vegetation, urban and infrastructure in the area in detail. Landslides, subsidence, mining waste dumps and other activities associated with mining hazard can easily and successfully be delineated and monitored.

Aerial photographs are taken in stereo, in which successive photos have a degree of overlap to enable stereo-interpretation and stereo measurements. Apparently, most mines are situated İn remote areas and aerial photograph covering such area is usually absent in many countries. Besides, running repetitive aerial photographic coverage of such area is always expensive to allow change detection mapping.

There has been phenomenal growth in the field of satellite remote sensing over *the last* two decades (refer to http '/«uelliies Melius se/ for further reading). While the importance of all existing Earth Observation Satellite (EOS) systems in mapping and zoning the impact of mining on the environment is acknowledged and can not be denied, in this paper, only the most widely used images are discussed.

Figure 2 Aerial photo coverage of the Tharsis sulphide mine in Southwest Spain The mine site for some years now is not in operation

Almost 30 years after the first EOS went to orbit, *Landsat MSS and TM* still remain the most popular and the most widely used remotely sensed data applied in many scientific researches and productions. Landsat MSS with its 80 m resolution comprise of four bands ranging from the visible to the near infrared region. Mining induced activities may affect large areas and in this respect Landsat MSS can be used to map such area. As is shown in Figure 3 however, the resolution been coarse, details might be absent.

Landsat TM on the other hand, has 30 m resolution and six broad spectral bands in the Visible, Near Infrared - NIR and Short Wave Infrared - SWIR region and a Thermal band of 120m resolution. The better resolution of TM (Fig.4) implies that better delineation and assessment of the features in mining induced activities is possible than with Landsat MSS. Besides, the compilation of resources and environmental maps up to a scale of 1:100 000 can be achieved with high reliability (Woldai & Fabbri, 1998).

__„_!__.!_'__________"._^ı„Ljıd4)ı n t -li • In • a in Southwest Spain. The very dark area correspond to mining dumps, open-pit mine and polluted lakes; dark grey to bushy vegetation (known as 'Jara' in Spain; grey corresponds to grass and white bare soil

rigure i. The ı ya ı Lanasat ı iw toand 3) data of tne ı naısıs area. Southwest Spain. The mining activities are much betler represented in this image than Figure 3. The change in land use is m a) where eucalyptus is growing as part of the afforestation programme in the area, b) represented the polluted lake showing different spectral signature than was possible m Figure 3 c) represents the bushy vegetation ('jara').

Woldai & Limptlaw (2000) used Landsat TM/MSS and aerial photographs taken at different time to monitor the changes inflicted by many years of mining activities in the Ndola mining area İn Zambia. By applying various image processing techniques including the Normalised Difference Vegetation Index (NDVI) to the images taken in 1989, 1993 and 1997, they were able to measure the presence of green vegetation, tailing dams and barren rock/soil including waterbodies and infrastructure (Limptlaw & Woldai, 2000). Besides, by including aerial photograph • of 1968 to the datasets, they were able to demonstrate the changes that have taken place in this area in the last thirty years (Woldai & Limptlaw, 2000). The degree to which spectral information derived from remotely sensed data (Landsat TM, MSS and aerial photos taken at different times) can be used to' extract land cover changes and contamination- from mining activities was also assessed by Woldai & Fabbri (1998) in the Tharsis mine area in Southwest part of Spain. By using Landsat MSS/TM and aerial photographs, all taken at different periods, it was possible to analyse:

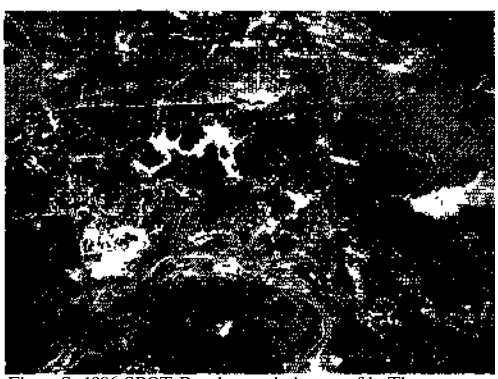
- Land uses changes associated with opencast mining.
- Evolution of dumping grounds.
- Delineation of the exact location of mine works, waste tips and land cover and land use changes.
- Landscape defacement as a result of mining activities.
- Barren wasteland.

The Thermal band, designated as TM6 can successfully depict higher temperatures related to underground coal fires (Prakash et al., 1995, 1999). Surface fires are nigh temperature phenomena, which show up also on short-wave InfraRed bands. By using a false colour composite generated by combining TM bands 7, 5 and 3 İn red, green and blue, respectively, they were also able to map surface fires and depict their aerial extent in the Jharia coalfield of India and North-west China. In an area like the Jharia coalfield (JCF), where extensive and rapid underground and opencast mining is going on continuously, land-use studies are of paramount importance. Prakash & Gupta (1998), on the basis image processed Landsat TM image data FCC of bands 4/3/2, 7/5/3, 5/4/2 (in RGB order) and ratio images was able to discriminate dense vegetation, sparse vegetation, fire, opencast mining (coal), overburden dump, subsidence and barren wasteland, settlement, transport network, river and water pond. Currently Landsat 7 ETM plus is added into the list with the same specifications as its predecessor, exception been, the introduction of an additional panchromatic band at 15 m resolution and an improvement of TM band 6 (thermal band) from 120 m to 60 m resolution.

Equally, various environmental analysts have used the *SPOT systems* (SPOT 1 and 2) for some years. The Panchromatic mode acquired in the visible region of the electromagnetic spectrum has a resolution of 10 m (Fig.5) while the M ulti-Spectral

(XS) has three bands in the visible to Near Infra-Red region.

SPOT 4, which is operational since March 24, 1998 offers a number of improvements over its predecessors; including the addition of a short wave infrared (i.e. mid IR) band. In order to allow on-board registration of all bands, the 10 metre panchromatic band on SPOT 1-2 has been replaced on SPOT 4 by 10 and 20 metre sampling of band 2 (red). The spectral bands measured by the HRVIR instruments have been carefully selected to match the SPOT missions requirements, particularly for: monitoring of crop and plant health, land use and land cover mapping, forestry, natural hazard and pollution monitoring, water resources evaluation, topographic and relief mapping, ecosystem monitoring, etc. In addition to the two high-resolution HRVIR instruments (each offering resolutions of 10 and 20 metres), SPOT 4 also carries the "Vegetation" wide-angle, medium-resolution payload offering a swath width of 2,250 km and a resolution of 1 km offering analysts with vegetation's excellent revisit capability. The stereo capability of SPOT panchromatic allows for precision mapping of various terrain features. It can assist in verifying the accuracy of existing maps and databases, updating other mapping information (e.g., vegetation, soil, hydrologie, settlement, etc.), serve as an up-to-date map for poorly mapped or unmapped areas and create highly accurate derived databases, such as land use or land cover maps



FigureS. 1986 SPOT Panchromatic image oftheTharsrs area (covering a window of Figure 3) showing the various mine dumps, open-pit mine, polluted lake and infrastructure

Chevrel & Coetzee (2000), using SPOT image and other data derived from various sources (e.g., digital elevation models, geology, airborne radiometry, data on water quality, etc.), were able to identify pollution, assess the environmental impact of mining activities and develop methods for assessing the risk and sensitivity analysis of surface and groundwater of the West Rand area - Gauteng Province, South Africa. Particularly by using digitally supervised classification of SPOT and Landsat TM images, they were able to classify the land-cover map, later recorded as an infiltration potential map. Accurate classification of the mining activity areas provided them with a map of potential pollution sources. Tailings dams were also easily identified and inventoried through a visual interpretation of SPOT image. The suitability of Landsat TM and SPOT Panchromatic data for assessing and representing changes in land cover (from 1986 to 1995) due to gold mining activities is also noted from the work of Ardanza (1999) working in the BOSAWAS Biosphere Reserve in Nicaragua. Ardanza İn his work was able to delineate several illegal mining. His research using SPOT and TM data revealed, the lost of 220 hectares of forest to illegal mining in about 10 years period.

In the list of Earth observation satellites used for mapping nowadays are the ***Indian Remote Sensing Satellites:*** IRS-1A (launched m 1988) and its identical follow-up 1RS-IB (launched in 1991). India's latest satellites IRS-1C and IRS-1D were launched in 1995 and 1996. In particular, the IRS-IC and IRS-ID introduced a heavier (1,350 kg), more capable Earth observation platform. The spacecraft bus is similar to those of IRS-1A and IRS-IB, but a slightly larger solar array generates more than 800 W. Both IRS-1C and ID produce 5.8-meter panchromatic (0.50.75 um - black and white) imagery, which is resampled to five-meter pixel detail, an improved spatial resolution than the SPOT panchromatic with 10 m resolution. The multispectral data acquired are comparable with Landsat's MSS and TM. These satellites are also equipped with two-band Wide Field Sensors (WiFS) that cover a 774-square-kilometer (481-square-mile) area in a single image, as well as LISS-3 4-band (0.52-0.59, 0.62-0.68, 0.77-0.86, and 1.55-1.70 urn) multispectral sensors that provide 23.5-meter resolution multispectral coverage. The 23.5-meter resolution imagery is resampled to produce 20-meter pixel detail. The spacecraft also carry a 2-channel (0.62-0.68 and 0.77-0.86 urn) wide-field sensor (190 m resolution).

3.2 *Microwave Remote Sensing*

For parts of the world, including some developing areas, that is habitually covered with cloud, the advent of airborne radar (Fig.6) and spaceborne radar (eg. Radarsat-1, ERS-1, JERS-1) has also provided excellent new tools for earth resources mapping, Unimpeded by haze and most cloud conditions and equally suitable İn day- and night-time, SAR could be used to survey, map and monitor various landuse and land cover activities relate to mining activities. A large archive of SAR data has been constructed since the launch of the European Remote Sensing Satellites *ERS-1 and 2* in July 1991 and April 1995, respectively, and this database is continually being updated now by ERS-2 with new acquisition. Colour composite created by combining ERSİ and ERS2 data acquired at different periods; may allow for delineating mine waste dumps, polluted tailings and hydrographie networks (Fig. 7).

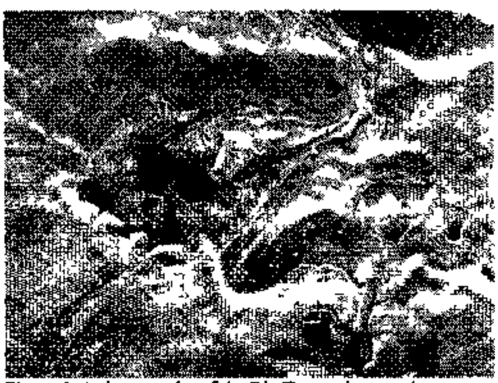


Figure 6 Airborne radar of the Rio Tinto mine area in Southwest Spain showing the different mine dumps.

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Similarly, the Canadian *RADARSAT* satellite launched on Nov. 4, 1995, is a powerful microwave instrument that can obtain high quality images of the Earth in all weather at any time. This provides significant advantages in viewing under conditions that preclude observation by aircraft and optical satellites. Using a smgle frequency, C-Band, the RADARSAT SAR has the unique ability to shape and steer its radar beam over a swath ranging from 35 to 500 kilometres with resolutions from 10 metres to 100 metres respectively. Incidence angles range from less than 20 degrees to more than 50 degrees The satellite's orbit is repeated every 24 days.

To protect but also exploit the forest areas in a sustainable manner during mining operations, radar surveillance is necessary. Radarsat data has assisted in mapping degraded and deforested areas around illegal mining activities from the Amazon in Brazil with success. It allowed spotting already rather small clearcuts. In this way every logging or deforestation activity can be controlled.

Pioneering studies so far carried out using SAR interferometry (InSAR) have already being earmarked as a new development in understanding our earth and its dynamics. InSAR can provide with unprecedented precision, high-resolution topographic data (DTM's using stereopairs of radar images with differing viewing angles). Differential interferometry allows one to measure surface movements with sensitivity of the order of a few centimetres over large surfaces (Massonnet et al., 1993). This technique has proven its importance in subsidence monitoring and landslides -- typical hazards often associated with mining activities. The ability to make these measurements from space without ground control is a fundamental advance in the management of natural disasters arising from mining activities.

3.3 *New Development*

Potential applications for one-meter satellite imagery in a GIS environment are limitless. Among the new development in Earth observation satellite data is the *IKONOS data*. The latter acquires panchromatic imagery with one-meter spatial resolution and multispectral imagery at four meters. With ground control, the imagery boasts a two-meter horizontal and three-meter vertical accuracy, equivalent to 1:2,400-scale map standards. The satellite's ability to swivel in orbit enables it to collect imagery anywhere on earth with a revisit frequency of just one-and-a-hal f days.

The imagery received from IKONOS can serve as an incredibly detailed basemap upon which other layers are laid, or it can be used as an up-to-date data source from which various land cover, soil degradation, mine wastes and other activities related to mining impact and elevation features are extracted to populate multiple GIS layers. IKONOS and aerial photography can complement the data sources that have traditionally served as sources of land cover and digital elevation models (e.g., Landsat, SPOT, and 1RS data).

Shuttle Radar Topographic Mapping (SRTM). In most developing and some developed countries accurate topographic and digital terrain model (DTM) of the area under investigation is missing. An exciting development towards solving this acute problem in GIS İs envisaged from the new SRTM acquired by Space Shuttle Endeavour in February of this year. The SRTM instrument will allow one to create very detailed topographic maps of the Earth's surface using interferomeöy. This radar system gathered data that will result İn the most accurate and complete topographic map of the Earth's surface that has ever been assembled. Once processed, the SRTM radar data will allow to obtain accurate knowledge of the shape and height of the land, and to assess: flood, soil degradation, deforestation and reforestation, landscape changes (due to mining activities), subsidence, landslides and most of all in monitoring land use and land-cover changes due to mining activities with high precision.

SRTM was launched into an orbit with an inclination of 57 degrees. This allowed most the Earth's land surface that lies between 60 degrees north and 56 degrees south latitude to be covered by the SRTM radar. This is about 80 percent of the Earth's landmass.

The *Advanced Spaceborne Thermal Emission and Reflection Radiometer (ASTER)* is an imaging instrument, which was built for the Ministry of International Trade and Industry of Japan İs flying on Terra, a satellite launched in December 1999 as part of NASA's Earth Observing System (EOS). ASTER will be used to obtain detailed maps of land surface temperature, emissivity, reflectance and elevation. The EOS platforms are part of NASA's Earth Science Enterprise, whose goal is to obtain a better understanding of the interactions between the biosphere, hydrosphere, lithosphère and atmosphere.

ASTER is the only high spatial resolution instrument on the Terra platform. It will be used with MODIS, MOPITT, MISR and CERES, which monitor the Earth at moderate to coarse spatial resolutions. ASTER's ability to serve as a 'zoom' lens for the other instruments will be particularly important for change detection, calibration/ validation and land surface studies.

The coming years will bring particularly exciting new, space-dependent, technologies to the service of the mineral exploration and mining industry. These will allow us to map the entire earth in more detail than ever before. Major new development in satellite remote sensing will be the products of *imaging spectroscopy or hyperspectral sensing*. The latter would employ from 100 to 300 contiguous spectral bands to record a "complete" and continuous reflectance spectrum for every pixel in the scene. This will allow one to determine the mineralogical composition of rocks and soils and some vegetation types as well as monitoring environmental processes and remediation and to locate waste products, such as sulphate minerals, causing acid mine run-off from mine tailings (Swayze et al., 2000). Hyperspectral sensing will bring the exciting new possibility of not just discriminating the material (possible with some current sensors) but actually identifying it and putting a name to the major mineral components present in every pixel of an image.

4 CONCLUSION

1. 30 years after the first Landsat went to orbit, remarkable advancement in spatial-, spectral- and temporal resolutions of the satellite data has been achieved. Spatial resolution has increased from a resolution of 80 m İn 1972 Landsat images to 10 m or 5 m of SPOT or 1RS panchromatic images and subsequently 4 m multispectral to 1 m panchromatic resolution of IKONOS-2 - the latter almost comparable to conventional aerial photograph. Such images with high spatial resolution represent a powerful Geo Information System (GIS) as they are, provided that there is sufficient good ground information with which to interpret them. In the case of IKONOS for example, the details and the amount of information that can be extracted has no limit making mining activities easy to monitor and timely assess wim high accuracy.

2. In most Developing Countries, topographic base maps and other maps related to environmental hazard are either missing or when available rather incomplete, inadequate or inaccurate for this purpose. Most of all, environmental maps are collected at different scales and wi± different legends by several actors making it impossible to reconcile and complement one another. Manual plotting of data on maps that exists lead to inaccuracies, inconsistencies between different organisations that plot the same data, and frequently this can lead to legal problems relating to such thing as the position of boundaries. The increasingly detailed digital elevation models (DEMs) coming from SRTM is at previously unheard of resolutions (e.g., I-5m) and something to look at to overcome this obstacles.

3. On the other hand, spectral resolution has allowed broadband spectral images to be reduced to hundreds of narrow bands in hyperspectral bands allowing for the first time an easy screening technique for potential acid mine drainage, for the discrimination of mineral types and chemicals often difficult to map using the other remotely sensed data (e.g., Landsat TM). Already the United States Bureau of Reclamation and the Environmental Protection Agency are using hyperspectral technology to guide them in their site characterisation and remediation activities.

4. In addition the improved temporal resolution of the satellite data has allowed for an increased frequency of observation and a better ability to observe changes at a point of measurements.

5. Satellite images and aerial photos, allow one to monitor mining event during the time of occurrence while the forces are İn full swing. The vantage position of satellite images and aerial photos makes it ideal to evaluate areas of historical mining operations as well as active mine sites to the extent of providing more sensitive, cost effective, and timely monitoring potential. Satellite images offer multispectral approach, synoptic overview and repetitive coverage. They provide very useful environmental information, on a wide variety of terrain parameters and range of scales.

6. Remote sensing data should generally be linked or calibrated with other types of data, derived from mapping, measurement networks or sampling points, to derive at parameters, which are useful in the study of mining hazards. The linkage is done in two ways, either via visual interpretation of via digital processing, manipulation and classification of the datasets. Modern image processing techniques are highly advanced and the software available to do that are readily available in the market nowadays.

7. The data required to understand the impact of mining from the environment is coming from different disciplines, which need integration in order to arrive at hazard map zonation. Data integration is the strongest element of Geo-Information System (GIS). The zonation of hazard might serve as the basis for impact assessment measures and should supply planners and decision-makers with adequate and understandable information within a relatively short period of time.

REFERENCES

Ardanza, Guilermo Chavez 1999. Mining activities in the BOS A W AS Biosphere Researve, Nicaragua: an assessment, unpublished *ITC MSc Thesis,* ITC. The Netherlands

Chevrel, Stephane & Coetzee, Henk 2000. A new tool for assessing the environmental impact of mining activities: Application to the surface and groundwater sensitivity analysis of the West Rand Area - Gauteng Province - South Africa. *Proc. 2Sih International Symposium on Remote Sensing of Environment and 3rd AARSE Conference.* 27'-31 March 2000, Cape Town, South Africa.

Limptlaw, Daniel & Woldai, Tsehaie 2000. Land use change detection as an initia) stage in environmental impact assessment on the Zambian Copperbelt. *Poster present. 28th International Symposium on Remote Sensing of Environment and 3rd AARSE Conference.* 27-31 March 2000, Cape Town, South Africa.

Massonnet, D., Rossi, M., Carmona, C, Adragna, F., Peltzer, G., Feigl, K. & Rabaute, T., 1993. The displacement field of the Landers earthquake mapped by radar interferometry. *Nature:* 364,138-142.

Poulin, R. and Sinding, K. 1993. Mining economics and the environment. *Natural Resources Forum.* Vol. 17(2): 157-163).

Prakash, A., Gens, R. & Vekerdy Z. 1999. Monitoring coal fires using multi-temporal nighttime thermal images İn a coalfield in Northwest China. *International Journal of Remote Sensing.* 20(14): 2883-2888.

Prakash, A. and Gupta, R.P.1998. Land-use mapping and change detection in a coal mining area - a case study of the Jharia Coalfield, India. *Internationa! Journal of Remote Sensing.* 19(3): 39M10.

Prakash, A., Saraf, A.K., Gupta, R.P., Dutta, M. and Sundaram, R.M. 1995, Surface thermal anomalies associated with underground fires in Jharia Coal Mine, India. *International Journal of Remote Sensing.* 16(12): 2105-2109.

Swayze , Gregg A., Smith, Kathleen S., Clark, Roger N., Sulky, Stephen J., Pearson, Ronald M., Vance, J. Sam, Hageman, Philip L., Briggs, Paul H., Meier, Allen L., Singleton, Michael J. & Roth, Shelly 2000. Using imaging spectroscopy to map acidic mine waste. *Environmental Science & Technolog}'.* 34(1): 47-54.

Woldai, T. & Fabbri, A. 1998. The impact of mining on the environment. - A Case Study From the Tharsis-Lagunazo Mining Area, Province Huelva, SW Spain. Workshop on Deposit and Geoenvironmental Models for Resources Exploitation and Environmental Security. *NATO Advanced Study Institute* September 6-18, 1998, Mâtrahâza, Hungary (In press).

Woldai, Tsehaie & Limptlaw, Daniel 2000. Impact of Mining on the Environment in the South African Development Community (SADC), Southern Africa (Issues and Approaches in Mapping with test case from Kitwe mine, Zambia), *ISPRS.* Vol. 32 part 6W8/2: 353 - 370, Ljubljana (Slovenia).

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Cyanides in the Environment and Their Long-Term Fate

B.Yarar
Colorado School of Mines, Mining Engineering Department, Golden, CO 80401. USA

ABSTRACT: A number of recent cyanide spills and tailings-dam failures have made world headlines. While hand cyanides are the primary chemicals used in the extraction of precious metals from low-grade resources, contributing substantially to the world economy, an appreciable anti-cyanide activism is in progress and search for alternative lixiviants is accelerating. Health effects of cyanide compounds other than free-cyanide are often insufficiently addressed or remain not required for reporting to regulatory agencies. This paper addresses cyanidation-extraction byproducts and their long-term fates in the ecosystem. Data show that a number of currently accepted cyanide abatement approaches can create long-lasting products that degrade at very low rates.

1 INTRODUCTION

The estimated amount of gold production in the world in 1998 was some 2,460,000 kg , more than 90 % of which was by cyanidation. The remaining 10% or so can be accounted for by gravity separations and such technologies (USGS, 1999; von Michaelis, 1984). Based on reaction (1) the molecular ratio of cyanide-to-gold consumption is 2:1 which works out as 0.498 kg of NaCN for each kilogram of gold recovered.

$$Au + CN^- \Rightarrow Au(CN)_2^- \qquad (1)$$

Table 1. World eold production usine cyanide; *basÉs for actual consumption ıs 0 4 kg NaCN/kg Au

Gold production	kg, NaCN equivalent needed		
	kg	Stoichiometric	Actual*
World production (including US production	2,460,000	1225080	984,000
US production	366,000	182268	146400
US production by cyanidation	341,000	169818	136400
US cyanidation, in tanks and closed containers	238,000	U8524	95200
US cyanidation in heaps and dumps	103,000	51294	41200

	NaCN Usage (kg /kg Au)
Stoichiometric kg NaCN/kg gold	0.498
Usage in Canada (average kg NaCN/kg Au)	0.450
Usage in S-Africa (average kg NaCN/kg Au)	0.280
Usage in Free world (average kg NaCN/kg Au)	0.400

Considering that some of the cyanide is recycled and some of it is lost to cyanicides, that is, cyanide-consuming-non-gold entities, the amount of sodium cyanide handled in the world for gold recovery comes out to be some 1 million kilograms per year. These numbers are summarized in Table 1 which excludes unrecorded numbers, likely, utilized in some developing countries.

Cyanide usage is not confined to the gold recovery industries but also includes metal recycling and metal plating and finishing industries. Notably, cyanide as NaCN and KCN, is a ubiquitous chemical used in the metallurgical industries since 1887. Flotation-concentration of Cu-Zn ores are also common users of cyanide as a modifying agent. Cyanides are also components of pesticide formulations and their degradation products and are generated by numerous plants and organisms as well (Towill, 1978; Huiatt, 1983). Yet, their high toxicity is firmly under control during usage. Control of pH and safe handling practices appear to be the sufficient requirements for safety in the workplace. Figure 1 is the speciation-diagram for dissolved HCN gas where it is seen that at pH >10 essentially all of the cyanide is in the ionic (CN") form. Under these conditions, in the plant, cyanide is not the "ogre" that it has been made to be, in recent years.

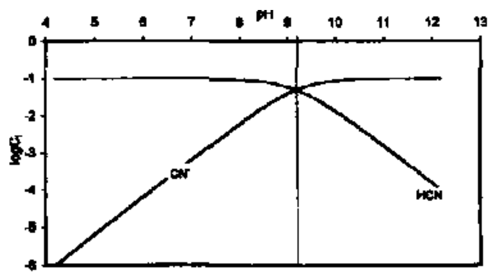

Figure 1. Speciation diagram for HCN in water (NaCN equivalent : 5 g /L)

Table 2. Some examples of recent high-profile cyanide accidents in the world.

Date	*Accident*
1992	Summitville, Colorado, USA; leaking mine abandoned, now a superfund site
1995	Omi gold mine, Guyana dam ruptures about 3.5 million tons of toxic waters are spilled with tailings.
1997	Nevada, gold quarry mine teach pad collapse releases 980 million tons of cyanide bearing slurry spills
1998	Kyrgyzystan-Issyk-Kul lake receives 1.7 tons of solid sodium cyanide from a truck overturn accident
1998	Homestake mining operation; Black-hills, S.D , USA 6 tons of cyamde-beanng tailings are lost to the environment.
2000	Baia Mare gold Mine, Romania, 100,000 tons of slurry is released into the rivers Lapus, and followed to Somes, Tizsa, and the Danube, from dam failure
2000	Papua, New Guinea, Dome Resources Freight helicopter spills some 100 kg of sodium cyanide during transportation

In this paper, the primary focus is on the use of cyanides as lixiviants in the recovery of gold from natural resources. In this context, primary problems associated with it arise, mostly, after usage as a leaching-agent, in tailings ponds, active and abandoned heaps and *at* disposal alter usage as a laboratory chemical and pilot plant material.

Numerous high profile and some secondary spills in the last decade have brought cyanide and its apparent perils to the limelight. A few of these are collected in Table 2. In all of the cases cited here, extensive efforts have been made for the remediation of the immediate damage to the environment with results considered satisfactory, while in some others such as the Summitville Mine İn Colorado large dollar numbers have been dedicated towards remediation efforts.

2 ENVIRONMENTAL CONTAMINATION

Potential mechanisms of contamination in mining and metallurgy related areas include the following:

1) During transportation of sodium cyanide to site of use
2) Tailings dam failures
3) Heap and dump failures
4) Seepage from heaps dumps and impoundments
5) Clandestine usage

Especially in the cases of dam failures and seepage processes, cyanides take a secondary seat to metals such as Pb, Hg, Zn, Cu etc. which gain special notoriety and exacerbate abatement and control costs. The Summitville Colorado, Superfund site is a case in point where the current effort includes extensive attention to heavy metals. The cost of cleanup for this site has been estimated to be in excess of $100 million.

The primary concern, in addition to the elimination of the immediate effects of such spills is the need to provide answers to a number of questions: i.e.: " what happens to the toxic material?"; " is it really rendered harmless and gone for ever?" and "where does it all go?"

3 ANALYTICAL REPORTING METHODS

Cyanide compounds associated with mining and metallurgy can be broadly grouped, (Scott, 1981) as shown in Table 3. The strength of the metal-cyanide complexes on the other hand İs better expressed in terms of their stability constants, that is, in essence their resistance to dissociation under ambient conditions. These values (recalculated after Scott, 1981 and Beck, 1987) are compiled in Table 4, where it is clearly observed that gold, cobalt silver and iron complexes are thermodynamically highly stable against dissociation.

%

Table 3 Examples of cyanide species in groups as used in assay and reporting procedures.

ÜEL	*Examples*
Free cyanide	CN",HCN
Simple cyanide compounds (some partly soluble)	NaCN , KCN, Ca(CN)î, $Zn(CN)_2$, $Cu(CN)_3$, Ni(CN)î Ag(CN)
Cyanide complexes	Weak complexes: $Zn(CN)^{\wedge 2}$, Cd(CN)"' Moderately strong complexes: Cu(CN)2 , $Ni(CN)*^{4}$", Strong complexes: $Fe(CN)_6$*", $CotCN^{\wedge 4}$", Au(CNV Ag(CN)j'

Table 4. Stability constants of some metal-cyanide complexes.

Species	*logio Ks*
$Au(CN")_4$	56
Au(CN>2	37.65
$Co(CN)_6^4$-	50
$Ag(CN)_2$-	39.1
FeCCNV'	35.4
$Cu(CN)_3^2$-	29.2
$Ni(CN)«^{2}$"	26
$Cu(CN)_z$"	19.95
$Zn(CN>4^2$-	19.01
$Cd(CN)_4^{2}$'	18.93
$AgCN_w$	13.80

For compliance with regulatory agency requirements (İn the USA) cyanides are reported in three forms:

a) Free cyanide, which includes HCN and CN'
b) WAD (weak-acid-dissociable) cyanide,
c) Total cyanide

It is also customary to determine and report metals such as Zn, Cu, Ag, Hg, Mn, Fe etc. together with pH and total dissolved solids.

As can be seen from Table 5, these techniques do not permit the reporting of all species among them cyanate (CNO') and thiocyanate (SCN") which are evidently not-negligible sources of toxic materials. Species missed by these techniques are considered innocuous although there is ample evidence indicating mat microorganisms, certain plants, crustaceans and other aquatic organisms such as fish and frogs are highly sensitive to their toxic effects.

Accumulation of cyanides is also known to occur in some plants and also fish. These include complexes and SCN* . Furthermore, not only fatal toxicity, but toxicity that reduces the agility of these organisms as well as their reproductive cycles should not be overlooked. Sometimes, low levels of ingested toxic concentrations, become lethal when temperature or oxygen content or heavy metals as well as nitrates and ammonia concentration in the aqueous environment change. Beside heavy metals such as Pb, Hg, Cu, Au, Co etc., sulfide species arising from the presence of sulfide minerals (e.g.; sulfide, poly-thionates, thio-sulfate and sulfate) can create thiocyanate (SCN") which is a reactive and persistent species of cyanide.

Persistent cyanide compounds present two types of hazard:

1 : They are constant sources of cyanide emission and
2: They act as sources of heavy metals which would otherwise have been dilute as they travel away from the source.

The stabilities of metal cyanides and complexes constitute a "hazard" because in effect, they act as repositories of cyanide until an opportunity arises for its release. Taking for example $Zn(CN)4^{2}$" with a dissociation constant of 1.02 $xlO^{"19}$. On its own in equilibrium with alkaline water it is an emitter of small concentrations of CN". In contact with a sulfide containing solution, in the same environment owever, it is liable to the formation of ZnS since the solubility product of ZnS equals 1.2 x $10^{"28}$ (Weast, 1979). The equilibrium thermodynamics of this system İs clearly in favor of the displacement of CN' by S^{2}". Ion-sensitive electrode measurements conducted at the author's laboratory show that such reactions occur at high rates.

Table 5. Cyanide types determined for reporting and species they may miss.

Assay type	*Reports*	*Test conditions*	*Misses*
Free Cyanide	OT, HCN	alkaline pH	Most compounds and complexes ofNi.Co, Au, and PGM
WAD-Cyanide	freeCN, HCN, compounds and complexes that break up at mild pH	pH ~ 4-4.5	Cyanates, SCN , complexes of Co, Ni, Au and Fe
Total Cyanide	Compounds and complexes that break up at pH £ 1 in hot water	prt< 1 in hot water	Cyanates, SCN\ complexes of Co, Nii or PGM

4 WHAT HAPPENS TO CYANIDE

Cyanides are disseminated and modified by two major routes: (1) Natural attenuation and (2) planned treatment procedures.

4.1 *Natural attenuation and breakup*

Processes such as evaporation, wind action, oxidation by oxygen from air, acidulation by COT / H2CO3 ; bacterial action, uv-radiation; dilution and attenuation by rocks and minerals, including adsorptions, precipitations and catalytic phenomena are operative natural processes. CO2 which results in HCN generation at shallow depths of disposal lagoons is the primary operator, accounting for about 90 of all cyanide loss.

Yet, the sum of these mechanisms has only limited effect on the total rate of natural de-cyanidation of process waters İn tailings ponds. Even then, the final products beside HCN, CNO", SCN" or NH3 are not environmentally-friendly. Furthermore, many of those processes that involve rocks minerals and dissolved ionic species create precipitates and adsorbed species that are longer-lasting than the original free-cyanide if it were to be treated.

On the optimistic side, İt appears that natural degradations in abandoned heaps can be utilized as a preliminary de-cyanidation stage followed by chemical destruction rinses of the residual cyanide. This is a promising prospect in the exploitation of natural processes to this end. Indeed, some test data indicate that an 84000-ton heap, after 3 months of abandonment retains about 11.5% of the applied cyanide. The calculated amount of total cyanide retained in the pores of this heap is almost 4.4 tons. What is more interesting is that after another 18 months, some 85% of the cyanide present had been naturally-degraded (Huiatt 1983).

Table 6. Natural cyanide attenuation processes and products.

ATMOSPHERE
Operators and reactants : Air currents, HCN, 0_2, $C0_2$ H_20, uv-radiation, air-bome-micro-organisms
Processes and Products : Dilution and dispersion processes and decompositions givingNH^, HCOO and Hi COj

WATER ENVIRONMENTS
Operators and reactants : HCN, CN\ O;, Metals (e.g.: Ni^2+, Cu^1*, $Fe^{1*'i+}$, etc.). S^{1}', microorganisms, mineral catalysts.
Processes and Products: Hydrolysis "HCOO ", NH/ Precipitations and complexations, e.g. Ni (CN)„', $Cu(CN)_2$" Fe(CN)fi "•'" SCN" and degradations by uv-radiation e.g. CN" ,CNO ,NH_3,HCO, NO, etc.

SOIL ENVIRONMENTS
Operators and reactants; HCN, CN", H_20, sulfur, micro-organisms and enzymes, metals, mineral catalysts
Processes and Products: Products, as in water-environments also numerous biodégradation products including NH_3, $C0_2$, CR,, SCN and MefCN), species

Table 7. Some methods used for cyanide abatement.

Effluent treatment method(s)		
Cyanide destruction by oxidation	*Cyanide destruction by photolytic methods*	*HCN vaporization reduced solution pH*
Direct radiation	Electrical potential	Hydrolysis, distillation
With Ozone	Electrical potential with chlorine in medium	AVR
With hydrogen peroxide		SO; /air
With catalysts	Chlorine/hypochlorite	Caro's acid
	Oxygen	
	Ozone	
	Hydrogen peroxide	
	Hydrogen peroxide plus Cu^{i+}	
	Hydrogen peroxide plus Castone (proprietary reagent)	

These natural processes and their products are compiled in Table 6. Thİocyanate, cyanate and metal cyanides in this table are the persistent ones which act, essentially, as constant reservoirs and emitters of cyanides and metal-ions.

4.2 *Planned treatment procedures*

The printed literature contains numerous cyanide removal and/or destruction memods (Davuyst, 1991 ; Botz, 1998; Robbins, 1996). These include :

Physical separations, biological procedures and chemical destruction methods.

4.2.1 *Physical separations*

Membrane filtration, electro and reverse osmosis, use of zeolites and ion exchange resins, or indeed adsorption on active carbon are typical methods of physical separation.

Ion exchange or reverse osmosis may be used as "polishing steps" for treated water. The problem is: they are complex and costly to operate and generate their own saline solutions which need special arrangements for disposal. Presently active carbon is a potential alternative when all else fails. Its additional advantage is that it removes tons beside free cyanide and by-products such as Pb, Cu, Hg and also WAD cyanides.

Admixture of metal salts or metal oxides, hydroxides or composite ores such as the iron-rich bauxite with contaminated solutions can remove cyanide by the formation of $K.3._4Fe\{CN)_6$, $Cu_2Fe(CN)_6$ or $Zn_2Fe(CN)6$ followed by filtration. Such substances during impoundment however, can decompose by the influence of solar radiation, bacterial action or catalytic effects and emit HCN.

These hybrid processes have not been demonstrated at industrial scale.

4.2.2 *Biological oxidation*

Micro-organism based oxidations can be represented by the following simplified equations:

$$CN^* + 1/2\ 0_2 \text{ [enzymatic catalysis] } \Rightarrow CNO \quad (2)$$

$$CNO- + 3H_aO \text{ [enzymes] } =^* NH_,^+ + HC0_3 + OH \quad (3)$$

Use of bacteria such as "Pseudomonas Pseudoalkaligenes" or "Bacillus Pumilus" isolated from mine-waters are known to degrade cyanide {Arps, 1994). One plant in the USA uses bacteria to remove cyanide and some of the heavy metals from the effluents of a metallurgical plant. In this practice a biological treatment by the *"attached* growth method" uses rotating biological contactors to facilitate the removal of cyanide, thiocyanate and some toxic metals from effluent water. The primary problems with this approach relate to the necessity of controlling temperature, bacterial nutrients such as phosphate and difficulty to maintain reproducible strains of bacteria.

4,2.3 *Chemical breakup methods*

Cyanide can be converted to other chemical compounds by numerous methods. A number of these are given in Table 7. Four widely-used industrial methods include the following: (i) Alkali chlorinarjon, (ii) Hydrogen peroxide treatment, (iii) $S0_2$ / air oxidation and (iv) Volatilization from acidified effluent solutions or slurries. Brief discussions of these methods are given below.

4.2.3.1 *Ozone treatment*

Ozone gas, is a strong oxidant and functions as shown in equations 4-6. It is used to a limited extent.

$$CN + 0_3 \Rightarrow CNO^- + 0_2 \quad (4)$$

$$3\ CN" + \Rightarrow 3\ OCN- \quad (5)$$

$$2CNO' + 30_3 + H_20 => 2HC0_3^- + N_2 + 3\ 0_2 \quad (6)$$

One property of ozone is that after the ininal formation of SCN in sulfur containing media, İt generates HCN from thiocyanate; it needs to be used at pH<l 1 and plus, it is a costly substance to utilize at the industrial quantities needed.

4.2.3.2 *Alkali chlorination*

This is a well-established method that uses chlorine gas (CI;), sodium hypochlorite (NaOCl) or calcium hypochlorite $Ca(OCl)_2$].

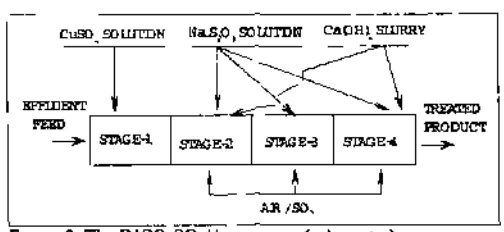

Figure 2. The INCO-SO_2/Air process (schematic).
pH=10, CaO 2-4 g/g cyanide, (SO_2/Air) · 2-10/100, SO_2 . 3-6 g/g cyanide, T=5-60 °C, Cu^{2+} = 0-50 mg/L

The basic reactions in this approach consist of a number of steps summarized in equations (7) and (8).

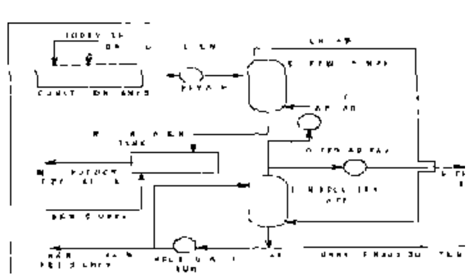

Figure 3 Scheme for the partial recycdng of cyanide by the Cyan l sorb technology.

$$2CN + 5Cl_2 + 10OH^- \Rightarrow 10Cl + 2HCO_3 + N_2 + 4H_2O \quad (7)$$

$$CN + NO \Rightarrow CNO + Cl \quad (8)$$

The problem with hypochlorite usage is thai it generates (CNO') which İs objectionable. Application of chlorine gas needs provisions for the capture of fugitive gas while sodium and calcium hypochlorites are water-soluble powders.

4.2.3.3 *Hydrogen Peroxide*

H2O2 as cyanide destructant, acts as follows :

$$CN^- + H_2O_2 \Rightarrow CNO'' + H_2O \quad (9)$$

$$CNO + 2H_2O \Rightarrow NH4^+ + CO_3^{2-} \quad (10)$$

Reaction (9) needs high pH, (which is already present in gold processing effluents) and Cu^{2+} ions to proceed, while reaction (10) occurs at slightly acidic aqueous environments, without *need* for cupric ions.

An alternative H2O2 - based reagent İs commercially available (DuPont) and primarily consists of a formulation of hydrogen peroxide with formaldehyde (HCHO) with some proprietary additives. The mixing of reagents is conducted at about 50 °C and the net reaction is given as:

$$HCN + HCHO + H_2O_2 \Rightarrow 3\ NH_3\ 5HCNO\ .\ 5H_2(OH)CONH_2 \quad (11)$$

The problem is that, not all of the CN" is removed from the medium. Additional treatment technologies are needed for the satisfactory de-cyanidation of effluents.

4.2.3.4 *SO2/air oxidation*

The use of sulfur dioxide-air mixtures in the presence of Cu^{2+} as catalyst forms the basis of the INCO (International Nickel Company) process (Robbins, 1996) where the reaction is represented as follows:

$$CN^- + SO2 + O2 + H_2O \Rightarrow OCN'' + H2SO4 \quad (12)$$

One of the arguments made against alkali chlorination (and in favor of the INCO process) is that if the effluent slurries contain iron arising from pyrite or pyrrhotite, ferro and ferricyanides are formed. These solids in turn can generate HCN when exposed to solar radiation at solid disposal sites. The INCO process is said to be among the ones that generate the least of harmful compounds, though, it may generate large quantities of $CaSO_4$ - rich sludge which adds to the cost of the process. A simplified flowsheet for the application of this technology is given in Figure 2.

There are more than 30 plants that use the INCO process in North America; worldwide, more than 50 projects are reported to have licensed this technology. The approach has been shown to be applicable to solutions, slurries and abandoned heaps (as heap rinse).

4.2.3.5 *Volatilization from acidic solutions*

The HCN speciation diagram given in Figure 1 shows that at pH<9.3, HCN gas is the predominating hydrolytic species. Thus the HCN gas can be driven off from the aqueous medium by distillation, or displacement by sparged air. Then HCN in the gaseous mixture can be captured by an alkaline solution such as NaOH or Ca(OH)î in a subsequent process step. This is in principle, forms the basis of AVR (acidification-vaporization-re-adsorption) technologies. A simplified flowsheet for the use of this approach is given İn Figure 3.

One potential drawback relates to me safety of operators of such plants because HCN gas which is highly toxic is present in concentrated form in the plant pipeline system though, no incident has been recorded in some 8 plants that have operated using this technology. A synopsis of

Treatment technologies with critical commentary are summarized in Table 8.

5 CYANIDE DESTRUCTION ECONOMICS

Two motivations that form the basis for cyanide abatement and control are obvious: 1) Recycling of a valuable resource and 2) Destruction of a toxic component added to nature by man.

One of the alternatives for the minimization of cyanide emission is to reduce its use whenever possible. For example in flotation technology where cyanide İs used as a selective-depressant for certain minerals such as copper or say, chromium-containing minerals sodium oxalate or sodium thiosulfate have been shown to be usable. Similarl in the recycling of scrap metal, or exposea photographic film, thiosulfate or nitric acid solutions can be readily used as alternative lixiviants.

If the economics of cyanide usage in a given precious metal recovery approach is considered successful, the problem then boils down to the toxicity of cyanide and its compounds and the resulting management of this situation. The toxicity problem can conceptually, be solved by three approaches two of which were cited above. "Replacement of cyanide by alternative lixiviants" would be an added method of cyanide abatement. (Yarar 1993; Yarar, 1999).

Table 8. Cyanide abatement and destruction technologies for CN^- - containing effluents.

Method	*Primary Mechanism(s)*	*Demonstrated scale*	*Advantages*	*Disadvantages*
Dialysis, E-osmosis	Membranes	Lab.	Future Potential	Cost; Maintenance
Ion Exchange Resins	Ion-exchange	Lab. ; pilot	Potential for cyanide recycling	Cost; Maintenance
Metal salts and/or Mineral powder addition	Precipitation, Adsorptions; Catalysis	Small scale and pilot	Capture of ions and suspended solids	Product solids removal and disposal; materials volume
Active carbon contact	Physical & chemical uptake; Catalysis	Lab. and small scale	Widely effective on ions and solids	Cost, Fouling of carbon; catalytic-byproducts
Ion or precipitate flotation	Surface chemistry of foams and solids	Lab. demonstration	Potential for treatment of large volumes	Large scale demonstration and; supplementary techniques needed
Direct and With O_3, H_2O_2 or Sensitizing *Solids* : ZnO, TiO_2	UV-and catalysis-induced redox	Commercially available small-scale	Demonstrated technology	Cost, UV radiation does not penetrate water, special reactor design needs.
Electrical potential	Oxidation	Lab and large pilot	Demonstrated technology, can be combined with O_2, O_3, Cl_2, or OCl^-	Potential for poisonous by products
Ozone	Oxidation of CN^-	Industrial	Cost -effective; minimal amount of harmful end-products	More effective at low pH, while CN-effluents are alkaline
Hydrogen peroxide	Oxidation of CN^-	Industrial, can be used with accelerators and catalysts	Cost -effective; proven technology	Cost of pH control when used with acids; Can produce nitrite and nitrate as byproducts
SO_2	Oxidation	Industrial	Proven technology can be accelerated with Cu^{2+}	pH-control needed; sludge formation and handling is problematic
Chlorine (Cl_2) and hypochlorite (OCl^-)	Oxidation of CN^-	Industrial	Versatile, usable at small and large scale	Minimal
H_2SO_4	HCN-generation	Pilot	Allows recycling of CN^-	
Bio-oxidation	Metabolic and enzymatic oxidation and metabolic adsorption	Industrial with some strains and pilot or lab. - scale with others	Partly proven technology, promising potential; partial removal of metal ions also occurs.	pH and nutrient control may be needed; biomass and bacterial strain control may create problems

The cost of conventional destruction technologies can be up to $1.50 per ton of ore treated. Recycling, provided by the AVR technology is an obvious abatement technique which is only partial at the present.

6 CONCLUSIONS

A study of the published literature on cyanidation-based metal extraction technologies and their cyanide-based products, allows the following conclusions:

1. Analytical data reporting İs not as best it could be because it omits numerous cyanide complexes and compounds.

2. It is assumed that free cyanide, in nature, breaks down completely to CO2 or nitrated compounds, which is not often the case. Compounds with heavy metals do form and resist destruction in natural environments for long periods of time. Cyanide compounds assumed to be "destroyed" or "not-present" are in fact, in place and are continually emitting harmful/toxic components.

3. Presently, a universal panacea to all cyanide-related problems does not exist although numerous technological approaches are available İn the marketplace and most commercialized cyanide destruction processes claim superiority to competitors, in some form or another.

4. Cyanide usage is not about to cease any time soon, nor is mere need for that although research is needed for a less controversial and equally-effective alternative lixiviant.

REFERENCES

Aips P. J. &. Nelson, M. G . 1994. Cyanide degradation by a bacterium isolated from mine wastewater: pseudomonas pseudoalcalİgenes (UA7), *SME Annual Mtg. Albuquerque , U.M. Feb. Preprint No.: 94-162*

Beck M. V. 1987. Critical survey of stability constants of cyanocomplexes. *PureandAppl. Ckem. 59(12). 1703-20*

Botz, M. C.&. Mudder, T. I. 1998. Cyanide recovery for silver leaching operations, application of CCD-AVR circuits, *Randol Silver Forum, Denver. Colorado: 5.*

Davuyst, E. A. & Robbins, (1991) G. cyanide pollution control-the INCO process. Randol-Caims Conf. Proceedings, Randol. III. Ltd. Golden, CO, Colorado. 145-147.

Huiatt, J. L. et. al. 1983. Proc. workshop on cyanide from mineral processing, *Utah MMRS Institute, Salt Lake City Utah.*

Robbins, G. H. 1996. Historical development of the INCO SO$_2$ / air cyanide destruction process. *CIMBull.. 89. (1003) 63-69.*

Scon, J. S. & J. Ingles, 1981. Removal of Cyanide From Gold Mill Effluents: *Proc. Can. Mineral Processors I3th Ann. Mtg.. Jan. 1981. Ottawa, Ontario.*

Tovill, L. E. et. al. 1978. Reviews of the environmental effects of pollutants: V.-Cyanide, *EPA/600/1-78-027. US-EPA, Cincinnati. Ohio.*

USGS, 1999. *Mineral industry surveys:* Gold, 1998 Annua! review. Pittsburhg, PA.

von Michaelis, 1984. Role of cyanide in gold and silver recovery. Cyanide in the environment, D. van Zyl (ed), CSU-Fort Collins CO, pp.51 -64.

Weast, R. C. (ed), 1979. Handbook of physics and chemistry. *CRC Press, Boca Raton Florida.*

Yarar, B. 1993. Alternatives lo cyanide in the extraction of gold. Precious Metals Economics and Refining Technology, J. p. Rosso, (ed) IPMI, Pensicole, Florida.

Yarar, B. 1999. Alternatives for cyanide in precious extraction and methodologies for the destruction of environmental cyanide. *Proc. International Meeting CAECO-99 (Central Asia Ecology-99) K. Karimov (ed.). Tamga, Issyk Kul, Kyrg}'zstan (in print).*

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Engineering Education at a Crossroad: An Integrative Approach

T.Ertekin
Department of Energy and Geo-Environmental Engineering, Penn State University, University Park, PA, USA

ABSTRACT: The engineering technology is a dynamic one. Therefore, the curriculum must continuously follow a suit. Tt must give the student a thorough background in the fundamentals with an eye on more challenging applications and it must engender sufficient versatility to permit the inclusion of subject areas that develop as the nature of the industry and its technology change. Creating a vibrant learning and teaching culture is admittedly an idealized vision that needs to be tempered with the reality of limited budgets, societal needs, and other pressure points such as employment opportunities and industrial demands. Nevertheless, great potential for a change of perspective exists within the universities if this vision of dynamic and broad-based learning and teaching culture is held up as a beacon by students to direct their learning and by faculty to direct their teaching.

1 INTRODUCTION

A modem engineering curriculum, beyond the traditional focus of a specific engineering discipline, must provide the graduating engineer with a working knowledge of thermodynamics, fluid dynamics, transport phenomena, numerical analysis, advanced mathematics, rock mechanics, safety, experimental methods, computer programming and computational devices, and information technology. The courses should be structured İn such a way as to meet this goal. The challenge is to incorporate this broadened working knowledge into an engineering curriculum without diluting the traditional emphasis of the specific field. Perhaps a vehicle to approach this challenging problem is the direct incorporation of some of this working knowledge into those courses where traditional areas and subjects are discussed. This strategy can give the engineer a greater versatility, in not only solving new problems in a specific engineering field but also solving engineering problems in general. Today, it is not uncommon to find a mining engineer modeling a contaminant transport problem both around active mine areas and in fresh water aquifers. Similarly, a drilling engineer can be found designing drilling programs for oil and gas extraction or for ore reserve estimates or for environmental remediation purposes or fresh water supply. A mining engineer can be found designing pipelines for water, gas, or coal slurry transport. The same engineer can be found designing a network system for information management. It is very clear that a modern engineering curriculum must be broad-based in such a way that all these various facets are accommodated

The current practice of engineering is a far cry from the days of slide rules, charts, and tables. With the quantum leap in computer technology, an engineer now has a very powerful tool at his/her disposal. With the strike of a few keys İt is possible to simulate various production scenarios of a plant. Voluminous amounts of information can be processed within a short period of time so that the engineers can quickly make decisions that may guide the overall project direction. Only a decade ago, this was inconceivable. However, inherent in every opportunity there is a danger. A curriculum can run the risk of over-emphasizing the use of 'canned' software packages to the extent of making the engineer a hostage rather than the prime mover. It is of paramount importance that a robust engineering curriculum equips the student with theoretical backbone as well as the mechanics behind the model. Thus, the engineer will be capable of properly interpreting the results generated. Furthermore, the engineer so trained should have little difficulties in implementing the same knowledge base to similar problems in other areas.

Why is a broad-based curriculum for today's graduating engineer advocated? Is it primarily because of the shift in the job market in whereby an engineer is expected to possess a good 'skill mobility*? Or, is it because the graduating engineer needs to be positioned in such a way as to be more marketable in our ever-changing economy? The answer to both questions is a resounding yes! A quick look at the changing nature, *albeit* structure, of the industry will shed some light on these issues.

2 CHANGING PHASES OF ENGINEERING PROFESSION

We do not have any choice. Change is all around us. Why are we so afraid of change? It was once said in a most elegant way: "It's not so much that we're afraid of change, or so in love with the old ways, but it is that place in between that we fear....It's like being between trapezes." (Ferguson 1986). Change is a fact of life. Engineers and the engineering profession are not immune. During recent years engineering has continued to evolve, resulting in what seems to be very different career trajectories for engineers than even two or three decades ago. Job change is much more frequent. Teamwork is the primary focus. Industry conducts less long-term, fundamental research. Engineers now spend much more time on management and other human resources related tasks. The workforce is more diverse and engineers today receive extensive continuing education and in house training.

What has remained unchanged in the engineering field is the design component It is universally accepted that the most important aspect of the engineering profession is the engineering process (sometimes called the engineering design process). If one were to review the engineering design process İn its entirety, immediately it would be realized that every aspect of the engineering profession had to be visited. The engineering design process will involve identification of customer need and opportunity, problem definition and specifications, data and information collection, development of alternative designs for various scenarios, evaluation of designs and selection of optimal design, and finally, the implementation of optimal design. The engineering profession over the course of years has completed a number of iterations on the design process, and one can trace the evolution of an engineering process through these iterations.

Engineering: where it has come from— Engineering_is as old as human beings and is concerned with everyday living, even survival, particularly in its ancient beginnings. It was once said that engineering represents the "desire of man and woman to harness and control the natural forces on earth to society's advantage—the wind, the seas, the tides, the soil, etc."

Engineering: where it is today—In engineering, yesterday's goals are still today's goals. However, today's engineers are working on problems of larger scales and greater complexities than hitherto. The significant increase İn population and resource use, greater awareness of our environment, and greater intellectual curiosity are the three main parameters that scale the size and degree of complexities of the goals of the engineering profession today.

Eneineenng: where it is going Today's challenges are tomorrow's opportunities and the engineer and engineering profession must rise to the occasion. Perhaps the most important challenge the engineering profession will be facing tomorrow is the presence of an information overload. At the touch of a button one can find millions of giga bytes of information. This information is available to everybody; thus, anybody can become a significant competitor. Therefore we will observe the emergence of a competition that will most probably become increasingly rougher. Yet, globalization efforts will have a compounding effect on the level of competition. The economic systems, eco-systems, and engineered systems will become more intricately tied together resulting in much larger systems. Therefore, tomorrow's engineer needs to be increasingly more adept in understanding the dynamics of interaction between these systems.

From this short discussion it is clear that engineering workforce and engineering profession are continuously evolving. It should be recognized that this evolution follows patterns that reflect the major shifts experienced in global economy.

3 CURRENT AND FUTURE FRAMEWORKS FOR ENGINEERING CURRICULA

The central goals of an engineering curriculum today and tomorrow should:

- provide students with skills to perform effective problem solving;
- assist the students to develop a logical thought process;
- introduce the students to basic engineering tools;
- increase students' spatial and temporal analysis skills;
- help students develop appropriate planning skills;
- teach the students how to read and/or interpret technical presentations; and
- help the students develop an ability to think both critically and creatively, in an independent and cooperative manner.

A typical contemporary engineering curriculum contains courses in general education, basic sciences, engineering sciences, and engineering design. Most of the materials covered in these different groups of courses are taught in a capsulated form. In other words, the existing interrelationships between these focus areas are not emphasized. In one classroom students leam how to solve a high order non-linear partial differential equation without realizing the equation represents various analogous physical phenomena in a variety of engineering fields. One or two years later, the same student in another classroom sees the same differential equation in a different form and even leams how to solve it using specialized software without recognizing that he/she is solving a partial differential equation that he/she had solved earlier. Similar examples can be found in fundamentals of ethics and engineering ethics or freshman physics laboratory experiment on equipotential contours and streamlines and transport problems in porous media. An effective integration of courses taught under different focus areas will result in the following enhancements to an engineering curriculum:

- students not only learn mathematics and science but also develop an understanding why they need to know it;
- it takes less time to cover the engineering material and the resulting time savings allow for team training and team development; and
- students develop a sense of community so that they regularly attend class, study in groups, and help each other.

It should be recognized that a vibrant curriculum with teachers and students energetically participating İn the learning process requires dedicated interaction between teacher and learner. Both quality teaching and quality learning require hard work, diligence, and major time commitment. Within the higher education institutions the focus has been essentially on improving the teaching process. The future curricula will have this attention redirected towards the students* role in the learning process by working on the question "what are the more essential elements required to learn?"

3.1 *Engineering versus science—a change of perspective*

Osborne Reynolds in 1868 said: "Science teaches us the results that will follow from a known condition of things; but there is always the unknown condition, the future effect of which no science can predict." Throughout the years engineering curricula always faced the challenge of striking a good balance between engineering topics versus science. The neophyte engineering student often asks the question why does he/she need to take the science courses if he/she is going to become an engineer. Osborne Reynolds' remark indicated that engineering is anchored on science.

Eric Laithwaite (1984) in his book Invitation to Engineering said: "An engineer İs a man who uses the earth and tries to capture the sun's energy more effectively. He controls the rate of destruction of matter and tries to find alternative sources of energy and new materials. He invents new shapes of matter and strives to improve the quality of life in whatever form he finds it."

If we attempt to combine Reynolds' and Laithwaite's remarks we will come up with the following deduction—scientists want to know the reasons behind the life and engineers want to improve the standards of life based upon what they have learned about it through science. This is why throughout the history of engineering education engineering and science have been hand-to-hand with some degree of variations in the relationship. Figure 1 shows the changes that have taken on the quality and the extent of relationship between science and engineering over a large time scale. In Figure 1, we see that engineering *comes* to an existence without using the scientific reasoning. Later, the engineering field recognizes the importance of science as science starts to push the envelop for the fundamental understanding of issues which start to appear in increasingly complex domains. At the present time, fields of science and engineering provide much needed feedback to each other. However, *the* presence of this symbiotic relationship does not necessarily imply that there is no polarity between the engineers and scientists. It is quite often possible to see the engineers with the perception that scientists have the tendency to become tangential to the needs of the society. At the same time scientists may think of engineers as narrowly focused practitioners because they are not dynamic in their thought process. This perceived controversy arises from misconception since in truth science forms the bedrock for engineering. However, the line of demarcation between science and engineering is opaque. This is why any effort that utilizes the synergy between science and engineering sets a trajectory that brings us to the solution in a much more efficient manner.

Figure 1. Engineering and science from isolated existence to a symbiotic relationship.

3.2 *The methodology of curriculum design*

Engineering is a field where one can experiment with creativity. In this field, the abstract ideas of the physical sciences and the insistent demands of society driven by a high-powered economy converge. In this conjunction the engineer is the synthesizer or the creator. The methodology of curriculum design has to honor this essential underpinning of the engineering field. It is a generally accepted fact that there is a mounting concern among the industrial organizations about the impact of traditional engineering education on the creative potential of future engineers. A lack of creativity is clearly problematic in a rapidly changing, technologically oriented world where generating new ideas is essential to survive. Therefore, as shown in Figure 2, the first step in curriculum design is the correct identification of the problems that are going to be addressed by the graduates of the engineering program. Obviously, the societal and industrial needs (which are fully linked to each other) have to be considered. In structuring the curriculum, the second step, the overall objective is to provide opportunities for students to learn meaningful concepts meaningfully. Here we identify three distinct components: (i) meaningful learning, where a person attaches meaning to the concepts under study, (ii) concept formation, where a learner organizes ideas and information to formulate new ideas and concepts, and (İÜ) problem solving, where an individual uses information and knowledge in various new ways to solve problems. While watching these three components closely, we note that these large scale adjustments in engineering education are put on new trajectories by the advancement of teaching and learning methods, controlled by the institutional resources, and watched by the accrediting bodies. Therefore, these existing external and internal forces will always provide the necessary and much needed boundary conditions. The implementation and evaluation phase constitutes the third step of the curriculum design methodology. The advisory board (ideally composed of industry representatives and academicians from other institutions), external examiners, visiting scholars, and feedback from the industry at large will provide the paths for infusion of ideas and much needed objective and constructive criticism during this phase.

3.3 *Attributes of an engineer and goals of engineering curricula*

The technical skills, inter-personal skills, and good citizenry skills constitute the three basic groups of skills that an engineer must gain during his/her engineering studies. These attributes of the engineer and goals of the engineering curricula must be in concert. Under the technical skills, the curriculum should provide opportunities to the student such that he/she is equipped with knowledge of mathematics and basic sciences, analytical and interpretive ability, empirical skills, system and process design skills, and problem solving skills. The communication skills, effectiveness as a team player, collegiality, and knowledge of contemporary issues will highlight the inter-personal skills that are acquired during the tenure of the student. Finally, the engineering curriculum must graduate engineers whose technical knowledge is tempered by professional ethics, professionalism, global perspectives, and environmental awareness which make the good citizenry skills.

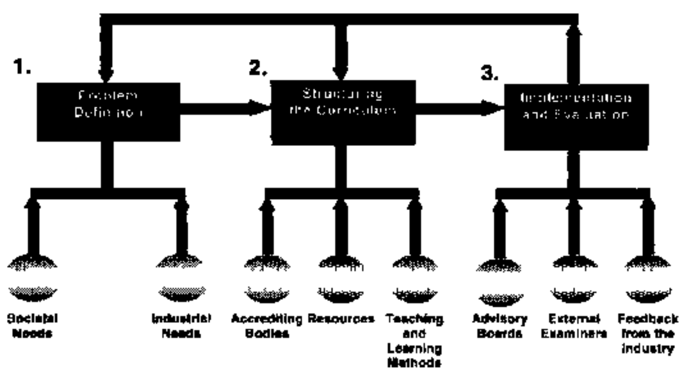

Figure 2. The methodology of curriculum design

3.4 *Framework of current engineering curricula*

The current engineering curriculum is designed to impart me attributes of the engineer as described in the previous section to its students. The framework for achieving this is well established as shown, in Figure 3. In Figure 3, engineering sciences, mathematics, basic sciences, and engineering design modules make up approximately two-thirds of the total number of credit hours covered in a typical engineering curriculum. Although it is very desirable to have all of the modules of Figure 3 integrated to each other, in reality today's engineering curricula follow a more or less fixed hierarchy. Most of the time, limited institutional resources will not allow each engineering discipline in the institution to redesign, optimize, and personalize the hierarchy shown in Figure 4 for each specific discipline. As a result of this, instead of an integrated curriculum, institutions end up with curricula in the form of a broken chain.

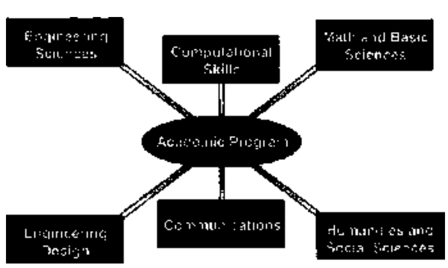

Figure 3. Framework of a typical engineering curriculum.

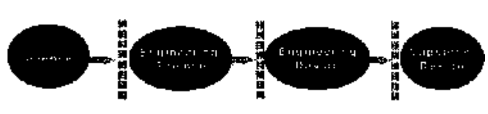

Figure 4. Broken-chain hierarchy of a typical engineering curriculum.

The following points capture the most salient weaknesses of a typical engineering curriculum:

- extensive compartmentalization;
- inadequate emphasis on communication;
- delayed exposure to the core curriculum;
- localized philosophy;
- inadequate emphasis on economics and management issues;
- intuitive rather than formal approach to ethics and professionalism; and
- overstrucfured and overloaded schedules do not permit independent learning.

3.5 *Winds of change*

In the previous section some of die well-known deficiencies of the current engineering curricula are highlighted. The good news is the wind of change is in the air. In this section we will look at the various forces dictating the changes.

Societal—One of the more effective factors forcing the change is society at large. The ever-increasing cross-cultural mobility and recognition of global dependence demand the removal of existing localized philosophy. Today's engineers are prepared not only to meet localized challenges but also towards the needs of the global village. It is also true that today's engineers are serving a society whose members are better informed and more demanding. Therefore, a greater emphasis on competitiveness and cost effectiveness will enter into the picture.

Industrial—The overall architectures of today and yesterday's industries in many respects do not conform to each otiier. In a traditional industrial organization profit was important, however, in today's industry 'profit' is always spelled in capital letters. In the past, industrial organizations have been known as insulated units working in their own spheres of interest. Today's industry, on the other hand, has to be more'competitive in a number of areas. While the hierarchical structure is a norm for the traditional industry, today's industry has a much more flat appearance. We also see a major philosophical change in 'the attitude of industry towards its employees. Traditional industry has always exhibited a parental attitude towards its employees. However, in today's corporate structure. we see a much more formal employee-emplmer relationship at every level of the organization. While traditional industry can be placed in the rich category in terms of its resources, today's industrial organizations, through its extensive restructuring efforts, surface as the ones with much leaner operations. During the last two decades, we also see that the traditional industry mat used to be thorough and perhaps a bit sluggish in its operations has assumed a much more rapid posture. In the traditional industry long-term projects were quite common, however today's industry is much more focused on current problems and issues. In consideration of all these changes, we see mat a much more hectic industrial work environment has taken over the yesterday's more stable traditional industry.

Technological—One of the more powerful winds of change is fed from the substantial technological advancements that we have been witnessing over the last two decades or so. The exponential growth in information technology, increased level of sophistication, information overload, and today's much more effective telecommunication all together place the industry on some new trajectories that were not in the radar scopes of the even most visionary organizations. The primary driver behind this forceful wind demanding changes in today's engineering curricula is the electronic explosion that we see in every facet of industry.

Measure of success—-As the societal, industrial and technological changes and developments dictate some pronounced changes in every aspect of the field of engineering, the yardsticks that are used to measure the level of success have to change as well. The perception that was used yesterday to measure the level of success has gone long and left its place to performance. Similarly, yesterday's "level growth" is replaced by "leverage" and so is the job security by employment potential. In the past the length of service was considered to be a powerful indicator of success, today, perhaps the "length of resume" is the corresponding new indicator. The degree of contentment that one was enjoying with his/her job is being changed by the degree of confidence. Perhaps, we can generalize these observations as the replacement of yesterday's romanticism with today's harsh realities.

Engineer—Whether one is graduated from the old school or the new school, there is no escape for the engineer from die effects of the forces summarized in the previous paragraphs. Yesterday's engineer who seemed to be more specialized is replaced by today's engineer who is expected to perform within a much wider spectrum of assignments. This is why today's engineer is in a more empowered status compared to the yesterday's engineer who would normally depend on his/her colleagues when the question Üiat needs to be answered is marginally out of his/her line of expertise. As a net outcome of this two statuses, we see that yesterday's comfortable engineer who felt more entitled and who is loyal to his/her company is being replaced by today's stressed engineer who is held more accountable and İn tum who has become much more loyal to himself/herself.

The question that still needs to be answered is how a new engineering curriculum can accommodate all these changes.

3.6 *Integration of research in undergraduate education*

At the present time undergraduate engineering students' exposure to research within the engineering programs across the world is quite minimal. Students who happen to gain some research exposure during their undergraduate studies are usually the ones who initiate this activity and are often the ones who are deemed most competent. There is no doubt that undergraduate students who join to a research group benefit from the experience in a number of ways including:

- developing domain expertise m science, mathematics, and engineering;
- gaining better appreciation as well as more sound understanding of the research process and the associated protocols and their implementation;
- enriching their decision making process especially at critical junctures;
- developing team member skills and appreciation for the teamwork; and
- becoming a more experienced technical communicator by writing technical reports and " making oral presentations.

There are several challenges in increasing the number of undergraduate research participants. First of all, funding is an issue for both research advisors and students. The rigid nature of the undergraduate engineering curricula also does not'permit for the student's participation in the research program for an extended period of time. Therefore, the mentor's investment on the student during the time the research program is initiated may be lost if the student does not have the flexibility to continue with the program for several consecutive semesters.

In consideration of the difficulties outlined above and the expected added values to the student's educational experience, faculty in their respective courses should provide ample opportunities to equip their students with some research experience. Engineers seek optimal solutions to problems. In making decisions analogical reasoning is at the heart of engineering thinking. By participating in research projects engineering students will receive training and be able to enhance their engineering skills in the use of analogy. This is a critical component of engineering education that should not be overlooked.

3.7 *e-Learning and engineering curricula*

Most of the e-learning is done at the graduate level education. This statement is also true for engineering education. By any measure e-leaming is booming, as it is becoming one of the fastest moving trends in higher education. Faculty members in different programs are putting their course materials in an electronic format and experimenting with various forms of interactive teaching forums such as web based learning and real time chat-room discussion groups. It is not a far-fetched imagination that within a decade or so all courses taught on-site will also be available on-line.

The burning question is whether the students learn as well on-line as they do on-site. Although only some relatively limited data (due to the relatively young nature of e-learning) have been collected, all of the data indicate that teaching and learning on-line does not dilute the educational experience of the students. However, it is also recognized that on-line learning requires more discipline and maturity than conventional on-site learning.

One of the questions that still needs to be answered in e-education is the question on the ownership of the intellectual property. Who owns the course taught on-line? The professor who designed it? or the school who helped putting it on-line by providing servers and/or courseware designers? While answers to questions like this are still being discussed, e-leaming will continue to become an increasingly attractive alternative to students both at the undergraduate and graduate levels. The engineering curricula should also be ready for this most significant transformation in the higher education institutions.

3.8 *Fully integrated engineering curricula*

A fully integrated engineering curriculum will respond to the external forces that demand significant changes in different components of the undergraduate engineering education. These changes will include (Everett *et al.* 2000):

- integration of basic sciences and mathematics into problem solving and engineering design;
- an increased emphasis on teaming and collaborative learning;
- use of computers to improve design and problem solving throughout the curriculum; and
- continuous outcome based assessment and evaluation methods.

Curriculum integration can be defined as the establishment of an educational protocol in which individual courses become integral components of a whole, while at the same time they are ensured to be interdependent with each other and are bound by a common thread of knowledge. Figure 5 schematically describes this educational protocol, One can visualize Figure 5 as a number of bowl-shaped water fountains that are concentrically placed within each odier. Water from the innermost fountain representing the basic sciences and mathematics cascades into the next fountain when it is filled up. The second fountain, engineering sciences in this case, while filled up from a direct connection to the main water line, It also receives the cascading water from the first fountain. This process in a similar manner continues as the water from the innermost fountains reaches the outermost fountains.

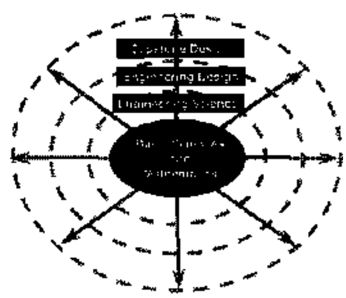

Figure 5. A possible structure for a fully integrated engineering curriculum.

In this process, it is important to note that water from the innermost fountain will reach the outermost fountain and will help having that portion of the fountain filled up. In this metaphor of cascading water fountains water represents the knowledge and information that are imparted to and utilized in the subsequent modules of the curriculum.

An integrated engineering curriculum as described here is expected to provide motivation for meaningful learning. It becomes more readily obvious to the student that mathematics and science are critically important to engineering as concepts learned during the freshman year are utilized in the first engineering course and the last capstone design course at an equal rate. A second advantage of an integrated curriculum lies in its inherent capacity for providing better control of the curriculum. In other words, concepts are taught In a much more umform manner and duplicative effort in teaching is minimized.

It is reasonable to conclude that a fully integrated engineering curriculum ensures that technical complexities are handled efficiently, and good problem analysis and problem solving skills are instilled to the students in an effective manner. Although these aspects of an integrated curriculum are necessary, are they sufficient to educate the engineer who is sought by the today's society and industry? Some of the missing links of an integrated curriculum as depicted in Figure 5 will include:

- flexibility in choosing courses from other disciplines;
- team experience and collaborative learning;
- written and oral communication skills;
- contextual perspective—can the engineer look at a problem In the context of a much larger problem?
- engineering ethics and professionalism;

- industrial safety and health;
- continuous learning;
- adaptability to broader spectrum of engineering problems; and
- decisiveness and judgment.

These aforementioned elements of an integrated engineering curriculum should be interjected into the curriculum at each level of the student's learning experience. In other words, our so called cascading water fountains should also be connected to other auxiliary water lines through which these equally important components öf the curriculum are introduced.

In the next and final section of tins paper, under the umbrella of integrated engineering curriculum, we will develop an inter-engineering disciplinary program.

4 SUBSURFACE ENGINEERING: AN ENGINEERING CURRICULUM FOR ACHIEVING SYNERGY

The concept of establishing an integrated undergraduate curriculum stems on the coexistence of Mining Engineering, Petroleum and Natural Gas Engineering, Geo-Environmental Engineering, and Mineral Process Engineering degree granting academic programs under administrative structure of the Department of Energy and Geo-Environmental Engineering (formerly known as the Department of Mineral Engineering) at the Pennsylvania State University.

4.1 *Motivation and driving force*

The shear diversity and complexity of the spatial domain called *earth* and of the potential for use of this domain for fulfilling human needs demand a broad-based view of the pertinent engineering activities anchored on a strong scientific foundation. This, no doubt, requires a broad and integrated multi-disciplinary approach. However, the present scientific culture for studying the earth and its resources is highly fragmented. While a number of disciplines focus on the very same spatial domain, they do not provide a coherent and integrated engineering education and research experience. The result of this fragmentation and rigid compartmentalization is a crop of young engineers and scientists with relatively narrow focus.

The approach being proposed under the umbrella *of subsurface engineering* takes a global view of the earth realizing that extracting mineral and other resources from it is only a subset of much broader set of activities all of which are interconnected. This is not only a radical philosophical shift in the way we think of the earth but also one which requires a new approach in the way we train engineers to think of it. If we treat the earth as a uniquely valuable resource that needs to be preserved and enhanced instead of being degraded by uncoordinated engineering activities, then optimal utilization of the earth can be achieved. It is envisaged that the engineers and scientists trained on this platform will be better equipped with broader and integrated thought processes and skills that will enable them to devise and build better engineering systems that satisfy a much wider spectrum of societal needs.

The driving force has three salient components whose resultant is the prime mover for tie new integrated engineering program proposed here.

These are:

- energy balance and society;
- environmental imperatives; and
- inter-/multi-disciplinary nature of the issues involved.

Whereas traditionally different groups of engineers focus on each of these forces without due cognizance by the others, the proposed approach will attempt to integrate all of them together with the eye on the prime mover. This is essential because the earth only recognizes the prime mover bufnot each of the components independently.

4.2 *Justification*

The fact that all human scientific and engineering activities are anchored by the earth is indisputable. The earth sustains the human being. However, uncoordinated human activities, whether on or in the earth, can only lead to a greater momentum towards non-substance of the earth. The challenge for the scientific and engineering community is to take an integrated view of the earth and its resources in such a manner that the parts will be greater than the whole. Although various disciplines are anchored on the earth, they tend to be compartmentalized as if there is no coupling between the various interest and activities. Since the sub-systems involved are physically coupled, it follows that the studies of these subsystems should take into account the interactions between them. This can only be achieved through a multi-disciplinary integrated approach.

4.3 *Vision, goal and objective*

<u>*Vision*</u>—-The proposed integrated curriculum aspires to shape future undergraduate and graduate education in earth related engineering through carefully orchestrated and fully integrated programs in subsurface engineering. This vision is anchored on the following premises:

- an engineered system that is based on a full understanding of the coupling with the other subsystems within the earth guarantees harmony between people and her environment; and
- engineers and scientists whose training and thought process emanate from an integrative approach to problem solving will become fully aware of the transportability of their knowledge based skills to a broad spectrum of problems.

Goal—The subsurface engineering curriculum focuses on the system of educating, training and research whose goal is to achieve optimal utilization of earth resources while preserving the pristine nature of the earth. Through creative and enlightening educational and research opportunities the program will promote the intellectual development of its students to improve the portability of their knowledge and skills. Along these lines, the specific goals are:

- to promote a full intellectual awareness of the interconnectivity between earth's subsystems and the corresponding engineered systems; and
- to produce a new breed of subsurface scientists and engineers whose understanding of the interconnectivity of the sub-surface sub-systems and the universality of the governing principles make them versatile and portable

Objective—The primary objective of the proposed integrated academic program is to construct a sound platform which will serve as a springboard from where the proposed integrative educational and research training programs In subsurface engineering can be launched.

4.4 *Barriers envisaged and catalyst needed*

It is certain that an ambitious initiative of this type will require a certain amount of activation energy to overcome the inherent energy barrier. As shown in Figure 6, it is quite probable that a high level of activation energy which includes:

- fear of moving away from traditions;
- uncertainty of success;
- threat to disciplinary individualism;
- unwillingness to share resources; and
- adapting to new realities

will be encountered.

It is strongly believed that the single most potent catalyst is the robust atmosphere of academic freedom. After all, the essence of academic freedom is to challenge established thought and traditions, and expand the knowledge envelope. It should not be forgotten that the primary motivation mat will propel this concept to a logical end for achieving the necessary synergy is the commonality of the parameters and the similarity of die systems currently being addressed within various disciplines.

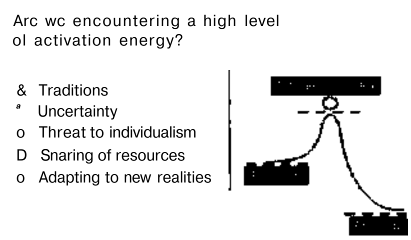

Figure 6. Potential barriers in structuring a new program.

4.5 *Framework/or the proposed multidisciplinary curriculum*

The practice of designing sub-surface engineered systems differs significantiy from that of traditional engineering systems. When designing sub-surface engineered systems one is invariably faced with far more unknowns than the number of relationships available. The challenge is always that the subsurface engineer is educated to work in any area in which all the answers are never known. That is why the subsurface engineer's education and logical minking must always be tempered with sound engineering judgments. In many cases, such judgments are emphasized. Perhaps, one of the reasons why the practice of subsurface engineering is partially shrouded mystery is a lack of understanding and accountability of interactions between the systems in question with other sub-surface systems. The framework of the proposed curriculum should be established on the aim of removing this shroud of mystery through integrated training and creative research.

Historically, the various disciplines that deal with some aspects of sub-surface engineered systems have operated within non-intersecting and non-interacting spheres, even when It is apparent that they may be addressing similar problems. This invariably translates into the training of their graduates to be narrowly focused. The proposed subsurface engineering curriculum is an attempt to redefine the needed intersections between the expertises of these various disciplines and map out the synergy needed to address the complex subsurface problems that must be solved. Inevitably, one realizes that the nature of interactions is strongly dictated by the problem at hand The major ingredients for the "dynamic positioning" needed to firmly anchor the new integrative education in subsurface engineering are subsurface domain characterization, the kinetics of the chemical activities, transport phenomena and geomechanics, economic risk factor considerations, impact of sub-surface operations on the environment and public policies. This *interdisciplinary synergy* and *intellectual cohesiveness* will shape the *modus operandi* of the subsurface engineering curriculum as shown in Figure 7.

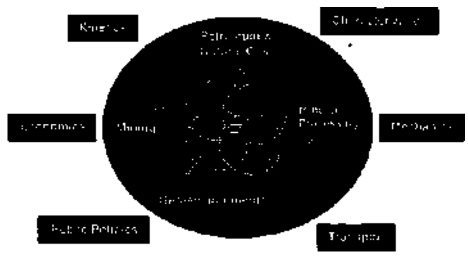

Figure 7. Intellectual cohesiveness and ingredients for a dynamic positioning.

4.6 *Typical thematic areas*

Within the framework of the subsurface engineering curriculum proposed, it is expected that engineers graduating from this integrated multidisciplinary program will be able to address problems and work on projects that span a wide variety of disciplines. A number of thematic areas is identified, examples include:

- groundwater resource management;
- solution mining;
- contaminant transport and control;
- waste storage and disposal
- coalbed methane recovery and containment;
- carbon dioxide sequestration;
- geothermal energy resource recovery;
- underground gas storage; and
- slurry transport.

The examples listed above to a certain extent epitomize the diverse and multi-faceted nature of sub-surface engineering issues. The cross-disciplinary challenge of addressing any of these issues is obvious. The subsurface engineer is not the one who is able to address one of these thematic areas in isolation but the one who is comfortable in addressing many of them and at the same time cognizant of the intricacies of their interconnectivity. Figure 8 schematically shows the thematic positioning of various engineering disciplines and their intersection on solution mining and coalbed methane recovery.

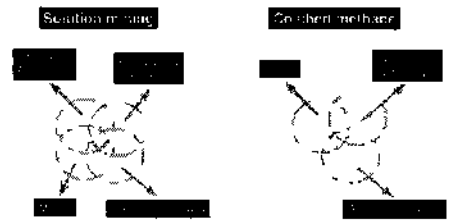

Figure 8. Intersection of traditional engineering programs on two interdisciplinary thematic areas.

4.7 *Process crystallization*

A broad-based engineering education for the subsurface engineer of the 21st century is necessitated by a need for skill mobility, flexibility and versatility. This is particularly important for the engineer to be well positioned in order to guarantee the continual marketability in the world's rapidly changing economical structure.

The colleges and universities who are best positioned to nucleate the subsurface engineering curriculum are the ones that have educational programs in petroleum engineering, mining engineering, geo-environmental engineering, geological engineering, mineral process engineering, and mineral economics. The necessary intellectual cohesiveness that emanates from these various disciplines will provide the scientific bedrock in the areas of exploration, characterization, mechanics, kinetics, economics, and public policies. In developing a new academic program in subsurface engineenng it is expected that the relevant faculty members from these various disciplines will form the core of this inter-disciplinary program so that no significant new faculty resources will be needed. Along the same lines, the existing instructional laboratories in these disciplines should provide the necessary infrastructure in fostering the objectives and goals of the subsurface engineering academic degree program as described earlier.

5 CONCLUDING REMARKS

In educating engineers we have come a long way and it has been a very successful journey. However, we should realize that we have come to a crossroad. It is now high time that we make a decision at this crossroad whether we should proceed on the same trajectory or make a departure. With the challenges of today, change is necessary to enable us to harness the opportunities of the future. An integrated engmeenng education anchored on strong scientific bedrock with an understanding of the social, cultural, geo-political context of our society and the globalization of the world economy provides the best competitive advantage for future engineers

ACKNOWLEDGEMENTS

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REFERENCES

Everett, L J, Imbne, P K. and Morgan J 2000 Integrated curncula purpose and design, *J of Engineering Education ASEE* (89)3,167 176

Ferguson, M 1980 *The Aquarian Conspiracy personal and social transformation in the 1980s* St Martin Press, Los Angeles, CA

Laithwaite, E R 1984 *Invitation to Engineering,* Bası! Blackwell, New York, NY

Reynolds O 1868 *The Progress of Engineering Considered With Respect to the Social Conditions of this Countn* Cambridge MacMillan and Co, London UK

1 f International Mining Congress and Exhibition of Turkey- IMCET 2001, ©2001, ISBN 975-395-417-4

Rock-Related Accidents, Investigations and Inquiries in South African Mines

K.S.Koldas
Department of Minerals and Energy Affairs of South Africa, Pretoria, Republic of South Africa

ABSTRACT: South Africa's mineral industry is supported by an extensive and diversified resource base. It has a mird or more of the world's reserves of alumino-silicates, chromium, gold, manganese, platinum-group metals, vanadium and vermiculate. This large reserve base allows it to play an important role in the world in respect of the production and export of many minerals and processed mineral products. Rock-related accidents have been a major problem in the South African mining industry for many years. According to 1999 statistics, 312 workers died in mines, in which 43% of the accidents were rockfall and rockburst related (rock-related). Although there has been a steady decline in accidents in the industry, rock-related accident rates are still far from being satisfactory. The author conducted 25 rock-related accident investigations and inquiries in various deep-level gold mines. The conclusions of the accident investigations and inquiries revealed that most of the accidents occurred as a result of a lack of support units in the working face area, poor mining practice, lack of hazard identification, poorly designed mining and support layouts and lack of strata control training for workers. The main objectives of this study are firstly to describe the importance of the mining industry for the South African economy and secondly to determine the causes of rock-related fatalities in gold mines.

1. INTRODUCTION

South Africa's mineral industry is an important contributor to the country's economy. In 1998, mining and quarrying contributed $6.6 billion, or 6.6% of the Gross Domestic Product (GDP), and $1.7 billion, or 8.6% of the Gross Domestic Fixed Investment (GDFT). The industry also contributed about 1.2% of state revenue. In 1998, South Africa's total primary sales were valued at about $12.8 billion, of which earnings from exports contributed 76.6%, while gold accounted for 44.2% of exports. Although primary minerals contributed some 37.2% of the country's foreign exchange earnings, the addition of processed mineral products would raise that figure to well above 50%.

2. STRUCTURE OF THE SOUTH AFRICAN MINING INDUSTRY

For more than a century, South Africa's mineral industry, largely supported by gold, diamond, coal and platinum production, has made an important contribution to the national economy. Furthermore, it has provided the impetus for the development of an extensive and efficient physical infrastructure, and has also contributed greatly to the establishment of the country's secondary industry.

The mineral industry is a well-established and resourceful sector of the economy. It has a high degree of technical expertise and the ability to mobilize capital for new development. It is recognized worldwide as a leading and reliable supplier of a large variety of minerals and mineral products of a consistently high quality. In 1998, some 55 different minerals were produced from 691 mines and quarries, of which 53 produced gold, 62 coal and 58 diamonds. Mineral commodities were exported to 80 countries. According to recent statistics, 470,372 people are employed in the mining industry.

The government's economic policies are based-on principles of private enterprise and a free-market mechanism. The system has enabled the mineral industry to develop without undue state influence, thereby allowing market forces to dictate the pattern of its development. The Department of Minerals and Energy Affairs (DMEA) is responsible for the administration of the Minerals Act, 1991 and Mine Health and Safety Act, 1996, which regulate the prospecting for, and optimal exploration, processing and utilization of minerals, and safety and health in the mining industry respectively. The DMEA's mission is to provide effective management of the mineral and energy industries for economic growth and development, thereby improving the quality of life of the people of South Africa.

The Chamber of Mines of South Africa is a private sector employers' organization with voluntary membership, founded in 1889, three years after gold was discovered on the Witwatersrand. The chamber is an association of mining finance companies and mines operating in the gold, coal, diamond, platinum, asbestos, lead, iron ore, antimony and copper mining sector.

The National Union of Mine Workers (NUM) is the biggest union in South Africa, and was formed in December 1982. The NUM is the largest recognized collective bargaining agent representing workers İn the mining and electrical energy sectors.

Table 1. South Africa's mineral industry reserves, 1998.

Commodity	World reserve %	World ranking
Alumino-silicate	37.4	1
Chrome ore	68.3	1
Gold	35.0	1
Manganese	80.0	1
Coal	10.6	5
Phosphate	7.2	3
Platinum	55.7	1

Table 2. South Africa's role in world mineral exports, 1998.

Commodity	World production %	World ranking
Alumino-sihcate	49.8	1
Chrome ore	28.9	!
Coal	11.9	3
Manganese	28.5	I
Vanadium	69.0	I
Zirconium	32.3	2

Table 3. Contribution of mining & quarrying to Gross Domestic Product (GDP), Gross Domestic Fixed Investment (GDFI) and state revenue of mining industry in RSA, 1995-98.

Year	GDP %	GDFI %	Total state revenue %
1997	6.5	8.7	1.4
1998	6.6	9.2	i.1

Table 4. Employment in the South African mining industry, i 998.

SECTOR	EMPLOYEES		£ Remuneration	
	Number	%	Rm	%
Gold	258815	55.0	9205	47.2
Platinum	89905	19.1	3474	17.8
Coal	60469	12.9	3532	18.1
Diamonds	14531	3.1	887	4.6
Other minerals	46652	9.9	2393	12.3
TOTAL	470372	100	19490	100

3. MINE HEALTH AND SAFETY ACT, 1996 (ACT NO 29 of 1996).

3.1 *Background to the Mine Health and Safety Act, 1996*

The unacceptably high number of fatalities and serious injuries in the South African Mining Industry was of grave concern to the government and subsequently, after a tripartite mining summit, the cabinet in 1993 approved the appointment of a commission of inquiry into health and safety in the mining industry. The commission recommended that the Mine Health and Safety Act, dedicated to regulating health and safety in the mines, be drafted. During 1995, the Parliamentary Mineral and Energy Affairs Portfolio Committee supported this recommendation and the cabinet approved the implementation of the recommendation.

3.2 *Nature and content of the Mine Health and Safety Act, 1996 (Act No. 29 of 1996)*

The act is dedicated solely to health and safety within the mining industry, which was not the case with the amended Minerals Act. The main objectives of the act are; -

- to provide for employee participation in matters of health and safety;
- to protect health and safety of persons at mines;
- to provide effective monitoring of health and safety conditions at mines;
- to provide for investigations and inquiries to improve health and safety at mines; and
- to require employers and employees to identify hazards and eliminate, control and minimize the risks relating to health and safety at mines.

Some of the important sections of the act are as follows; -

1. Health and safety İn mines concerning employer's duty.
2. Codes of practices.
3. Risk management.
4. Record of medical surveillance.

5. Health and safety representatives and committees.
6. Tripartite institutions.
7. Establishment of inspectorate.
8. Minister's power.
9. Legal proceedings and offences.

4 ACCIDENTS IN SOUTH AFRICAN MINES

The South African mining industry has for the past years been trying to reduce expenditure on all major and minor costs in an attempt to either keep profits up to acceptable levels, or in many cases, to prevent the mine from making a loss. Great pressure in the field of heath, and safety has also been exerted by unions, employers and state in order to bring about a significant reduction İn the fatality and injury frequency rates. Although there has been a steady decline in accidents in the South African mining industry, both in fatality rates and injury per thousand workers, the health and safety situation is still far from being satisfactory.

Table 5 Type of accidents ui South African mines, 1999

Type of accident	Fatality	Injury	Total accidents
Rock-related	137	1517	1529
Machinery	11	294	303
Transport	62	1103	1158
Electricity	8	37	41
Fires	0	3	3
Explosives	5	26	28
Subsidence /caving	0	1	1
Heal sickness	2	8	10
Gas/fumes	28	54	39
Conveyance	8	43	43
General	45	2402	2443
Rate	0.77	13 42	
Total	315	5488	5598

General, miscellaneous, occupational diseases, diving sickness, inundation, struck by objects etc

Table 6. Total accidents in South African mines

	1995	1996	1997	1998	1999
FATAL	533	463	415	371	315
INJURY	7717	7426	7095	6064	5466

Table 7. Total accidents in specific mines, 1999.

Year	1997	1997	1998	1998	1999	1999
	I	F	I	F	I	F
Gold	5707	277	4650	252	4202	213
Coal	270	40	257	42	207	28
Platin	755	53	785	44	766	39
Other	363	45	372	28	291	35
Total	7095	415	6064	366	5466	315

Table 8. All mines rock-related accidents, 1999

	1995	1996	1997	1998	1999
Fatal	222	247	192	181	137
Injury	2102	2184	2012	1819	1517

As it can be seen m Table 5, the most significant problem in the mining industry is rock-related accidents such as roekburst and fails of ground. Table 7 also shows that most of the accidents occur in gold mines.

In this research, most attention will be focused on the South African gold mines, which not only has higher total casualties than other hard rock mines, but also experiences more severe hazards, notably a high incidence of roekburst in deep-level gold mines. The author conducted 25 rock-related roekburst and fall of ground accident investigations and inquiries in order to determine the causes of these accidents.

4.1 *South African Gold Mining Industry*

Gold is synonymous with South Africa. Approximately 31% of the world's gold has been mined in the country over the past decade. Today, the gold fields form a discontinuous arc 430 km long, extending through the Gauteag, the North-west, the Mpumalanga and the Free State provinces. In 1999, 475 t of gold (at 5-6 g/t average grade) was produced by primary gold mines, tailings re-treatment operations and as a by-product of the production of other metais.

Gold mining m South Africa, from its humble beginnings in the first recorded mine in Eesterling in the Northern Province in 1871 to its pre-eminence as the largest gold mining industry in the world, has played a significant role in the economic development of the country over the past 120 years Through gold minmg, many towns and cities have come into being. One notable example is Johannesburg. Much of the İnfrastructural development of roads, electricity generation, water reticulation, telecommunications, housing and the development of industry to provide input for the gold mining industry have resulted directly from gold mining.

Most gold mining companies exploit more than one reef-vein in the Witwatersrand Supergroup. Further exploration, although at a reduced level, is expected to ensure that recent production levels are maintained for at least the near future. Precise age estimation in the Witwatersrand Basin is difficult since the rocks were deposited by sedimentation approximately 2700 million years ago, before the age of fossils. Experts believe that a great inland sea existed in what are now the Highveld and the Free State plains. Successive layers of conglomerate containing pebbles and gold were washed down into the sea and spread over the bottom by wave action. The gold particulars subsequently settled in successive layers of pebbles along the shoreline of this sea, which later silted up.

4.2 *Mining gold m South Africa*

South Africa's thin but extensive gold reefs often lie several kilometers beneath the earth's surface and usually slope through the ground at up to 25°. The country's gold mining industry has to sink the deepest mine shafts in the world, sometimes close to 4 km in depth, in order that miners can reach and extract these reefs. The mining method in deep gold mines is the 'longwalT mining method and the mining operation is carried out in hard brittle quartzitic rock, often at extreme depth. The great bulk of this rock mass behaves elastically, but stress concentrations around the excavations cause stable as well as sometimes unstable fracturing to take place. The vertical component of the virgin stress in South African mines tends to increase with depth approximately 0.027h(MPa) rock density 2.75 t/m^3 in each meter. The rock temperature can reach up to 50 C°, and this necessitates the use of ice and refrigeration facilities.

Today, with the tremendous pressure on profit margins in the gold mining industry, which is mining steadily declining grades at ever-greater depths, there is more emphasis on mechanization then ever before. Among the many aspects of mechanization, which are the focus of ongoing research, are technologies like trackless mining, backfill, non-explostve breaking and hydropower.

On average, only 5 ppm of every ton of ore mined is actually gold. It is, therefore, necessary to separate the precious metal from more than 100 million tons of ore milled each year in South Africa. The carbon-in pulp (CIP) method, which is increasingly widely used, makes use of the tremendous physical affinity "activated" carbon has for gold, which it readily attracts to its surface in cyanide solution. After smelting, which takes place in individual mines, bullion bars containing about 85% gold are then taken to the Rand Refinery near Johannesburg and processed to either 99.5% purity or 99.9% purity to meet specialized demands from certain industries.

Despite the fact that the gold industry's contribution to mining and the economy has declined, it remains a vital sector in the South African economy. Gold mines provided 211,200 direct job opportunities in 1999. Moreover, through its links to the domestic economy, about 220,000 additional jobs are also maintained in the rest of die economy. The direct contribution in terms of salaries and wages amounted to $1.4 billion during 1999.

Table 9 clearly indicates that South African is a pioneer in world gold production and the country will remain a world-class gold producer in the 21$^{s'}$ century.

Table 9. Annual world gold production by country and region, 1996-1998 - m metne ions.

Country /Repon	1996	1997	1998
South Africa	494.6	492.5	464 4
USA	329.3	359.0	364 4
Canada	1639	168.3	164
Oceania	305 6	329 2	325 0
China	144.6	153 0	161.0
Russia	132.8	138.0	127 3
Peru	64.8	74 8	89 2
Brazil	64.2	59.1	55.4
Chile	56.4	52.9	46.7
Other Asia	229.7	246 0	258.7
Other America	188.4	126.3	144.6
Other Africa	129 1	144.5	169.4
Other Europe	300	32.9	33 6

Table 10. Total labour force in South African mines.

MINE	1997	1998	1999
GOLD	292 HO	235940	211200
COAL	55297	57585	54820
DIAMOND	14274	14903	14537
PLATTN	80164	81734	85921
OTHERS	56193	55775	56919

4.3 *Accidents in South African Gold Mines*

In 1999, 213 fatalities occurred in gold mines in which 43% of these fatalities were rock related.

Accident statistics in South African gold mines cannot be compared with those of other countries for die following reasons:

1. Most of the gold mines are currently working 2500 km below the surface.
2. Deep mines are subject to very high states of rock stress, causing seismicity and rockbursting.
3. In all deep mines, the exposed rockwalls are highly fractured, which results in fall of ground accidents.
4. The heavy faulting encountered in many deep gold mines generates strata control problems and seismicity.
5. Gold mines in South Africa are the largest employers in the industry and employ more than 210000 workers.

Table 11. Number of rock-related fatalities in South African gold mines.

ACCIDENTS	YEAR			
	96	97	98	99
GRAVITY	125	73	81	68
ROCKBURST	65	80	62	41
TOTAL	190	153	143	109

In South Africa, rock-related accidents are classified in two groups. - Firstly, *gravity accidents* are accidents which occur mainly when an unsupported rock or portion of fractured rock falls in the working environment. Secondly, *rockburst* accidents occur as a result of stress-strain build up in the rock face or geological discontinuities such as faults or dykes, sometimes causing fatality and/or damage to underground workings.

In gold mines, rockfall and rockburst accidents represent the most significant cause of all fatal accidents. An important finding of the investigations here was that 72% of all rock-related accidents were seismic, and 28% of them were gravity-related accidents. In 25 accidents, the total deaüı toll was 38. The most important finding of these investigations was that all gravity-related accidents could have been prevented if the support had been installed prior to the accidents. Accident investigations and inquiries also revealed that most of the damage mechanism of the seismic-related accidents could also been minimized if the design of the mining layout and support had been adhered to by the production staff. Another important finding of these investigations was that most of the seismic-related fatalities occurred between the support units due to seismic shakedown.

5 ACCIDENT INVESTIGATIONS IN MINES

5.1 *Accidents to be reported*

Whenever an accident results in the death of any person or an injury to any person likely to be fatal as a result of rockfall or rockburst, the manager of the mine shall give notice thereof to the inspector of the DMEA without delay.

When an accident causes the immediate death of any person(s) as a result of rockfall or rockburst, the place where the accident occurred cannot, without the consent of the inspector of the DMEA, be disturbed or altered before such place has been inspected by an inspector, unless such disturbances or alteration İs unavoidable to prevent further accidents, to remove corpses and injured persons or to rescue persons " from danger, or unless me discontinuance of work at such place would seriously impede the working of the mine or works: provided that should an inspector assigned by the Chief Inspector fail to attend within three days after notice of the accident has been given, work may be resumed at the working place concerned.

5.2 *Accidents to be investigated*

In terms of the Mine Health and Safety Act, 1996, (Act No. 29 of 1996) the Chief Inspector of Mines of the DMEA instructs an inspector to investigate any accident or occurrence at a mine that results in any death, serious injury, any occurrence, practice or condition concerning the safety of person(s) or any actual or suspected contravention of, or failure to comply with, any provision of the acf.

5.3 *Initiating accident investigation*

The purpose of carrying out investigations is often poorly understood by all the parties concerned. As a result, they can degenerate into finger-pointing, blame-fixing and fault-finding exercises, which seldom determine the actual causes of the accidents. When the purpose is poorly defined, investigations are often poorly done. The purpose of the accident investigation should be:

- to determine what actually happened in order to prevent conflict,
- to determine the real causes jn order to develop corrective/remedial actions,
- to demonstrate concern in order to show the importance of the employees in mines.

All rock-related accidents must be investigated by an inspector of the DMEA, who is a specialist in the field of rock mechanics." Only a specialist and well-trained inspector can determine the real cause of the accident. An incompetent inspector always causes conflict during an accident investigation or inquiry. The inspector must also be well acquainted with all die legal aspects of the Mine Health and Safety Act and Regulations.

When a rock-related accident occurs in a mine, the responsible manager reports the accident to the DMEA for in-loco inspection (accident investigation). It is the responsibility of the inspector to obtain all me details concerning the accident prior to me accident investigation. The information that is obtained form the mines must also be reported to the Chief Inspector immediately. It is always good practice to conduct in-loco inspection on the same or next day, as the working place may be under die influence of potential instabilities such as seismic activity.

All the responsible mine staff and trade union representatives should accompany the inspector of the DMEA during the accident investigation. The investigating officer should also inform people of the purpose of the in-loco investigation prior to the investigation.

The following methodology is recommended for the investigating officers for an effective investigation: -

1. Establish a small leading group. •
2. Take many photos at the accident scene.
3. First determine the location of the injured or deceased.
4. Measure support distances, size and excavation height at the accident scene as well as non-damaged area.
5. Determine the damage mechanism of the event in the excavation and on the support units.
6. Take some rock samples.
7. Determine the geological discontinuities around the accident scene.
8. Observe fracturing at the accident scene.
9. Continuously question all the witnesses and responsible persons.
10. Do mt allow anybody at the accident scene due to danger.
11. Discuss all sub-standard acts and conditions with die manager and request remedial actions.
12. Thank all people concerned for their valuable contributions.

After completing the in-loco inspection, the responsible inspector must also issue a notice of instruction concerning the accident scene and determine a date for an inquiry.

5.4 *Accident inquiries*

The Mine Health and Safety Act, 1996 makes provision for the conduct of an inquiry into any accident or occurrence at a mine that results in die deam of any person. The Act also provides uSat a person questioned at such an inquiry has die right to be protected against self-incrimination and die right to be represented.

The purpose of the inquiry must be the following:

- To develop control and make recommendations in order to prevent the recurrence of a similar accident.
- To define risk in order to determine frequency and consequences.
- To enable the Attorney General to decide whether a prosecution should be instituted.

In terms of me Act, all inquiries must be held in public. The person .presiding at an inquiry may of that people's own accord or at the request of a witness exclude members of the public from attending the proceedings. The people entitied to participate are:

- Any person who has a material interest in die inquiry.
- A representative of any registered umon with members at the mine.
- Any health and safety representative responsible for the working place.

The person presiding at die inquiry may -

- instruct or summon any person to appear at any specified time and place,
- question any person under oath or affirmation,
- instruct any person to produce a book, plan, record or other document or item necessary for the purpose of the inquiry.

Witnesses and persons called to produce documents or other items have the same rights as they would have in a' court of law. Witnesses may therefore refuse to answer any incriminating questions. The presiding officer should always advice a witness of his/her right to so refuse.

It is recommended that the following documentation should be taken by me presiding officer at all times:

1. Statements from all the legally appointed individuals.
2. Details of the mine and accident place.
3. Plan of the accident area.
4. Particulars of the people present at die inquiry.
5. Particulars of the deceased.
6. Training record of the deceased.
7. Post mortem report.
8. Related codes of practices.
9. All rock-related risk assessment modules.
10. Safety statistics of the mine.
11. Logbooks of the responsible production staff.
12. Seismic history of the accident area.
13. All numerical modeling works concerning the accident area.
14. Minutes of all production and seismic review meetings concerning the accident area.

5.5 Inquiry reports

A person presiding at an inquiry must -
- record the evidence given at the inquiry;
- at the conclusion of the inquiry, prepare a written report of the findings, recommendations and any remedial steps;
- submit a copy of the report and the record of the inquiry to the Chief Inspector of Mines;
- supply a copy of the report and the record of the inquiry to the employer, Attorney General and on request to any person who has a material interest in the inquiry.

6 SOME STATEMENTS *WHICH* WERE TAKEN IN ACCIDENT INQUIRIES

Many pathetic statements have been taken from witnesses and responsible persons during inquiry proceedings. Here are some examples: -

Need 1 : All code of practices for stope layout design should be properly and clearly defined. It should be based on rock mechanics principles. Production and rock mechanics matters should be separated.

From a mine's code of practice:

"Standard size for up-dip mining to be 11.2m at all times, but wider up-dips only to be established as per section manager's discretion. "

Need 2: Supervisors often have a workload that is too heavy and cannot properly supervise their working places. The supervisors need very close contact with the crews under their care.

Question: "How many working places were you in charge of prior to the accident? "

Answer: "3 Panels in Level 31, 3 Development ends in Level 32 and 3 Panels in Level 33. "

Need 2: Do not pass the buck. Stop blaming others.

Section Manager:

Question: "Why was the bracket pillar mined out prior to the accident? "

Answer: "I really do not know The rock mechanics officer would be the best person to answer this question. "

Need 3: Think before talking. Don't escape from the fact. Seisimicity in deep-level mines is a reality.

Rock Mechanics Officer:

Question: "Did you consider that this working place was in a seismogenic region? "

Answer: "No, I did not. "

Question: "During the past 3 months, how many seismic events occurred in the area where the fatality occurred?

Answer: "I486events."

Need 4: Everyone needs training.

Team Leader:

Question: "Did you ever think that you needed to be trained in the field of strata control? "

Answer. "No, I never requested training. I am a big man. "

Need 5: The training officer's duty is to train the workers in the mine. The identification of job-related hazards is his/her main responsibility.

Senior Training Officer:

Question: "Didyou train the in-stope team members in mechanical prop installation and removal procedures?"

Answer: "They normally learn it by themselves. "

Need 6: All mines need outside consultation for the safe operation of the mine. Consulting companies should not regard safety matters as money-making business. They are also responsible for the safe design of the works.

Question: "Did the consulting firm ever visit the area for the applicability of their work? "

Answer: "No, they did not. They sent us a quotation. "

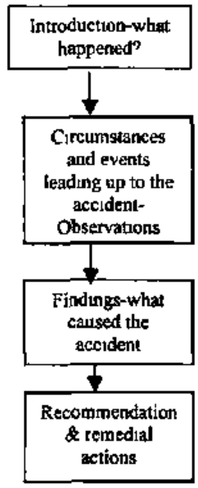
Need 7: Inspectors must be impartial during the investigations and inquires. (Please note that this statement was observed during another inspector's inquiry).
Question by manager to inspector for clarification:

Question- "Can you tell us the purpose of leaving crush pillars along the old working area? "

Answer: "I think you and I must discuss this matter outside. "

All these quotations above, taken from witnesses following falls of ground and rockburst accidents clearly indicate inadequate human response to strata control needs.

Proposed Accident Investigation Report



7. SE1SMICITY AND ROCKBURST IN SOUTH AFRICAN MINES

The • term "rockburst" first received official recognition in 1924, with appointment by the government of safety inspectors. The terms of reference were "to investigate and report upon the occurrence and control of rockburst in mines and the safety measures to be adopted to prevent accidents and loss of life resulting therefore".

Rockburst can be defined as "a seismic event which involves brief, violent movements of the rock mass and which causes fatality and noticeable damage to an excavation. "

Mining excavations induce elastic and then inelastic deformation within the surrounding rock mass. The elastic strain energy accumulated in a portion of the rock mass may be gradually unloaded due to„the passage of mining, or it may be released gradually or suddenly during the process of inelastic deformation. Therefore, *seismic event is a "sudden inelastic deformation (release of the strain energy stored in the rock mass) within a given volume of rocks, i.e., seismic source that radiates detectable seismic waves. "*

Table 12. Rock-related accidents, death and injuries in all mines, 1999

	Accidents	KilledB	Injured
Gravity	1185	94	1138
Rockburst	344	43	379
TOTAL	1529	137	1517

Rockburst has been a matter of concern in South African Mines, especially in deep-level gold mines, for many years. Whilst the total number of injuries and fatalities has been dropping steadily, the rates have remained essentially constant for many years. Table 12 shows that the number of fatalities resulting from rockburst is 31% of all rock-related accidents. Table II also indicates that 109 rock-related fatalities took place in gold mines, which is almost 80% of all rock-related fatalities that took place in all South African mines. Table 11 also shows that 38% of all rock-related accidents that took place in gold mines were rockburst related. Mines in South Africa are planning to extract ore at depths of 4.5 km and deeper m the next 10 years, and it is clear that serious measures are needed to minimize the risks implicit in mining at such great depth. In essence, the safety of the underground workers is paramount.

7.1 *Seismic monitoring in gold mines*

Most of the seismicity in the South Africa mining region is mining induced. Most of the seismic events are categorized as being face-driven, geological-driven (local) and regional-driven. That is why most of the African deep-level gold mines are equipped with a seismic network system for warning, prevention and design purposes.

The recognition of the hazards posed by seismicity can be quantified through seismic observation, i.e., experience over time in a particular environment. That is, by obtaining seismic information from a particular environment for a period of time. The hazard of large events associated with major geological discontinuities can also be inferred by knowledge of the structure and the mine layout. It can be recognized without having had prior seismic information, which then implies that it can be estimated even before mining starts in an area, i.e., 'non-monitor' recognition of hazard. Substantial investment in research is being made in South Africa in order to understand the physical processes, development and evaluation of the early warning concept.

7.2 *Seismic emission and rockburst control in gold mines*

Mines must adopt all reasonable procedures and techniques to prevent or reduce seismic emissions. These can be achieved by implementing stabilizing or bracket pillars, backfilling, proper mining configuration and sequencing, limitation of excess shear stress (ESS) on the geological feature, mining of dykes, face shapes, limitation of energy release

%

rate (ERR), face advance rate, remnant removal, mining of dykes, mining away from structures, hydraulic props, seismic monitoring for prediction, or any other preventive procedures.

8 ROCK-RELATED RISK MANAGEMENT IN MINES

The Mine Health and Safety Act, 1996 requires that every employer must:

identify the hazards to health and safety to which employees may be exposed while they are at work;

assess the risk to health and safety to which employees may be exposed while they are at work;

record the significant hazards identified and risk assessed; and

make those records available for inspection by employees.

Rock-related risk management is an ongoing and necessary process that must be implemented to address hangingwall hazards, support installation, seismicity etc.

Before risk management can be implemented, it must be folly understood by all people concerned. All risk assessment works should be based on a tripartite approach because employees and workers especially are joint owners of the risk management process. Employers wanting successful results form their rock-related risk management processes are advised to consult workers from the outset.

8.1 Rock-related risk assessment process (İRAP)

Nobody comes to work to be injured. The mine management therefore has a moral obligation to provide workers with the tools and knowledge to enable them to work in a safe manner, avoiding accident and injury; risk assessment facilitates this. The causes of rockburst and fall of ground accidents as experienced in deep-level gold mines in South Africa are controlled by a variety of factors, such as the magnitude of the seismic event, rock mass conditions, geological features, mining layout, support effectiveness, and safety and management strategies. Rock-related risk assessment could be greatly enhanced by analyses of accidents and incidents. This type of approach helps the mine management to determine *"hot spots "* and *"critical job tasks "*.

The guidelines issued in terms of the Mine Health and Safety Act, 1996 by the Chief Inspector for compilation of a mandatory code of practice to combat rockfall and rockburst accidents in metalliferous mines also require managers to identify and describe rock-related hazards which are likely to arise from the mining of each gee-technical area identified. This information and information arising from the above accident analysis will enable the manager to develop strategies.

The rock-related risk assessment process provides the basis for decision making and enables management to create a safer environment. Therefore, roçk-related risk assessment process in a work place must be carried out continuously and not be regarded as once-off exercises- There are three types of rock-related nsk assessment techniques namely; -

1. *Base Line Risk Assessment*

This must be done to identify major risk for future risk control. (Analysis of historical data, rock-related accident reports, inspections and information-sharing between mines, etc.).

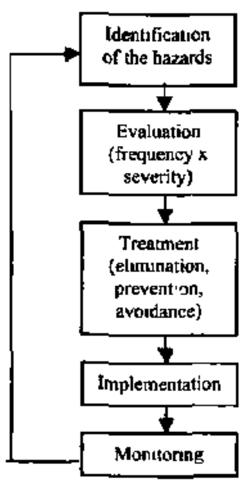
2. *Issue-Based Risk Assessment*

As circumstances and needs arise, separate risk assessment studies will need to be conducted. (Introduction of a new support after an accident, process or technique introduced to the underground environment, etc.).

3. *Continuous Risk Assessment*

This is the most important form of risk assessment, which should.take place continuously, as an integral part of day-to-day management. It may not use the more sophisticated hazard identification and risk assessment tools. It will mainly be used by frontline supervisors (audits, strata control checklists, etc.). Checklists can also provide input to an ongoing working area risk classification process mat will monitor ground conditions, control of conditions and serve to highlight any detenotion that requires remedial action.

<u>*Risk Assessment Model for Mines*</u>



Many mines, especially deep-level gold mines, in South Africa develop their own rock-related risk management system according to their needs. Many parameters such as seismicity, mining layout, sloping width, ERR, ESS, face shapes, support, gully shape, ground condition, drilling & blasting, etc. are taken into account in the design of a rock-related risk management process.

9 EXPECTATION FROM ALL CONCERNED

What I strongly believe is that rock-related safety in mines should be everybody's concern and responsibility. Each employee, all employee representative unions, all levels of management and the inspectorate are required to be committed to rock-related safety, to acting safely at all times and to promoting safety. The accident investigations and inquiries that I conducted in deep-level gold mines revealed that if the following responsibilities were carried out by all concerned, Aère would not be any problems in managing rock-related issues in mines.

What we should expect from employers.

- define practical and adaptable rock-related safety objectives;
- define rules and rock-related safety standards;
- define realistic rock-related safety training and hazard identification so that all employees can perform assigned duties safely;
- consult and discuss rock-related safety matters with workers and state at all times;
- encourage safe behavior by disciplining unsafe behavior,
- appoint all rock mechanics officers to prevent conflicts between production section and rock mechanics department.

What we should expect from employees:

- participate İn the continuous improvement of the safety process of the company;
- comply with safety regulations and safe work procedures;
- identify and initiate action to correct unsafe work practices and conditions;
- challenge work procedures, practices or behaviors that you believe to be inappropriate;
- take reasonable care to protect your safety as well as that of other people.

What we should expect from unions:

- proper participation and involvement in accident investigations and inquiries;
- properly trained union representatives concerning rock mechanics issues.

What we should expect from rock mechanics officers:

- properly qualified and trained staff;
- properly equipped service department;
- safe, economic design of mine workings and support systems;
- identification *of* potentially dangerous situations (seismic monitoring) or recommendation of remedial action before workers are injured or working place damaged;
- develop a strategy to reduce the incidence of as well as ameliorating the effects of rockburst;
- routine monitoring and investigations;
- quality control of support units;
- provide basic strata control training for production personnel;

What we should expect from the state:

- continuous participation and co-operation concerning rock-related issues at all times;
- enforcement of regulations in mines;
- drafting and implementing more rock-related safety regulations for die Act;
- properly trained and qualified inspectors.

10. CONCLUSIONS

Over the years, the South African rock mining industry sustained, and continues to sustain, a high level of rock-related accidents, and the resulting rate of human casualties (injuries and fatalities) has been, in worldwide terms, unacceptably high. Of these accidents, 43% of all accidents have been rock-related, that is, the result of rockfalls or rockbursts. In gold mines rockfall and rockburst accidents also constitute the most significant cause of all fatalities. In 1999, 68% of all fatalities took place in gold mines, of which 51% of all accidents that occurred in gold mines were rock-related.

The author conducted 25 rock-related accident investigations and inquiries in order to determine the actual cause of 38 fatalities. The investigations and inquiries revealed that most of the fatalities and seismic damage mechanism occurring in deep-level gold mines could have been minimized if the following points had been in order: -

1. Installation of the support as per standards.
2. Additional support installation and areal coverage
3. Proper removal of the support units.
4. Non-adherence to designed mining layout & sequences-poor mining practice.

5. Hazard identification & training.
6. Appreciation to seismic monitoring and prevention in mines.

It is my considered opinion that the amount of seismicity and rock-related fatalities in mines can only be reduced if the following are taken in to account. -

- keeping or introducing *backfill support* in all deep-level gold mines in terms of strata control, regional support, environmental control, ERR control, etc. This conclusion is based on my routine underground inspection and accidents investigations.
- rock engineering services can make great contributions to the rock-related safety of the mine. All rock mechanics personnel should be legally appointed so that they can speak and express themselves in management language in order to get their rightful recognition.
- rock-related risk management should be integral to the management system of the mine to reduce accidents.
- rock-related accidents can be reduced significantly by training the workers in strata control and support issues.
- in-stope face support systems must be closed to the face as much as possible and die support should have sufficient areal coverage.
- adherence to layout design and extraction principals of the mine's rock mechanics department.
- the concepts and tools for seismic prediction in South African mine have been developed, and the results are appreciated and valid in a more holistic approach towards the assessment and management of seismic risk.

The mining industry in South Africa, especially deep-level gold mines, has no choice other than to radically improve its safety and productivity records. If the gold mines are to be in a position to enjoy continued long-term success, they must utilize the most advanced technology available to reduce rock-related fatalities and injuries.

ACKNOWLEDMENT

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REFERENCES

Department of Minerals and Energy of South Africa 1999. *Safety Statistic in Mines.*

Department of Minerals and Energy of South Africa 1996. *Guideline for Compilation of a Mandatory Code of Practice to Combat Rockfall and Rockburst Accidents.*

Department of Minerals and Energy of South Africa 1999. *Invest in an Intense and Diverse Minerals Industry.*

Koldaş, K.S. 1998. Proposed Risk Management Regulations for MRAC, *Department of Minerals and Energy Affairs of South Africa.*

Koldaş, K.S. 1999. Safety and Health Methodology for Mines, *Department of Minerals and Energy Affairs of South Africa.*

Koldaş, K.S. 2000. Notes of Rock-Related Accident Investigations and Inquiries, *Department of Minerals and Energy Affairs, Gauteng Region of South Africa.*

Mine Health and Safety Act, 1996 (Act No 29 of 1996), *Department of Minerals and Energy Affairs of South Africa.*

Ortlepp, W.D. 1984. Rockburst in South African Mines, *SAIMM Seminar on Rockburst and Seismicity in Mines.*

SIMGAP 608, 2000. Survey and Assessment of Techniques Used to Quantify the Potential for Rock Mass Instability.

* SIMRAC, 1999 "A Hand Book on Rock Engineering Practice for Tabular Hard Rock Mines."

SIMRAC, 1997. Practical guide to the risk assessment process, *Tripartite Working Group on Risk Assessment.*

1 f International Mining Congress and Exhibition of Turkey- IMCET2001, © 2001, ISBN 975-395-417-4

Open Pit Optimization - Strategies for Improving Economics of Mining Projects Through Mine Planning

K.Dagdelen
Mining Engineering Department, Colorado School of Mines. Golden, Colorado, USA

ABSTRACT: The open pit design and scheduling problem is a large-scale optimization problem that has attracted considerable attention during the last 40 years The development of the "know-how" to improve economics of open pit mining projects through the use of mathematical optimization techniques goes back to early 1960's. Unfortunately, up until recently, many of these "optimizing algorithms" could not be implemented due to the limited capacity of the computer hardware used in many mining operations. During the last 10 years, advancements in the computer hardware technology along with developments in software technology allowed open pit mines to have powerful desktop computers that can solve complex optimization problems on site. Due to applications of optimization techniques developed in the early 1960*s, for example, Chuquicamata Open Pit Mine in Chile re-evaluated their cutoff grade strategy and improved Net Present Value (NPV) of their ' operations by USS800M. Newmont Gold Corporations in Nevada, USA has implemented large scale Linear Programming Model mat was developed in early I980's to schedule their entire mine and mill production in the Carlin District, resulting in significant process costs savings. This presentation will outline open pit optimization techniques that are available today and how they can be used to improve overall economics of projects that are being planned or in production.

1 INTRODUCTION

The current practice of planning of hard rock open pit mine begins with a geologic block model (see Figure 1) and involves determination of: 1) Whemer a given block in the model should be mined or not; 2) If it is to be mined when it should be mined; 3) Once it is mined then how it should be processed. The answer to each of these questions, when combined within the whole block model, define the annual progression of the pit surface and the yearly cash flows that will be coming from the mining operations during the life of mine. There can be many different solutions to the scheduling problem depending on how the decision is made for each of the blocks. Decision as to which blocks should be mined in a given year, and how they should be processed (i.e. waste, run of mine leach, crushed ore leach or mill ore etc.) defines not only the cash flows for that year but also impacts the future annual schedules. What is decided today has long-term implications as to what can be done in the future and all these decisions link together in defining the overall economics of the a given project. The objective of the planning process for an open pit mine is usually to find optimum annual schedules that will give the highest Net Present Value (NPV) while meeting various production, blending, sequencing and pit slope constraints.

Traditionally, the scheduling problem described above is solved by dividing the problem into sub problems similar to one shown in Figure 2. The solution step starts with the assumption of initial production capacities in the raining system and the estimates for the related costs and commodity prices.

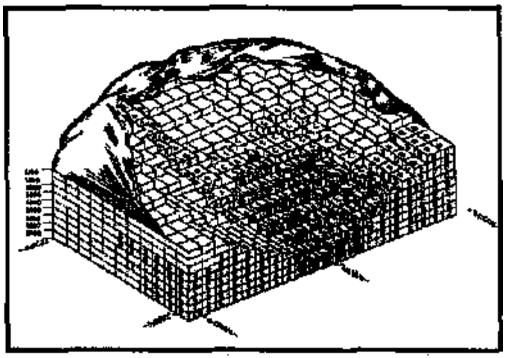

Figure I 3-D geologic block model representation of a copper deposit.

Once the economic parameters are known, the analysis of the ultimate pit limits of the mine İs undertaken to determine what portion of the deposit can economically be mined. Within the ultimate pit limits, pushbacks are further designed so that deposit is divided into nested pits going from the smallest pit with highest value per ton of ore to largest pit with the lowest value per ton of ore. These pushbacks are designed with haul road access and act as a guide during the scheduling of yearly productions from different benches. The cutoff grade strategy is defined as to differentiate ore from waste and further to determine how the individual blocks should be processed. These steps are repeated in a circular fashion as further improvements are made with respect to adequacy of the production capacities and the estimated costs.

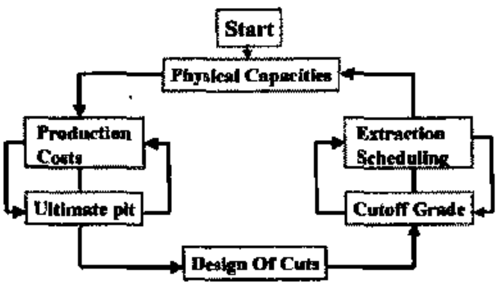

Figure 2. Steps of traditional planning by circular analysis.

There are many sophisticated software packages in the mining industry to perform ultimate pit limit analysis, design of pushbacks and to determine yearly mine plans and schedules. These computer programs are regularly used by the mining engineers in generating mine plans and schedules that are feasible. These plans are regularly implemented in actual operations without questioning whether they are the best that one can do m obtaining the highest returns possible on the capital invested.

The underlying principal for the analysis of each step İn these packages tend to be similar. The ultimate pit limits, the pushbacks and the cutoff grades are all designed and analyzed on the basis of breakeven analysis first without any consideration given to time value of money. There are serious shortcomings with these commonly followed practices if the goal of the enterprise is to maximize NPV of a given project. It İs not realistic to believe that plans and schedules obtained on the basis *of* breakeven analysis will give the highest NPV possible for a given project. This paper will discuss why certain mine planning practices result in sub optimal exploitation of resources when NPV is used as the evaluation criteria and provide suggestions and alternative solutions to overcome the shortcomings of current open pit planning and scheduling methods and practices.

2 ULTIMATE PIT LIMIT DETERMINATION

The final pit limits define what is economically mineable from a given deposit It identifies which blocks should be mined and which ones should be left in the ground. In a effort to identify the blocks to be mined, an economic block model is created first from the geologic grade model. This is done by assuming production and process costs and commodity prices at current economic conditions (i.e. current costs and prices). Then using the economic block values, each positive block is further checked whether its value can pay for the removal of overlying waste blocks. The analysis is based on the breakeven calculation that check if undiscounted profits obtained from a given ore block can pay for the undiscounted cost of mining the waste blocks. This analysis is *done* by using computer programs that either utilizes the "cone mining" method or the Lerchs and Grossmann (LG) algorithm (Lerchs and Grossmann 1964; Zao and Kim, 1992). The LG algorithm guaranties the optimality with respect to defining the pit limits that maximize the undiscounted profit while cone-mining routine is heuristic and may give sub optimum results.

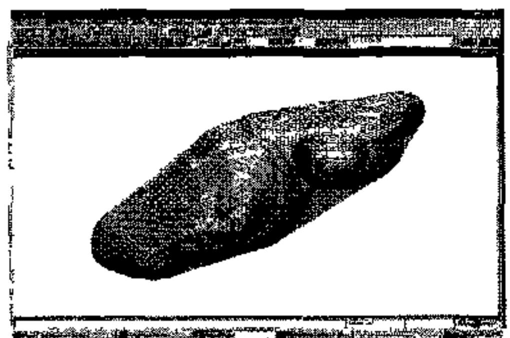
Figure 3 Ultimate pit limits with designed haul roads

The decision as to what should be mined within the ultimate pit limits is time depended and proper solution needs to take into account the knowledge of when a given block will be mined and how long one needs to be stripping the waste. The analysis of pit limits which maximize NPV requires that the time value of money is taken into account in defining which blocks should be mined and which blocks should be left in the ground during the life of the project. The pit limits that maximize the undiscounted profits for a given project will not maximize the NPV of the project.

To overcome this, it is suggested mat one carries out a preliminary complete pit design and annual scheduling first. Then determines a new economic block model by using time dependent revenues and costs knowing when a given block will be mined and how it will be processed. Using this new economic block model, ultimate pit limits are determined again to reflect the effect of time value of money on the final pit limits. It has been our experience that this new pit is always smaller than the previous one in terms of both contained ore and waste tons and give higher NPV for the cash flows generated from İt. This is due to the fact that the discounting effect on the economic block value calculation tends to reduce ore block values to be mined in the later years of the deposit while the waste mining costs to reach these blocks have to be incurred sooner. As such, the ore blocks that are very marginal in value drop out from *the* ultimate pit.

3 PUSHBACK GENERATION

As part of the planning and scheduling process, the intermediate pits leading to ultimate pit limits are determined to see how the pit surface will evolve through time. The procedure followed in the existing software packages to generate nested pits is by varying commodity price, costs or cutoff grades gradually from a low value to a high value. By changing the commodity price, for example, from a low value to a high value, one can generate a number of pits in increasing size and decreasing average value per ton of ore contained in the pit. Since the smallest size pit contains the highest valued ore, the production is scheduled by mining smallest pit first followed by the production in larger pits (see Figure 4). The incremental mining from the smallest pit to larger pit is referred to as pushback mining and there are cases where production is scheduled from more than one pushback simultaneously. Once the nested pits are generated, smoothed and haul roads are added, they are used as pushbacks underlying practical plans from which yearly schedules are generated.

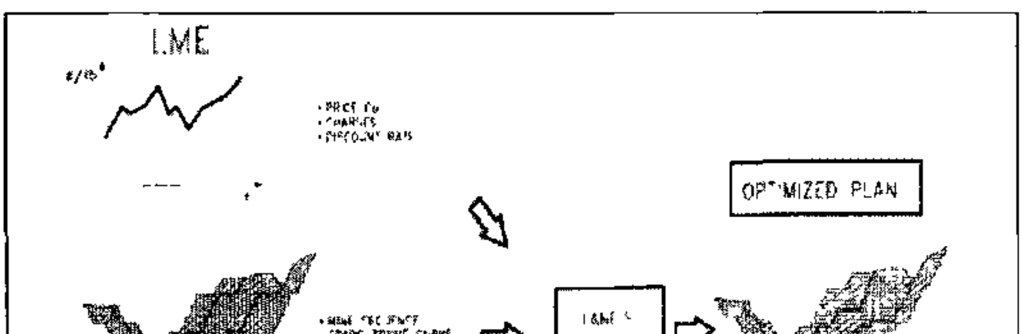

Figure 4: The nested pits showing pit progress leading to ultimate pit limits (Camus and Jarpa, 1996).

The nested pit generation also does not take into account time value of money. They are generated assuming undiscounted value of the blocks. The pushbacks that will maximize the NPV of a project can be significandy different than the ones found by using existing procedures. It can be shown (See Bemabe, 2001) that the nested pit generation on parametrizing a single factor such as the metal price or the production costs or the metal grades will lead to sub-optimum results when more than one process type exist for the ore types in the deposit.

4 LONG TERM YEARLY SCHEDULES

Once the pushbacks are generated and designed for haul roads and minimum width requirements, the next step is then to come up with yearly progress maps within the pushbacks by dividing the pushback further down into smaller increments. The yearly progress maps are usually generated by taken into account annual waste and ore raining tonnage requirements for different material types. Ore and waste discrimination is normally done on the basis of breakeven cutoff grades. In the simplest case, yearly schedules are determined by mining from the top bench of the smallest pushback towards the bottom bench. Once a given pushback is exhausted then the mining from the top bench of the next pushback starts and continues until it is exhausted. In many cases, this approach does not result in best yearly schedules that maximize the NPV of the cash flows. Realizing this, the newest schedulers in the mine planning packages are designed to work with multiple pushbacks simultaneously and the mining activity can be scheduled from 3 or 4 pushbacks at the same time. In one scheduling package (see Cai, 1993) a schedule for a given year is determined by generating plans for all the possible mining scenarios between benches of the pushbacks and choosing one plan that gives the highest profit. This process is repeated for each year one year at a time until whole deposit is mined out In another scheduling package possible (see Tolwinsky, 1998) yearly mine plans between pushbacks are further linked together year by year and analyzed with respect resulting overall NPV. The overall plan that links together yearly schedules resulting in highest NPV is chosen as the optimum. In another package (Whittle, 1999), yearly ore mining is scheduled wimin die individual pushbacks in the pushback sequence by mining ore from the benches of the pushbacks without any consideration given to waste tonnages. The schedule obtained by using this process results in fluctuating waste tonnages from one year to another. As such, these fluctuations are further smoothed by mining from multiple pushbacks in a given year.

The underlying concept in determining yearly schedules in all of the commercially available scheduling packages assumes mat the previously designed pushbacks will guide the scheduling process to result in distribution of cash flows that will give the highest NPV. Of course this is not the case for many open pit mines, particularly for the ones where strip ratio varies significantly from one area of the pit to the other areas as well as for open pit mines that require blending of different material types.

5 CUTOFF GRADE SRATEGIES

The cutoff grade is the grade that is used discriminate between ore and waste during scheduling. The most open pit mines are designed and scheduled by using cutoff grades that are calculated by using breakeven economic analysis. The use of breakeven cutoff grades during open pit planning results in schedules that maximize the undiscounted profits (Dagdelen, 1992). The cutoff grade that maximizes the NPV of the cash flows is not only a function of economic parameters but also mining, milling, and refinery capacity limitations as well as the grade.distribution within the deposit.

Lane (Lane, 1964) proposed an algorithm to determine cutoff grades that maximize the NPV of a project subject to mine, mill and refinery capacity constraints.

The cutoff grade strategy that results in higher NPV for a given project starts wim high cutoff grades during the initial years of the deposit. As the deposit matures the cutoff grades gradually decline to breakeven cutoff grade depending upon the grade distribution of the deposit

Various computer packages are developed using Lane's algorithm (Lane, 1988; Dagdelen 1992; Whittle, 1999). Application of these programs İn determining optimum cutoff grade strategy has resulted significant improvements to NPV of the projects (see Jamus and Jarpa 1996).

6 FUTURE

The ultimate pit limits cannot be determined without knowing when the individual blocks will be mined. Determination of when a given block will be mined cannot be done without knowing pushback sequence and the cutoff grade strategy. The pushback sequence and the cutoff grade strategy are themselves a fonction of when the blocks will be mined in *he block model. As such, the optimum solution to the problem we have identified initially deals witfi many interdependent variables and currently solved by using heuristic techniques that are trial and error.

The determination of ultimate pit limits, yearly mine schedules and the cutoff grade strategies for a given open pit mine can be formulated using large scale LP/IP models (see Johnson (1968) and Dagdelen (1985)). These models include over one hundred thousand variables and fifty to one hundred thousand constraints (see Akaika and Dagdelen, 1999; Hoerger, 1999).

The hardware and software technology with respect to implementation of the optimization techniques based on Linear (LP) and Integer Programming (IP) have advanced to a point that we can now solve some of these problems without any difficulty.

A good example of a large-scale LP application is Newmont Mining's Carlin operations involving multiple .open pit mines and plants. The implementation of a large scale LP model by the Newmont engineers in actual operations involved over 100K variables and close to 30 K. constraints. The model is proved to be successful resulting in significant improvements in maximizing NPV of these projects (see Hoerger 1999).

7 CONCLUSIONS

The large-scale open pit operations are looking at ways to improve economics of their operations using NPV as a criterion. The mine planners of the new millennium are looking beyond the optimization techniques that traditionally provided the highest undiscounted profits. The available commercial packages are retooling their programs to overcome. shortcomings of traditional mine planning techniques in providing NPV maximized mine plans and schedules. It is matter of time before the latest operation research based optimization tools become commercially available and regularly used. The use of these optimization tools by mine planners provides great opportunities for increased returns on large amount of capital being invested on these projects.

REFERENCES

Akaike, A. and Dagdelen, K.. 1999, "A strategic Production Scheduling Method for an Open Pit Mine", Proceedings of 28"[1] APCOM, pp. 729 to 738.

Bemabe, D., 2001, "Comparative Analysis of Open Pit Mine Scheduling Techniques For Strategic Mine Planning of a Copper Mine in Southern Peru", MSc. Thesis, Colorado School of Mines, Golden, Colorado, 2001.

Cai, Wen-Long, and Banfield, F., 1993, "Long Range Open Pit Sequencing - A Comprehensive Approach", APCOM Proceedings - Montreal, 1993, pp. 11-18.

Camus, J. P. and Jarpa, S. G., 1996, "Long Range Planning at Chuquicamata Mine", Proceedings of 26[lh] APCOM Symposium, pp 237 to 241.

Dagdelen, K., 1985, "Optimum Mûti-Period Open Pit Mine Production scheduling", Ph.D. Thesis Colorado school of Mines, 1955.

Dagdelen, K., 1992, "Cutoff Grade Optimization", Proceedings of 23"* APCOM, University of Arizona, Tucson, Arizona, pp. 157-165.

Hoerger, S. 1999, and et al "Long Term Mine and Process Scheduling at Newmont's Nevada Operations, Proceedings of 28* APCOM, Colorado School of Mines, Golden, Colorado, E999, pp 739-748.

Johnson, T.B., 1968, "Optimum Open-Pit Mine Production Scheduling", Ph.D. Dissertation, University of California, Berkeley, Operations Research Department.

Lane, K. F., 1964, "Choosing the Optimum Cutoff Grade", Colorado School of Mines Quarterly, Vol. 59, pp. 811-829.

Lane, K. F., 1988, Economic Definition of Ore - Cutoff Grades Theory and Practice", Mining Journal Books Limited, London.

Lerchs, H. and Grosmann, I. F.. 1965, "Optimum Design of Open Pit Mines", Canadian Institute of Mining Bulletin. Vol. 58, no. 633, Jan., pp. 47-54.

Tolwinski, B., 1998, •'Scheduling Production for Open Pit Mines", 27™. APCOM Proceedings - London, UK, !998, pp. 651-662.

Whittle, J., 1999, "A Decade of Open Pit Mine Planning and Optimization -The Craft of Turning Algorithms into packages", 28[Th] APCOM Proceedings - Colorado School of Mines, Golden, CO, 1999, pp 15-23

Zhao, Y. and Kim, Y. C, 1992. A New Optimum Pit Limit Design Algorithm", Proceedings of 23'" APCOM, pp. 423-434.

17* international Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-395-417-4

Stope Optimiser - A FORTRAN Program to Optimise Stope Boundaries

E.Y.Baafi & M.Ataee-pour
Faculty of Engineering, University of Wottongong, Wollongong, NSW 2522, Australia

ABSTRACT: A computer program has been developed to implement a heuristic algorithm for optimisation of underground stope geometries. The *Stope Limit Optimiser (SLO)* software integrates *FORTRAN 90* computer program with *WINTERACTER* user interface features to provide a Windows based interactive environment to define and modify the underground mine design parameters such as, block model parameters, stope geometry constraints and economic factors. The heuristic algorithm termed *Maximum Value Neighbourhood (MVN)* and its *FORTRAN 90* implementation procedure are presented with examples.

1 PROBLEM DEFINITION

The problem of finding the best combination of desirable and non-desirable blocks that result in the maximisation of profit of a stope, may be expressed by Equation (1).

Objective fonction:

$$\left.\begin{array}{l} \textit{Maximise } SEV = \sum_{ijk} F_{ijk}\, BEV_{ijk} \\ \text{subject to:} \\ \quad \textit{slope geometry constraints} \end{array}\right\} \quad (1)$$

where
SEV: total stope economic value,
BEV_{ijk}: the economic value of block B_{ijk},
F_{ijk}: an indicator function showing whether the block, B_{ijk}, is mined or not and defined by Equation (2).

$$F_{ijk} = \begin{cases} 1 & \textit{if } B_{ijL} \textit{ Is selectd,} \\ 0 & \textit{otherwise.} \end{cases} \quad (2)$$

2 STOPE GEOMETRY

The geometry of the working spaces in underground mines is restricted in each orthogonal direction by both a minimum and a maximum size (length, width or height). Figure 1 depicts the minimum stope size constraints in ID, 2D and 3D directions. Physical parameters, such as the geo-mechanical properties of the ore-body and the surrounding rock, the dip, the depth and the thickness of the ore-body, affect the proposed underground mining methods, which, in turn, impose some practical restrictions on the extraction limit of the orelwdy. The *block caving* method imposes different constraints to the stope geometry from that imposed by a *cut-and-fill* method. A *cut-and-fill* stoping is flexible allowing the extraction of the high-grade ore whilst leaving the low-grade material İn the stope as fill. In practice, the minimum size of the stope must be designed so that sufficient space is provided for the activities of drilling, blasting and loading, as well as the traffic of personnel. The maximum limits of the stope dimensions are usually dictated by the geo-technical factors.

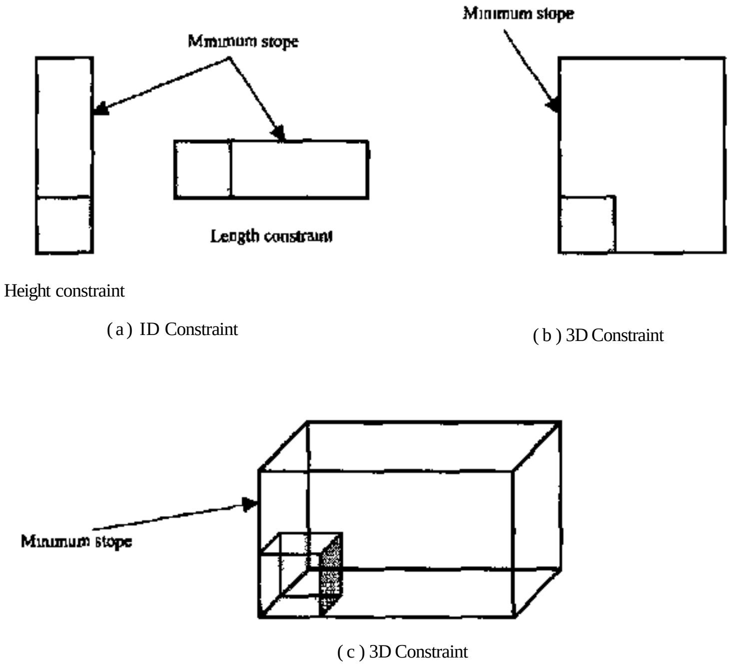

Figure \ Stope Constraints (a) ID, (b) 2D and (c) 3D Problems

3 THE NEIGHBOURHOOD CONCEPT

The minimum stope size represents by the number of mmeable adjacent blocks of a fixed geological block model. This leads to the definition of the two terms the *"neighbourhood"* and the *"order of neighbourhood"*. When considering a one-dimensional stope constraint, a *"neighbourhood" (NB)* is the set of all the sequential blocks, including the block of interest in the specified direction, which may be mined to satisfy the minimum sfope size requirement. Figure 2 shows a row of seven blocks *(Bj\ j = 1, 2 , ... 7}*. If the mininram stope size is 20 m and the block size of a fixed block model is 5 m, then a possible *neighbourhood* for the block, 8_4, may consist of the set of four sequential blocks *{Bi,* B_4, *Bi,* B_6*J*. It is clear that the illustrated neighbourhood is not the only possible neighbourhood for the block, *B4*. One can easily assume that the set *{*B_3*, Bi,* B_4*, B$)* could be another possible neighbourhood for the same block, *B4*. Assummg that the stope geometry is restricted in length or height only, then the term *"stope block* fig/io" *fSBR)* may be expressed by Equation (3).

$$SBR = \frac{\text{minimum stope size (length or height)}}{\text{fixed block size (length or height)}} \qquad (3)$$

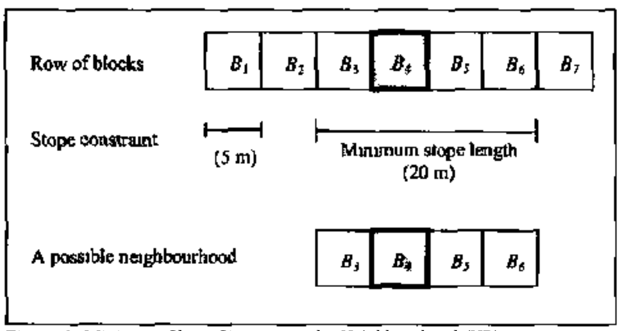

Figure *2* Minimum Slope Size versus the *Neighbourhood (NB)*

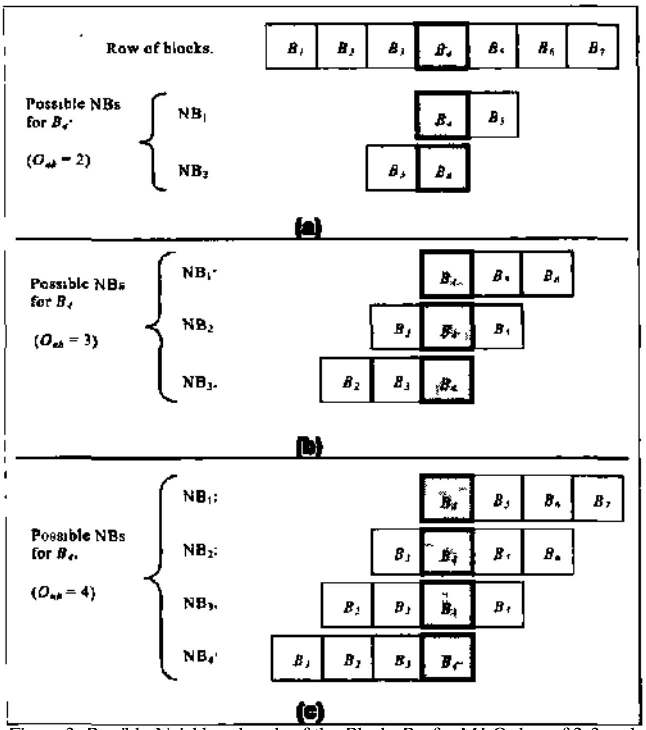

Figure 3. Possible Neighbourhoods of the Block, B_4, for MJ Orders of 2,3 and 4.

A stope block ratio of 3.8 suggests that the minimum stope length (height) must be 3.8 times the block size of the fixed block model. The stope block ratio is a continuous real function, which suggests an inclusion of partial blocks. Since the proposed *MVN* algorithm does not allow partial blocks, the value of *SBR* is rounded up. A positive integer value termed the *"order of neighbourhood" (0„b)* is used to represent the rounding up value of the stope block ratio. In other words, the *order of neighbourhood* represents the size of the neighbourhood set. That is, the total number of sequential blocks, including the block of interest, in a specified direction, which may be mined to satisfy the minimum stope size requirement. There are a number of combinations of blocks forming a neighbourhood for any given block. Figure 3 shows the possible neighbourhoods of the block, *B4,* for the neighbourhood orders of 2, 3 and 4. For the *NB* order of 4, four neighbourhoods may be defined, so that each neighbourhood contains four members {Figure 3c).

The set of neighbourhoods, *NB„„* for any block, *Bj,* with any order of neighbourhood, "/", may be defined by the set of "/" sequential blocks In ascending order, starting from the block, B_{j-m+1}, and ending with the block, B_{j+l-m}, where $\leq$ m $\leq$/, as expressed by Equation (4):

$$NB_{m,j,l} = \{BEV_\beta \in U \mid \beta = j-m+1, j-m+2, \ldots, j-m+l\} \tag{4}$$

where

β: the column number,^, used for blocks in the m'* neighbourhood and

BEV_β: the economic value of a typical block, located in the m^{th} neighbourhood.

The definition of any neighbourhood requires'

(i) the location of the block, for which the neighbourhood is defined (the *j* address of the block in a ID constraint problem),

(ii) the order of neighbourhood, *"0„h"*> and

(iii) the neighbourhood number (that is, which neighbourhood is required).

In general, each neighbourhood, $NB_{m,j}, i,$ is assigned a value, denoted by $NBV_{mj}i$ which is defined by Equation (5)

$$NBV_{m,j,l} = \sum_{\beta=j-m+1}^{j-m+l} BEV_\beta \tag{5}$$

For any block, *Bj,* with the neighbourhood order of "/", the maximum neighbourhood value

(MNBV), among all the elements of the set of neighbourhood values, is defined by:

$$MNBV_{j,l} = Max.\ \{NBV_{1,j,l},\ NBV_{2,j,l},\ ...,\ NBV_{m,j,l},\ ...,\ NBV_{l,j,l}\}$$

or

$$\begin{aligned} MNBV_{j,l} &= Max\ NBVS_{j,l} \\ &= Max\ \left\{NBV_{m,j,l} \mid m = 1,2,...,l\right\} \end{aligned}$$

As a numerical example, consider the row of blocks shown in Figure 4. If the minimum slope length is ten meters and each block is three meters long, the stope block ratio is 3.3, which results in an order of neighbourhood of 4. All neighbourhood values for the block, B_j, with the order of 4 are collected in the set $NBVS_{j4}$. Then, the 4th neighbourhood value *(NBV)* provides the maximum neighbourhood value, termed *MNBV* for this example.

4 MAXIMUM VALUE NEIGHBOURHOOD ALGORITHM

The neighbourhood concept locates the optimum neighbourhood for a block. This heuristic optimisation technique requires the following seven basic steps for each block within the economic model, with a value *BEV:*

(i) determine the neighbourhoods of the block, based on the order of neighbourhood, i.e. construct the set of possible neighbourhoods *(NBS);*
(ii) evaluate the feasibility of each neighbourhood within the *NBS* set;
(iii) calculate the economic value for each neighbourhood, i.e. neighbourhood value *NBV,* and determine the set of neighbourhood values *(NBVS);*
(iv) locate the maximum neighbourhood value *(MNBV)* within the *NBVS* set;
(v) determine the maximum value neighbourhood *(MVN);*
(vi) flag the blocks of the *MVN* and
(vii) update the stope economic value *(SEV).*

The generalised flow-chart for the optimisation procedure in the *MVN* algorithm is illustrated in Figure 5 whereby blocks are taken into consideration one by one, in the order of rows (X direction), columns (Y direction) and finally sections (Z direction). FORTRAN 90 implementation of MVN algorithm has been provided by Ataee-pour (2000). If a block has a non-negative value and is not already flagged, the procedure will continue to construct its set of neighbourhoods, *(NBS),* based on the order of neighbourhood "*l*"; calculate the values of neighbourhoods and finally locate *the* maximum neighbourhood value *(MNBV)* as well as the maximum value neighbourhood *(MVN)* of the block.

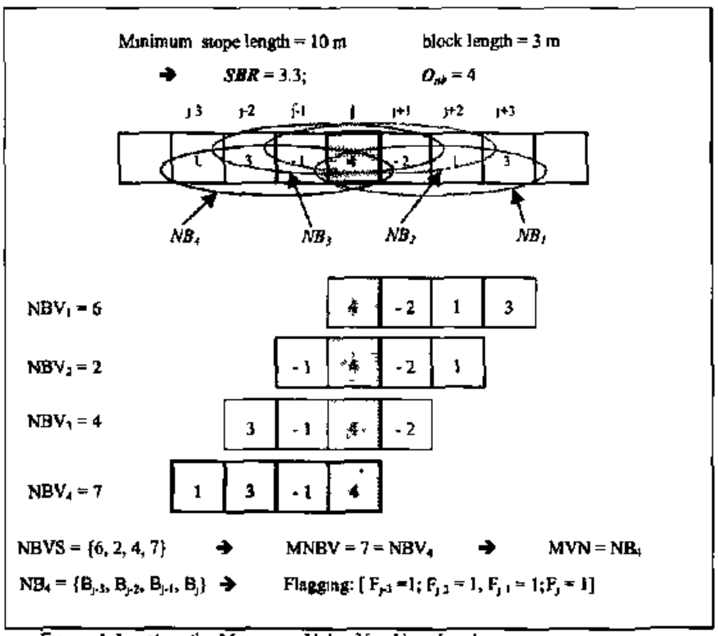

Figure 4. Locating the Maximum Value Neighbourhood.

The algorithm then determines whether or not the maximum neighbourhood value is non-negative, i.e. it contributes to the stope value. Ignoring the blocks with a negative *MNBV*, the procedure continues for the non-negative ones. The *MVN* of each block represents its marginal contribution to the stope value. This marginal value may be negative, zero or positive. Negative marginal values are ignored and the algorithm selects only a block with an *MVN* that provides a non-negative marginal value. The elements of that *MVN* are then flagged and included in the stope. The above process is repeated for all blocks withm the block economic model. After all the blocks are exammed, the final optimum stope boundaries are displayed. Four checks are used in this algorithm to exclude unnecessary blocks. A block is ignored:

(1) if the block is flagged already;
(2) if the block has a negative dollar value;
(3) if the maximum neighbourhood value *(MNBV)* of the block is negative; and
(4) if the marginal value provided by *the MVN* is negative

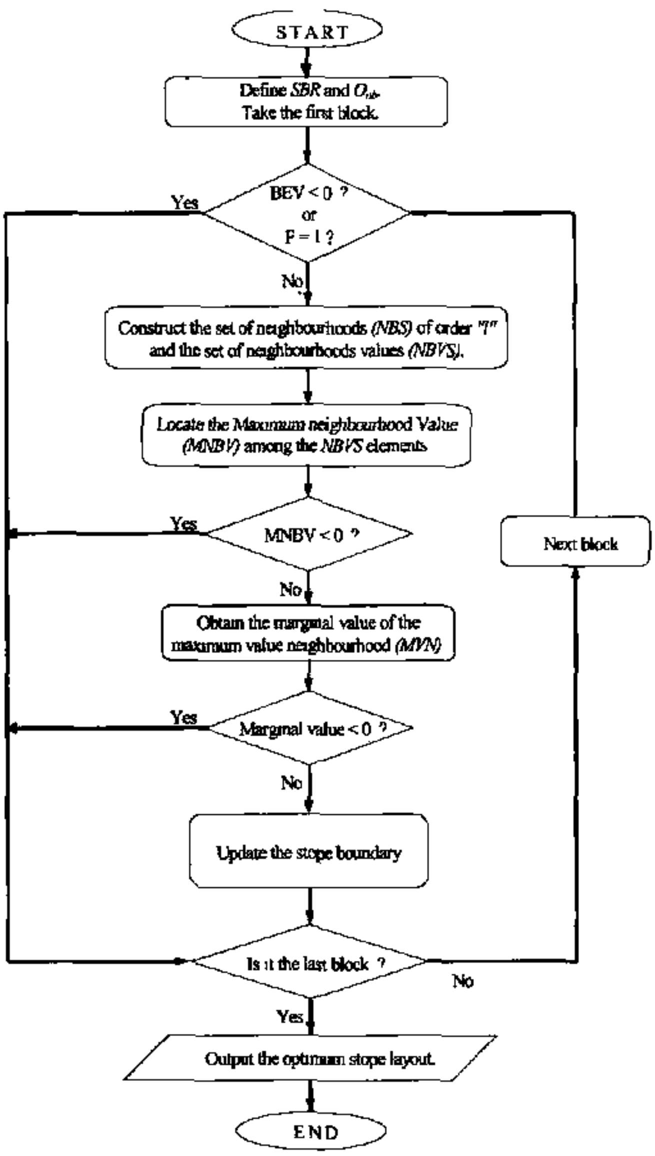

Figure 5. *Generalised* Flow-Chart for the Optimisation Procédure

5 NUMERICAL EXAMPLE

Consider a row often 10 m blocks, $\{B_j, j = 1,2, ..., 10\}$, shown in Figure 6. The number inside each cell represents the economic value, BEV_j, of the respective block. A 25 m minimum stope length gives a stope block ratio is then 2.5, which results in the order of neighbourhood of 3. The pseudocode of the *MVN* algorithm with this example is provided below:

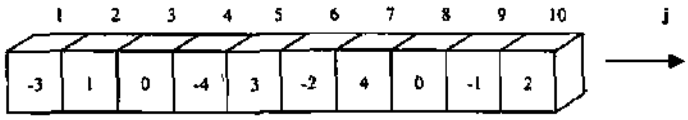

Figure 6 A Row of Blocks with 10 Columns taken from an Economic Block Model

Step 1 : Initialise variables.
Block length = 10 m; Minimum stope length = 25
$SBR = 2.5$; $O_{nb} = 3$
$SEV = 0$; $F = 0$; $FF = 0$

Step 2: Examine the first block.
B_1: $BEV_1 = -3$; $F_1 = 0$

Step 3: Check the negativity and the flag of the block.
$BEV_1 < 0$ ➔ Block B_1 is exempted from further process.
Go to Step 9.

Step 9: Check the end of the model.
B_1 is not the last block. Continue.

Step 10: Examine the next block.
B_2: $BEV_2 = 1$; $F_2 = 0$
Go to Step 3

Step 3: Check the negativity and the flag of the block.
$BEV_2 > 0$ and $F_2 \neq 1$ ➔ Continue.

Step 4: Locate die maximum neighbourhood value *(MNBV)*
$NB_1 = \{1, 0, -4\}$ ➔ $NBV_1 = 1 + 0 - 4 = -3$
$NB_2 = \{-3, 1, 0\}$ ➔ $NBV_2 = -3 + 1 + 0 = -2$
NB_3 = Not feasible
➔ $MNBV = -2$; $MVN = NB_2$

Step 5: Check the negativity of the *MNBV*.
$MNBV < 0$ ➔ Block B_2 Is exemptedfrom further process.
Go to Step 9.

Step 9: Check the end of the model.
B_2 is not the last block. ➔ Continue.

Step 10: Examine the next block.
B_3: $BEV_3 = 0$; $F_3 = 0$
Go to Step 3

Step 3 : Check the negativity and the flag of die block.
$BEV_3 \geq 0$ and $F_3 \neq 1$ ➔ Continue.

Step 4: Locate the maximum neighbourhood value *(MNBV)*.
$NB_1 = \{0, -4, 3\}$ ➔ $NBV_1 = 0 - 4 + 3 = -1$
$NB_2 = \{1, 0, -4\}$ ➔ $NBV_2 = 1 + 0 - 4 = -3$
$NB_3 = \{-3, 1, 0\}$ ➔ $NBV_3 = -3 + 1 + 0 = -2$
➔ $MNBV = -1$; $MVN = NB_1$

Step 5: Check the negativity of die *MNBV*.
$MNBV < 0$ ➔ Block B_3 is exempted from further process.
Go to Step 9.

Step 9: Check the end of the model.

B3 is not the last block ➔ Continue

Step 10: Examine the next block

B_4. BEV_4 = -4; F_4 = 0

Go to Step 3

Step 3: Check the negativity and the flag of the block.

BEV_4 < ➔ Bl«^ % Ä^einplei feıâ farther process.

Go to Step 9

Step 9: Check the end of the model.

B4 is not the last block. T> Continue.

Step 10: Examine the next block.

B_s: BEVs = 3; F_s = 0

Go to Step 3

Step 3: Check the negativity and the flag of the block

BEVs £ 0 and F_5 # 1 •* Continue.

Step 4: Locate the maximum neighbourhood value *(MNBV)*.

NB, = (3, -2,4} • • NBVi = 3 - 2 + 4 =5

NB_2 = H, 3, -2} -» NBV_2 = - 4 + 3 - 2 = -3

NB_3 = {0, -4,3} •* NBV_3 = 0 - 4 + 3 = -1

* MNBV = 5; MVN = NB,

Step 5: Check the negativity of the *MNBV.*

MNBV > 0 * Continue.

Step 6: Calculate the marginal value.

MVN = NB, = {BEVj, BEV_6, BEV_7}

F_s = 0 •» FF5 = 1; F_6 = 0 •» FF_6 = 1; F_7 = 0 •* FF_7 = 1

Marginal Value - BEV_S. FF_S + BEV_6. FF_6 + BEV_7. FF_7

= (3x1) + (-2x1) + (4x1) = 5

Step 7: Check the negativity of the marginal value.

Marginal Value > 0 * Continue.

Step 8: Update the stope.

SEV = SEV + Margin Value = 0 + 5 = 5 ➔ SEV - 5

F_3 -1; F_6 =1; F_7 =1

Step 9: Check the end of the model.

B_s is not the last block. ➔ Continue.

Step 10: Examine the next block.

B_6: BEV_6 = -2; F_6 =1

Go to Step 3

Step 3: Check the negativity and the flag of the block.

BEV_6 < 0 ➔ Slock Bg is exempted from feather proeess.

Go to Step 9.

Step 9: Check the end of the model.

Be is not the last block. ➔ Continue

Step 10: Examine the next block.

B_7: BEV_7 = 4; F_7 = 1

Go to Step 3

Step 3 : Check the negativity and the flag of the block.

F_7= 1 ➔ Block B? is exempted from fî&ïhesr process*

Go to Step 9.

Step 9: Check the end of the model.

B_7 İs not the last block. ➔ Continue.

Step 10: Examine the next block.

B_8: $BEV_8 = 0$, $F_8 = 0$

Go to Step 3

Step 3: Check the negativity and the flag of the block.

$BEV_8 > 0$ and $F_8 \neq 1$ ➔ Continue.

Step 4: Locate the maximum neighbourhood value *(MNBV)*.

$NB_1 = \{0,-1,2\}$ ➔ $NBV_1 = 0 - 1 + 2 = 1$

$NB_2=\{4,0,-1\}$ ➔ $NBV_2 = 4 + 0 - 1 = 3$

$NB_3 = \{-2,4,0\}$ ➔ $NBV_3 = -2 + 4 + 0 = 2$

➔ $MNBV = 3$; $MVN = NB_2$

Step 5: Check the negativity of *the MNBV.*

$MNBV > 0$ ➔ Continue.

Step 6: Calculate the marginal value.

$MVN = NB_2 = \{BEV_7, BEV_8, BEV_9\}$

$F_7 = 1$ ➔ $FF_7 = 0$; $F_8 = 0$ ➔ $FF_8 = 1$; $F_9 = 0$ ➔ $FF_9 = 1$

Marginal Value $= BEV_7 . FF_7 + BEV_8 . FF_8 + BEV_9 . FF_9$

$= (4x0) + (0x1) + (-1x1) = -1$

Step 7: Check the negativity of the marginal value.

Marginal Value < 0 ➔ Block B_8 is exempted from further process.

Go to Step 9.

Step 9: Check the end of the model.

B_8 is not the last block. ➔ Continue.

Step 10: Examine the next block.

B_9: $BEV_9 = -1$; $F_9 = 0$

Go to Step 3

Step 3: Check the negativity and the flag of the block.

$BEV_9 < 0$ ➔ Block B_9 is exempted from further process.

Go to Step 9.

Step 9: Check the end of the model

B_9 is not the last block. ➔ Continue.

Step 10: Examine the next block.

B_{10}: $BEV_{10} = 2$; $F_{10} = 0$

Go to Step 3.

Step 3: Check the negativity and the flag of the block.

$BEV_{10} > 0$ and $F_{10} \neq 1$ ➔ Continue.

Step 4: Locate the maximum neighbourhood value *(MNBV)*.

NB_1 = Not feasible

NB_2 = Not feasible

$NB_3 = \{0, -1, 2\}$ ➔ $BV_3 = 0 - 1 + 2 = 1$

➔ $MNBV = 1$; $MVN = NB_3$

Step 5: Check the negativity of the *MNBV.*

$MNBV > 0$ ➔ Continue.

Step 6: Calculate the marginal value.

$MVN = NB_3 = \{BEV_8, BEV_9, BEV_{10}\}$

$F_8 =$ ➔ $FF_8 = 1$; $F_9 = 0$ ➔ $FF_9 = 1$; $F_{10} = 0$ ➔ $FF_{10} = 1$

Marginal Value $= BEV_8 . FF_8 + BEV_9 . FF_9 + BEV_{10} . FF_{10}$

$= (0x1) + (-1x1) + (2x1) = 1$

Step 7: Check the negativity of the marginal value.

Marginal Value ≥ 0 ➔ Continue.

Step 8: Update the stope.

SEV = SEV + Marginal Value = 5 + 1 = 6 ➔ SEV = 6

F_a=1; Fç=I; $F_{,0}$= 1

Step 9: Check the end of the model.

Bio is the last block. ➔ Go to Step 11.

Step 11 : Output the results.

Stop»» (B* Bs, Bfc *Bs,* Bs*»8ış)î **Stope Value-*6***

STOP

END

Figure 7 shows the optimised row of blocks, in which the flagged blocks are shaded. The first four blocks, including one positive and one zero value block, are excluded from the final stope as they do not improve the stope economic value. In addition, two negative blocks have been included in the optimum stope to insure that the minimum stope size constraints are met.

The following example (Figure 8a) shows a section of six rows with six columns of blocks, that has been optimised using the algorithm. In this example, the order of neighbourhood equals 3 in the vertical direction, that is, the stope has a minimum height restriction of three blocks. The optimised section has been shown in Figure 8b, where the flagged blocks have been shaded. Table 1 summarises the step-by-step application of the *MVN* algorithm to the blocks in the third column of Figure 8a.

6 CONCLUDING REMARK

The *"maximum value neighbourhood"* concept uses a fixed economic block model to outline the optimum geometry of a stope. After expressing the optimisation objective function as maximization of net stope economic value, the imposed stope geometry constraints were formulated In terms of *"neighbourhood"* and *"order of neighbourhood"*. For an individual block, the *MVN* approach outlmes a small island of mineable blocks within the whole ore-body.

REFERENCE

Ataee-pour, M 2000 A Heuristic Algorithm to Optimise Stope Boundaries, PhD Thesis, University of Wollongong, 313 PP

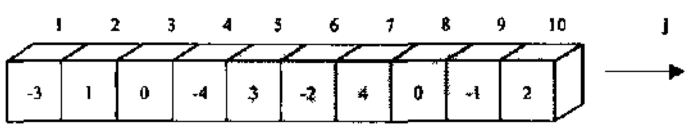

Figure 7. The Optimised Example Manually using the *MVN* Algorithm.

3	-2	3	-2	1	3
4	-1	2	4	3	5
-2	5	*4*	5	-1	3
3	-2	3	-2	6	-2
4	1	3	2	3	2
2	-3	-2	5	-2	3

(a) Model Section

3	-2	3	*2	1	3
4	-1	2	4	3	5
-2	5	«	*S*	-I	3
3	-2	3~	-2	6	-2
4	1	3	2	3	2
2	-3	-2	5	-2	3

(b) Optimum Stope Section

Figure 8. Optimising a Stope Section using the *MVN* Algorithm with a 1D (height) Constraint.

Table 1 Summary of the Algorithm Applied for Column 3 of Figure 8

block (i, j)	BEV	negative?	flag?	Neighbourhoods	old F	FF	BEV	FF BEV	old SEV	new SEV	new F
(1, 3)	3	no	no	1 2 3 [3] [ ] [ ] [2] [ ] [ ] [6] [ ] [ ]	$F_{13}=0$ $F_{23}=0$ $F_{33}=0$	1 1 1	3 2 6	3 2 6 11	0	11	$F_{13}=1$ $F_{23}=1$ $F_{33}=1$
(2, 3)	2	no	yes								
(3, 3)	6	no	yes								
(4, 3)	3	no	no	[3] [6] [2] [3] [3] [6] [-2] [3] [3]	$F_{33}=1$ $F_{43}=0$ $F_{53}=0$	0 1 1	6 3 3	0 3 3 6	11	17	$F_{43}=1$ $F_{53}=1$
(5, 3)	3	no	yes								
(6, 3)	-2	yes									

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Application of Advanced Technologies to Delineate Ground Hazards in Coal Mines

K.Hanna, D.Conover, & D.Neil
NSA Engineering, Inc., Golden, Colorado USA

ABSTRACT: Early identification of adverse geologic conditions and high stress concentrations during underground excavation using modem mechanized mining systems remains a challenge to cost-effective mining operations. Often, unexpected geologic anomalies and related stress concentrations are encountered more rapidly, leaving the mine operator with insufficient time or information to deal with difficult ground conditions. Two high-tech detection systems developed by NSA Engineering, the RockVision3D™ seismic tomography panel imaging system and the GeoGuard™ shield load monitoring system, allow for continuous assessment of geologic structures and stress conditions ahead of me longwall face without interference to daily mining operations. This paper focuses on the applications of seismic tomography to delineate stress and geologic anomalies for hazard mitigation and full panel shield pressure monitoring to detect high-stress zones, roof caving, shield weighting and periodic loading, and bump occurrences along the face and in adjacent gate roads. Case studies for bump-prone coal mines in Australia, Germany, and the United States where these technologies were used will be discussed.

1 INTRODUCTION

Characterization of geotechnical and geological conditions and responses to high-production demands are essential to mine management to maintain cost-effective mining operations, high mining rates, and workers* safety in today's mechanized mining systems. High-speed longwall mining can cause rapid buildup of ground stress ahead of the face. One common phenomenon in longwall mining, is the periodic occurrence of high loading conditions along the face associated with geostructural conditions in the roof and gateroads, such as caving of cantilevered roof layers at regular intervals and floor heave and pillar yielding in the gateroads. Problems can arise from these conditions in the form of face bumps, spalling and overloading of face supports (shields) leading to severe safety hazards and production delays. Conventional ground hazard detection and mitigation technologies have fallen short in fully addressing both personnel safety requirements and the high-production demands of today's mining. Advances in electronic sensors, computer technologies, and data processing allowed geophysical seismic tomography and shield pressure monitoring techniques to become more popular for identifying potential ground hazards ahead of a mining face (Hanna et al. 1999, Demarco et al. 1997, & Conover et al. 1994). With appropriate warning of impending high-load conditions, the mine operator can take steps to mitigate these problems, such as removal of personnel, increasing the production rate and inducing caving.

Two high-tech detection systems developed by NSA Engineering-the RockVision3D™ seismic tomography panel imaging system, and the GeoGuard™ shield load monitoring system-have been used successfully in coal mines worldwide. These technologies offer substantial benefits over conventional detection methods by providing continuous assessment of geologic structures and stress conditions ahead of the longwall face without interference to daily mining operations. Results are presented in near-real time and are easy to interpret, allowing mine operators to deal with potential hazards on a day-to-day basis.

RockVision3D™ is a high-tech tomographic imaging method that utilizes seismic energy and processes the information using techniques similar to those used in medical CAT (Computer-Aided Tomography) scans (Rock et al. 1997). The principle behind RockVision3D™ is that seismic energy travels through /different material types with different attenuation and velocity levels. Seismic waves will travel faster through competent or highly stressed rock than through broken or fractured rock and voids (Westman et al. 1995, & Yu 1992). RockVision3D™ is the only commercially available tomographic imaging system that utilizes mining equipment such as the shearer in coal mines as the seismic source. This allows continuous operation of the iongwall while images are being generated. In general, RockVision3D™ effectively provides real-time graphic representations of (1) relative stress concentrations as they migrate across an underground rock mass; (2) structural discontinuities, such as faults, joints, or shear zones; and (3) geological anomalies (sand channels or rolls in coal mines).

The GeoGuard™ shield monitoring system is designed to provide warning of high loading conditions on the Iongwall face. The system collects leg-pressure data from the face supports in real-time and provides a variety of tools for displaying and analyzing the data, both in real-time and off-line. The history of face loading conditions is continuously updated and analyzed to identify the development of high-load conditions and anticipate the occurrence of periodic weighting zones.

This manuscript focuses on the applications of seismic tomography and shield pressure monitoring for delineating hazards caused by stress and geologic anomalies. Results from case studies in bump-prone coal mines İn Australia, Germany, and the United States will be presented.

2 THE ROCKVISION3D™ SYSTEM

The RockVision3D™ system consists of commercially available hardware and proprietary software. The system hardware for Iongwall operations is simple and İs designed to measure acoustic noise generated by the shearer cutterhead during coal excavation. In general, the hardware is comprised of an intrinsic safety barrier (for gassy mine applications), geophones that are attached to roof bolts in gate roads, and cables that carry the signal to a seismic data acquisition system. The entire system is located İn the mine, and the data can be easily transferred to the surface or any mine office location through a monitoring and control network.

The system software combines wavefront and curved ray theories to reconstruct color-coded attenuation or velocity tomograms of the region to be imaged. The velocity and attenuation of seismic waves are directly related to the elastic constants, which characterize the type and condition of the rock medium. The paths followed by seismic waves from a source (i.e. shearer) to individual geophone receivers are represented as velocity or attenuation rays. The ray paths can be straight or curved depending on differentiation of the velocity and attenuation of seismic waves in the rock. In a uniform rock, the velocity and attenuation ray paths are generally straight. However, the ray paths are not normally straight, but rather bend (refract, curved ray) depending on the physical properties of the medium or the velocity contrasts between various material units (Rock et al. 1997). For example, in mining, as the seismic signals (waves) travel through rock, geological anomalies or highly stressed/fractured zones ahead of excavation absorb the vibrations, bend the ray, and attenuate me signal. As a result, variations in the measured magnitude of the seismic signals at the sensors are produced. Typically, in Iongwall mining, attenuation tomograms are generated and are related to fracturing and stress under the assumption that an area of higher stress results in microfracture closure and, mus, lower attenuation levels. The results of these tomograms are used to observe stress behavior and ground conditions ahead of mining (Westman et al. 1996).

2.1 *Field applications*

The RockVision3D™ technology has been used successfully in mines İn the United States, Australia, Germany, Ireland, South Africa, Poland, and Canada for stress and geological mapping, ore deposit delineation, solution cavity location, water migration pathways identification, and characterization of ground conditions within tunnel alignments. The following are some RockVision3D™ applications in coal mines.

2.1.1 *Rock VisionSD ™ monitoring in Germany*

At a 1,130-m-deep Iongwall mine in Germany, where high stress zones on the face are known to contribute to severe ground control problems such as face bumps, RockVision3D™ was applied to detect stress concentration zones and to delineate bump-prone areas in the Iongwall panel. The Iongwall panel utilizing the single-entry system was 1,300 m long and 323 m wide. The entries were 5.9 m wide and 4.2 m high. Mining height at the face was approximately 2 m. The panel was oriented approximately S50° E, advancing from SW to NE, and was located between two major fault zones. The immediate roof consists of a massive sandstone layer approximately 20 m thick and contributes to stress concentration along the Iongwall face. The immediate roof is overlain by strong competent shale/sandstone layers of various thicknesses.

After mining 780 m, the Iongwall face approached these two major faults. The first fault had a displacement of 4.3 m and was filled with sandstone. This fault appeared between shields 202 and 205, approximately 12 m from the tailgate rib.

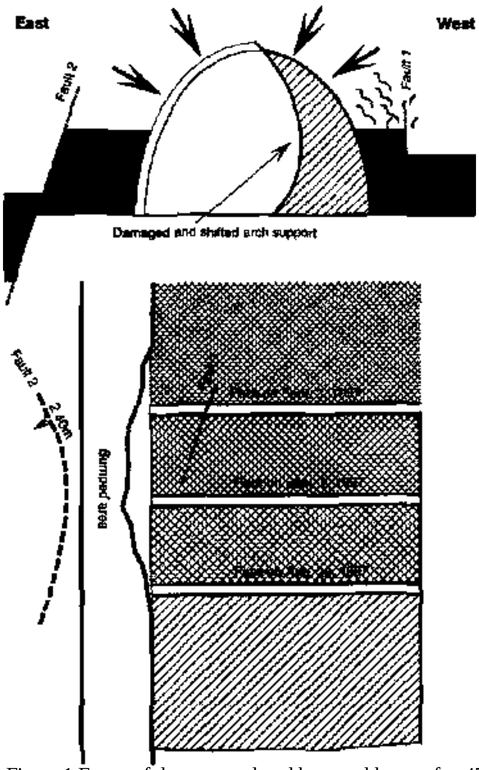


Figure 1 Extent of damage produced by a coal bump after 450-m advance of the longwall panel and location of associated faults.

The second fault, with a displacement of 2.4 m, was located approximately 72 *m* from the headgate rib. Slippage along faults within the coal seam had contributed to significant bumps. For example, after 450 m of advance, when the tailgate was between these two faults at approximately 10 m ahead of the face, a significant bump occurred. The released force pushed the panel rib into the tailgate entry, as shown in Figure 1. "This caused substantial damage to the arch support system. The damage extended approximately 13 m along the tailgate entry.

The factors contributing to structural stability problems m this mine are numerous and present very complicated problems in predicting potential bump locations; however, most of the bump problems are typically associated with high stress conditions and occur on the face. Timely mapping of high-stress concentration zones and geological anomalies ahead of mining can be beneficial to ground control and safety at this mine.

2.1.2 Analysis of roof caving

To evaluate the face bump potential for the mine setting and to characterize roof caving behind the shields, an analytical evaluation was conducted of cantilever and fixed beam span over elastic foundations subject to exponentially distributed abutment stresses. The mined seam and overlying strong roof units configuration were modeled using the Coal Bump Potential Evaluation Program (CBPEP), developed at Virginia Polytechnic Institute and State University, Blacksburg, Virginia (Wu et al. 1995, & Haramy et al. 1988). The program allowed determination of (1) critical roof span, (2) induced foundation stresses, and (3) strain energy stored in the roof and foundations. Dynamic failure potential immediate to the failing span is characterized by local Richter magnitude (ML) and induced maximum stresses in the roof and foundation.

The following summarizes beam lengths and local Richter magnitudes (ML) calculated for the current longwall panel. If the local Richter magnitude exceeds *2.0,* bump-prone conditions are assumed to exist.

- Fixed beam {first cave) length, m 114.91
- First cave Richter magnitude, (M_L) 5.26
- Periodic weighting cantilever beam length, m 8.96
- Cantilever beam failure Richter magnitude, (M_L) 3.31

The above results indicate that, based on the input parameters, the sandstone setting at the mine is bump-prone, and may incur substantial damage from dynamic events associated with die first cave and/or cantilevenng channels "periodic weighting" during panel retreat. Although the existence of a 20-m-thick competent roof layer in close proximity *to the* coal seam contnbutes to bump conditions, a combination of many other factors may act independently or together to promote the following high-stress conditions *on the face:*

a. High depth of cover;
b. Presence of geological structures;
c. Mining of a seam 170 m above;
d. Massive 20-m-thick sandstone above the seam;
e. High strength gob-sealing walls built along the tailgate entry, affecting caving of the gob;
f. Panel orientation with respect to fracture zone in the roof.

2.1.3 *Interpretation oftomogfaphic images*

The RockVision3D™ system was installed to determine if tomographic images produced on a daily basis could be used to detect zones of high stress concentrations on the longwall face and delineate bump-prone areas in the panel. Sixteen geophones were installed, with eight in the headgate and eight in the tailgate. The geophones were attached to angled roof bolts at 15- to 20-m spacings, with the closest geophones 10 to 15 m head of the face. The vibration signal produced by the shearer appeared to be transmitted clearly through the rock mass. The data were transmitted through a modem to the surface so that the mine could remotely monitor and analyze data on the surface quickly and cost-effectively.

The tomograms produced over a period of five days showed lower stress zones extending from the middle of the face toward the tailgate. Figure 2 shows tomograms produced on February 10 and 11. The fault present at the tailgate (left) side of the face diverts tbe velocities. This fault does not appear to build strain energy. The fault at shield 50 (right-hand side) had a major effect on how the seismic rays traveled. This fault appeared to contain highly fractured rock to within 25 m ahead of the face. The fractured zone continued to move ahead of the face, and a high-stress zone developed 50 m ahead of the face between shields 1 and 48. The fault, in combination with the overlying massive sandstone, contributes to a high stress concentration between shield 50 and 120. This zone disappears when the roof member caves. The tomograms indicate that the forward abutment extends up to 40 m, widi the highest stress zone occurring between 10 to 15 m ahead of the face (Hanna et al. 1998).

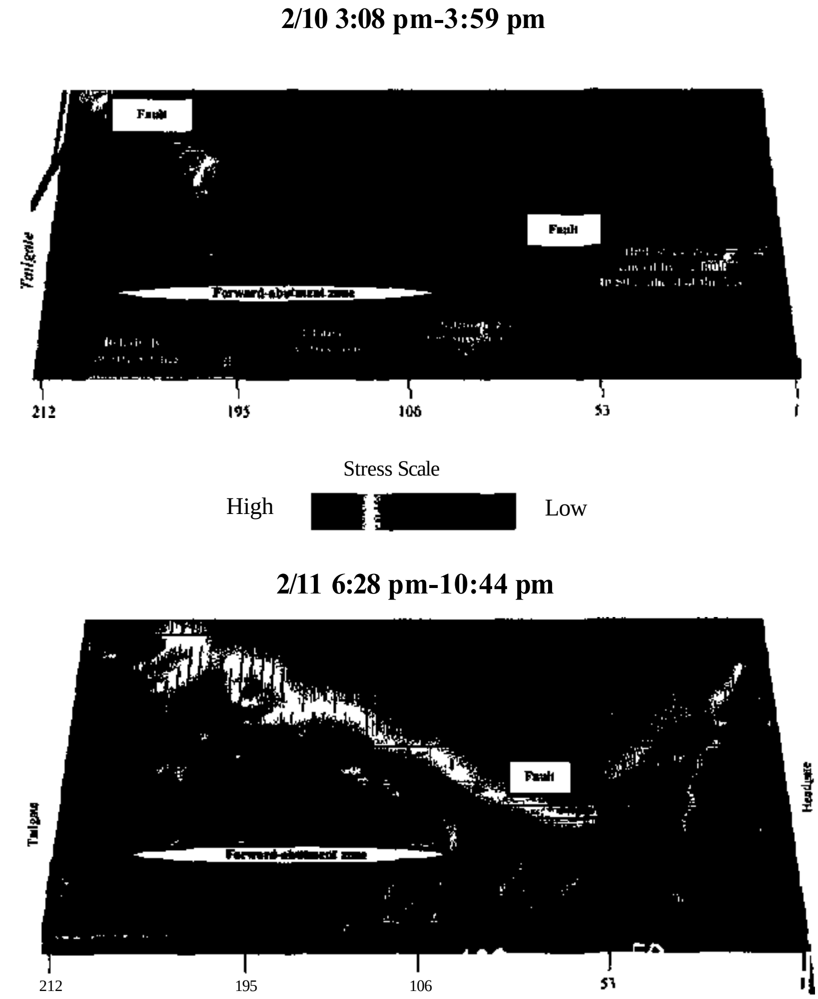


Figure 2. Fault and forward abutment mapping ahead of mining - Germany.

2.1.4 *Comparison between tomographic images and drilling*

Typically, the drilling-yield method is used by the mine operator to locate the areas of high-stress zones on the longwall face. Once the high-stress zones are identified, the auger drilling method is then used to destress such areas. This method can also give reliable information on the general stress conditions in the coal seam. Results indicate that on the average, the high stress zones occur between 10 and 13 m ahead of mining. The extent of the high-stress zones obtained from the auger drilling method coincides with the measurements obtained from the tomographic imaging data.

To further verify the accuracy of the tomographic images, the mine decided to use the drilling-yield method in areas depicted in the tomograms as "Very High, High, and Low." The results are summarized in Figure 3 and show high accuracy in correlating stress areas obtained from seismic tomography with those obtained from the drilling-yield method. Using the tomogram produced on February 9, the stress results obtained from RockVision3D™ and the drilling-yield method matched perfectly, and the data from February 10 matched at about 80%. Other comparisons were conducted and gave a similar high level of correlation. This information indicates that the technology can be used for real-time detection of stress concentration on longwall faces and the determination of the effectiveness of destressing methods.

2.2 *RockVision3D™ monitoring in Australia*

At an Australian longwall coal mine, a RockVision3D™ study was conducted to determine if the system could predict periodic shield loading. The mine was experiencing difficulties with face control due to the onset of periodic loading on the shields, possibly caused by the presence of massive sandstone in the roof.

Figure 4 shows the tomograms that were produced from the study. The first tomogram (2/10, 11:12 a.m.-12:34 p.m.) shows an elevated stress zone (red) in the mid-face area, approximately 30-50 m from the face. High attenuation zones (purple to blue) are shown on the outer portions of the area near the tailgate and maingate. The second image (2/11,4:36-5:38 a.m.), covering a period of one hour early the next day, shows enlargement of the high stress zone from the previous day. The zone is similar in shape to the previous day, but has grown outward as the face approached. The third image (2/18, 5:20-6:25 a.m.), seven days later (the face only advanced a short distance due to production delays during the intervening period), shows that the high stress zone has changed shape slightly and moved closer to the face. The lower left lobe of high stress has faded away however. The fourth image, showing the situation 11 hours later, indicates a significant stress relief in the mid-part of the face. This suggests that the bridging sandstone had broken, lowering stress on the immediate face line. Areas of high stress developed to the lower left and right, similar to the pattern for the first image, indicating a cyclical pattern of stress development.

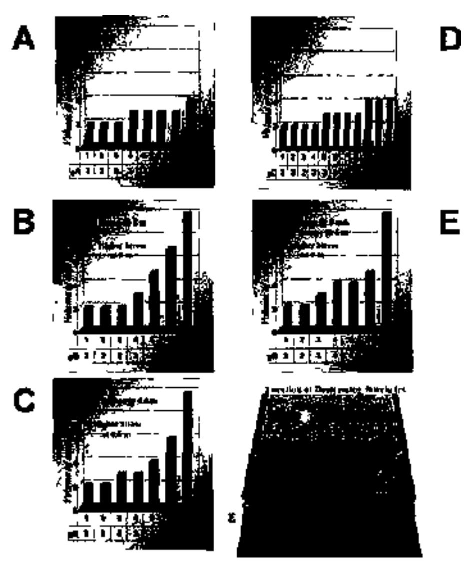


Figure 3 uestressing in a deep coal mine - Germany.

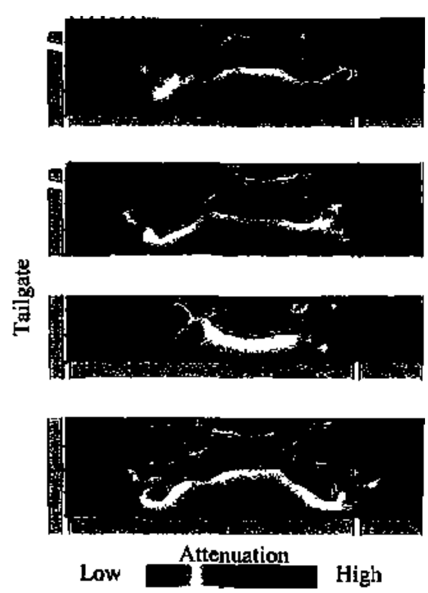


Figure 4. Detection of penodic shield loading - Austrasha.

During the period monitored, failure of fractured roof rock ahead of the face was noted, which corresponded to the blue areas on the tomograms. The blue areas on the tomogram represent high attenuation, indicating that the rock had fractured, and was no longer competent.

The Rock3D™ images obtained during this short demonstration give an indication of the nature of cyclical weighting on the longwall fece. A longer period of monitoring would be required to better define the exact nature of the loading.

At another Australian longwall coal mine, fracture zones were causing occasional guttering in front of shield canopies and consequent delays. Several roof falls occurred that extended up to 80 shields along the face. Position and severity of roof falls were typically unpredictable. RockVision3D™ was deployed to determine if roof structural anomalies or fracture zones, along with stress concentrations, could be readily mapped ahead of the face. The longwall panel investigated was 250 m wide by 1,900 m long, with a mining height of 4.2 m and an overburden depth of 170 m. Eight geophones were installed on roof bolts in die tailgate only of the longwall panel at 15-m spacings.

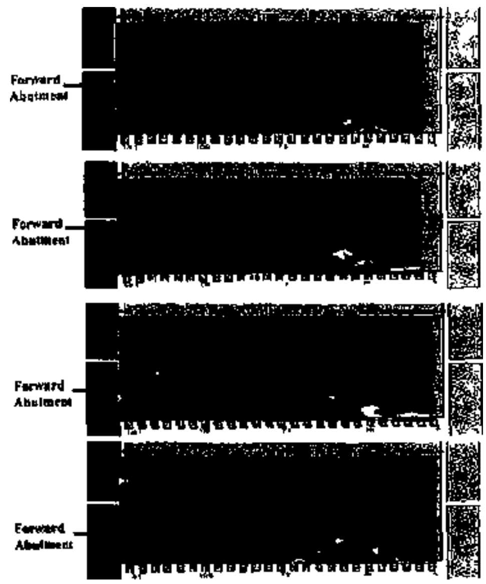
Tomograms developed from the data showed a pattern forming when roof falls occurred. A high-stress zone developed at approximately 50 m ahead of the face, which created a high shear zone close to the face. Due to the weakness of the roof rock, failure then occurred. After the fall, the high-stress zone moved closer to the face, and a smaller fracture zone was detected. Figure 5 shows four consecutive tomograms indicating the partem Üiat developed. The first, at 3:55 to 4:09 pm shows areas of high attenuation (dark blue) indicating areas of fractured rock not under appreciable stress. An area of light green, indicating low attenuation and high stress, is developing around this blue region. In the next tomogram for the period between 6:08 and 6:23 pm, the area of high stress that probably represents the front abutment is further developing and looping around the fractured material toward the tailgate.

The third tomogram at 6:59 to 7:17 pm shows the beginning of. a second high-stress zone, and the fourth tomogram at 10:32 to 11:14 pm shows the high-stress zone expanding and spreading across the face. During the period of these tomograms, sporadic hard cutting and yielding of shield legs occurred.

Subsequent tomograms show an increase in stress, which then migrates toward the maingate side of the face, and the cycle appears to terminate. For the two days following this, difficult face conditions were encountered, with significant spalling and guttering, and failure of most of the mid-face zone.

Over the following days, tomograms were produced that confirmed the pattern of stress abutment development, with stress circumscribing the tailgate corner. The elevated stress zone appears to migrate toward the face until me rock became fractured and developed high attenuation, ultimately leading to failure on the face.

Image analysis and underground observation indicate that the extent of the mining-induced stress zone extended 45 to 50 m ahead of the face with significant mining-induced loads at 18 m. The most critical zone is approximately 2 m from the face in the tailgate area. This agreed well with the RockVÌsion3D™ results. At this mine, RockVisiorßD™ was considered a reliable tool in mapping patterns of stress buildup ahead of the mining face.



FigureS. Detection ofa pattern developing prior to roof failure - Australia.

3 THE GEOGUARD™ SYSTEM

The principal goal of the shield monitoring implementation is to provide an early warning system to alert the operator to an imminent occurrence of severe periodic weighting events. The average loads and/or load increments experienced by the face supports during each cycle have been shown to provide a reliable indicator of weighting conditions. Automated analyses of historical data provide estimates of the weighting interval, and warnings are generated when the face approaches a predicted weighting zone and when loads became unusually high.

Earlier research in this area (Conover et al. 1994) had success using support loads to predict ground conditions; however, a separate monitoring system was required, and the software analysis functions were performed off-line, requiring significant operator time to process and interpret. Modem installations include the necessary interfaces to permit collecting and transmitting the support data through existing monitoring and control networks. To obtain maximum advantage from existing installations, NSA modified the various software components into an integrated application that combines the functions of monitoring, analysis, interpretation, prediction, and warning.

3.1 *GeoGuard™ components and system layout*

Although GeoGuard™ is a software system that accesses real-time monitored data, the interface to the monitoring hardware is an integral part of the system and significantly influences the capabilities and performance of the software. Data from the supports are accessed using hardware and software components supplied by the support manufacturer, including an interface between the support controller and an Allen-Bradley monitoring and control network. The support data are periodically scanned and transferred to a PLC using custom, system-specific control programs. These data typically include the pressures of both support legs and possibly support convergence. Also, the shearer position İs normally captured and stored in a separate PLC location. Typically, updated data are collected every few minutes and whenever each support is reset.

GeoGuard™ connects to an Allen-Bradley network using the RS-Linx software bridge, available from Rockwell Software. RS-Linx permits accessing the network via a data highway, serial, or Ethernet connection. GeoGuard™ communicates with the RS-Linx program using software calls to set up data pathways and initiate the transfer of data to and from specific PLC memory locations. The GeoGuard™ and RS-Linx software are installed on a dedicated PC, and the system runs continuously, capturing data in real time.

Data are read at a user-specified scan interval (normally 1 minute) and are written to daily files to provide a permanent record. At the end of each support cycle, the time-weighted average pressure (TWAP) is calculated and written to a file. This loading history is analyzed to identify the pressures accompanying peak (periodic) loading zones and the intervals between zones.

The locations of future weighting zones are predicted using the average of past weighting intervals. A multi-stage alarm is generated based on the proximity of the face to the predicted weighting zone and the current support loading intensity relative to a pre-set threshold. The weighting interval predictions and pressure threshold values are automatically updated in order that the warning system adapts to changing ground conditions.

A variety of graphical displays are provided to evaluate the data visually. A real-time graphics display (Fig. 6) shows the current leg pressures and other parameters for the entire face. A two-dimensional trending function permits trending any of the monitored data points versus either time or face position, as shown in Figure 7. Three-dimensional plots permit viewing the support loading data superimposed on base maps of mine layout or geology to identify correlations between loading behavior and any conditions that may contribute to excessive weighting occurrences.

3.2 *GeoGuard™ monitoring in the United States*

GeoGuard™ was installed at a longwail operation İn the western United States (Hanna & Conover, 1998). Support leg pressures were monitored on 12 shields, evenly spaced along the face, during die mining of five panels. An independent monitoring system transmitted the data in real-time to the main office, 240 km from the mine. Processing was conducted off-line and correlated with mine geologic maps and reports of ground control conditions.

The coal seam was 320 m deep, 3 m thick, and had a slight dip oblique to the panel layout. The seam and surrounding strata were relatively uniform throughout the monitoring area. The roof consisted of three strong sandstone layers extending approximately 14 m above the seam. The main ground control problems were bumps that occurred in the tailgate, resulting İn significant floor heave and subsequent down-time required for cleanup. Some rapid shield-loading events associated with the bumps were also observed.

Review of the TWAP data revealed a pattern of periodic loading, with pressure peaks of different intensities occurring at different intervals. It was determined that the pattern was indicative of the caving behavior of the tiiree main roof layers. Figure 8 shows an idealized plot of TWAP versus face position and a diagram of the roof strata. The low-intensity pressure peaks, at intervals of 2.5 m, coincide with caving of the immediate, lowest roof layer (A). As mining progresses, the upper layers form cantilevers extending behind the shields and fail when the underlying layers cave and their support is removed. The middle layer (B) caves at intervals of 7.5 m accompanied by somewhat higher support pressures, and the upper layer (C|) caves at intervals of 30 m and produces the largest pressure peaks.

The current version of GeoGuard™ contains an automatic system for identifying pressure peaks, calculating the periodic weighting interval between peaks, and alerting the operator when the face approaches periodic weighting zones.

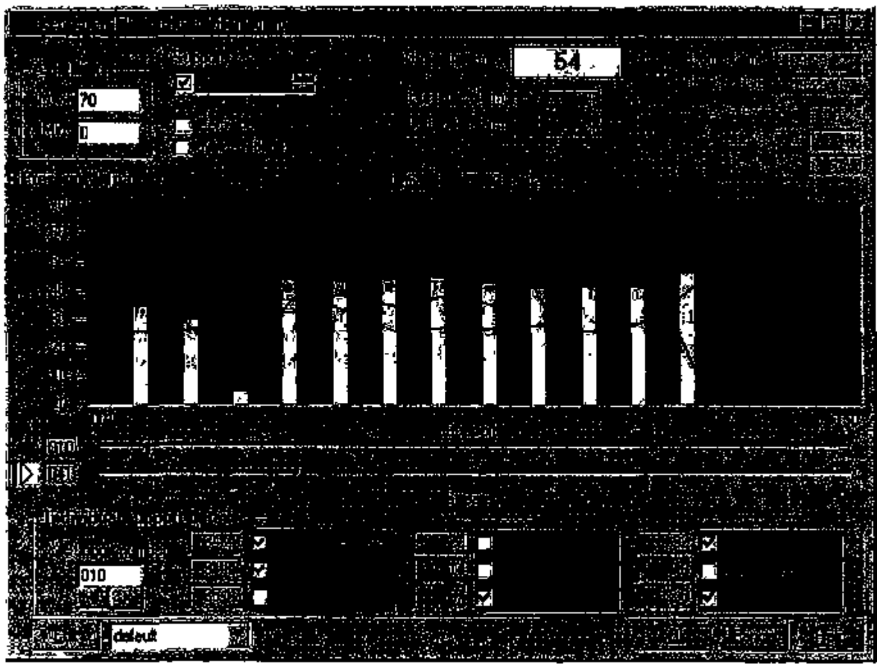


Figure 6. GeoGuard™ real-time monitoring display showing shield leg pressures, alarm state, and other parameters related to detection of penodic weighting pressures and intervals.

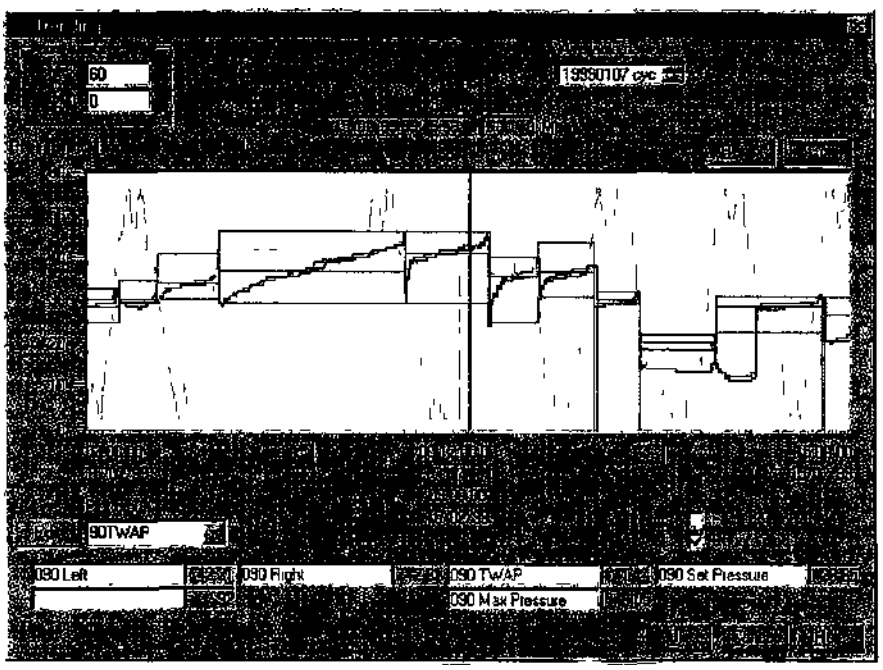


Figure 7. Two-dimensional GeoGuard™ trend plot of let pressures including calculated cycle parameters and shearer position

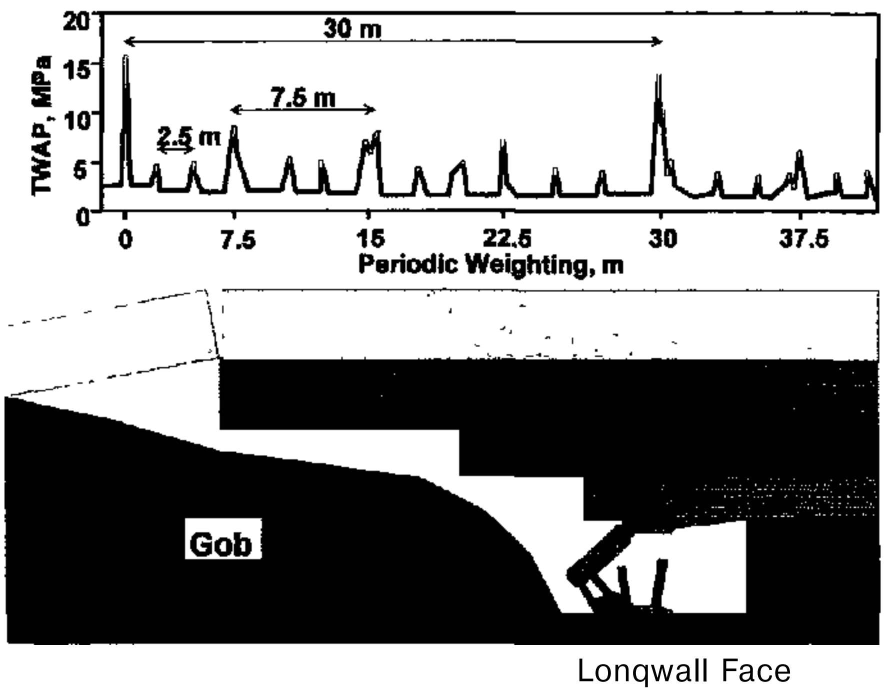


Figure 8. Correlation of periodic shield weighting with roof caving sequence Layer A caves every 2 5 m, layer B every 7.5 m and layer Ci every 30 m.

3.3 *GeoGuard™ monitoring in Australia*

GeoGuard™ was installed at a longwall operation in New South Wales, Australia (Conover & DeMarco, 1999). Transducers were installed on every support, and convergence meters were installed on several supports. The data were accessed from the surface through an existing Allen-Bradley control network.

The longwall panels were located under a 24-m-thick sandstone unit immediately overlying the coal seam, which resulted in several sudden and severe face weightings, three of which resulted in severe overloading of the supports, with significant equipment damage and several weeks of downtime. During these weighting events, vertical convergence of up to 200 mm was measured along the panline.

The strata overlying the seam were generally characterized by a strong, massive sandstone residing immediately above the seam, overlain by moderately strong sandstone and shale sequences, with interbedded weak coal and shale layers. The primary contributors to the weighting behavior were determined to be the existence of the massive sandstone unit, minimal cover depth, and a lack of significant jointing in the roof.

The first face-stopping weighting event involved approximately 30 shields toward the tailgate end of the face. Operational delays caused the face to stand idle for more than 90 minutes, during which time the face converged to such a point that the shearer could no longer travel the face. Mining was suspended for two weeks while the collapsed shields were cleared and repaired.

A second face-stopping event occurred approximately 200 m outby the first event. Once again, operational delays in excess of one hour contributed to the eventual stoppage of the face, and another week of downtime.

Face width is another factor that contributes to difficulties In mining through weighting events. Wider faces require additional shearer tramming per pass, thereby extending the time required to advance the wall through a weighting area. Over the course of a major weighting, 60 to 90 mm of closure can take place per shearer pass, with a rapid-onset event experiencing more than 200 mm per shearer pass.

GeoGuard™ can alert the operator when the face approaches weighting areas and when pressures increase prior to weighting events. Review of three-dimensional plots of the support loading history, such as Figure 9, can quickly identify patterns of loading that accompany periodic loading and panel advance through specific geologic conditions. With this knowledge, the operator can take steps to ensure that the area is mined-through rapidly, before excess convergence can occur.

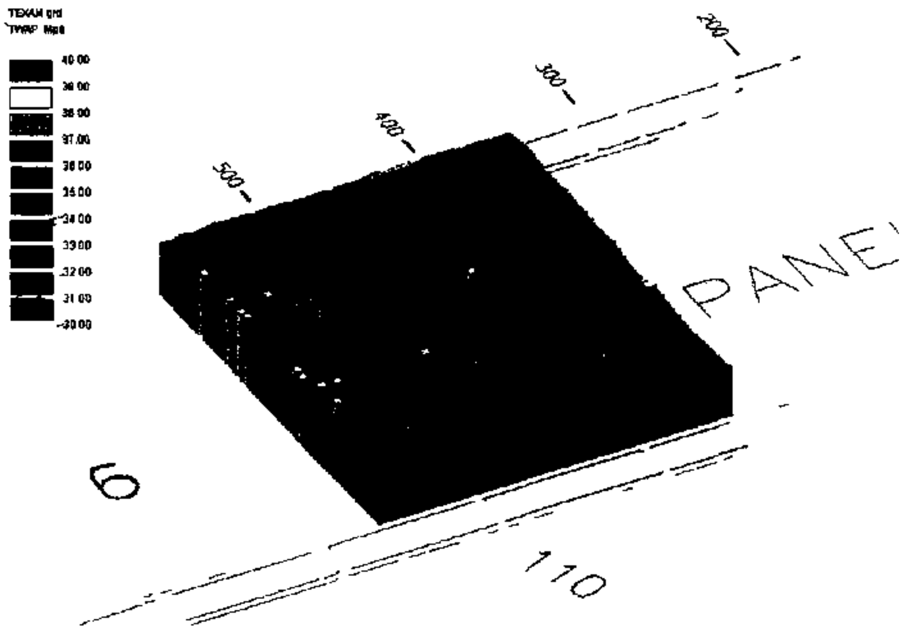


Figure 9 Three-dimensional GeoGuard™ plot of Time-Weighed \\i.rage Pressure (TWAP) overa penod of 22 days (210 m) Gale road shown at top is the headgate and race advance is toward the right

4 CONCLUSIONS

The conventional ground hazard detection methods to delmeate high stress areas and geologic anomalies have fallen far short of meeting both personnel safety requirements and the high production demands of today's mechanized mining operations The RockVısıon3D™ seismic tomographic technology and GeoGuard™ shield monitoring system have evolved into powerful predictive tools for continuous non intrusive assessment of ground conditions without interference to daily mining operations, and without putting mine personnel at risk

REFERENCES

Conover, DP, & DeMarco, M J 1999 Shield Monitoring to Forecast Severe Face Weightings at the South Bulga Colliery *Proc of W Internationa! Conference on Ground Control in Mining,* Morgantown, WV

Conover, D P, McDonnell, J P , & Hanna, K. 1994 Computer assisted ground control management system USBM 1C 9408 25 pp

DeMarco M J & Haramy, K.Y 1997 New technologies for coal bump hazard detection and mitigation assessment Pres 28th Ann Inst On Mining Health, Safety and Research, Salt Lake City, UT, 15 pp

Hanna, K & Conover, DP 1998 Shield Pressure Monitoring A Key to High-Production Longwall Mining Systems *Proceedings of Advanced Toob & Monitoring Techniques for Mechanized Longwall Mining Conference,* Poland

Hanna, K. & Haramy, K- 1998 Ground control and geologic assessment in mining through the use of geophysical tomographic imaging *Proc Geomechantcs/Ground Control m Mining and Underground Construction Conf Wollongong NSW Australia 543 pp*

Hanna, K & Haramy, K- 1999 RockVısıon applications to delineate hazardous geologic and stress conditions in coal mines Case histones SME Preprint 99 94

Haramy K Y Magers J A & McDonnell JP 198S Mining under strong roof Proc of 7th International Conference on Ground Control in Mining Morgantown WV pp 179-194

Westman, E, Haramy, K & Rock, A 1996 Seismic tomography for longwall stress analysis *Proc 2nd North American Rock Mechanics Symposium Montreal Quebec Canada*

Wu, X, Karfakis, M G , Haramy, K. V & Brady, B T I995 Ground motion parameters associated with coal mine tremors SME Preprint No 95-49

1 f International Mining Congress and Exhibition of Turkey- IMCET 2001, © 2001, ISBN 975-395-417-4

New Approach to Study Load Transfer Mechanisms of Fully Grouted Bolts

N.I.Aziz, A.Dey & B.Indraratna
Faculty of Engineering, University of Wollongong, NSW 2522. Australia

ABSTRACT: The load transfer mechanisms across the rock/resin/bolt interfaces are governed by the surface properties of the bolt. Although there are various bolt surface configurations available in the market, very little work is reported on die bolt/resin interface failure mechanism. This paper examines the behaviour of bolt surface roughness under constant normal stiffness conditions which is considered as being a realistic way of evaluating bolt surface roughness. To study the shear behaviour of bolt/resin interface, laboratory tests were conducted on the bolt surfaces of two most popular types currently in use in Australian coal mines, at an initial normal stress of 0.1 to 7.5 MPa. The study snowed different shearing characteristics of both bolt types under similar loading environment. Field investigations carried out in a local coal mine produced similar results as obtained from the laboratory testing.

1 INTRODUCTION

The load transfer characteristics of the bolt play an important role in the design of effective support system for stabilising rock mass in various types of excavation. The load transfer characteristics of a bolt in the field largely depend on the behaviour of its surface properties among other parameters. In the recent past, the shear stress developed at bolt-resin interface has been calculated by the strain gauged instrumented bolts (Fabjanczyk & Tarrant 1992, Fuller & Cox 1975, Gale 1986 and Signer, Cox & Johnston 1997), and the shear stress developed at any point along the bolt length could then be calculated by the following formula:

$$\Delta\tau = \frac{F_1 - F_2}{\pi d l} \qquad (1)$$

where,
$\Delta\tau$ = Shear stress at bolt-resin interface,
F_1 = Axial force acting on the bolt at strain gauge position 1, calculated from strain gauge reading,
F2 = Axial force acting on the bolt at strain gauge position 2, calculated from strain gauge reading,
d = Bolt diameter, and
1 = Distance between strain gauge position 1 and strain gauge position 2.

One of die major shortcomings of die above mediod is that, it does not consider the effect of horizontal stress or the confining pressure on the shear stress at bolt/resin interface. As the confining pressures or the horizontal stresses around the opening play an important role İn the failure mechanism of grouted rock, incorporation of the confining pressure in the above formula would result in a better approximation of the İn-situ condition. Following the installation of a bolt in the field, the relative movement, however small, between the rock and the bolt causes the load to be transferred on the bolt, and as a result the normal stress is applied on the resin/rock interface through the ribs of the bolt. The magnitude of normal stress İs dependent on the relative displacement, shape of the rib profile and the composite stiffness of the bolt/resin/rock interface. The Constant Normal Stiffness (CNS) condition thus represents a better approximation of the deformation behaviour in the field as compared to conventional Constant Normal Load (CNL) condition. The above hypothesis has been suggested by many researchers (Benmokrane &c Ballivy 1989, Indraratna, Haque & Aziz 1998 and Ohnishi & Dharamaratne 1990). A novel approach was, therefore, adopted to study the shear behaviour of bolt/resin interface under Constant Normal Stiffness condition.

2 BOLT SURFACE PREPARATION

A 100 mm length of a bolt was selected for the surface preparation for CNS shear testing. The specified length of bolt was cut and then drilled through. The hollow bolt segment was then cut along the bolt axis from one side and preheated to open up into a flat surface as shown in Figure 1. The surface features of me bolt (ribs) were carefully protected while opening up the bolt surface. The flattened surface of the bolt was then welded on the bottom plate of the top shear box of the CNS testing machine. Although these flattened bolt surfaces may not ideally represent the complex behaviour of circular shaped bolt surface observed in the field, nevertheless, they still provided a simplified basis for evaluating the impact of the bolt surface geometry on the shear resistance offered by a bolt. Table 1 shows the specification of two types of bolt used in the study, known as type I and type II bolts respectively.

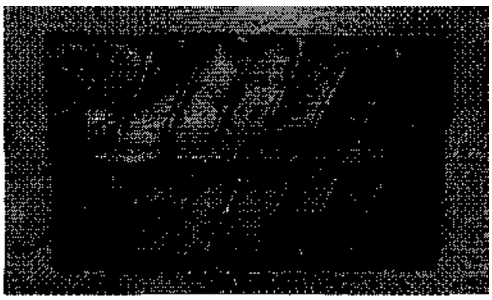


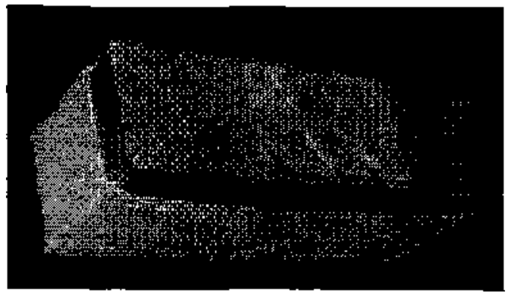
Figure 1. Flattened bolt surface.

Table I Specification of bolt types.

Bolt	*Core Diameter (mm)*	*Finished Diameter (mm)*	*Rib Spacing (mm)*	*Rib Height (mm)*
Type I	21.7	24.4	28.5	1.35
Typen	21.7	23 2	12.5	0.75

3 SAMPLE CASTING

The welded bolt surface on the bottom plate of the top shear box was used to print the image of bolt surface on cast resin samples. For obvious economic reasons, the samples were cast in two parts. Nearly three-fourth of the mould was cast with high strength casting plaster and the remaining one-fourth was topped up with a chemical resin commonly used for bolt installation in underground coal mines. A curing time of two weeks was allowed for all specimens before testing was carried out. The properties of the hardened resin after two weeks were, uniaxial compressive strength (σ_c) = 76.5 MPa, tensile strength (σ_t)= 13.5 MPa, and Young's modulus (E) = 11.7 GPa. The cured plaster showed a consistent σ_c of about 20 MPa, σ_t of about 6 MPa, and E of 7.3 GPa. Such model materials were suitable to simulate the behaviour of a number of jointed or soft rocks, such as coal, friable limestone, clay shale and mudstone, and were based on the ratios of σ_c/σ_t and σ_c/E applied in similitude analysis (Indraratna, 1990). The resin sample prepared in this way matched exactly with the bolt surface, allowing a close representation of the bolt/resin interface in practice as shown in Figure 2.



Hgure 2. A typical cast sample.

4 CNS SHEAR TESTrNG APPARATUS

Figure 3 is a general view of the CNS testing apparatus used for the study, which was a modified version of the similar equipment reported by Johnstone and Lam (1989). The equipment consisted of a set of two large shear boxes to hold the samples in position during testing. The size of the bottom shear box is 250x75x100 mm while the top shear box is 250x75x150 mm. A set of four springs are used to simulate the normal stiffness (k_n) of the surrounding rock mass. The top box can only move in the vertical direction along which the spring stiffness is constant (8.5 kN/mm). The bottom box is fixed to a rigid base through bearings, and it can move only in the shear (horizontal) direction. The desired initial normal stress (σ_{no}) is applied by a hydraulic jack, where the applied load is measured by a calibrated load cell. The shear load is applied via a transverse hydraulic jack, which is connected to a strain-controiled unit. The applied shear load can be recorded via strain meters fitted to a load cell. The rate of horizontal displacement can be varied between 0.35 and 1.70 mm/min using an attached gear mechanism. The dilation and the shear displacement of the joint are recorded by two LVDT's, one mounted on top of the top shear box and the other is attached to the side of the bottom shear box

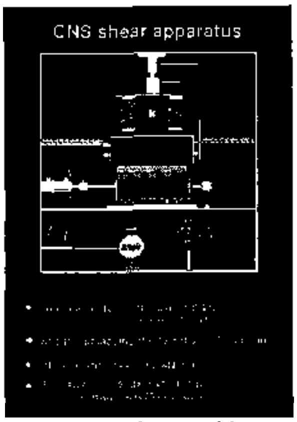


Figure 3 Line diagram of the CNS apparatus (modified after Indraratna et al., 1998).

5 TESTING OF BOLT/RESIN INTERFACE

A total of 12 samples were tested for two different types of bolt surface at initial normal stress (σ_{no}) levels ranging from 0.1 to 7.5 MPa. Each sample for bolt type I was subjected to five cycles of loading in order to observe the effect of repeated loading on the bolt/resin interface. Samples for bolt type II were subjected to only three cycles of loading, as it was found that me stress profile did not vary significantly after the third cycle of loading. The stress profile, as described above, is defined as the variation of shear (or normal) stress with shear displacement for various cycles of loading. The σ_{no} applied to the samples represented typical confining pressures, which might be expected in the field. A constant normal stiffness of 8.5 kN/mm (or 1.2 GPa/m when applied to a flattened bolt surface of 100 mm length) was applied via an assembly of four springs mounted on top of the top shear box. The simulated stiffness was found to be representative of the soft coal measure rocks. An appropriate strain rate of 0.5 mm/min was maintained for all shear tests. A sufficient gap (less than 10 mm) was allowed between the upper and lower boxes to enable unconstrained shearing of the bolt/resin interface.

6 SHEAR BEHAVIOUR OF BOLT/RESIN INTERFACE

6.1 *Effect of Normal Stress on Stress Paths*

Figure 4 shows the shear stress profiles of the bolt/resin interface for selected normal stress conditions for the type I bolts. The difference between stress profiles for various loading cycles was negligible at low values of σ_{no} (Figure 4a). This was gradually increased with increasing value of σ_{no} reaching a maximum between 3 and 4.5 MPa (Figure 4b). Beyond a 4.5 MPa confining pressure, the difference between stress profiles for the loading cycles I and II decreased again (Figure 4c). A similar trend was also observed for the type II bolt surface (not shown in figure).

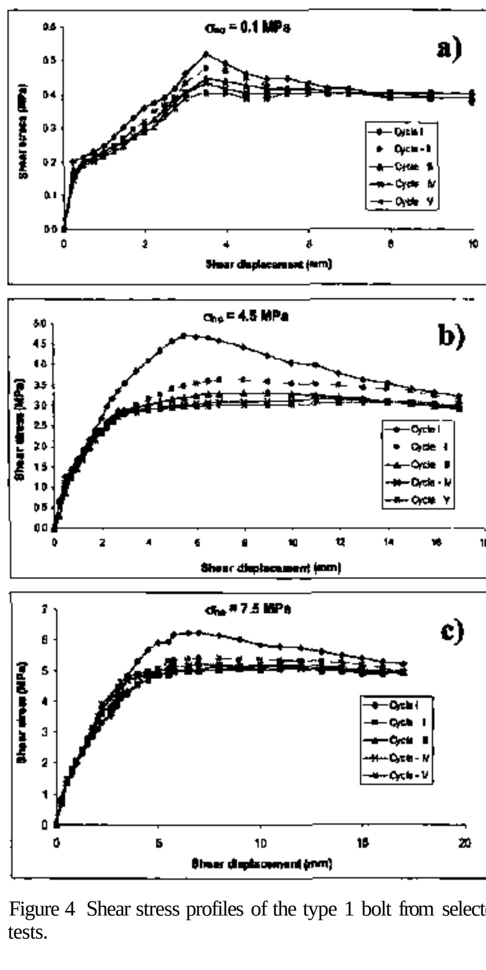


Figure 4 Shear stress profiles of the type 1 bolt from selected tests.

At low σ_{no} values, the relative movement between the bolt/resin surfaces caused an insignificant shearing and slickensiding of the resin surface, thus keeping the surface roughness almost intact. For each additional cycle of loading, the shear stresses marginally decreased, especially in the peak shear stress region. However, as the value of σ_{no} was increased, the shearing of the resin surface was also increased, and the difference in stress profiles for various cycles of loading became significant.

6.2 *Dilation Behaviour*

For the first cycle of loading, Figures 5a and 5b show the variation of dilation with shear displacement at various normal stresses for type I and type II bolts, respectively. For various values of σ_{no}, the maximum dilation occurred at a shear displacement of 17 - 18 mm and 7 - 8 mm, for type I and type II bolts, respectively {Figures 5a and 5b). The distance between the ribs for both bolt types İs shown in Table 1. Therefore, İt may be concluded that die maximum dilation occurred at a shear displacement of about 60% of the bolt rib spacing.

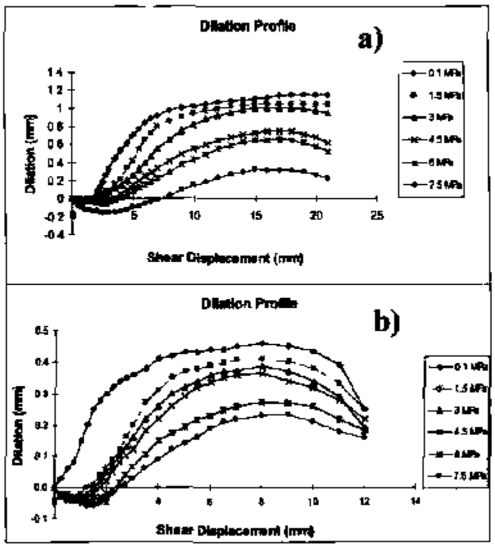


Figure 5. First loading cycle dilation profiles at various σ_{no} values., a) type I bolt, and b) type II bolt

6.3 *Effect of Normal Stress on Peak Shear*

Figures 6 and 7 show the variation of shear stress *with* shear displacement for the first cycle of loading at various normal stresses, for both type I and type II bolts, respectively. The shear displacement for peak shear stresses increased with increasing value of σ_{no} for both bolt types. This was due to the increased amount of resin surface shearing with the increasing value of σ_{no}. However, there was a gradual reduction in the différence between the peak shear stress profiles with increasing value of σ_{no}. The shear displacement required to reach the peak shear strength is a function of the applied normal stress and trie surface properties of the resin, assuming that the geometry of the bolt surface remains constant for a particular type of bolt as evident from Figures 6 and 7.

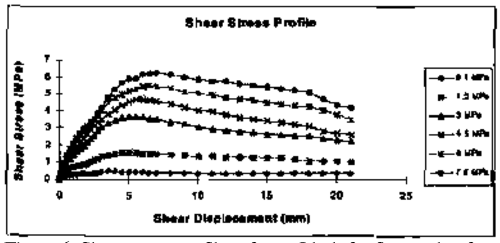


Figure 6. Shear stress profiles of type I bolt for first cycle of loading.

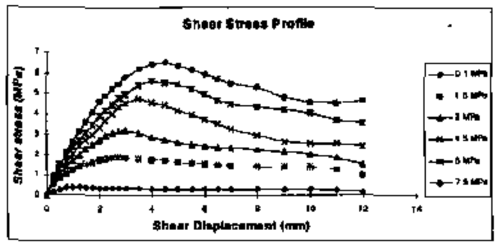


Figure 7. Shear stress profiles of type II bolt for first cycle of loading

6.4 *Effect of Cyclic Loading on Peak Shear*

Figures 8 and 9 show the variation of peak shear stress with normal stress applied for type I and type II bolts for various loading cycles. For the type I bolt surface, the graphs of cycle I through cycle III show a bi-linear trend, whereas the graphs representing cycles *TV* and V show only a linear trend. For the type II bolt surface, only cycle I shows a bi-linear trend and cycles II and III show a linear trend. At low initial normal stress, the shearing of resin surface is negligible, and hence, the rate of increase of peak shear stress with respect to normal stress is high. At higher normal stress, the degree of shearing of resin surface is greater, and some of the energy is thus utilised to shear off the resin surfaces. As a result, there is retarded rate of increase in peak shear stress with respect to the normal stress.

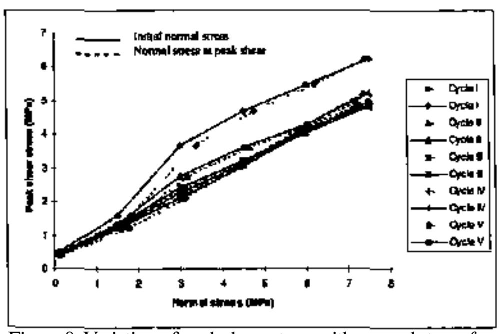


Figure 8. Variation of peak shear stress with normal stress *for* type I bolt.

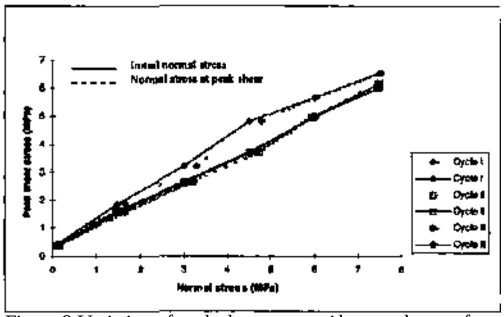


Figure 9 Variation of peak shear stress with normal stress for type [[bolt

As the samples are loaded repeatedly, the resin surfaces become smoothened reducing the surface roughness, and as a result, the rate of increase of peak shear stress is likely to remain constant with respect to the normal stress.

6.5 *Overall Shear Behaviour of Type I and Type II Bolts*

Figure 10 shows the shear stress profiles of both type I and type II bolts for the first cycle of loading. The following observations were noted.

- The shear stress profiles around peak were similar for both bolt types However, slightly higher stress values were recorded for the bolt type I at low normal stress levels, whereas slightly higher stress values were observed for the bolt type II at high normal stress levels, in most cases.
- Post peak shear stress values are higher for the bolt type I indicating betteT performance in the post-peak region.
- Shear displacements at peak shear are higher for the bolt type I indicating the safe allowance of more roof convergence before instability stage is reached.
- Dilation is greater in the case of bolt type I

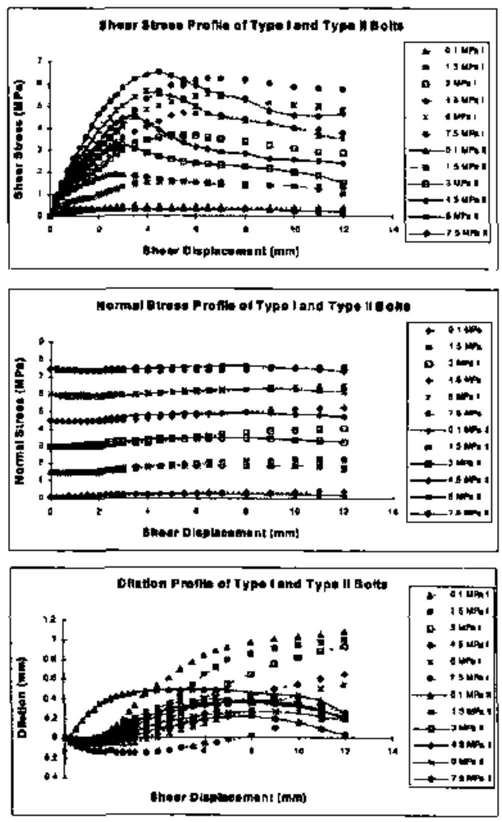
6.6 *Effect of Normal Stiffness*

The laboratory experiments were carried out with spring assembly with an effective stiffness of 8.5 kN/mm. In practice, the stiffness of resin/rock system will be usually higher than the laboratory simulated stiffness. As the stiffness increases, the effective normal stress on the bolt/resin interface at any point of time will also increase, as per the following equation:

$$\sigma_n = \sigma_{no} + \frac{k_n . \delta_v}{A} \qquad (3.1)$$

Where,

σ_n = effective normal stress,
σ_{no} = initial normal stress,
k_n = system stiffness,
δ_v = dilation, and
A = area of the bolt surface



FigurelO Comparison of stress profile and dilation of type 1 and type II bolts for first cycle of loading

In general, higher values of effective normal stresses should be observed for the type I bolt as compared to the type II bolt as long as the confining pressure remains low. Higher values of effective normal stress will have a direct positive impact on the peak shear stress values and, therefore, when installed in the field, the type I bolt would outperform the type II bolt, particularly at low confining pressure conditions. However, in deep mining conditions, where the high stress conditions prevail, the reverse situation may occur because of the greater contact zone between the resin surface and the closer spaced bolt ribs in bolt type II. This increase in contact surface area would require greater shearing force necessary to fail, which might not he the case in wider spaced ribs in bolt type I, and that explains the reason for the stability of the bolt type II in deep mine applications.

7 FIELD INVESTIGATION

In an attempt to verify the laboratory findings, with respect to the influence of bolt surface profiles on load transfer mechanisms under different testing environments, a program of field investigation was undertaken in a local coal mine, known as Mine A. The mine is located at Douglas Park about 80 km South-West of Sydney, NSW, and mines coal from Buili seam, 3m thick, at a depth of about 480m. The mine produces around 1.5 mt of coal from a 230m longwall face and heading development operations. The Buili seam is overlain with a succession of moderately strong roof layers consisting mainly of sandstone, mudstone, siltstone and shale. Figure 11 shows the general plan of the test site. The direction of the principal horizontal stress (σ_1=25 MPa) at the instrumented site was nearly parallel to the headings axis. However, and because of the presence of the dyke, there were some variations in the overall principal horizontal stress orientation, particularly at the outbye of the dyke as shown in Figure 11.

The Geld investigation program consisted of installing 12 instrumented, 2.4m long, strain gauged bolts in two adjacent main entries to die longwall panel as shown in Figure 12. Eighteen strain gauges were housed in each bolt in two diametrically opposite channels (6mm x 3mm). In addition to the instrumented bolts, three extensometer probes were also installed between the two rows of instrumented bolts in each gateroad. Each extensometer location housed 20 magnetic reference points above the roof. The notations adopted for the bolts and extensometer probes in Mine A is explained in Figure 12.

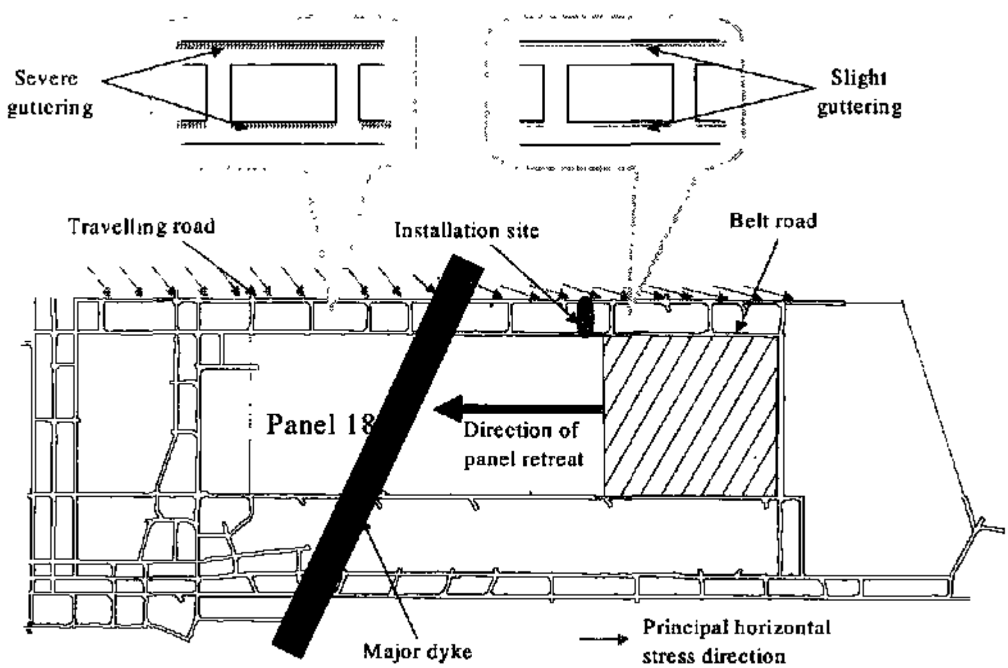


Figure 11. Detail layout of the panel under investigation at Mine A.

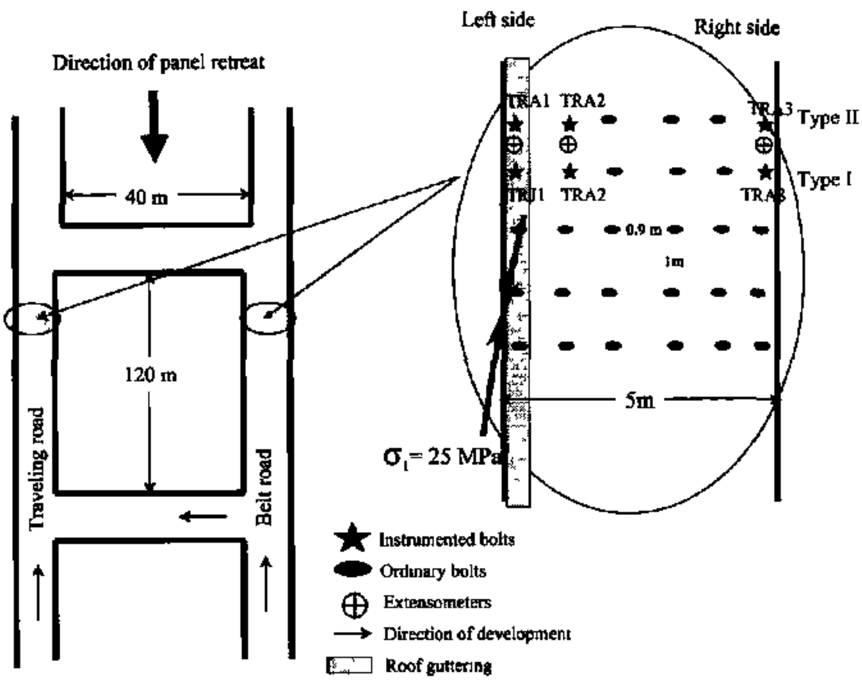


Figure 12. Detail layout of the instrumentation site at Mine A

8 RESULTS AND DISCUSSIONS

Field monitoring commenced immediately following the completion of site instrumentation, monitoring ended when the site was overrun by the retreating longwall face. The total period of site monitoring was S months.

8.1 *Load transfer during the panel development and the longwall retreating phases*

Figure 13 shows the overall load transferred on the type II bolts during panel development and subsequent longwall retreating phases, installed at the left side of the travelling road (numbered as TRA1 in Figure 13). As expected, the load transferred on the bolts during the panel development stage was relatively low as compared to the longwall retreating phase in both the travelling and belt roads. The load build up due to the front abutment pressure of retreating longwall face was, on the average, 5 to 8 times higher than that of panel development phase (see Figure 13).

The maximum load transferred to the bolt BRJ2 at a particular face position, during panel development and longwall retreating phases, is shown in Figure 14. The load transferred during the panel development stage became constant within 40m advance of development headings, away from the instrumented sites. However, the load build up due to the front abutment pressure of the approaching longwall face began to increase significantly, when the distance was 150m from the test sites.

8.2 *Load transfer m the belt and travelling road*

Figure 15 shows the maximum load transferred on to BRJ2 and TRJ2 for a particular face position, during the panel development and the longwall retreating phases. In both gateroads, the load on the bolts started to build up immediately after their installation during the panel development stage, and then became constant when the heading development face was about 50m away from the test sites (see darker lines in Figures 15a and 15b). During the longwall retreating phase, however, the impact of front abutment pressure was observed (by sharply increasing load), when the approaching longwall face was around 60m away from the test site in case of travelling road (Figure 15a). In case of belt road, the same was observed when the approaching longwall face was about 150m away from the test site (Figure 15b). Thus, the impact of longwall face movement was more prominent in case of belt road as compared to the travelling road.

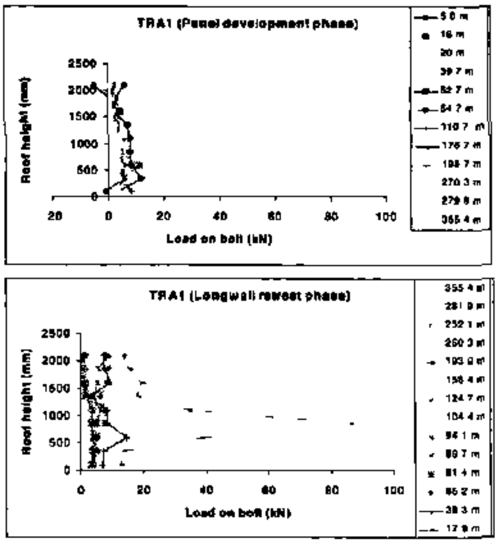


Figure 13. Load transferred on the bolt TRA1, during the panel development and longwall retreating phases, Mine A

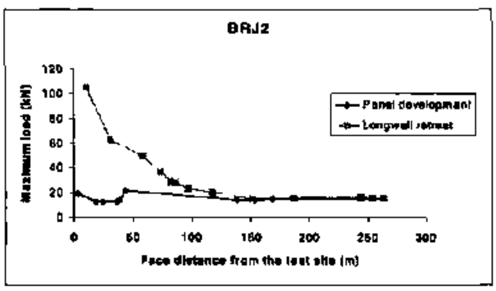


Figure 14 Maximum load transferred on the bolt BRJ2, for a particular face position, Mine A.

8.3 *Behaviour of strata deformation*

Figure 16 shows the overall roof deformation recorded from the extensometry readings in the travelling road. As expected, the maximum deformation was recorded in the middle of the road (Figure 16b) because of the prevailing near parallel principal horizontal stress direction, and was aggravated by the deadweight of the separated sagging roof. The amount of roof deformation recorded at the left side of the gateroad (Figure 16a) was relatively small as compared to the middle section (Figure 16b), but was greater than that on the right side (Figure 16c) of the roadway. Also, the acute angle between the horizontal stress direction and the axis of the gateroads caused some shearing in the immediate roof at the left side of the gateroads, while the other side of the gateroads was relatively stable, with negligible amount of strata deformation (Figure 16c). No significant strata deformation was observed at a horizon level of more than 3m from the roof level. Thus, it may be suggested that, in addition to the regular bolt pattern at Mine A, occasional use of longer secondary reinforcement units (e.g. cable bolts) may be required for effective heading stabilisation. However, it is difficult to suggest similar strata reinforcement pattern at outbye side of the dyke, because of varying stress conditions.

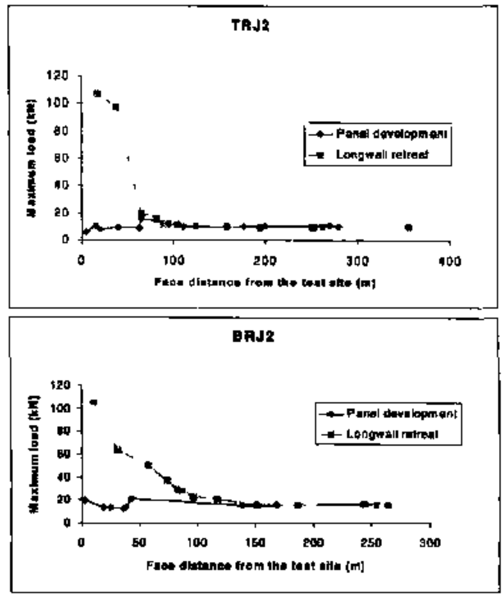


Figure 15- Maximum load transferred on bolts TRJ2 and BRJ2, for a particular face position, Mine A

8.5 *Comparison of load transfer in type I and type II bolts*

Figure 17 shows the load transferred on the bolts TRJ1, TRJ2 and TRJ3, installed at the left, middle and the right side of the travelling road. The maximum load recorded on the above bolts was 39 kN, 97.6 kN and 33.5 kN, respectively. The bolt at the left side was subjected to relatively higher load as compared to the bolts at the right side, which may have been due to the influence of the orientation of principal horizontal stress, striking the gateroads with an acute angle from the left side (Figure 12). When compared with other bolts, the bolt at the middle of the road recorded the maximum value because of the dominant role of excessive strata deformation in the middle of the gateroads.

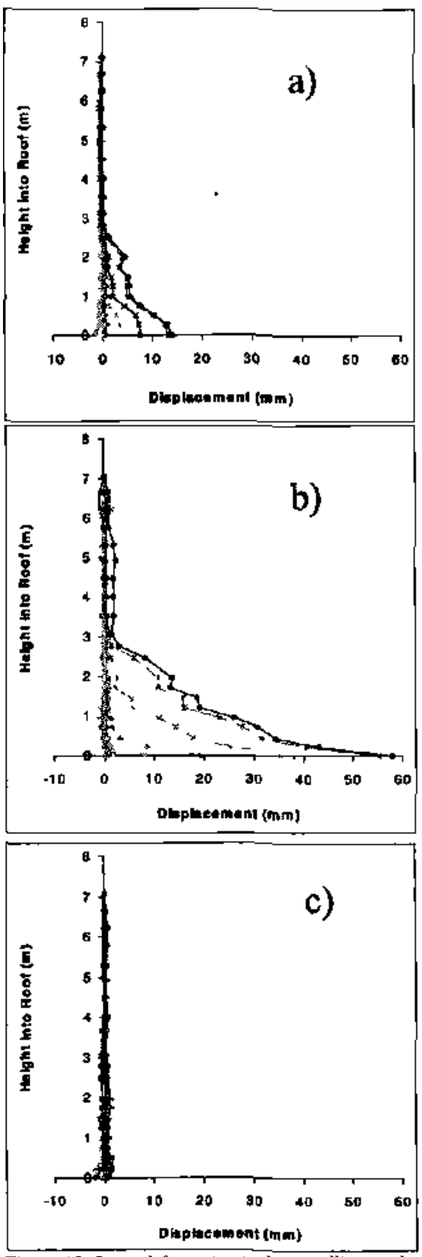


Figure 16. Strata deformation in the travelling road, a) left side (TRI), *b)* middle (TR2), and c) nght side (TR3).

Figure 18 shows the pattern of load transferred on the type I (BRJ1) and type II (BRA1) bolts, installed at the left side of the belt road. The load transferred on the type I bolts was relatively smaller as compared to the type E bolts. The maximum load transferred on BRJ1 and BRA1 was 41.7 kN and 84.6 kN, respectively. The corresponding shear stress developed at the bolt/resin interface for botJi bolts is shown in Figure 19. Thus, it can be inferred that, the location of the neutral point is independent of the bolt type. The comparative values observed from the shear stress profiles of BRJ1 and BRA1 suggests that, the bolt type II offered better load transfer characteristics when subjected to higher shear loading, caused by the influence of the horizontal stress.

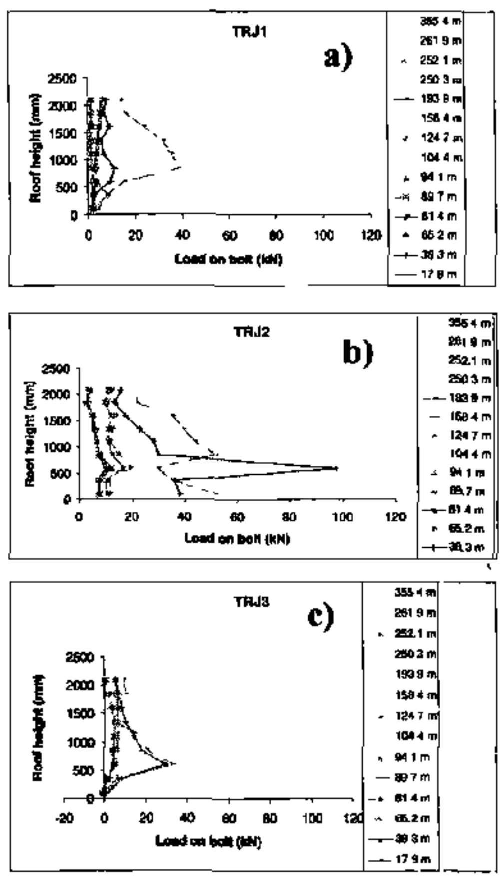


Figure 17 Load transferred on the type 1 bolts, installed *m* the travelling road. Mine A

Figure 20 shows the load transferred m type I (BRJ3) and type II (BRA3) bolts, installed at the right side of the belt road. As expected, the load transferred in type I bolt (38.1 kN) was relatively higher as compared to the type II bolt (18.6 kN). Because of the lower influence of the horizontal stress on the bolts, the shear stress developed at the bolt/resin interface in type I bolt was relatively higher than in type II bolt (see Figure 21), thus, reconfirming the superior load transfer characteristics of the type I bolts under lower levels of prevailing horizontal stress. Based on both laboratory and field study, it is clear that bolts with deeper and wider rib spacing should be used in section of the roadways subjected to the influence of low horizontal stress, whereas, bolts with shallower and narrower rib spacing should be used in areas under the influence of high horizontal stress (as evidenced by the excessive roof guttering).

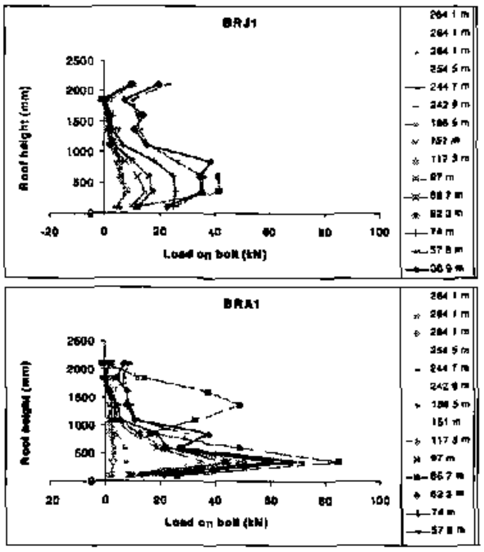


Figure 18. Load transferred on type I and type II bolts, installed at left side of the belt road, Mine A

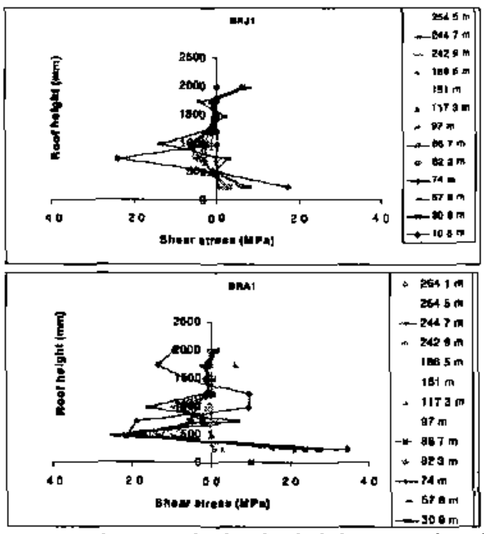


Figure 19. Shear stress developed at the bolt/resin interface of the type I and type II bolts, installed at the left side of the belt road, Mine A.

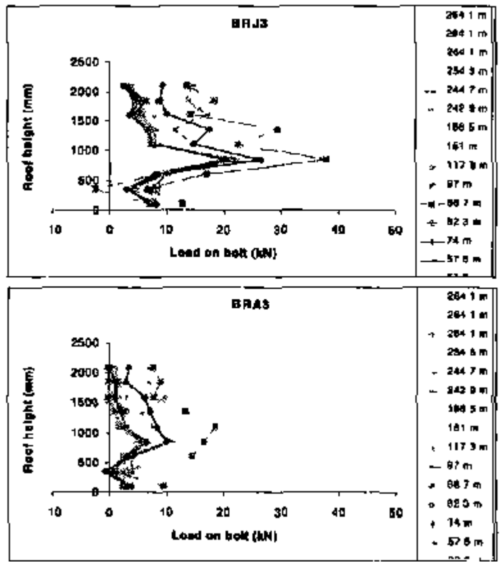


Figure 20 Load transferred on the type I and type II bolts, installed at the right side of the belt road. Mine A

9 CONCLUSIONS

It can be inferred from this study that:

- The shear behaviour of the bolt surface at various confining pressures directly affects the load transfer mechanism from the rock to the bolt.
- The type I bolt offered higher shear resistance at low confining pressure (below 6.0 MPa), whereas, the type II bolt offered greater shear resistance at high normal stress conditions exceeding 6.0 MPa. This was attributed to the surface profile configuration of the bolt i.e., the spacing and the depth of the rib.
- The bolt with deeper rib offered higher shear resistance at low normal stress conditions, while the bolt with closer rib spacing offered higher shear resistance at high normal stress conditions.
- The impact of repeated loading on the effective shear resistance of the bolt/resin interface was influenced by the magnitude of the applied normal stress, the number of loading cycles and the surface geometry of the bolt.
- The maximum dilation occurred at a shear displacement of nearly 60% of the rib spacing.
- The bolt type I showed better performance than that of bolt type If when considering the shear behaviour at low normal stress, dilational aspects, and the post-peak behaviour.
- The load transfer on the bolt was influenced by; a) the confining stress condition, b) the extent of strata deformation, and c) the surface profile roughness of the bolts.

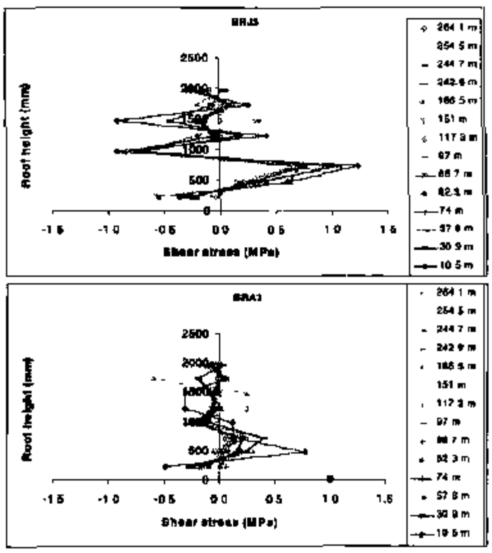


Figure 21. Shear stress developed at the bolt/resin interface of the type I and type II bolts, installed at the right side of the belt road, Mine A.

- The load transferred on the bolts, during the longwall retreating phase, was relatively greater than that of panel development phase.
- The face movement did not influence the load transferred on the bolts, when the development face moved beyond 50m away, or the approaching longwall face position was 150m away from the test sites.
- The influence of front abutment pressure build up on die gateroads appears at different face positions. The load build up on the bolts in the belt road occurs when the longwall face is less than 150m from the test site, whereas, the same build up on the travelling road starts when the face position Is less than 60m.
- The field study showed that, under the low influence of horizontal stress (both in magnitude and die direction), the type I bolt offered higher shear resistance, whereas under high influence of horizontal stress, the type II bolt offered better shear resistance at die bolt/resin interface. Such findings were also observed in the laboratory studies as indicated before, and can provide useful guidelines for future selection of appropriate bolt type for given stress conditions.

REFERENCES

Benmokrane, B. & G. Ballivy 1989. Laboratory study of shear behaviour of rock joints under constant normal stiffness conditions. *Proc. of 3& US symp. on Rock mechanics.* Tuscon:157-164.

Fabjanczyk, M. W. & G. C. Tarrant 1992. Load transfer mechanism in reinforcing tendons. *Il"' international conf. on Ground control in Mining.* Wollongong:212-219.

Fuller, P. G. & R. H. T. Cox 1975. Mechanics of load transfer from steel tendons to cement based grout. *Fifth Australasian conf, on the Mechanics of structures and materials.* Melbourne: 189-203.

Gale, W. J. 1986. Design considerations for reinforcement of coal mine roadways in the Illawarra coal measures. *AusIMM symp. on Ground movement and control related to coalmining.* Wo lion gong: 82-92.

Indraratna, B. 1990. Development and applications of synthetic material to simulate soft sedimentary rocks. *Geotechnique,* Vol. 40, No.2, pp. 189-200.

Indraratna, B., A. Haque & N. Aziz 1998. Laboratory modelling of shear behaviour under constant normal stiffness condition. *Journal of Geotechnical and geological engineering.* 16: 17-44.

Johnstone. I. W. and Lam, T. S. K. 1989. Shear behaviour of regular triangular concrete/rock joints - analysis. *Journal of Geotechnical Engineering,* Vol. IIS, pp. 711-727.

Ohnishi, Y. & P. G. R. Dharamaratne 1990. Shear behaviour of physical models of rock joints under constant normal Stiffness conditions. *Proc. Int. symp. on Rock joints (Barton & Stephanson eds).* Leon:267-273.

Signer, S. P., D. Cox & J. Johnston 1997. A method for the selection of rock support based on bolt loading measurements. *!tf' Conf on Ground control in mining.* Morgantown:183-190.

Skinas, C. A., S. C. Bandis & C. A. Demiris 1990. Experimental investigations and modeling of rock joint behaviour under constant normal stiffness. *Proc. Int. symp. on Rock joints joints (Barton & Stephanson eds).* Leon:301-308.

Strata Control Technology. 1996. *Influence of Sandstone Roof on Longwall Mining at Tower Colliery.* Report No. TOW1130,4p.

if International Mining Congress and Exhibition ofTurkey- IMCET2001, ©2001, ISBN 97S-395-417-4

Application of Tunnel Boring Machines in Underground Mine Development

M.Cigta, S.Yagiz & L.Ozdemir
Excavation Engineering & Earth Mechanics Institute, Department of Mining Engineering, Colorado School of Mines, Golden. Colorado, USA

ABSTRACT: Because of Üieir demonstrated capabilities in attaining high rates of advance in civil tunnel construction, the hard rock mining industry has always shown a major interest in the use of TBMs for mine development, primarily for development of entries, as well as ventilation, haulage and production drifts. The successful application of TBM technology to mining depends on the selection of the most suitable equipment and cutting tools for the rock and ground conditions to be encountered. In addition to geotechnical investigations and required rock testing, cutterhead design optimization is an integral part of the machine selection to ensure a successful application of the machines in a specific underground mine environment. This paper presents and discusses selected case histories of TBM applications in mining, the lessons learned, the process of laboratory testing together with machine selection and performance estimation methods.

1 INTRODUCTION

There is a constant and growing demand in the mining industry for rapid excavation to develop new orebodies faster in order to reduce overall development costs. The majority of civil engineering tunneling projects İs now carried out by mechanical excavation rather than drill and blast methods. Drill and blast is also a well-developed technology, and the choice of excavation method becomes a matter of economics for both civil construction and mining. Faster development schedules and lower costs can be achieved with mechanical excavators when supported by adequate mine planning and detailed performance analysis. Accurate estimation of production rates and costs for mechanized excavation systems increases the economic confidence and provides justification for the high capital investment associated with mechanical excavator use. Mechanical excavation offers numerous advantages over drill and blast for all types of mine development, including:

- Personnel safety is greatly improved due to elimination of blasting and toxic fumes.
- With machine excavation, ground disturbance is drastically reduced, which results in significantly tower support requirements to provide a safe and stable opening.
- The smooth walls created by machine boring also mean reduced ventilation requirements.
- Unlike drill and blast, machine generates a uniform muck size, which allows for the implementation of continuous material haulage systems, such as conveyor belts.
- Machine excavation provides a continuous operation, making it highly suitable for remote control and automation.

In this paper, recent case studies, geological effects and current state of the art in machine performance prediction methods are discussed.

2 TBM APPLICATIONS IN UNDERGROUND MINING

TBMs have been used in mining operations from time to time almost as long as TBMs have been operation. Unfortunately, they have not been nearly as successful as in civil tunneling. The earliest application of TBMs in mining dates back to late 50's. This was followed by several attempts in 60's and 70's in other underground mining operations (including coal mining in England). The penetration rates achieved by the machine in most of these projects were far above the capabilities of any drill and blast operation, yet overall tunneling costs could not be justified for widespread use of these machines in other mining operations. The common shortcomings of these earlier applications were:

- The usage of smaller disc cutters with low load capacities, which was far less than what, was required for efficient cutting of the rock formations encountered.

- Limited power available on the machines
- Lack of ability to cope with unexpected ground conditions, such as very broken ground or fault zones.
- Some of these machines were just refurbished to fit to their new job. They were not originally designed for mining applications.
- Lack of experience of the crew with these machines caused very low machine utilization rates.
- Forcing the machines to sometime follow the formation and consequently, frequent need for turning and steering.

In recent years, the mining industry has witnessed several breakthroughs resulting in successful application of the TBMs in mining operations. The main reason for the success is due to the experiences gained within over two decades of applying these machines in different projects, new technological advances in cutter design, machine components, hydraulic and electrical systems, increased use of computer and electronic controls. And finally, willingness of the manufacturers to design machines specifically suited for mining operations.

Two most recent successful applications of TBM technology in mining have been in the San Manual copper mine and the Stillwater platinum mine in the United States, as discussed below.

2.1 *San Manual Mine, Magma Copper Company*

Magma Copper Company's San Manuel mine is located approximately 72 km north of Tucson, and 11 km west of San Manuel, Arizona.

Geological interpretation developed during mining In the San Manuel ore body led to the discovery of the Kalamazoo orebody in the mid 60s. The Kalamazoo is the upper half of the San Manuel ore body, sheared and moved nearly 2,500 m down dip at a 25-degree angle. Geologically, Kalamazoo ore body is the result of intrusion by late Cretaceous age granodiorite porphyry into a Frecambrian age porphyritic quartz monzonite. Both are members of the granite family of rocks, differing mineralogical only slightly in composition. The Kalamazoo and San Manuel ore bodies were once part of a single ore body created as a nearly vertical, cylindrical shell surrounding the contact of the host quartz monzonite and me invading granodiorite porphyry.

The Lower Kalamazoo was being developed for a block caving operation, with modified stope design and belt conveyor ore haulage. As shown in Figure 1, the mine development plan called for approximately 12,800 m of drifting in two levels. After a detailed trade-off study, the mine elected to use a TBM for all development work.

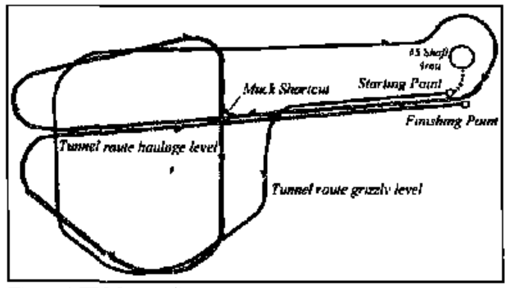


Figure 1 TBM tunneling route

The Magma Copper selected a hard rock, main beam TBM from Atlas Copco Robbins (Figure 2) for the Lower Kalamazoo development. The cutterhead and main beam were built in two pieces to meet the space and weight limitations imposed by the mine's shaft. Some of the features of the machine are given in Table 1.

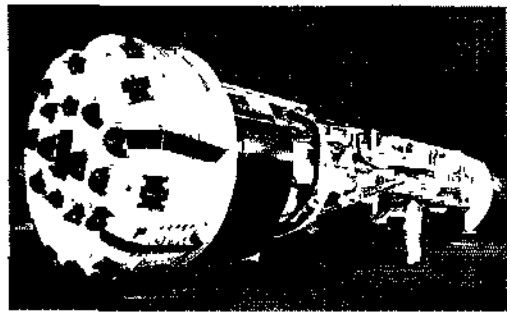


Figure 2. San Manual TBM (Robbins).

Table 1 Robbins TBM Specifications

Boring diameter		4.62 m
Cutterhead	Installed power	1259 kW
	RPM	4 – 12
	Thrust	7,340 kN
Cutters	Number	33
	Tip width	15 875 mm
	Diameter	432 mm
	Maximum load	222 kN/cutter
Boring stroke		1 575 m
Minimum turning radius		105 m
Weight		225 t

The TBM's excavation route included the stable quartz monzonite of 150 MPa to 180 MPa unconfmed compressive strength. The bore path also crossed the San Manual Fault six times and The Virgin Fault five times. The San Manual is a flat dipping fault with a 1-m wide clay zone that follows the bore some 30 m at a time but influences a total of 190 m; The Virgin Fault dips steeply but a series of related minor faults produces poor rock conditions for about 500 m. In addition, where dacitic, andesitic and rhyolitic dikes contact the granodiorite and quartz-monzonite, weak zones 0.2-m to 0.6-m wide affect the bore approximately 180 m. Along most of the TBM's path, hydrothermal metamorphosis has weakened the rock further by veining, fracturing and jointing. Unsupported wall stability ranged from 30 minutes to months. Magma Copper chose to take a systematic approach to rock support and installed ring beams the full length of the bore path.

Clay plugging the cutterhead was one of the problems. Other difficulties encountered included were trouble starting the cutterhead and keeping it rotating in soft, collapsing ground. This was partly due to ground subsiding onto the top of the cutterhead, partly due to sidewall collapse. It was also found that the machine's side supports gouged die tunnel walls in weak ground. Not only did this contribute to sidewall collapse, it also made it impossible for the machine to be steered along the narrowest of the curves along the tunnel route.

Due to these problems, the TBM could not initially reach its planned rate of advance and it became obvious that modifications were needed to make the machine reach its design performance under very difficult ground conditions. All options were reviewed and as a result, the following modifications were made:

- Two cutters were removed to permit enlarging the buckets for a better muck flow.
- To improve cutterhead "starting" and "pull-through" capabilities, cutterhead drive torque was increased with better stall characteristics.
- The cutterhead speed was reduced from 12 to 9.3 rpm. This improved the running torque by some 29 %.
- The roof support was extended forward by a canopy over the rear of the cutterhead to within a short distance from the outermost gauge cutter.
- Fingers were added to the side supports to increase the surface area of those supports.
- The side roof supports were separated from the side supports to allow for independent operation.

All the modifications were executed in 23 days, very close to the planned schedule. The modifications greatly improved the performance of the TBM in all areas. Results since the modifications are shown in Table 2, which compares the TBM progress before the modifications with results following the modifications.

Table 2. Comparison of machine performance

	Before	After
Average Daily Advance, m	6 46	22.6
Best Shift, m	19.2	21.6
Best Day, m	37.5	44 5
Best Week, m	141.7	263
Best Month, m	333.5	831 2

Overall, this was a successful application of TBM in underground hard rock mining and allowed the mine to meet its accelerated schedule for the development of the new orebody.

2.2 *Stillwater Mine, Montana*

Stillwater Mining Company (SMC) is developing a second underground palladium and platinum mine about 21 km west of their existing Nye operation in Montana. A 5,650-m tunnel is required to gain access to the ore-bearing horizon (J-M reef). Associated by-product metals include rhodium, gold, silver, nickel and copper.

As shown in Figure 3, the access tunnel (Adit #1) is being driven at an azimuth of 208° with a plus 1/2% vertical grade approximately 5,650 m to reach the reef.

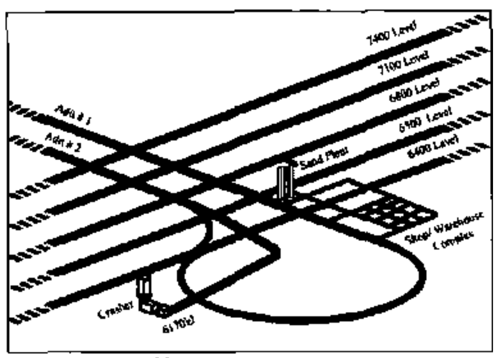


Figure 3 East Boulder Project

The tunnel geology consists of approximately 1,006 m of sediments, 2,530 m of gabbro, 430 m of norite, 1,140 m of anorthosite, 335 m of olivine gabbro, and 210 m of troctolite. Rock uniaxial compressive strengths are as follows: limestone 60-140 MPa, gabbro 70-190 MPa, norite 100-150 MPa, and anorthrosite 60 -190 MPa. In general, the unconfined compressive strength test results show a wide range of values from 60 to over 190 Mpa.

Because of their earlier successes with TBM use, the Stillwater mine decided use TBMs for all development work for the mine. The first TBM was custom designed and built by Construction and Tunneling Services (CTS) of Seattle Washington. Table 3 lists the basic specifications of this machine. The machine was delivered to the project site in July 1998 and reassembled and field-tested. Independent contractors were used to drive the 5,650-m long, 4.6-m diameter tunnel, starting from Adit #1. The excavation was completed at the other end of the reef on July 29, 2000. Crews and equipment will access to western section of the reef through this tunnel.

Table 3 Specifications of the CTS Machine

Bonng Diameter		4 58 m
Cutter head	Installed Power	1345 kW
	RPM	11.6/3 8
	Thrust	8,545 kN
Cutters	number	26 - 432 mm
		3 - 406 mm
	Tip width	15.875 mm
	Maximum load	222 kN/cutter
Bonng Stroke		1,220 mm
Minimum turning radius		61 m
WeiRht		2751

Efficient access into the East Boulder Mine was essential to achieving a 1,800-t/day-production rate. The most essential element in the acceleration of the project schedule was to establish a second means of access and the ventilation required for underground development. Accordingly, SMC purchased a second TBM for the excavation of adit #2. This was the same machine as used in San Manual Mine in Arizona. An independent contractor refurbished this machine to the specifications shown in Table 4.

Table 4 Specifications of Robbins Machine

Bonng Diameter		4 62 m
Cutter head	Installed Power	828 kW
	RPM	12/4
	Thrust	7,300 kN
Cutter	Number	33
	Tip Width	15 875 mm
	Diameter	432 mm
	Maximum load	222 kN/cutter
Bonng Stroke		1,550 mm
Minimum turning radius		105 m
Weight		225 t

A 914-mm wide conveyor belt was used for material haulage from the second drive. A key feature in the rapid development of the East Boulder Mine was to use tins conveyor for all development and production rock haulage when the tunnel reaches the East Boulder workings.

Initial production is expected to begin in 2001, with mine operation at 1,800 t/day. When East Boulder reaches full production, the annual production rate will be 450,000 to 500,000 oz of palladium and platinum annually.

3 TBM PERFORMANCE PREDICTION METHODS

As noted before, a key factor İn TBM applications to any tunneling project is the accurate prediction of attainable penetration rates and the cutter costs. This is necessary in order to compare the economics of TBM use vs. drill and blast excavation.

3.1 *Factors Influencing TBM Performance*

The parameters influencing Tunnel Boring Machine performance can be summarized as'

- Intact rock properties
- Rock mass properties
- Cutter geometry
- Cutting geometry
- Machine specifications
- Operational parameters

Proper application of the TBMs to any mining or civil tunneling operation depends on the detailed understanding of the parameters given above The following İs a brief discussion of these parameters when assessing the economics of a TBM application.

3.1.1 *Intact Rock Properties*

It is well known and established that the uniaxial compressive strength (UCS) of rock is the most commonly measured rock property. UCS can be used to evaluate the resistance of the rock against the indentation of the cutting tool into the rock surface. Great attention must be paid as to how the sample failed during UCS testing. Figure 4 illustrates a typical structural and non-structural failure of UCS sample. Those samples, which are observed to fail along existing rock defects, such as joints, fractures, bedding or foliation, are classified as structural failure. Where the sample failure is not controlled by any defects and occurred in an "intact" manner, the sample İs noted as having failed in a non-structural manner. This classification is of crucial importance since the structural failures do not represent the actual rock strength and therefore, are excluded from any boreability predictions. However, mechanical cutting predictions relying only on the compressive strength may provide inaccurate results. Several other intact rock physical property tests need to be performed to increase the accuracy of performance predictions for TBMs.

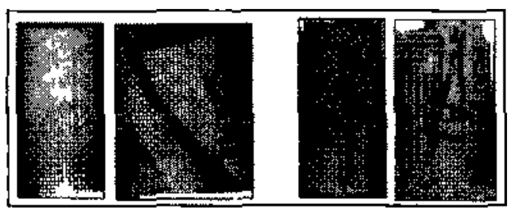


Figure 4. Typical structural (left) and non-structural (nght) failures on cores.

Tensile strength is another common rock property, which is used in making boreability predictions along with the uniaxial compressive strength of the rock. Brazilian Tensile Strength (BTS) is generally intended to provide an indication of rock toughness from a viewpoint of crack propagation between adjacent cutter paths. As in UCS testing, great attention must also be paid as to how the sample failed during BTS testing. Figure 5 illustrates a typical failure of structural and non-structural failure. All the structural failures are excluded to present the true tensile strength of the rock.

Figure 5. Typical failure modes in 1__L !_!

Another rock property, which affects boreability, is the brittleness or the plasticity, which the rock exhibits when subjected to the mechanical forces generated by the cutting action of an excavator. In general, rock cutting efficiency of any mechanical tool improves with increasing brittleness exhibited by the rock formation. Thus, brittleness is a highly desirable feature of the rock from a boreability standpoint. One of the tests, which help assess the brittleness of the rock in the laboratory, is the Punch Penetration Index test. In this test, a standard indentor is pressed into a rock sample that has been cast in a confining ring as shown in Figure 6. The load and displacement of the indentor are recorded with a computer system. The slope of the force-penetration curve indicates the excavatibility of the rock, i.e., the energy required for efficient chipping.

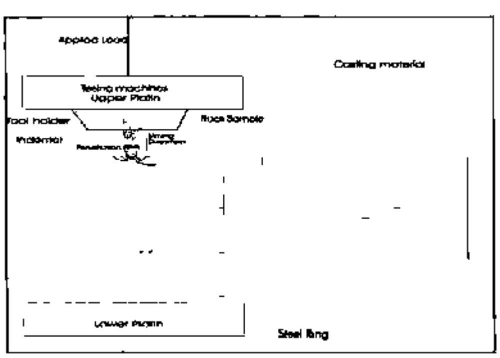


Figure 6. Punch Penetration Test Set-up

Rock abrasivity together with strength İs used to estimate cutter wear during TBM excavation. The Cerchar Abrasivity Index (CAI) has proven to be fairly accurate and is commonly used for cutter life estimation. A series of sharp 90° hardened pins of heat-treated alloy steel are pulled across a freshly broken surface of the rock, as shown in Figure 7. The average dimensions of the resultant wear flats are related directly to cutter life in field operation. The geometry of the planned excavation then allows calculation of the expected cutter costs per unit volume of material.

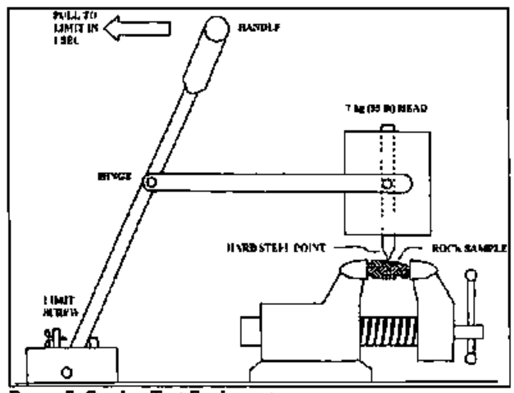


Figure 7 Cerchar Test Equipment.

3.1.2 *Rock Mass Properties*

In foliated/bedded rock, foliation can play a significant role İn rock fracture propagation between cuts, depending on the foliation direction with respect to the direction of machine advance. Figure 8 illustrates the orientation of foliation planes with respect machine advance. Angle "a" is defined as an angle between tunnel axis and the foliation planes

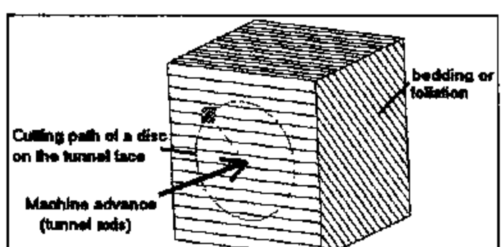


Figure 8. Definition of Foliation angle with respect to machine advance.

When machine advances parallel to foliation planes (Figure 9), crack propagation İs forced to occur across the foliation planes. This reduces machine penetration because of increased difficulty of rock breakage.

When the foliation is perpendicular to direction of machine advance, rock failure occurs along foliation planes as shown in Figure 10. This case generally represents the most favorable boreability as the foliation planes assist crack initiation and growth between adjacent cuts.

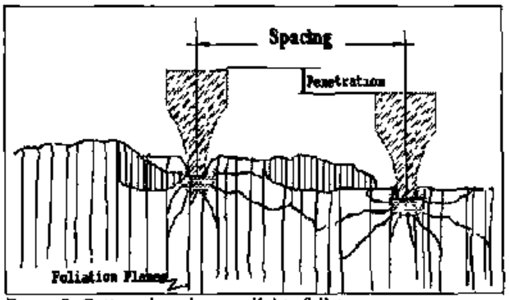


Figure 9 Cutting direction parallel to foliation

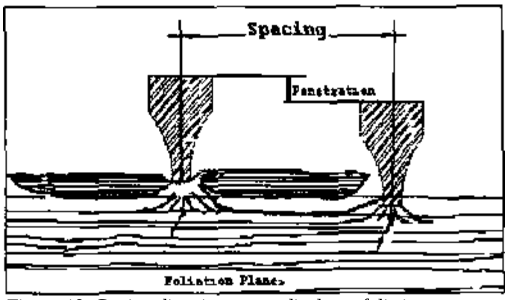


Figure 10 Cutting direction perpendicular to foliation.

One way to integrate the foliation effect in machine performance modeling is to measure the tensile strength of the rock in different directions as shown in Figure 11. The loading direction of the sample can be selected based on the machine advance with respect to foliation/bedding planes in order to represent the crack propagation across or along the weakness planes.

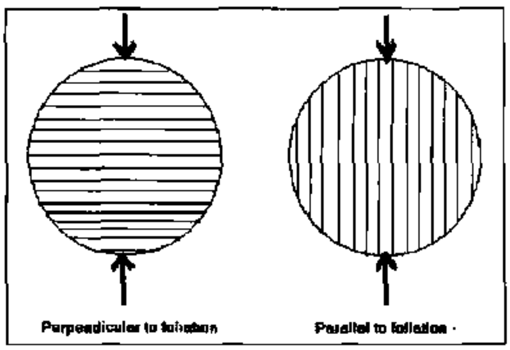


Figure 11. Loading direction for tensile testing.

Joint sets in the rock mass can also have a major effect on machine performance. The studies performed by NTNU (Norwegian University of Science and Technology) provide one of die roost extensive database for estimating the ROP based on rock mass properties (Bruland & Nilsen, ISRM 1995). Rock mass fracturing means fissures and joints with little or no shear strength along the planes of weaknesses. Joints are continuous; they may be followed all-around the tunnel Contour. Fissures are non-continuous; they can only be followed partly around the tunnel contour. Table 5 shows the class for joints and fissures used in NTNU model. The smaller the distance between tie fractures, the greater the influence on the penetration rate of the machine.

Table 5. Joint and Fissure Classes.

Class	Spacing (cm)
0	-
0 - 1	160
I -	80
I	40
It	20
m	10
IV	5

The fracture factor k_s for fissures and joints is shown in Figure 12, as a function of fissure or joint class and angle between the tunnel axis and the planes of weakness (a angle). From this figure, "k," can be determined for each set of weakness planes. Based on this information, the effect of fissures and joints is evaluated on the penetration rate of the machine.

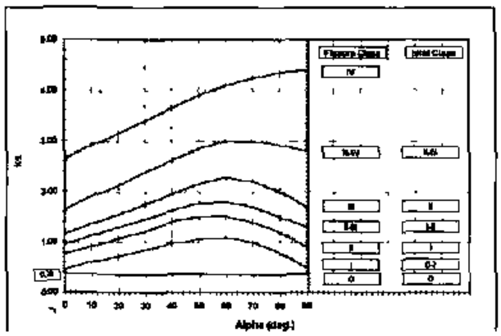


Figure 12 Determination of k, fracturing factor

Geological conditions to be encountered such as faults, and groundwater can have a major impact on the machine selection, application, operation and the production rate. These parameters must be accounted for when estimating the machine utilization, which is a key parameter in scheduling. Analysis of field performance of different TBM projects is the foundation for estimating the effect of these geological features in the rock mass.

3.1.3 *Cutter Geometry*

Cutting tools provide for the transmission of energy generated by the machine to the rock in order to cause fragmentation. As a result, the geometry and wear characteristics of the cutting tool have a significant effect on the efficiency of energy transfer to the rock and the attainable rate of penetration.

Single disc cutters are the most commonly used roller cutters for hard rock Tunnel Boring Machines (Figure 13). They are the most efficient types of rolling cutters since the entire capacity of the bearing is concentrated into a single, narrow edge.

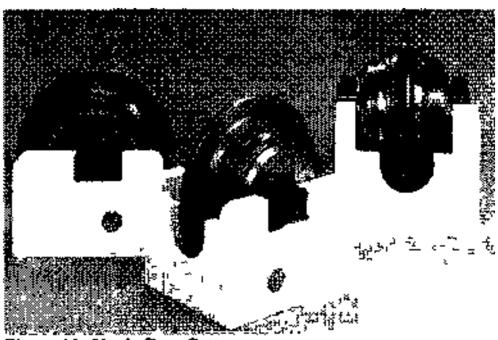
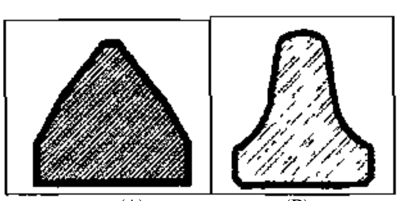


Figure 13. Single Disc Cutters.

Until about two decades ago, disc cutters utilized on TBMs featured a V-shaped edge profile (Figure 14A) with an included angle varying from 60 to 120 degrees. Although this profile provided for high rates of advance when the cutter was new, its performance, as expected, was found to drop rapidly, as edge wear developed and the rock-cutter contact area became larger. To ensure a more consistent cutting performance with increasing edge wear, the so-called constant-cross section (CCS) cutters were developed. The CCS cutters (Figure 14B) are designed to maintain a more or less constant profile as edge wear occurs. This means the machine performance does not decline as rapidly with cutter wear. These advantages have led to the CCS cutters becoming an industry standard on TBMs.



(A) (B)

Figure 14. (A) V-shape and (B) CCS disc cutter profiles.

The ring diameter is another variable, which affects disc cutter performance. Since their introduction, disc cutters have steadily grown in size from about 305 mm to present-day 483 mm cutters. Development work for even bigger size cutters is underway. For the same thrust load on the cutter, increased diameter causes a reduction in the depth of cutter penetration into the rock because of larger cutter footprint area. However, larger cutters provide for higher bearing capacity, which more than offsets the performance loss brought about by the wider cutter-rock contact area. In addition, larger cutters rotate slower for a given machine rpm which means less heat generation during boring. They also contain more cutter material to wear out before replacement becomes necessary, again contributing to longer ring life. All these features combined thus lead to improved cutter life and reduced excavation costs.

3.1.4 *Cutting Geometry*

The cut spacing and the depth of the cutter into the rock per cutterhead revolution define the efficiency of the cutting by disc cutters.

As would be expected, the spacing of cutters has a significant impact on the chipping mechanism and the efficiency of boring. As shown In Figure 15, there exists an optimum spacing for a given cutter penetration where the interaction between adjacent cuts is maximized. This optimal spacing is usually expressed as the ratio of spacing to penetration Extensive past research and field data analysis have shown that to achieve optimal cutting efficiency, this ratio should be maintained between 10 to 20; with lower ratios used for tougher rocks and the higher ratios approaching to 20 for more hard and brittle rock. The manufacturers extensively use this ratio when they design the cutterhead.

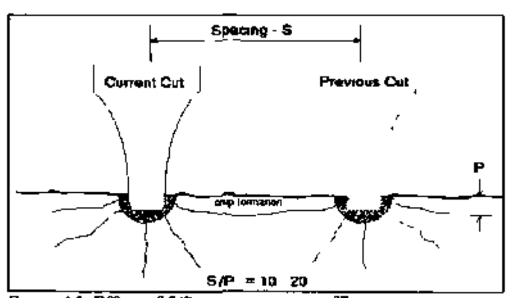


Figure 15 Effect of S/P ratio on cutting efficiency.

3.1.5 *Machine Specifications*

The machine specifications, such as thrust and power are the key to providing sufficient amount of forces and torque to support the excavation operation. Machine thrust should provide the enough force to efficiently penetrate the tools into the rock surface. Also, the cutterhead torque and power requirements to rotate the head at the required penetration rate and overcome the rolling force resistance of the cutters has to be determined and installed on the head.

3.1.6 *Operational Parameters*

In every mechanical mining operation, there are some operational constraints, such as the haulage capacity, ground support requirements, water-handling, etc. that limit the productivity of the machine. In addition, other factors such as tunnel grade and curves impact machine utilization and consequently productivity. All these factors must be taken into account when application of mechanical excavator to a particular operation is considered.

3.2 *Performance Prediction for TBMs*

The CSM/EMI computer mode] for hard rock TBMs is based on the cutterhead profile and rock properties. The model utilizes semi-theoretical formulas developed at EMI over the last 25 years to estimate the cutting forces. The output of this model consists of the cutterhead geometry and profile, individual cutting forces, thrust, torque, and power requirements, eccentric forces, moments, and finally variation of cutting forces as the cutterhead rotates.

The first step in performance estimation involves characterization of the rock and the geologic conditions. This is provided by the intact rock and rock mass properties mentioned earlier.

The next step is to select the proper cutting tool and cutting geometry. Disc cutters are the most commonly used cutting tools for TBMs. The only parameters need to be taken into account are cutter diameter and cutter tip width. Table 6 gives the commonly used disc cutters on today's TBMs.

Table 6 Disc Cutters for TBMs

Cutter Diameter, mm	Max. Cutter Load, kN	Cutter Tip width mm
432	222	15.875
	267	19.05
483	311	19 05

After selection of cutter and cutter geometry, the forces acting on the cutters are estimated or measured. The algorithms have been derived from extensive full scale testing performed over the two decades at Earth Mechanics laboratory of Colorado School of Mines.

If rock samples are available, cutting forces can be also measured through full scale testing on the LCM. The LCM (Figure 16) features a large stiff reaction frame on which the cutter is mounted. A triaxial load cell, between the cutter and the frame, monitors forces and a linear variable displacement transducer (LVDT) monitors travel of the rock sample. The rock sample is cast in concrete within a heavy steel box to provide the necessary confinement during testing.

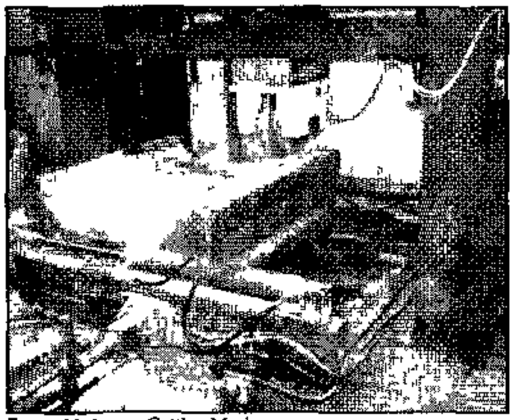


Figure 16 Linear Cutting Machine

LCM test can be used not only for measurement of cutter forces, but also to select the most suitable cutter in order to achieve die most efficient cutting. This means different diameter cutters with different tip widths can be tested in order to observe their performance at different spacings and penetrations. Actual rock sample from the field can also be tested. This eliminates the need for any scaling effects.

After the selection of cutter type, cutting geometry and detennining the cutting forces, the' next step is to optimize the cutterhead design and cutter lacing on the head. Among the parameters influencing the performance of a mechanical excavator, the easiest parameter to control is the cutterhead design. The input data for cutterhead design and simulation comes from the previous steps, which are the cutter type, cutting geometry and the required cutting forces to achieve the desired rate of penetration and the mimmum-specific energy. Simulation of the cutterhead is also used to calculate the machine parameters, such as thrust, torque and power requirements. Rather than working with average cutter spacings and forces, CSM/EMI model analyzes each cutter individually since actual cutter forces vary across the cutterhead depending on their location and angle with respect to machine axis (Figure 17). In case of an existing machine, required machine parameters are first calculated and men machine parameters are first calculated and then evaluated to determine if the machine is able to sustain the estimated or desired rate of penetration.

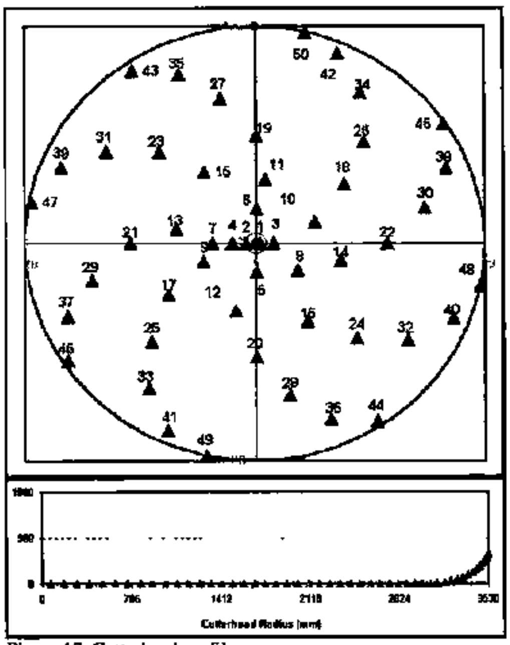


Figure 17 Cutterhead profile.

The program then calculates the penetration rate on an iterative basis until one of the machine limits in terms of thrust, power or cutter load capacity is exceeded. The same calculation is repeated for each rock type in order to evaluate the machine performance in a given geology. An example output for one of the geologic zones is presented in Figure 18. The program also includes a plot, which describes the capability of the given machine at different rock strengths as shown in Figure 19.

If the rate of penetration and machine parameters are known, back up and mucking systems of TBM can be designed to match the tunnel advance rate.

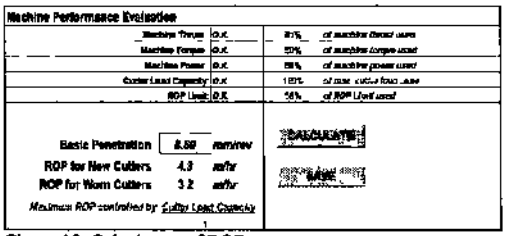


Figure 18 Calculation of ROP.

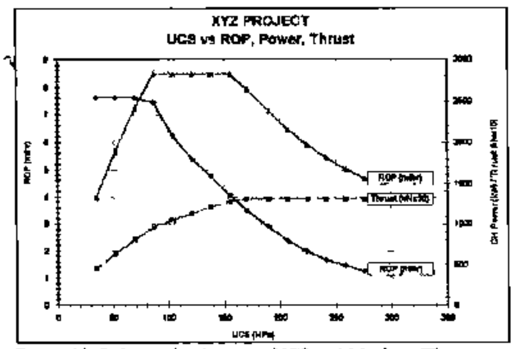


Figure 19 Relationship between UCS and Machine Thrust, Torque, Power.

The last step in modeling of any TBM application is the scheduling and cost analysis. There are two important numbers in modeling of a TBM job. Those are rate of penetration, which is estimated from rock, and machine properties that the machine can achieve, and machine utilization, which is the net boring time as a percentage of the total working time. The total available working time cannot be entirely utilized to excavate rock. It is reduced by equipment and job site related down times. Once machine utilization is estimated, daily advance rate (m/day) can be calculated from rate of penetration and machine utilization. This calculation needs to be repeated for all the reaches if the tunnel drive is divided into sections in terms of geological conditions and technical requirements of the project. This will help the designer determine the time required to excavate the tunnel and the overall costs. Table 7 illustrates an example of CSM/EMI model output for scheduling of a typical TBM tunnel.

Table 7. Example output of a scheduling

Rock TYM	Section #	Length	Ships 1%)	Cumhtfa (YESiNO)	ROP feruîir)	US'-Kına Unliiatran S	AdYancs flail	Completion Urn« dap	manth^
Sandstone	1	aao	1	NO	54	35*	16 3	155	6
Gunıl e	1	H O	1	NO	26	30%	58	253	ID
Gneiss	1	457	1	YES	21	25%	38	122	5
Gneiss	1	163*	1	NO	21	30%	45	338	U

4 CONCLUSIONS

There are several special issues that have to be identified and dealt with when introducing a new piece of equipment in mining operations. The past experience with TBMs in mining industry has shown that a thorough study is needed to match the machine and the surrounding environment. The issues, which need to be considered, include:

1. Multiple speed cutterheads to provide a high torque at lower speed for operating in highly fractured/blocky ground

2. The use of back-loading cutters to avoid unnecessary delays and potential safety problems associated with cutter changes in the face.

3. Ability to install a variety of ground support measures such as bolts, wire-mesh, shotcrete, and steel sets where the ground is first exposed. Rock drills for roof bolting, ring beam erector for steel sets installation, and probe hole drilling equipment too see what is ahead of the machine are some of the features that should available on the machine.

4. The circular shape created by TBM may not be suitable for certain mining applications in particular the haulage drifts. There are options available to resolve this problem. One alternative is to utilize TBM generated muck to backfill behind the machine to provide a flat floor. The shape of the TBM drift can also be modified to a more suitable geometry by a secondary excavation operation using drill and blast Another option is to mount a set of boom cutters on sides of the TBM to produce a horseshoe shaped entry.

5. A better understanding of expected ground conditions will allow for optimization of machine design to fully cope with the ground conditions to be encountered.

6. Improved performance prediction so that project economics and completion schedules can be assessed more accurately.

REFERENCES

Allum, R. &. VanDerPas, E. 1995. TBM Technology in a Deep Underground Copper Mine. *Rapid Excavation and Tunneling Conference: 129-!43*

Alexander, C. 1999. Development of Stillwater Mining Company's East Boulder Project Using Tunnel Boring Technology. *SME Annual Meeting.* Denver CO, USA: 1 -8

Cigla, M. &. Özdemir, L. 2000. Computer Modeling For Improved Production of Mechanical Excavators. *SME Annua! Meeting. Sail iMİce City UT. USA*

Bruland, A., Nilsen, B. 1995. Tunneling Performance Estimation based on DrillabİlÍty Testing. *Proc. 8"" ISRM Congress on Rock Mechanics.* Vol. I: 123-126

Rostami, J. Özdemir, L. and Nilsen, B. 1996. Methods for Predicting Mechanical Excavation Performance and Costs, *Proceedings of Annua! Technical Meeting of the Institute of Shaft Drilling Technology (1SDT), Las Vegas. May 1-3, 1996.*

Özdemir, L. 1990. Recent Developments in Hard Rock Mechanical Mining Technologies" *4 th Canadian Symposium on Mine Automation, Saskatoon, Canada, Sept. 1990.*

if International Mining Congress and Exhibition of Turkey- IMCET2001, ©2001, ISBN 975-395-417-4

In Situ Measurements in a Paste Backfill: Backfill and Rock Mass Response in the Context of Rockburst

F.Hassani, J.Ouellet & S.Servant
McGill University, Department of Mining & Metallurgical Engineering

ABSTRACT: Rockburst can be a major problem in deep underground mines and has the potential for causing injury to mine operators as well as damage to underground workings. An extensive investigation was conducted to study in situ behaviour of mine backfill, in response to mining induced stresses with special reference to die phenomenon of rockburst. The field program included various in situ instrumentations, laboratory testing, and a detailed geomechanical study. Within die time frame of mis study a rockburst event occurred within *the* mine that was monitored for this investigation. When the field observations are analysed a definite stress trends is detected prior to this event. The potential of in situ instrumentation will be presented and the role of die paste backfill will be discussed in relation to the case descnbed, from die point of view of ground support.

1 INTRODUCTION

From die standpoint of operations, backfilling of mined out openings serves multiple functions such as working surface, roof, secondary pillar or ground support by maintaining the stability of the openings. It also impacts on the overall stability of the mine by limiting convergence interacting widi die rock and can also reduce the impact of rockburst.

The backfill/rock interaction is complex. In ground control we will be concerned with the rock mass behavior around openings (convergence, induced stresses, sismicity) but also on the loads supported by the backfill and its stress-strain response to the rock mass deofrmations. To achieve an understanding of Ulis complex interaction a good knowledge of both rock and backfill behavior is required (Gürtunca et al, 1989, Gürtunca et Adams, 1990, Hassani et al., 1998).

In order to better understand the role of mine backfill a monitoring of both the backfill and rock mass is required. Obtaining field data on botii will alow a quantitative appreciation of the backfill role as a ground support element. To this end, more specifically in a mining operation having rockburst incidents, it will be shown how die use of in situ instruments can contribute to better understand the backfill impact on the rock response to mining. A brief review will be given on the possibilities of in situ instrumentation in backfills and the various recent experiments published so far. For the case study presented here from mine Chimo, it will be shown how in situ instrumentation could help forecast a rockburst event by monitoring stresses and strains. The combination of field measurements with geomechanical and geophysical characterisation of the mine site gives the possibility of comparing numerical analysis results with actual field performance. This m tum gives the possibility to validate die numericals models and dispose of a valuable tool to understand the rock mass response to mining and predict eventually how mining operations will impact on the ground stability.

Numerical software and powerfull computers available today allow the engineers to make very complex analysis. These tools can be very usefull to perform back-analysis or be used as predictive tools to foresee ground response to mining operations. But to get meaningfull results from these numerical tools good field data are essential. This paper presents a review of the means used in a mine to obtain in situ data required to understand the in situ response of a paste backfill to the rock mass in the context of rockburst and to better understand its role and impact as a ground support element.

2 BACKFILL INSTRUMENTATION

There is limited documentation about backfill instrumentation in underground mines. A research program was started by the US Bureau of Mines to monitor the in situ backfill behaviour (McNay and Corson, 1961). The in situ monitoring work included recording the pressure build-up in the backfill and measurement of vein wall deformation in the mines of the Coeur d'AIene district in Idaho, USA. This research was carried out to assess the backfill pressure caused by deformation of the stope's sidewalls. The stability of stopes was made mainly by the intact rock pillars left between mined-out stopes as well as ore body crown pillars. Having an idea of backfill performance in mining stability would mean a reduction in the pillar size.

They reported that backfill pressure was measured directly by hydraulic pressure cells which were placed in the backfill (McNay and Corson, 1961). Laboratory tests were conducted to interpret in situ data at maximum and minimum backfill densities. The working capacity of the pressure cells were İn the range of 3 MPa to 6 MPa. The recorded data was reliable İn the range of 2.5 MPa to 4 MPa. The rock pressure changes in areas around the backfilled stopes was measured using borehole pressure cells. The deformation of stope wall was measured between rockbolts installed İn opposite walls. The closure was recorded using two techniques : a) a dial extensometer, b) a measuring stainless steel wire attached to rockbolts; total stope wall closure was measured by a hydraulic cylinder which displaced fluid into a container.

Corson and Wayment (1967) reported a monitoring program in an area 60m x 40m in a near vertical vein at the Star Mine in Idaho, USA. The stress versus displacement measurement in a backfill stope of 3 meters wide at a depth of 2000 m were described. The data was collected using 22 total pressure cells and six closure measurement instruments. Stresses in the backfill initially build up hydrostatically but men pressures across the vein increased more rapidly than in the plane of the ore. The maximum pressure measured across the vein was 3.5 MPa, while the pressure in the plane of the vein İn the horizontal and vertical directions only reached 1 MPa, then the cells broke down rendering further measurements impossible. The amount of closure recorded in the stope was about 10 cm.

A monitoring of backfill stresses and strains at the Lucky Friday silver Mine at Mullen, Idaho, USA, was recorded by McNay and Corson{1961). Five backfilled areas were instrumented at a depth of 1300 m at this level. Very little increase in backfill pressure was observed at the beginning due to a production delay because of a miners strike. However, when mining recommenced, rockbursts were experienced and the pressure in the backfill increased. The rate of strain increased following the rockbursts. A maximum pressure of 5 MPa in the backfill was reported.

A field evaluation of backfill compaction at the Luck Friday Mine, Idaho, USA was also reported by Corson (1971). This monitoring was conducted in two backfilled stopes separated by 30 m. on the same elevation at a depth of 1000 m. Both sites were filled with cemented fills but at the ore site backfill was placed normally while me other was vibrated. Each stope was instrumented with six pressure cells and a mechanical closure device to measure the vein wall movements. The density and moisture of the backfill was measured at three elevations using an access pipe.

Thibodeau (1989) reported an extensive field instrumentation at Levack Mine in Canada. The instrumentation included 36 electronic total earth pressure cells, three electronic piezometers and seven convergence monitors. The mechanical behaviour of the fill under load was studied. It was found that arching and the elastic beam behaviour could not be applied since the principal stresses İn the fill were much larger than the maximum fiber stresses calculated using arching equations. The principal stresses measured in the backfill were then attributed to pillar convergence.

The stresses measured at backfill site in deep levels South African gold mines are much higher than the results fron other mines. The first known measurements of stresses and displacements in backfill in South African mines were reported by Gay et al (1986). They discussed the development of many instruments suggested by Greig Et. al. (1979) of COMRO (Chamber of Mine, Research Organization) for recording stresses and strains in harsh underground environment. Gay et al. (1986) reported backfill pressure in two ranges: a) backfill acting as a local support would have a pressure less than 10 MPa. b) backfill acting as a regional support with a pressure exceed up to 100 MPa. at 4-5 km depth.

Bruce and Klokow (1988) explained a backfill instrumentation program at West Driefontein Mine in South Africa. A closure meter was placed in backfill underground, some 13m. from the backfill edge. A Glötzl pressure cell was installed alongside the closure meter. The stress-strain curve from the field data and laboratory was compared. The stress generation at a particular load in underground was lower than the results from similar material tested in the laboratory.

Clark et al. (1988) reported the development of instruments for backfill by COMRO. These included stress meters to record stresses up to 100 MPa and closure meters to monitor up to 0.5 m of closure inside the backfill. They presented closures and stresses data in three orthogonal directions in classified tailing backfill. The highest vertical stress recorded in backfill was almost 30 MPa. They showed that the ratio of the average of the two horizontal stresses to the vertical stress remained fairly constant above a vertical stress of about 3 MPa. They also showed a good agreement between the in situ measurements and a laboratory confined compression test on similar material.

A backfill monitoring program was carried out by Gürtunca et al. (1989) to investigate the behaviour of three types of backfills, comminution waste, classified tailings and dewatered tailing backfill. A good agreement between the in situ and laboratory data in confined compression was reported. They suggested that the lower the starting or placement porosity of any particular backfill, the suffer that backfill will be. The ratio of dip or strike stresses in backfill, to the stresses acting at right angles to the roof plane, was showed to be between 0.3 to 0.6.

Using the data available to them, Gürtunca et al. (1989) present simple three dimensional models for the closure and stress profiles across complete backfill ribs. The closure profiles shows the maximum closures occurring near the edge of the backfill with lesser closure occurring near the backfill center. The three dimensional stress profile model suggests that stresses in the backfill increase with increasing distance from the stope face. On any section parallel to the stope face, the stresses build up slowly from the edge of the backfill and reach a plateau of maximum stress in a central section of the backfill rib. This is in contrast to the stress profile across a rock pillar which has the highest stresses at or close to the edge of the pillar, with the stresses reaching a plateau of lowest values near the pillar center (Hoek and Brown, 1986).

A backfill instrumentation result including the stresses in three orthogonal directions as well as closure at two sites was reported by Clark (1989). He showed mat the backfill appeared to go through phases of stiffness related to the ratio of the horizontal to vertical orthogonal stresses. It was suggested that the backfill experienced failure at certain times when the deviatoric stresses (a ı - o 3) dropped. The aim of work conducted by Clark (1989) was to obtain a constitutive model of backfill behavior for use in numerical modelling.

Squelch (1990) explained a field measurement in a classified tailings backfill at 2030 m. underground mine with a dip angle of 10° for reef. Three triaxial stressmeters with mechanical closure meter were installed İn a backfilled stope with an average width of 1.22 m. A closure-ride station was also placed outside the backfill for comparison with that inside the backfill. The maximum recorded pressure and closure was 4.5 MPa and 130 mm respectively.

Gürtunca and Adams (1991) reported a field instrumentation in classified and dewatered tailing backfills at West Driefontein gold mine. They proposed an explanation of the effect of backfill in reducing the closure of stopes. They also stated that the behaviour of backfill İs determined to a large degree by whemer the material is partially confined or fully confined. The field results presented by Gürtunca and Adams (1991) showed that at constant strain curves, the highest stresses are developed at or near the center of backfill rib, and stresses decreased towards the edge of the backfill rib. They also showed that the maximum closure recorded at the edge of the backfill rib with the minimum closure at the center.

A backfill instrumentation result by laboratory and numerical modelling result was reported by Adams et al. (1991). He presented the performance of the three dimensional behaviour of three different types of backfill; a) de-watered tailings, b) full plant classified tailings and c)comminution waste. This included measurement of stresses and displacement inside backfill in three directions. These results show that the highest stresses are expected near but not at the edge of the backfill rib. The stresses drop slightly toward the center of the rib and fall significantly towards the edge of the rib. He also showed that stress cells may be inclined up to 25° to the principal stress plane and measure a decrease in stress from the principal stress of only 5%.

3 ROCKBURST

A seismic event is a transient earth motion caused by a sudden release of potential or stored strain energy in the rock. Rockburst is a seismic event which can cause injury to mine operators as well as damage to underground workings. The general and essential feature of rockburst is its sudden and violent nature. It is sudden, failure occurs within a very short period of time, and violent in the sense that it can result in significant damage around mine openings. The various types of rockburst have been classified as follows:

a) *Strain bursts:* These are caused by high stress concentrations at the edge of mine openings.

b) *Pillar bursts:* These are usually associated with the sudden failure of complete pillars when they become relatively small and/or overloaded.

c) *Crush bursr.* These are commonly observed in room-and-pillar mining where sudden, multiple pillar failure occurs.

d) *Fault-slip bursts:* These take the form of sudden slippage along a geological plane of weakness.

The above rockburst types can generally be grouped into two categories:

i) *Inherent bursts* (replaced strain bursts) : occurs where the pre-mining stresses are high enough to cause failure when the initial development openings are driven.

ii) *Induced bursts* (replaced pillar bursts and crush bursts) : are caused by mining operations transferring and concentrating stress on the remaining structures such as pillai.

Several techniques are currently available for the estimating of rockburst potential of underground excavations. From the review conducted, it appears that the "Stress" and "Strain Energy" approaches tend themselves easily to numerical modelling. Both methods were used in this project as the main analytical tools for the evaluation of rockburst potential.

3.1 *Effect of Backfill on Rockburst Alleviation*

There is very little information available on the effect of backfill on either the number of rockbursts or the damage caused. The use of backfill for reducing rock wall closure and hence the incidence of rockbursts in the Coeur D'Alêne mining district of the north-western United States of America is reported in papers by Pariseau and Kealy (1972), Board and Voegele (1983) and Corson et al. (1983). Comments are made by these authors about the benefits of using backfill and that higher density backfill improved the degree of control provided. However, no quantitative results, which could assist towards understanding backfill impact on rockburst, were documented.

Dhar et al. (1983) reported on a mine in the Kolar Gold Fields of India, with depths exceeding three kilometers, where a modified cut-and-fill method was developed and reduced the occurrence of rockbursts. Quesnel et al. (1989) reported on the use of a stiff consolidated rock fill for the alleviation of rockbursts at a mine İn Ontario, Canada had exceeded all expectations. They stated that the use of the rock fill appeared to have achieved significant reduction of the frequency of rockbursts. The use of semi-stiff (uncemented rock fill) backfill resulted in minimizing the damage caused by rockbursts but appeared to have littie effect on the frequency of rockbursts.

Close and Klokow (1986) reported that a mined out panel which had been backfilled for two years was less severely damaged than adjacents unfilled panels after a major rockburst. Bruce and Klokow (1986) reported a reduction in rockburst damage due to backfilling but indicated that it was not possible to identify a reduction in the number of seismic events or radiated energy because of the limited extent of backfilling which had taken place at that time. Gay et al. (1986) reported on visits made to inspect rockburst damage at a number of mines using backfill. In all cases, they reported that the use of backfill had been beneficial in that damage at the slope face had been restricted compared to unfilled areas, and gullies had remained open.

Recorded information on the effect of backfill on the number and magnitude of rockbursts is limited. In a paper by Gay et al. (1986) seismicity is compared for a backfilled and an unfilled area. The data suggested that, although seismic events occurred more frequently in the backfilled areas than in the conventionally supported area, the magnitude of the events in the backfilled area were smaller and generally more uniform.

When Cook et al. (1966) first proposed a method for evaluating the rockburst potential of a mining layout, they used it to quantify the effect of waste-filling and partial extraction (using regularly spaced stabilizing pillars) on the rockburst hazard. No direct comparisons between the two regional support systems were made by the authors but they concluded that the rockburst hazard can be reduced very effectively by partial extraction. The authors added that, on the basis of the criterion used, waste-filling is capable of reducing the hazard by an estimated 50 per cent in deep mines. However, it will only influence the hazard in shallower mines if the span of the excavation is exceptionally large. In addition, they commented that the waste-filling should be carried out continuously if the maximum benefits, İn terms of the rockburst hazard, were to be achieved.

DeJongh(1986) conducted a comparison of stabilizing pillars and backfill for regional support. The study stated that, for the mining layout modelled, backfill could achieve approximately the same reduction in the rockburst hazard as stabilizing pillars covering 15 percent of the area at an average depth of approximately 3300 m. A criterion known as energy release rate was used by all the above authors to estimate the level of the rockburst hazard from their calculations. Although these studies provide an indication of the possibility of replacing stabilizing pillars with backfill as a means of regional support, none of them have determined the influence of backfill quality or depth below surface on stabilizing pillar requirements, both of which are important parameters in establishing the efficiency of backfilling.

Undoubtedly, the most comprehensive analysis of backfill on the rockburst hazard was carried out by Ryder and Wagner0978). First, they developed a more accurate and direct method of modelling the effect of backfill on energy release rates after establishing that the methods used previously under-estimated the effects of backfill considerably. They then conducted numerous simulations of mining layouts, primarily to establish the effect of backfilling parameters on energy release rates, and hence the rockburst hazard. They concluded that high quality backfill can provide up to a six-fold reduction in the energy release rate, compared to a layout with no backfill, and that the distance at which backfill is placed from the face is a critical parameter in achieving energy release rate reductions. In a report written by Lloyd (1978), mention is made that backfill should be placed within 5 metres of the face and that high quality backfill can reduce the energy release rate from 80 MJ/m^2 to 20 MJ/m^2 at a depth of 4 kilometres. Lloyd suggested that the rockburst hazard would be reduced from 0.7 to 0.1 seismic events per 1000 square metres mined. Adams et al. (1989) evaluated the use of concrete pillars as a possible replacement for stabilizing pillars in deep mines. They concluded that the substitution is possible, from a rock mechanics point of view, with energy release rate and average pillar stress values being similar at a depth below surface of 3800 metres.

Pariseau and Kealy (1972) modelled the effect of backfill on the closure in a deep isolated vertical stope. Their results showed that relatively low quality backfill can reduce stope closure by approximately 50 per cent. When high quality backfill was used İn the model, the stope closure was only one-third of that obtained with a lower quality backfill and one-fifth of the closure for the same stope with no backfill. A reduction in stope closure has been related to the number and magnitude of rockbursts by Cook et al. (1966). From this relationship, it can be assumed that the reduction in closure reported by Pariseau and Kealy will result in a reduction in the rockburst hazard. Whyatt et al. (1989) determined that, for the mining layout modelled, backfill reduced energy release rates by 42 percent. They also established that a one per cent increase in the density of their backfill reduces the energy release rate by approximately one per cent and that an additional 28 percent reduction in energy release rate can be achieved by using a high quality backfill. They reported that the addition of cement in small quantities does not appear to affect energy release rates.

The following can be concluded from the literature review of the effect of backfill in rockburst alleviation: First, it appears that backfill has the potential to provide important regional support benefits in addition to substantial local support benefits which have already been recorded. Second, despite many decades of backfilling, there are no quantitative results from underground measurements which conclusively demonstrate the effect of backfill and backfilling parameters on the number and magnitude of rockbursts. The quantitative results which have emerged have been derived from the application of analytical solutions and numerical models. Third, the theoretical investigations which have been conducted to date have all used the concept of energy release rate as a mean of relating the analytical results to the rockburst hazard.

4 CASE STUDY OF MINE CHIMO

Chimo Mine is located 50 km east of Val D'Or, near Louvicourt, Quebec. Native gold is found associated to quartz and sulfides (arsenopyrite and pyrhotite). The deposit is part of a sequence of massive volcanic rocks, surrounded in the north by a sequence of metasedimentary and volcanic rocks, in the south by sedimentary rocks, grauwakes and pelitic schists.

About six ore bearing zones have been identified m Chîmo Mine. These ore zones are generally striking east-west and dip steeply (around 70°) to the north. The ore body between levels 16 and 17 included zones B, F and A with 4 meters wide and an average of 90 m long. Mining method is open stoping longhole with 70 m high of stopes. Hanging wall and foot wall are generally strong except in some situations where they contain a graphite bearing schist material. Blasting holes are 4 mches diameter.

4.1 *Field instrumentation*

The instrumentation at Chimo Mine consisted of rock mass instrumentation and backfill instrumentation. In the rock mass extensometers and vibrating wire Stressmeters were used to measure the displacements and change of stresses respectively.

In the backfill pressure cells were installed in the stopes prior to backfilling to measure the actual pressure within backfill. In order to choose the instrumentation location İn the mine, a preliminary numerical modelling was conducted to study the displacement and mining induced stresses in the rock mass and backfill during the mine-and-fill sequence. A 3-dimensional finite elements program was used to create 10 models presenting the mine-and-fill sequence of the mine. The results of numerical modelling were used to determine the stopes where significant build up of backfill pressure could be expected. The mme-and-fill sequence were also studied to determine when instruments had to be installed in order to respect the mining schedule.

A total of 6 hydraulic pressure cells were installed successfully in two backfilled stopes as follows:

- Three pressure cells were installed in 20 meters depth at middle of the stope 17-5B-5 in level 17.
- Three pressure cells were installed in 20 meters depth at 1/4 width from the east side wall at the stope 18-5F-8 in level 18.

A special procedure was used to install the pressure cells from the top of stope. This was the first experience for installation of pressure cells from the top of an open stope in a Canadian underground mine. The procedure was as follows:

- A cubic frame of elbow steel with dimension of (2" x 2" x 2") w/s prepared.
- 3 hydraulic pressure cells were fixed in x, y and z planes of the cubic frame.
- A cable was passed by the top of the stope using bow, arrow and fishing line. The arrow was shot to an access on the other side of the stope from the top access. The fishing lines was then used to pass cables from the top and other side of the stope. These cables were then fixed at both sides of the stope.
- The cubic frame together with 3 fixed pressure cells was lowered to the 20 meters depth from the top of the stope using cables and pulleys. Reflectors positioned on the corners of the box were used to target the box and allign it properly with surveying equipment once the target level in the stope was reached

Backfilling of the stope started after hanging the frame inside the stope. The pressure cells in North-South (across ore), East-West (along ore) and vertical directions were monitored and data registered once they were covered with backfill. The installation procedure of the pressure cells with frames prooved to be very successful.

4.2 *Pastefill Performance*

Field data from the pressure cells in two stopes were recorded over 200 days and excellent results were obtained. These data showed that a significant amount of energy was absorbed by paste backfill materials. The energy absorbed by backfill during the mining operation as well as rockburst activity was related to the geological condition of surrounding rock mass, mining activities around the backfilled stope and Young's modulus of the backfill. A performance analysis of these data is presented in below.

4.3 *Bacffilled Stope 17-5B-5*

Three pressure cells were installed in the center of the backfilled stope 17-5B-5 at a depth of 20 meters, on September 13 1995. Backfill operation started on October 20 1995 and finished on November 18 1995 (7 meters of backfill on top was left for later operation). Figure 1 presents the recorded backfill pressure in 3 directions versus the number of days since installation.

4.4 *Pressure build up in backfill*

When backfill operation was finished, the recorded backfill pressures were as follows:

$\sigma_{(N\text{-}S)}$ (across ore body) = 245 kPa
σ_{v} (vertical) = 180 kPa
$\sigma_{(E\text{-}W)}$ (along ore body) = 149 kPa

In last day of the operation, backfill was in a liquid situation with hydraulic pressure. Considering the 7 meters left on top, the vertical pressure on top of the pressure cell should be:

$$h = 20 - 7 = 13 \text{ m.}$$
$$\sigma_v = \gamma * h = 0.0145 \text{ MN/m}^3 * 13 \text{ m.}$$
$$= 0.189 \text{ MPa} = 189 \text{ KPa}$$

This is almost the same value that was collected from the field (180 kPa). In general, after termination of the backfill operation in stope 17-5B-5, vertical pressure in the backfill was equal to gravity loading.

The ratio of the build up pressure in different directions inside the backfill was compared with in situ stresses and presented in Table 1. This comparison shows that whereas the ratio of $\sigma_{(N\text{-}S)}/\sigma_{(E\text{-}W)}$ for backfill pressure was in close agreement with in situ stresses, the ratio of $\sigma_{(N\text{-}S)}/\sigma_{(v)}$ was different. This comparison tends to indicate that the ratio of build up pressure in the horizontal section (or plane) is related to the ratio of the in situ stress in the horizontal section. Because, if only gravity loading was involved, the stress ratio in the horizontal plane should have been 1.

Table 1 Stress ratio rock/backfill (after termination of the backfill operation).

	$\sigma_{(N\text{-}S)}/\sigma_{(E\text{-}W)}$	$\sigma_{(N\text{-}S)}/\sigma_{(V)}$
Backfill	1.64	1.36
In situ Stress	1.52	2

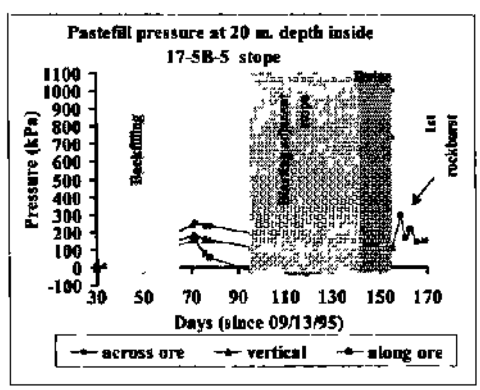


Figure 1 Pressure inside the backfilled stope 17-5B-5 since the day of installation.

4.5 *Curing Period of Backfill*

During the curing period, the temperature inside the backfill increased. The backfill temperature at the beginning of the backfill operation was 18° C. During the next 60 days, backfill temperature increased to the 38° C and then started to decrease very smoothly.

During the curing period {from the termination of the backfill operation and over the next 40 days), the backfill pressure decreased in 3 directions. Decreasing of the backfill pressure in N-S and vertical directions was very smooth, keeping a ratio between 1.3 to 1.7. But, the backfill pressure in E-W direction dropped significantly to a nominal zero pressure. This was due to the opening of the adjacent stope (17-5B-3) on December 16 1995.

4.6 *Opening the Adjacent Slope*

During the mining advance of the adjacent stope 17-5B-3, the backfill pressure in N-S and vertical directions increased, but in E-W the pressure remain close to zero. After blasting the last layer of the ore body in stope 17-5B-3, backfill pressure increased by 400 and 275 kPa in N-S and E-W directions respectively. The pressure cell in the vertical direction was damaged. It seems that the backfill pressure in vertical direction continued to keep the same ratio as the N-S direction. The backfill pressures on day 155 (before rockburst) in 3 directions were as follows:

$\sigma_{(N-S)}$ {across ore body) = 1.0 MPa
σ_v (vertical) = 0.7 MPa
$\sigma_{(E-W)}$ (along ore body) = 0.1 MPa

During the 20 days (since day 140 to 159 in Figure 1) preceding the rockburst activities, the backfill pressure in N-S direction increased from 500 to 1000 kPa. This might be used as an indicator for the potential of rockburst. The backfill pressure increased by a rate of 32.5 kPa/day.

4.7 *Backfill Pressure vs Stresses in Hanging Wall*

The backfill pressure can be used as an indicator to check the movement of the surrounding rock mass toward the backfill. Figure 2 presents a vertical section of the backfilled stope 17-5B-5. The hanging wall of the stope was instrumented using one extensometer and 2 VBS to measure the displacement and change of the stresses respectively. Figure 2 shows that the extensometer 17-3 intersected with a thick graphite zone. This graphite zone had a major effect on stability of the hanging wall.

Field data recorded from extensometer 17-3 is presented in Figure 3. Anchors 1 and 2 were located above the graphite zone and anchors 3, 4 and 5 between graphite zone and stope wall. Figure 3 shows that anchors 3, 4 and 5 haved moved up to 4 times more than anchors 1 and 2 prior to the rockburst. This was due to the fault separation that caused a significant pressure increase on backfill.

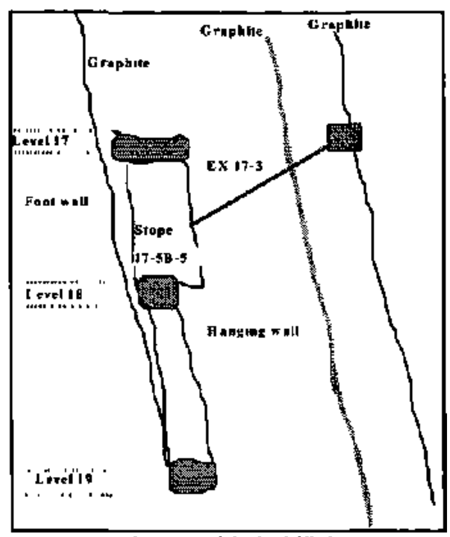


Figure 2 Venical section of the backfilled stope 17-5B-5

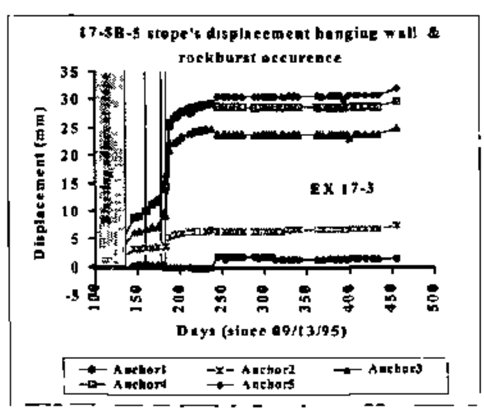


Figure 3 Displacement at the hanging wall of the stope 17-5B-5.

The maximum measured displacement of the hanging wall of backfilled stope 17-5B-5 was 32 cm. The hanging wall of the adjacent backfilled stope (17-5B-6) had a 8 cm displacement. At the hanging wall of the other adjacent open stope (17-5B-3), closest anchor to the opening was lost, this would indicate a displacement of more than 1 meter. A comparison among the displacement of these three stopes showed that the backfill had prevented a major collapse of the hanging wall.

Figure 4 shows a comparison between displacement of hanging wall in N-S direction and build up pressure in backfill since pressure cell installation. The backfill pressure grew due to the displacement of the hanging wall. Figure 5 shows a comparison between the change of vertical stress in hanging wall and vertical backfill pressure. It can be observed that the reduction of the vertical stress İn the hanging wall was parallel to the increasing vertical stress İn the backfill. This indicates that, through convergence, the backfill was being solhcited increasingly.

Figure 5 shows a comparison between the change of horizontal stress in the hanging wall and the horizontal pressures measured in the backfill. It can be observed that changes in the backfill stresses were synchronized with stress changes in the surrounding rock mass.

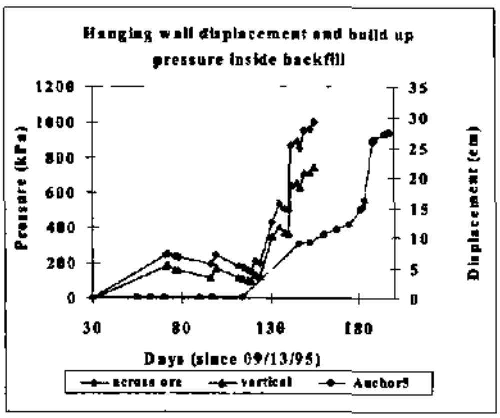


Figure 4. Comparison between displacement of hanging wall in N-S direction and build up backfill pressure.

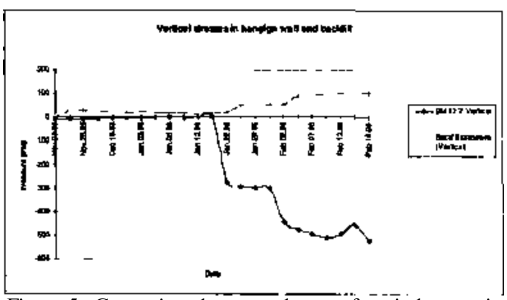


Figure 5. Comparison between change of vertical stress in hanging wall and backfill pressure

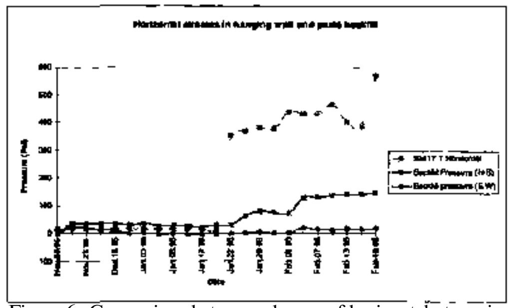


Figure 6. Comparison between change of horizontal stress in hanging wall and build up backfill pressure

A comparison between Figures 5 and 6 shows that in general, the vertical stress in the hanging wall was decreasing as follows; a) a rotation of stresses inducing higher horizontal stresses within the hangingwall, b) a certain amount of stress transfer from the rock to the backfill. It is apparent that there was intense stresses redistributions in the rock around these stopes and from the rock to the backfill.

4.8 *Stress Calculation Inside the Backfill*

The stress m N-S direction inside the backfill can be calculated as follows:

0 - E * e

where E İs the Young's Modulus of the backfill (E nominal = 22 MPa) and £ is the strain of the backfill due to the displacement of hanging wall in N-S direction. Assuming that the major part of backfill strain was related to the hanging wall displacement then, considering the displacement of the hanging wall prior to me rockburst 10 cm and the width (vertical distance between two side walls) of the stope 4 meters:

dX = 10 cm

e = d *XI* L=10 cm / 400 cm = 0.025

σ = 22 MPa * 0.025 = 550 kPa

This value is within the range of data collected in the field in N-S direction right before the rockburst.

These results explain mat the pressure applied inside the backfilled stope 17-5B-5 was related to both the displacement of the hanging wall and the Young's Modulus of die backfill. The fault movement towards me backfill causes higher backfill strain and pressure.

4.9 *In Situ Paste Backfill Behavior (Stress Versus Strain)*

The paste backfill inside the stope with a size of 4m.x 15m.x70m. can be assumed as a large specimen that is tested by a testing machine in laboratory. The major source of the force was the displacement and bending of die hanging wall due to the fault separation. The value of strain in the middle of the backfill was obtained by dividing the displacement of the anchor 5 by the stope wide:

$$\varepsilon = d\,X / L = dX_{anchor\,5} \;/\; 400\ cm$$

Considering that the angle between the extensometer and stope wall is 70° then, the value of the stress vertical to the stope side (Force applies in the direction of extensometer) was calculated as follows:

$$\sigma = \sigma_{(N\text{-}S)} \times \cos 70^{\circ} + \sigma_{(v)} \times \sin 70^{\circ}$$

A graph of the backfill stress in direction of extensometer versus displacement of anchor 5 was prepared and presented in Figure 7.

4.10 *Strain Energy Density*

Having the stress and strain inside the backfill, the strain energy density was calculated as follows:

Strain Energy Density = 1/2 x E x a

Figure 8 presents the strain energy in backfilled stope 17-5B-5 in a period of one month before starting the rockburst activities. This graph shows that the strain energy density increased significantly prior to the rockburst activity. The rate of absorption of the strain energy into the backfill might be used as an indicator to study the potential of rockburst. This method needs further investigation İn field sites.

4.11 *Energy Absorbed by Pastefill*

The field data of pressure cells show that backfill absorbed a large quantity of the energy from the surrounding rock mass. This energy transfers to the backfill through the work done by displacement and build up pressure.

Work = Displacement x Force
(W = D x F)

Force = Stress x Area
(F = o x S)

To calculate the energy absorbed by pastefill in stope 17-5B-5, with 70 m. height and 15 m. width:

$S = 15 x 70 = 1{,}050\ m^2$

The recorded backfill pressure m N-S direction before the rockburst was 1 MPa, therefore:

Force = 1MPa x 1,050 m^2 = 10^6 x 1.05

Pascal x m^2 = 10^6 x 1.05 Newton

Force = 10^6 x 1.05 Kgf

Considering the 10 cm displacement in hanging wall at the end of rockburst period:

$$Work = (10^{6} \times 1.05\ Kgf) \times (10 \times 10^{-2})$$
$$= 32.5 \times 10^{3}\ Joule = 105\ kJ$$

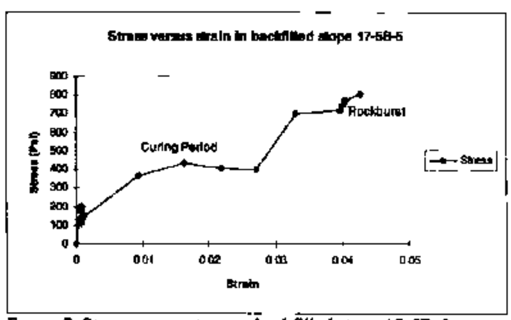


Figure 7 Stress versus strain at backfilled stope 17-5B-5

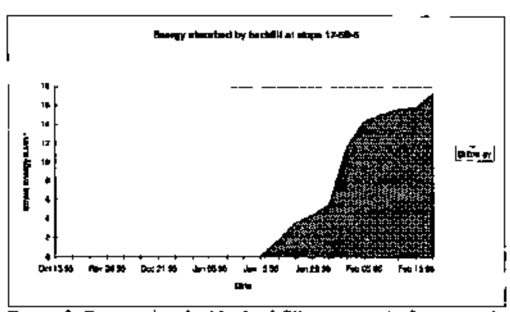


Figure 8 Energy absorbed by backfill in a period of one month prior to rockburst

5 CONCLUSION

The performance of backfill İn the alleviation of rockburst can be concluded as follows:
The build up vertical pressure after termination of backfill operation was equal to hydraulic pressure in the case of no mining activity around the stope. In the case of mining activities around the backfilled stope, the vertical backfill pressure was 3 times more than hydraulic pressure.

The ratio of build up backfill pressure in different direction was related to the in situ stress in the case of no mining activity around the stope. In the case of mining activities around the stope, the convergence had a major role in the magnitude of the backfill pressure İn each direction.

During the curing period, the backfill pressure and temperature decreased İn the case of no mining activity around the stope. In the case of mining activities around the stope, the temperature still dropped but the backfill pressure increased and the some stress transfer took place from the rock to the backfill.

Significant stress redistributions took place due to the mining activities and stress transfer occurred from the rock toward the backfill.

The pressure inside the backfill was related to both the displacement of the side walls of the stope and the Young's Modulus of the backfill. The fault movement in side walls of the stope caused higher backfill strain and pressure.

The strain energy increased significantly in a period of one month before the start of rockburst activities. The rate that strain energy was absorbed into the backfill could be used as an indicator to study the potential for rockburst. This method needs further investigation in field sites. The rate of stress increase in the backfill might be used as an indicator to predict the potential of rockburst.

Backfill protected the stope walls from sudden movement. During the rockburst period, the hanging wall displacement for two backfilled stopes were 8 and 20 cm. At the same period, anchor 1 (the closest anchor to the stope) in the adjacent open stope was lost (indicating a potential displacement of more than 1 meter). The presence of backfill inside the stope prevented these large movements.

Backfill absorbed energy through the compressive strain. The movement of the side walls compressing the backfill, increased the backfill strain as well as the pressure inside the backfill. This deformation work done on the backfill by the convergence reduced the concentration of energy in surrounding rock mass. By decreasing the concentration of stress İn surrounding rock mass, the rockburst intensity probably reduces.

The change of strain energy density of backfill, in a period of one month prior to the rockburst activity, was presented. This graph showed that the strain energy density increased significantly prior to the rockburst. The rate that strain enrgy was absorbed into the backfill might be used as an indicator to examine the potential of rockburst. This method needs further investigation in field sites.

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REFERENCES

Adams, D.J.. Giirtunca, R.G, Jager, A.J and Gay, N.C, 1989. "Assessment of a new mine layout incorporating concrete pillars as regional support", *Proceedings of the 4th International Symposium on Mining with Backfill,* Montreal, October 1989. pp. 199-208.

Adams, DJ., Giirtunca, R.G. and Squelch, A.P., 1991. "The 3-dimensional in situ behaviour of backfill materials". Proceedings of the 7th International Congress on Rock Mechanics, Witlke(ed.), Balcema, Rotterdam.

Board, M.P. and Voegele, M.D., 1983. "Examination and demonstration of undercut and fill sloping for ground control in deep vein mining", *Proceedings of the 3rd International Symposium on Mining with Backfill,* Lulea, June 1983. pp. 300-306.

Brace, M.F.G and KJokow, J.W., 1986. "A follow-up paper on the development of the West Driefontein tailings backfill project". *Proceedings of a Symposium on Backfill in South Africa Mines,* South African Institute of Mining and Meiallurgy, Johannesburg, November 1986.

Clark, I.H., 1989. "The strength and deformation behaviour of backfill in tabular deep-level mining excavations. PhD Thesis, University of the Witwatersrand.

Clark, I.H., Gürtunca, G.R. and Piper, P.S., 1988. "Predicting and monitoring stress and deformation behaviour of backfill in deep-level mining excavations. *Proc.5th Aust New-Zealand Conference on Geomechanics,* pp 214-218.

Close, A.J. and Klokow, J.W.. 1986. "The development of the West Driefontein tailings backfill project", *Association of Mine Managers of South Africa, Papers and Discussions,* vol.3/85, November 1985.

Cook, N.G.W., Hoek, E., Pretoius, J.P.G, Ortlepp, WD. and Salamon, M.D.G., 1966. "Rock mechanics applied to the

study of rockbursts". *Journal of Che South African Institute of Mining and Metallurgy,* Johannesburg, vol.66, May 1966,pp.436-528.

Corson, D.R., Dorman, K..R. and Sprute, RH., I983. "Improving the support characteristics of hydraulic fill". *Proceedings of the 3rd International Symposium on Mining with Backfill,* Lulea, June 1983, pp.93-99.

Corson, D. R. and Wayment, 1967. "Load-displacement Measurements in a Backfilled slope of a Deep Vein Mine". Report of Investigation 7038. U.S. Bureau of Mines, Washington, D.C.

Corson, D.R., 1971. "Field evaluation of hydraulic backfill compaction at the Lucky Friday Mine, Mullen, Idaho". Report of investigation 7546. U S Bureau of Mines, Washington, D. C.

De Jongh, C.L., 1986. "The potential of backfill as a stope support in deep mines", GOLD 100 - *Proceedings of an International Conference on GOLD, South African Institute of Mining and Metallurgy,* vol.1, pp.271-297, Johannesburg, 1986.

Dhar, B.B, Shankar, K.V. and Sastry, V.R., 1983. "Hydraulic filling - an effective way of ground control", *Proceedings of the 3rd International Symposium on Mining with Backfill,* Lulea, June 1983, pp.113-120.

Gay, N.C, Jager, A.J. and Piper, P.S., 1986. "A quantitative evaluation of fill performance in South African gold mines", *Proceedings of a Symposium on Backfill in South African Mines, South African Institute of Mining and Metallurgy,* Randburg, June 1986.

Greig, J. D., Lloyd, P. J. D., Quinn, M. and Blight, G. E., I979. "Instrumentation to measure various parameters of fill in deep-level excavations and a preliminary assessment of the performance of a cemented slimes fill".

Gürtunca, R.G. and Adams, D.I., 1991. "Results of a rock engineering monitoring programme at West Driefontein Gold Mine". Journal of SAIMM, Vol 91, No 12, December 1991.

Gürtunca, R.G., Jager, A., Adams, D.J. and Gonlag, M„ 1989. "The in situ behaviour of backfill materials and the surrounding rockmass in South African gold mines". *4th Int. Symp. on Innovations in Mining Backfill Technology,* Montreal, Canada.

Gürtunca, R.G and Squelch, A.J., 1990. "Benefils of backfilling". Unpublished research report *Chamber of Mines of South Africa. Johannesburg,* November 1990.

Hedİey, D.G.F., 1992, "Rockburst handbook for Ontario hard rock mines", CANMET special report SP92-1E.

Hoek. E. and Brown, E. 1986. "Underground excavations in rock". IMM pp 123.

Lloyd, P.J.D., 1978. "The potential of integrated mining and extraction systems on the Witwatersrand", Unpublished research report. *Chamber of Mines of South Africa,* Johannesburg, 1978.

McNay and Corson, 1961. "Hydraulic sandfill in deep metal mines". Bureau of mines, Spokane, Washington.

Pariseau, W.G. and Kealy, CD., 1972, "Support potential of hydraulic backfill", *Proceedings of the 14th Symposium on Rock Mechanics,* Pennsylvania, June 1972, pp.501 -526.

Ryder, J.A. and Wagner. H., 1978. "2D Analysis of backfill as a means of reducing energy release rates at depth". Unpublished research report, *Chamber of Mines of South Africa,* Johannesburg, 1978.

Squelch, *A.* P., 1990. "Results of a monitoring program al vaal Reefs, 5 Shaft". COMRO Reference Report 24/90,1990.

Thibodeau, D., 1989. "In situ determination of high density alluvial sand fill". *Innovations in mining backfill technology: 4th international symposium,* Montreal.

Quesnel, W.J.F., De Ruiter, H. and Pervik, A., 1989. "The assessment of cemented rockfill for regional and local support in a rockburst environment", *Proceedings of the 4th International Symposium on Mining with Backfill,* Montreal, October 1989, pp.217-224.

Whyatt, J.K., Board, M.P. and William T.J., 1989 "Examination of the support potential of cemented fills for rockburst control". *Proceedings of the 4th International Symposium on Mining with Backfill,* Montreal, October 1*989, pp.217-224.

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Management of Waste in the Mineral Processing in the Baia Mare Area

S.Arad, V.Arad & S.Irimie
Petrosani University, Petrosani, Romania

D.Cosma
The West University of Timisoara. Romania

ABSTRACT: It can be stated that there is a constant preoccupation with solving risk problems in mineral exploiting and processing activities, which have a powerful impact on the environment. This paper reports a preliminary study in which sources of pollution and waste in precious metal processing activity were identified in the Baia Mare area. Data from water, air and soil samples were analyzed by statistical methods, and the global pollution indexes of environmental factors were also established. Using the results of the study, it is possible to appreciate the impact on the environment factors, and a waste management plan is suggested.

1 INTRODUCTION

Taking into account the great interest in environment protection and internal and international laws imposed in recent years, ecological aspects are being studied more and more in Romania.

The Baia Mare mining basin İs made up of metalliferous deposits of the following types:

1. Gold-silver deposits (Bäita, Säsar, Valea Roşie, Dealul CnıciÎ, Şuior) in the upper part.
2. Complex metallic sulphide deposits with a Pb and Zn content, to which Cu, Au and Ag are subsidiarily associated. At Cavnic and Bäiut, the gold- silver contente are the object of mining.
3. Cupriferous deposits, to which Au and Ag are subsidiary associated (Nistru, Baia Sprie, Toroiaga, Baia Borsa).

At Şuior, auriferous arsenous pyrites are stockpiled, forming a waste dump of 300,000 t over an area of 1,000 m^2. The stockpiling of the ores is being carried out until an adequate processing technology is found, taking into account the As content.

The dressing of auriferous-silver deposits in the Baia Mare area is done in the ore-processing plants: Säsar and Flotatia Centrals. The Meda waste dump was established by stockpiling the waste resulting from flotation and cyaniding activities at the Säsar processing plant.

The Romanian-Australian joint venture, Aurul SA, which has the production of precious metals as the object of its activity, focuses on retreating the waste that has resulted from the two ore-processing plants in Baia Mare.

The establishment of the Aurul SA. project İs related to the idea of moving waste dumps from urban areas to areas farther from the inhabited zones. Owing to the ore-dressing technology used, the above-mentioned waste has a high recoverable content of precious metals, and this has enabled the recovery process to be carried out efficiently.

The Bozanca slurry pond stores, in a safe, environmentally friendly manner, the tailings resulting from cyaniding after the recovery of gold and silver with active carbon according to the CIP method.
The Bozânta slurry pond lies 6.5 km from the plant for retreating waste on level ground, near the operating Bozanca pond and the Säsar conservation pond belonging to the Baia Mare Remin Company.

2 POLUTION LEVEL OF THE ZONE

2.1 *The Bozanca slurry pond*

The Bozanca pond is made up of two embankments of permeable material from the Säsar preserve pond. Although the uncovered ground where the pond was built was composed of clay rocks, it was completely covered with a high-density material. This has a thickness of 0.5 mm inside the starting embankment and a thickness of 1 mm under the starting embankment, on the outline and between the two embankments, in order to ensure the pond is watertight given the ground, and to avoid pollution of the environment.

Inside the outline embankment, there is a drainpipe, through which the infiltration waters are collected through the starting embankment and through the embankment built during the hydrocycloning operation.

In the Bozanla slurry pond, the waste resulting from cyaniding and the recovery of gold and silver with active coal is stockpiled in a safe and ecological way.

The working technology is based on the construction embankment of the raising by hydrocycloning with the deposition of the heavy fraction in the embankment and the light fraction on the internal beach of the pond. The clearing waters are evacuated through a reverse derrick to the pump station, from which the water is sent to the plant and used in the technological process.

In case of any technological accidents, there is a functional damage pond, on the east side, built like the big pond.

3 POLLUTION SOURCES

3.1 *Soil pollution*

The waste dumps in the area, notably Meda and Bozanca, constituting the main source of raw material for the retreating plant, have been constructed directly on the soil and are not insulated against the soil or groundwater layer.

The slurry ponds in the area have polluted the soil with metals and caused an increase in soil acidity due to the phenomenon of natural biological leaching that occurs in the waste mass. According to the assays carried out before the activity had begun, the high concentration areas exceeding the intervention thresholds are situated north-west of the ponds. The northeastern area, according to the same study, is within allowable limits, and the increasing tendency of tibe overall pollution index is from the east to the west.

The principal metals for which increases in the intervention threshold have been recorded are: Mn, Zn, Pb.

Taking into consideration the type of pollution the soil has been exposed to in the surveyed area, namely, industrial pollution due to precious metal recovery activity, the degree of pollution is quite well defined by pH, by the heavy metal content and by cyanide content.

The main effects of the Bozânfa slurry pond on the soil are:

> taking up the area (94ha);
> potential pollution with fine waste material borne by the wind and later deposited on the neighbouring terrain.

Therefore, the soil sample were assayed as follows:

- determining the pH;
- determining Cu, Zn, Pb, Mn, and Cd;
- determining cyanides.

3 2 *Water pollution*

Although the Meda dump is located in the vicinity of the Săsar river and may constitute a potential source of pollution, quantifying die impact of this pollution source is difficult due to the accumulation of several sources of pollution in the area, sources from neighbouring units.

The quality change in the groundwater layer in the surveyed area in the existing piezometers, shown mainly by the cyanide indicator, could be a signal that the surface water quality should be examined.

3.3 *Air pollution*

Atmospheric pollution in the area of the Meda waste dump may be due, first of all, to the fine waste particles carried from the pond outline, resulting in an increase in the airborne powder content.

This air pollution has been felt for a long time and is not specific to the activity carried out by S.A Aurul at the Meda waste dump.

The pollution by powders is due, to a large extent, to the transport activities in the area, taking into consideration the fact that the access ways have not been modernized. The content of airborne powders has been determined. The polluting elements in the atmosphere according to regulation STAS 12574-87 are only *Câ* and Pb. The studies undertaken at the Baia Mare dumps have shown that, in general, the Pb content is below 0.1%, while Cd is present only in trace amounts.

The dust emission at the Bozanca pond resulting from fine fractions carried by the wind is low due to the moisture of the material, and is largely due to the heavy traffic on the gravelled access ways.

4 STRATEGY REGARDING MANAGEMENT OF WASTE ÎN THE BAIA MARE AREA

In order to prevent or reduce environmental pollution in an ecological reconstruction and rearrangement of the effected zones in Baia Mare, a project of environmental management and protection is at present being undertaken by specific institutions according to environmental legislation.

When starting upon the plan of environmental management, it was necessary to define the notions of "significant impact" on the environment and the "importance of the impact".

According to the situation in the Baia Mare area, the main problem was how to define the notion of "significant impact". If the significant impact has already occurred, it İs necessary to allocate money and time resources to evaluate it. For non-significant impact, as assessed by an objective evaluation, a report regarding this impact should be prepared so as to gain important savings in time and money.

The evaluation was structured as a three-level analysis:

- Level no. 1 - a list of categorical exclusion regarding the activities with no impact on the environment.
- Level no. 2 - an evaluation report identifying the non-significant impacts.
- Level no. 3 - a final report about me main environmental impacts.

5 RESULTS AND CONCLUSIONS

In order to measure the extent of degradation of the environment in the pond area, soil, water and air samples were processed and analyzed physically and chemically in terms of the main pollution factors which could affect the quality of the environment as a result of the activity at S.C. AURUL SA.

By analyzing and interpreting the date obtained by the chemical analysis of the soil, underground water and air samples taken from the area, we came to the conclusion that the soil and the groundwater layer are polluted as a result of previous activities.

Compared to the original degree of pollution, because of the activity carried out by S.C. Aurul SA, additional pollution is possible, namely:

> The waste dislodging technology by hydromonitoring results in die emergence of suspension, which is leaking towards the dam.
> The fine waste on the transport ways arranged along the pond during waste dislocation may be blown away by the wind.
> The dislocation technology by hydromonitoring uses process water under pressure; aerosols generated with a high cyanide content can be taken up by die wind, polluting the atmosphere.
> Taking into consideration the object of the activity at the waste retreating plant, the main potential sources of soil pollution and the principal pollution sources, the principal pollutants analysed are non-ferrous, heavy metals and cyanide.

On the basis of the concentrations determined in the soil, groundwater and air samples from Meda waste dump and Bozân{a slurry pond, the pollution index was calculated and each element was given a note. The results are shown in Table 1 and Table 2.

Due to the great amount of precipitation in January 2000 at the Bozanta mud-setting pond, a technical accident occurred, which had a great impact on the environment. This event created a breach İn me starting embankment from 20-35 m and there was a discharge of 100,000 m^3 of water, as illustrated in Figure 1. All measures for minimising pollution in this situation were taken immediately. It was necessary to conduct a waste management study for this objective.

Table 1 Analysis of environmental factors at Meda waste dump.

Environmental factors	Analyzed Elment	Level I (0 05m) IP*	Level I (0 05m) NB*	Level II (0 3m) IP*	Level II (0 3m) NB*
Soil	CN	0 0009	9	0.0002	9
	Cu	1,89	6	1,035	6
	Zn	0 86	7	0 847	7
	Mn	0,42	8	0 359	8
	Pb	8.13	3	5 07	4
	Cd	5.67	4	5 8	4
	IPG	2.32	-	2 17	-
Ground water layer	Fe	89 6			
	Cu	2.7			
	Zn	2 4			
	Mn	825 3			
	Pb	4.8			
	Cd	25			
	IPG	9.09			
Air	Airborne powders	0.23			
	Cyanide	0			
	IPG**	1.05			

Table 2 Analysis of environmental factors at Bozanta slurry pond

Environmental factors	Analyzed Element	Level I (0 05m) IP*	Level I (0 05m) NB*	Level II (0.3m) IP*	Level II (0.3m) NB*
Soil	CN	0.0002	9	0 0001	9
	Cu	0 76	7	0.68	7
	Zn	0.86	7	0 95	7
	Mn	0.33	8	0 38	8
	Pb	10.8	3	19 85	2
	Cd	5.16	4	5 52	4
	IPG	2.22	-	2 39	-
Ground water layer	Fe	706 2			
	Cu	10 08			
	Zn	9.61			
	Mn	2306 2			
	Pb	9,74			
	Cd	78 6			
	IPG	23 07			
Air	Airborne powders	0 255			
	Cyanide	0			
	IPG**	1.11			

*Index pollution and beneficiation note,
**Global pollution index

Figure 1 Technical accident at Bozanta slurry pond

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Linking Long-Term Environmental Liability and Closure: A Necessary Development Towards Walk-Away Closure

E.C.Packee Jr.
Travis/Peterson Environmental Consulting, Inc., Fairbanks, Alaska, U.S.A

S .Bandopadhyay
University of Alaska Fairbanks

ABSTRACT: Long-term liability is a risk that must be considered prior to permanent mine closure. Current definitions of closure do not link liability release with the attainment of environmental performance standards. Indeed, contaminated sites laws of major mineral producing nations hold me mine owner, and long-term corporate profitability, accountable not only for current environmental regulations but for future regulations as well. This is known as absolute and retroactive liability. This paper uses the South African situation to address the impact that absolute and retroactive liability has on the long-term profitability of the raining industry and then uses internationally published definitions of mine closure to demonstrate the desirability of linking 'closure' with liability release. Once liability and closure have been linked, all that prevent 'walk-away' closure is the determination of the financial implications of future environmental degradation at the site.

1 INTRODUCTION

Internationally, published definitions of mine closure generally indicate that closure occurs following the cessation of extractive operations at the site. South African, Canadian, and Australian mines reach closure following decommissioning and final rehabilitation, while closure initiates decommissioning and final rehabilitation at mines in the United States of America. Current definitions do not address the long-term liability for the site following closure. Linking closure to liability release means that mine closure occurs when the state accepts responsibility for the decommissioned site. In the case of South Africa, site liability varies between the three principal acts governing mine environmental management and is currently preventing closure from occurring.

Closure is a necessary step in the management of South African mines because mine assets cannot be sold until a Certificate of Closure has be issued by the Department of Mineral and Energy Affairs. The current confusion surrounding closure threatens the economic survival of mining corporations in South Africa and must be resolved. Industry should take the initiative by assuming that mine owner's have absolute and retroactive liability for the impacts of mining. Although this represents 'worst case liability', it insulates the mine owner and operation from changing environmental performance standards. The Australian concept of 'safe and stable' mines provides site specific and quantifiable means for determining the impacts of mining following closure.

2 THE DEFINITION OF MINE CLOSURE

Traditionally, the environmental management of collieries ceased following permanent closure. The long-term liability imposed by the 'polluter-pays-principle' fundamentally alters the 'operating environment' of coal corporations. This 'new' reality forces are-examination of conventional definitions of'closure* and 'impact' and the mining lifecycle.

2.1 *Current international definitions of mine closure*

The operating environment of coal corporations is a dynamic system. Individual operations are sensitive to changes in corporate priorities, commodity prices, and extraction costs. Mining is venture capitalism, albeit on a grand scale. Traditionally, closure represented the final stage of the mining life cycle. Corporate involvement and site responsibility ceased following closure and the property lapsed into abandonment. Indeed, this view pervades the mining industry. Consider the following definitions:

United States of America (Hardrock)

"Closure - Closure involves the removal of structures/buildings, and other infrastructure, and initiation of reclamation on the yet unreclaimed ' portions of the mine" (Taggart and Kieth, 1997).

"Closure is defined as the activity of a mining company related to the shutdown and reclamation of mining projects in a cost-effective manner" (Licari, 1997).

"Closure/post-closure Phase - Structures are removed and ground surfaces are recontoured and revegetated. Underground mines may be plugged and other measures for the control of acid mine drainage are initiated" (Murray, 1997).

"Closure entails:

- End of processing, deposition, or use;
- Drain-down, treatment, and release of process water;
- Construction of containment structures;
- Plugging of drill holes, adits, or drifts;
- Detoxification of process equipment;
- Machinery salvage;
- Removal of buildings, pipelines, and structures; and
- Final reclamation/revegetation" (Williams, 1998).

United States of America (Coal)

"Upon permanent cessation of operations, the operator shall complete the reclamation plan submitted under 11 AAC 90.083 - 11 AAC 90.099 as approved by the commissioner" (Anon, 1999).

Australia

The Minerals Council of Australia (Anon., 1997) has determined that closure is the "permanent cessation by a company of operations at a mine or mineral processing site after the completion of decommissioning process which is signified by tenement relinquishment." Closure follows the rehabilitation and decommissioning of the mine site.

Canada

The regulatory authority for Canadian mining operations resides, primarily, with provincial governments. However, the closure definition, used and implied, is similar to the Australian definition. The adopted definition is that closure follows decommissioning and final rehabilitation. However, the mine owner's liability following closure varies significantly between provinces (Cowan, 1996; Bourassa, 1996; Daigneault, 1996; and Overholt, 1996).

2.2 *Current South African mine closure definition*

The Environmental Management Programme Report Process (EMPR) sets fort a definition of closure for South African Mines. The Aide Memoire (Anon. 1992) indicates that: "Closure, in the case of mining operations discontinued or abandoned prior to the coming into force of the Minerals Act, 1991, means that a closure certificate has been issued in terms of Regulations 2.11 under the Mines and Works Act, 1956, or in a.iy other case, mat a closure certificate has been issued in terms of Section 12 of the Minerals Act, 1991, or in terms of regulation 2.11 thereunder, and that a closure certificate provided for in Section 32(2) of the Atmospheric Pollution Prevention Act, 1956, has been issued."

As in the case of Australia, closure at South African mines occurs following site decommissioning. More importantly, there is no link between closure and liability.

All of the above definitions indicate that once extractive operations cease, the mine enters closure. In the case of definitions used in the United States, this point demarcates the beginning of site decommissioning and final rehabilitation. In Canada, Australia, and South Africa closure follows decommissioning and final rehabilitation. Thus, closure represents a discreet point m time. Because economically viable mineral deposits are finite, all mines will eventually cease extraction and close. This perception is implicit in the expectation that closure is merely a formality for mines, assuming that proper environmental control was exercised during operations (Greef, 1993; Anon 1993). The assumption or expectation that mine closure releases the owner from long-term environmental liability is inconsistent with the 'polluter-pays-principle'. Commenting on the closure of a mine near Faro, Yukon Territory, White (1996) states that:

"Closure is not an open and shut case. It is a process of reconciliation for, at any point in time, [minmg] costs that may represent the price of past failures that were not seen [as failures] at the time. Looking back, it is tempting to blame unwillingness, neglect, ignorance, or attitude. This is unfair. In looking at the past, there is clear evidence that the standards of the day and the knowledge of the times were frequently applied. So, perhaps the attempt to find blame is really overreaction that ignores the historical evolution of simple economics imposed upon us by uncontrollable, external factors that have nothing to do with what we would like to do, but more with what can be afforded - then and now."

White's observation introduces two fundamental issues that must be addressed prior to permanent closure. First, mine owners will be held financially accountable for the failures of environmental management during operations regardless of cause. Second, mine closure is not simply a phase of the mining cycle but is inextricably linked to environmental liability.

3 LEGAL LIABILITIES FOLLOWING PERMANENT CLOSURE

The 'polluter-pays-principle' introduces the concept of site responsibility following permanent closure. Legislated liabilities and economic necessities compel mine owners to reconsider the conventional view of closure. The liability issue at, and following closure, is not unique. In fact legislated long-term liability is commonplace for mineral producing nations.

3.1 *Legislated long-term environmental liability in mineral producing nations*

The regulatory environment defines the constraints placed on mines following closure. Pertinent constraints include long-term liability provisions, scope of liability, and the duration of liability. The following section contains selected examples of long-term liability provisions in mineral producing nations.

United States of America

The Comprehensive Environmental Response, Compensation, and Liability Act of 1980 (CERCLA) is the most notorious environmental liability statute in die world. CERCLA and subsequent amendments, commonly referred to as Superfund, were enacted to force potentially responsible parties (PRPs) that cause or have caused contamination of soil, air, or water to pay for clean-up efforts. CERCLA establishes strict liability for the site, which means that no evidence of wrongdoing is necessary for enforcement action against PRPs. In general, the government, acting through the Environmental Protection Agency (EPA), establishes a 'joint and several liability' claim against one or more of the PRPs. 'Joint and several' means that any one of the PRPs is responsible for the costs of clean up. It is then up to the affected PRP to seek reimbursement from other PRPs through legal action. This statute applies to all industrial facilities mat generate hazardous wastes, including mining operations (Cowan, 1997). According to Cowan (1997) typical mining problems that result in CERCLA enforcement include:

- Acid mine drainage;
- Trace metal releases from tailings impoundments;
- Contaminated soils; and
- Radioactive mine wastes.

CERCLA enforcement action in the United States is not limited to historical operations. In a study of mine sites on the National Priorities List (NPL) Housman and Hoffman (1992) found that of the 52 mine sites listed, 12 were active. In their estimation only half of the 52 sites represented 'historical mining practices'. It is important to bear in mind that these 52 sites are estimated to cost the U.S. mining industry 21 billion dollars in clean-up costs. These costs do not include the costs of legal action (Housman and Hoffman, 1992). Indeed, legal costs in the form of class-action lawsuits may dwarf the actual expenditure for site clean-up (Kumamoto and Henley, 1996). Thus, in the United States of America a mine site is never closed from a liability standpoint.

Australia

In Australia, the closed mine site must be both safe and stable before the mine owner is absolved of fiscal liability. Safe refers to "the condition of a closed mine such that the risk of adverse effects to people, livestock, other fauna and the environment in general has been reduced to a level acceptable by all stakeholders (Anon., 1997). Stable refers to "the condition of a closed mine such that the rate of change of reference parameters does not exceed those rates occurring on the site prior to mining or on comparable unmined land in the same locality. Stability is dependent on the geomorphology of the surrounding landfonn and the proposed post-mining land use. The reference parameters can cover fields such as geotechnical slope stability, soil erosion, downstream water quality, or sustainability of revegetation" (Anon., 1996). Thus, a mine owner can achieve liability free closure if the site meets these conditions. The major problem with this approach is that this may take many decades to achieve, during which time the mine owner remains financially responsible for the site.

Canada

Canada, unlike the United States of America and Australia, does not have national legislation mandating long-term liability for contaminated sites. Instead, Provincial legislation determines the requirements for long-term environmental liability. These requirements vary in scope from no responsibility to full responsibility. The Province of Ontario represents the former, while the Province of British Columbia represents the latter.

Provincial legislation in Ontario grants mine owners an 'exit-ticket'. In Ontario, the crown accepts responsibility for the site following the voluntary surrender of the land from a proponent on the conditions specified by the Minister of Mines (Cowan, 1996 and Bourassa, 1996). Significantly, this provision exempts the mine owner from other legislation requiring long-term liability following closure. The only caveat to this 'exit-ticket' is discretionary and that "the site is closed out and that necessary fiinds are placed in a special purpose account for use in rehabilitation of mining lands in general" (Cowan, 1996). Cowan makes the following observation regarding the financial provision:

"It must be stressed, that where long term monitoring, care, and maintenance are required, the determination of costs has a high risk factor and the accrual of public financial liability will be a primary consideration in decision making."

It is therefore possible for mine owners in Ontario to obtain 'walk-away-closure' if financial provision is made.

The long-term liability issue in British Columbia resembles that of the United States. Non-mining legislation establishes that Responsible Parties (RPs) "are absolutely, retroactively, and jointly and severally liable for clean-up costs (Overholt, 1996). By establishing 'Absolute liability", British Columbia has exceeded even the CERCLA (Superfund) program of the United States of America because "Absolute liability precludes 'due diligence' defences (Overholt, 1996). According to British Columbia law, all parties, including current and former owners and operators of closed or abandoned mine sites, are fiscally responsible for environmental degradation following closure "regardless of whether the original mining remediation activity complied with the laws of the day or with permits held at the time of activity"(Overholt, 1996). From this Overholt (1996) concludes that "in British Columbia a closed mine is never a closed mine for liability purposes."

It is clear that, internationally, mine owners remain financially liable for the environmental impacts of mining following closure. Even Ontario requires financial provision to be made for site maintenance, mitigation, and treatment following the assumption of liability by the province. South Africa is a microcosm of the current issues surrounding closure and liability affecting mineral producing nations world-wide.

3.2 *Legislated long-term environmental liability in the republic of South Africa*

Environmental management at South African collieries is performed in accordance with the Environmental management Programme Report (EMPR) process. When a mine applies for permanent closure, Section 5 {Impact Assessment and Section 6 (Environmental Management Programme) of the EMPR assume a dominant role in the decision making process.

Section 5 of the EMPR contains the potential environmental impacts of the project. All impacts are ranked İn terms of significance, which is determined by the context and intensity of effects. The significance is evaluated in terms of present and future site conditions in terms of direct, synergistic, and cumulative effects on the environment. As part of the EMPR process, the proponent is required to assess the residual impacts of mining. Specific effects that must be evaluated include (Anon., 1992):

- The potential for acid mine drainage or poor quality leachates emanating from the mine or residue deposits (Sect. 5.4.1);
- The long-term impacts to groundwater (Sect 5.4.2);
- The long-term stability of rehabilitated ground and residue deposits (Sect. 5.4.3); and
- The long-term impacts arising from river diversions (Sect. 5.4.4).

The list of residual impacts contained in the Aide-Memoire is incomplete (Packee, 1997). There are provisions in the Mineral Act, Water Act, and Atmospheric Pollution Prevention Act that impose long-term liability on collieries.

The Mineral Act stipulates that the owner of lands affected by mining activity is entitled to compensation, under certain circumstances. According to the Minerals Act "such person is entitled to compensation if [he] has suffered or is likely to suffer damage as a result of:"

- Subsidence (§ 42 Ss. 1 paragraph i (aa));
- An obstruction that is placed on the land by the mine (§ 42 Ss. 1 paragraph i (bb)); or,
- Agricultural losses attributable to mining activity (§ 42 Ss. 1 paragraph a. (i and İİ)).

The Minerals Act does not define 'subsidence*. For the purpose herein, subsidence "implies the total phenomenon of surface effects associated with the mining of minerals and not just only the vertical displacement of the surface as is sometimes inferred in the literature" (Singh, 1992). When identifying the residual impacts of mining all of the effects of subsidence must be considered (both surface and subsurface).

The Minerals Act defines 'obstruction' as "any immovable property established on land for mining operations or operations in connection therewith by the person entitled to mine on such land, and includes any dam, or dump of slimes, rock, or any other residue produced in the course of mining operations on such land" (§ 42 Ss. 1 paragraph a. (i and ii)). Compensatory requirements for waste deposits are, as a rule of thumb, best dealt with prior to construction. Following closure, compensation may be required if there is a geotechnical failure, which needs to be considered when negotiating financial remuneration with the landowner.

Compensatory damages for agricultural losses fall into two categories. The Minerals Act requires compensation when either

- "the use or intended use of such land, or any portion thereof, by such person for the mining of minerals or purposes in connection therewith, prevents, hinders, or is likely to prevent or hinder the proper use of such land or such portion for farming purposes" (§ 42 Ss. 1 paragraph a. (i)) or
- "any portion such land which is not being used or not likely to be used by such person for mining purposes or purposes in connection therewith, is or is likely to become an uneconomic farming unit" (§ 42 Ss. 1 paragraph a. (ii))

Settlements for mining induced damage are negotiated in accordance with Expropriation Act of 1975 as if "an expropriation of property or the taking of a right has taken place." Compensatory awards must take into account both rehabilitation performed, or to be performed, and compensation paid to the owner previously. Once a dispensation has been made for a particular effect, the mining company is absolved of responsibility for that particular impact. The limits of financial responsibility transcend land title transfers and exclude reservations existing prior to mining activity.

Additional liability is imposed on mine owners by the Water Act of 1956. Prior to closure, the mine owner must take steps to:

- prevent water pollution;
- prevent further degradation of water quality, if pollution has already occurred;
- bear the financial responsibility for clean-up costs on adjacent land;
- bear the financial responsibility for clean-up costs at the site; and,
- bear the financial costs of ' emergency clean-up actions' undertaken by the Department of Water Affairs and Forestry.

Similar to other liability assessing legislation (Overholt, 1996 and Cowan, 1997), the Water Act defines a responsible party (RP) for site clean-up costs. The RP is "any person who wilfully or negligently does any act which could pollute public or private water, including underground water, or sea water, in such a way as to render it less fit" (Water Act. 'Less fit' implies that the impacted waters do not meet the South African Water Quality Standards, as defined by regulation, upon discharge to a public or privet water body. This liability is strict, not absolute, and may not be transferred to a party other than the responsible party.

The Atmospheric Pollution Prevention Act (APPA) has special implications at collieries. As the name of the act implies, the purpose of the APPA is to prevent air pollution. Like the Water Act, this legislation establishes liability for degradation of atmospheric quality. The RP is defined as:

"Any person who in a dust control area -

> caries on any industrial process the operation of which in the opinion of the chief officer causes or is liable to cause a nuisance to persons residing or present in the vicinity on account f dust originating from such process becoming dispersed in the atmosphere; or
>
> has at any time or from time to time, whether before or after the commencement of this act, deposited or caused to be deposited on any land, a quantity of matter which exceeds, or two or more quantities which together exceed twenty thousand cubic meters in volume...shall take the prescribed steps or where no steps have been prescribed adopt the best practicable means for preventing such dust from becoming so dispersed or causing such nuisance"

The liability for dust control and mitigation is strict, and resides with the mine owner or operator following closure. If the mine owner is insolvent or cannot be determined, moneys from dust levy accounts are used for remedial action. Typical dust sources at closed mines include waste dumps, tailings impoundments, and denuded areas.

Smoke and combustion by-products, originating from burning waste deposits or residual coals, fall under the provisions of the APPA. If smoke or combustion by-products become a nuisance, "the local authority shall cause to be served on the person responsible for such nuisance a notice calling upon him to abate the nuisance within a period determined by the local authority...and to take all such steps as may be necessary to prevent a recurrence of the nuisance"([ref!)]. The APPA does not make a distinction as to whom the responsible party is or as to the transferability of the liability.

South African legislation is non-committal regarding the mine owner's long-term environmental liability. Like other mining nations, the paradox between limited liability and absolute liability is contained in national legislation. The Minerals Act indicates that the mine owner's liability ends with receipt of a closure certificate. This is similar to the 'exit-ticket' of the Province of Ontario. However, the South African Closure Certificate does not exempt the mine owner from the provisions of other legislation. The Water Act, specifically Section 22, establishes strict and absolute liability for water pollution prevention and mitigation. In respect to mining's liability for water resource degradation, South Africa more closely resembles the United States CERCLA program and the contaminated sites legislation of the Province of British Columbia. The fact that no large colliery has received a Certificate of Closure from the Department of Mineral and Energy Affairs (DMEA) indicates that conflicting liability provisions between the three principal acts have created an impasse with respect to colliery closure in South Africa.

4 RESOLVING LIABILITY CONFLICTS BETWEEN SOUTH AFRICAN LEGISLATION

At first blush, the problems being experienced with divergent liability appear irreconcilable. The divergence is caused by the degree of liability imposed by individual legislation. The most stringent financial responsibility following closure is 'strict, absolute, and retroactive' liability. This is where the mine owner held solely responsible for impacts attributable to mining, including those attributable to changing regulatory standards. This is a worst case assumption. However, the concepts of 'safe and stable' used İn Australia and the 'exit-ticket' of Ontario, provide a conceptual approach to resolving the issue of liability following closure.

4.1 *Complying with absolute liability*

Absolute liability implies that the financial responsibility for clean up remains the owner or operator that caused environmental degradation. In effect, Üris is strict interpretation of the "polluter-pays-principle'. In practical terms, absolute liability is a mechanism for insuring that mines are not abandoned following closure, which İs the objective of the Minerals Act (Oberholzer, 1995).

The exit ticket strategy of the Province of Ontario indicates that 'walk-away-closure' may be obtained at the discretion of regulatory agency and providing mat sufficient funds exist to cover the environmental risks associated with the site. As stated by the Director of Mine Rehabilitation for the Ontario Ministry of Northern Development and Mines (White, 1996) "the accrual of public financial liability will be a primary consideration İs decision making." Thus, if the mine owner can demonstrate that no financial liability resides with the government following the transfer of liability, the mine will be granted 'walk-away-closure'. This would satisfy the of the liability provisions contained in all South African legislation, past and future, including the Water Act. The advantage is that the mine owner has addressed financial responsibility from the most stringent possible perspective. This however does not resolve the issue of retroactive liability following closure.

4.2 *Complying with retroactive liability*

Retroactive liability implies that the mine owner is responsible for all die environmental impacts of mining, irrespective of standards in place at the time of closure. In effect, this prevents changing environmental standards from affecting the long-term financial returns of mining companies. Currently, The South African government holds mining companies responsible for impacts, attributable to mineral extraction or associated activities, which violate current environmental performance standards. However, precedents exist which hold mine owners financially responsible for activities mat were legal at the time they were performed (Overholt, 1996 and Cowan, 1997). As was the case with strict liability, this assumption represents worst case legislation from an industrial perspective.

The use of safe and stable' as a closure objective in Australia is a statement of retroactive liability. By definition (Anon., 1997) the site conditions must be similar to those existing prior to mining or existing at adjacent immined sites. In simple terms, stable implies that the site is in equilibrium with the environment. A stable condition may take years or decades to attain. In the interim, the mine owner remains fiscally responsible for the site.

4.3 *Definition of impact for absolute and retroactive liability*

Strict and retroactive liability forces the mine owner to consider all potential environmental impacts of the project. Often, impact is linked to regulatory compliance, which leads to complaints of shifting 'goal posts' for closure (Williams, 1998; Swart and Pulles, 1997). Using 'stable' as a closure objective simplifies impact identification. An impact is simply any change from either baseline conditions or surrounding sites resulting from the project. It is important to note that these changes can be positive and negative. The advantage to non-compliance related impacts identification is that changing regulatory performance standards will not affect environmental management before or after closure. Additionally, the use of baseline or reference site data provides the mine owner with discreet and quantifiable environmental standards that do not vary significantly over time.

5 FINANCIAL CONSTRAINTS AT CLOSURE

In general, mine cash flow peaks during production and steadily declines as the operation approaches the 'end of mine life' and closure. The mine owner is in the precarious position of balancing decommissioning and final rehabilitation costs and the long-term liabilities imposed by regulation with declining revenues. Internationally, governments seek to ensure that decommissioning and closure will occur by requiring financial provision be made for diese tasks. South Africa takes this one step further by requiring that all assets remain in place until a Closure Certificate is granted. For South African collieries closure is not merely a desirable situation but is an economic necessity.

5.1 *Cashflow of mining operations from exploration through closure*

The activities occurring at a mine change dramatically over time. The conventional mine life cycle is depicted in figure 3.1. Mining operations begin with Discovery. During the first stage of discovery, the mineralised area or district is identified (Figure 1 'A'). The second stage of discovery, geologists and engineers investigate and map the mineral district (Figure 1 'B'). If subsequent investigations indicate the existence of a potentially commercially viable deposit, the mine progresses into Exploration (Figure 1 'C').

Exploration "includes all activities involved in the discovery and evaluation of a mineral deposit, establishing the size, grade, initial flow sheet, and annual output of the new extractive operation. Once exploration is completed, site activities focus on operations planning (Figure 1 'D').

The Pre-operational stage is die interval between Exploration and Production and includes feasibility studies, mine financing, and construction. Once construction İs complete, the mine begins Operations. Hartman and Lacey (1992) subdivide mine operations based on changes in gross production rate. Following the 'pre-production' stage, mine gross production rates increase rapidly. The rapid growth in production following construction is the 'Expanding Production' stage of operations (Figure 1 'E'). During this initial growth the mine is focused on debt repayment and vertical and horizontal growth The Expanding Production stage ends when production rates stabilise near the production target set in the feasibility study. At this point, production İs considered Mature (Figure 1 'F'). During Mature Production, the mine focuses on local exploration and cost reduction. The local exploration is required to determine the limits of the mineral deposit. Cost reductions in terms of innovation and efficiency related work allow the mine to enhance its cash flow. At the end of Mature Production, the gross mine production starts to fall as the mineral deposit is depleted

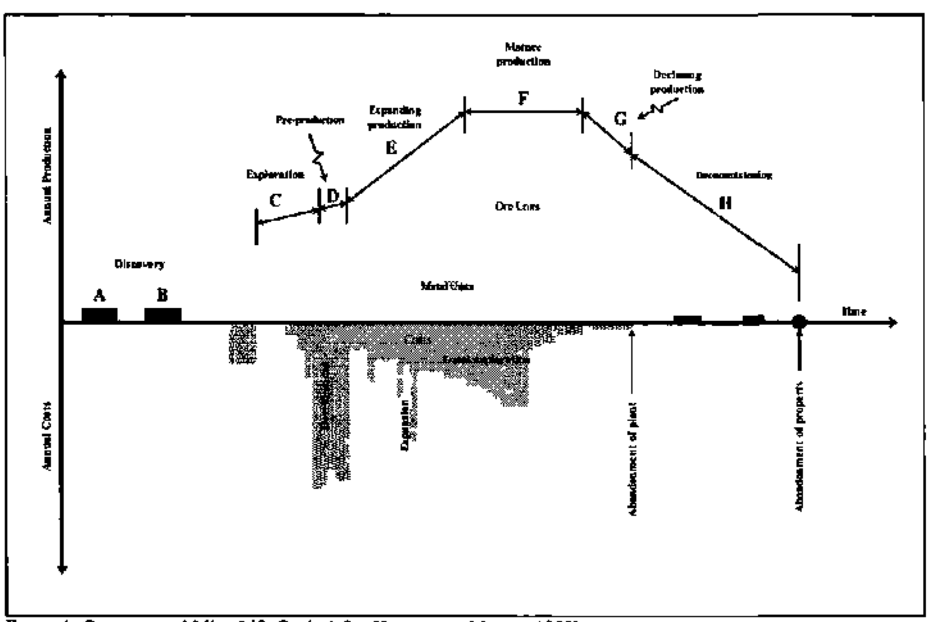

Figure 1 Conventional Mine Life Cycle (after Hartman and Lacey, 1992)

The mine enters a period of Declining Production as the end of the mining cycle approaches (Figure 1 G '). Hartman and Lacey refer to the period following operations as Abandonment (Figure 1 'H'). The focus of mine efforts during abandonment is equipment salvage and site restoration. Hartman and Lacey (1992) use the common definition of mine closure encountered in the United States. According to the mine life cycle presented the mining cycle ends with abandonment (closure in the South African context) of the property.

In the South African context, die final stages of mine life are decommissioning and closure. The EMPR stipulates evaluation and contingency planning for impacts following site abandonment, which is consistent with absolute and retroactive liability. Effectively, the mining cycle now extends past the point of closure and die final phase is Post Closure (Figure 2) Although Figure 2 reflects mineral policy in South Africa, it accurately portrays conditions m countries mat adhere to die polluter-pays-principle.

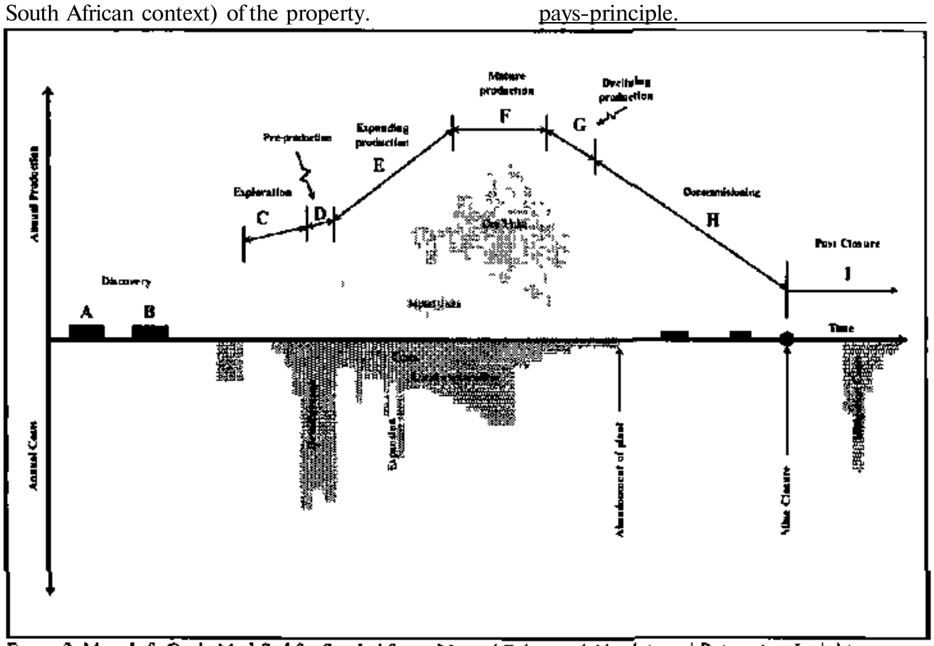

Figure 2 Mine Life Cycle Modified for South African Mineral Policy and Absolute and Retroactive Liability.

5.2 *Financial implication of closure in South Africa*

Short-to-medium term effects of environmental compliance costs on corporate profitability effectively reduce the rate of return on most mining projects. "In recent years, the national and, to a lesser but growing extent, the international operating environment of mining properties has been impacted significantly by environmental and other regulatory requirements. These constraints have invariably increased operating and capital cost requirements for the industry and have reduced or delayed production of mineral commodities. The operating environment of mining operations is also affected by direct economic variables such as royalties and taxes...All these costs, whether direct or indirect, impact profit margins, ore reserves, mineral conservation, and ultimately project viability" (Gentry and Jarnigan, 1993). Additional costs incurred following operations and decommissioning will negatively affect corporate profitability. Potential costs following decommissioning include asset depreciation, site administration, and monitoring. These costs eventually translate themselves into decreased economic activity in the mining sector (Parrish, 1991).

5.3 *Financial provisions for decommissioning and closure*

In today's global economy, transferring environmental liability to the public sector is perceived as government subsidies that confer competitive advantage in the marketplace. Consider two mines, Mine 'A' has absolute and retroactive environmental responsibility and the other (Mine 'B') has no responsibility for the site following closure. The competitive advantage conferred to Mme 'B' İs immeasurable. Mine 'B' not only saves the costs of environmental compliance during operations, but also can apply the full net profit of the mine to purchase and construct 'new* operations. Mine 'A' commodities not only cost more, due to operational expenditure for environmental compliance, but the mining corporation must retain part of its net profit in reserve for unanticipated costs arising at closed mines. Thus, environmental liability places Mine 'A' at a tremendous disadvantage in the marketplace. It is not a mistake that industrialised nations of the Northern Hemisphere have demanded, through treaty, that all nations adopt environmental legislation for industrial activities.

The intent of the polluter-pays-principle[1] is to insure that governments do not aid industry with environmental compliance. Thus, the primary interest of the government at closure is to ensure that financial liability does not accrue to the public sector. The fundamental problem İs that, especially in the case of mining, liquid assets must be used to finance the next project. Thus, closure forces two parties with opposing self-interests to the negotiating table. The subject of these negotiations is the amount of the closure provision: Industry will seek to minimise the amount, while government will seek to maximise it. Currently, no process, procedure, or policy exists to determine what the actual amount will be. This has created an impasse at closure that must be resolved if mines are to achieve 'walk-away' closure.

6 CONCLUSION

World-wide, mine closure is accepted as being the last step in the active management of mines. Legislation mandating long-term liability forces mine owners to consider the post-closure environmental impacts of mining. These stipulations range from none (Ontario) to absolute and retroactive liability (United States of America and British Columbia). With respect to long-term liability, South African legislation displays the same variability encountered at the international level. This has created considerable confusion as to post-closure site responsibility of mine owners. The situation, as existing, is not conducive to closing mines. Indeed, the current situation threatens the long-term survival of mining companies by holding assets and profits hostage until closure is attained. The fundamental issue is protecting the mine owner from the long-term financial implications of post-closure care without transferring them to the public sector.

It is suggested that the mine owner assume absolute and retroactive liability until the site is 'safe and stable'. This represents the worst possible case from an industry perspective. However, the advantage is that impacts identification, environmental management, and site liability becomes independent of legislated or regulated environmental performance standards. Further, the general performance standard following closure (safe and stable) is site specific and based upon quantifiable environmental attributes. To insure that financial responsibility does not default to the public sector, mines will have to provide financial surety that covers the costs of post-closure care, maintenance, and mitigation of the site. On the basis of these assumptions, all that remains is determining site costs following closure.

REFERENCES

Anon. 1992. *Aide-Memoire for the Preparation of Environmental Management Programme Reports for Prospecting and Mining.* Department of Mineral and Energy Affairs, Pretoria, Republic of South Africa.

Anon., 1996. *The Integrated Environmental Management Procedure.* Department of Environmental Affairs, Pretoria, Republic of South Africa.

Anon. 1997. *Discussion Paper Mine Closure: Towards a* Strategy *for the Australian Minerals Industry.* Minerals Council of Australia, Dickson, Australia.

Anon. 1999. *Alaska Surface Coal Mining Program.* State of Alaska Department of Natural Resources, Division of Mining and Water Management, Anchorage, Alaska.

Bourassa, M.J. 1996. Article I - Ontario's Mine Closure Plans: 1996 Revisions to Part VII of the Mining Act. *Proceedings of the Insight Conference: Mine Closure Plans and Liability Issues.* Insight Press, Toronto, Ontario, pp 13-58.

Cowan, J. 1997. The Environmental Response, Compensation and Liability Act. *Mining Environmental Handbook* Ed J.J. Marcus, Imperial College Press, San Mateo, California, pp 73-78.

Cowan, W.R. 1996. Article IV - New Directions for Mine Rehabilitation Legislation in Ontario. *Proceedings of the Insight Conference: Mine Closure Plans and Liability Issues.* Insight Press, Toronto, Ontario, pp 121-140.

Daigneault, R. 1996. Article II - Mine Closure Plans in Quebec: The New Requirements, One Year Later. *Proceedings of the Insight Conference: Mine Closure Plans and Liability Issues.* Insight Press, Toronto, Ontario, pp 59-80.

Gentry, D.W. and Jamigan, L., 1993. Environmental Aspects an Increasing Part of International Mining Projects. *Mining Engineering,* August, pp 1009-1011.

Greef, R.I. 1994 The Environmental Management Programme Route to Integrated Environmental Management: A South African Mining Industry Perspective. *Proceedings of the International Land Reclamation and Acid Mine Drainage Conference and Third International Conference on the Abatement of Acidic Drainage.* Ed NTIS, United States Bureau of Mines Special Publication SP06A-94, Pittsburgh, Pennsylvania, pp 269-274.

Hartman, and Lacey, 1992. Mineral Prospecting and Exploration, Section A., *SME Mining Engineering Handbook,* 2nd Edition, pp 205-209.

Housman, *VB.* and Hoffman, S.D. 1992. Mining Sites on Superfund's National Priorities List - Past and Current Mining Practices. *Risk Assessment/Management Issues in the Environmental planning of Mines.* Ed D. Van Zyl, M. Koval, and T.M. Li, Society for Mining, Metallurgy, and Exploration, Inc., Littleton, Colorado, pp 55-62.

Kumamoto, H. and Henley, E.J. 1996. *Probabilistic Risk Assessment and Management for Engineers and Scientists.* IEEE Press, New York, New York.

Licari, 1997. Closure and Post-Closure Requirements. *Mining Environmental Handbook.* Ed J.J. Marcus, Imperial College Press, San Mateo, California, pp 388.

Murray, 1997. Economic Impact of Current Environmental Regulations on Mining. *Mining Environmental Handbook.* Ed JJ. Marcus, Imperial College Press, San Mateo. California, pp 630-641.

Oberholzer, J.W.O. 1995. Problems Experienced in the Closing of South African Collieries. *Proceedings of the A 12th Annual Meeting of the American Society for Surface Mining and Reclamation.* Ed O.E. Schuman and G.A. Vance, American Society for Surface Mining and Reclamation, Gillette, Wyoming, pp 659-666.

Overholt, D.H. 1996. Article III - Mine Closure Plans in British Columbia. *Proceedings of the Insight Conference: Mine Closure Plans and Liability Issues.* Insight Press, Toronto, Ontario, pp 81 -120.

Packee, E.C. and Oberholzer, J.W. 1997. The Financial Valuation of Closed Collieries. *Mine Closure Colloquium.* The South African Institute of Mining and Metallurgy, Randburg, Republic of South Africa, pp 69-82

Parrish, C, 1991. Environmental Compliance Costs in Mining. *Environmental Management for the I990's.* Ed D.J Lootens, W.M. Greenslade, and J.M. Barker. Society for Mining, Metallurgy, and Exploration, Inc., Littleton. Colorado, pp 69-78.

Singh, M.M. 1992. Mine Subsidence. *SME Mining Engineering Handbook: Volume I.* Ed H.L. Hartman, Society for Mining, Metallurgy, and Exploration, Inc., Littleton. Colorado, pp 938-971.

Swart, S.J. and Pulles, W., 1997. Environmental Risk Assessment as the Basis for Mine Closure al 1SCOR Mining. *Mine Closure Colloquium.* The South African Institue of Mining and Metallurgy, Randburg, Republic of South Africa, pp 53-68.

Taggan and Kieth, 1997. Aesthetics. *Mining Environmental Handbook.* Ed J.J. Marcus, imperial College Press, San Mateo, California, pp 174-178.

White. A.M.S. 1996. Closure: An Open and Shut Case? Is It that Simple? *Proceedings of the Insight Conference; Mine Closure Plans and Liability Issues.* Insight Press, Toronto, Ontario, pp 141-153.

Williams, 1998. *Mine Life Cycle Environmental Management; Short Course Notes.* Short Course offered at 1998 Annual Meeting, Society for Mining, Metallurgy, and Exploration, Inc., Orlando, Florida.

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Methodology for Estimating the Costs of Treatment of Mine Drainage

V. Raj aram
Tetra Tech EM Inc., Chicago, IL, USA

A.Glazer & G.Coghlan
Tetra Tech EM Inc., Washington, DC. USA

ABSTRACT: Tetra Tech developed worksheets for the U.S. Department of the Interior, Office of Surface Mining (OSM) to allow a consistent, accurate, and rapid method of estimating the costs of long-term treatment of mine drainage at coal mines, in accordance with the Surface Mining Control and Reclamation Act (SMCRA) of 1977. This paper describes the rationale for the worksheets and how they can be used to calculate costs for site-specific conditions. Decision trees for selection of alternative treatments for acidic or alkaline mine drainage are presented.

1 INTRODUCTION

The Surface Mining Control and Reclamation Act of 1977 (SMCRA) established bonding requirements for operators of coal mines. Regulations under SMCRA require that operators of coal mines prepare site-specific estimates of the costs of reclaiming areas affected by mining operations and managing any pollutants mat may emanate from them. The cost estimates are reviewed by me U.S. Department of the Interior, Office of Surface Mining (OSM) or authorized state regulatory authorities to determine the amount of bonding that is necessary for reclamation activities at each mine. OSM has determined that the estimated costs of reclamation for a number of coal mines have been significantly lower than the actual costs of completing reclamation activities at those facilities. To a large extent, such discrepancies have been the result of failures in the development of reasonable estimates for the long-term treatment of mine drainage-

Acid mine drainage (AMD) forms when sulfide minerals in rocks are exposed to oxidizing conditions. AMD is found İn coal mining and metal mining areas and also in highway embankments where sulfides in geologic materials are exposed to water and oxygen. The predominant acid-producing minerals are pyrite and marcasite. Significant work in methods of predicting, controlling, and treating AMD has been done by Skousen (1996, 1998) at West Virginia University and by the Pennsylvania Department of Environmental Resources (1998). Earle and Callaghan (1998) have described the effects of AMD on aquatic life, on potable and industrial water supplies, and on metal and concrete structures. In an effort to prevent the adverse effects of AMD in the future and hold companies that produce AMD responsible for their actions, OSM contracted with Tetra Tech EM Inc. (Tetra Tech) to develop a methodology for estimating the long-term costs of treating mine drainage. Costs determined through the use of the methodology will serve as the basis for establishing the bonding requirements for coal mine operators.

2 TECHNOLOGIES FOR PREVENTION AND REMEDIATION OF AMD

Coal mine operators are required to meet the discharge permit requirement established under the National Pollution Discharge Elimination System (NPDES). NPDES permits for coal mines generally require the monitoring of pH, total suspended solids, and concentrations of iron and manganese. Depending on the drainage quality of the particular site, monitoring of other parameters may be required. The technologies for preventing and remediating AMD include the following:

- Source control by addition of alkaline substances, capping of acid-producing materials, hydrologie controls, or grouting (for underground mines)
- Active treatment by neutralization processes and aeration basins
- Passive treatment by alkalinity-producing diversion wells, anoxic limestone drains, or aerobic or anaerobic wetlands

The life-cycle costs of the technologies vary considerably, with active treatment methods the most expensive and passive treatment systems the least. The selection of a technology depends on several site-specific conditions, including the quality and quantity of mine drainage to be treated and the discharge requirements to be satisfied. There are two types of mine drainage, net acidic and net alkaline. Net acidic mine drainage occurs under conditions under which the total acidity of the drainage exceeds its total alkalinity, while net alkaline drainage occurs under conditions under which total acidity is less than total alkalinity. Figures 1 and 2 provide decision trees to assist the user in identifying alternative treatment technologies for net acidic mine drainage and net alkaline mine drainage, respectively.

3 COST-ESTIMATING METHODOLOGY

The methodology described in this paper can be used to estimate the costs of long-term treatment of mine drainage for both new and existing mining operations.

To generate cost estimates for the treatment of mine drainage, the methodology uses third-party costs and assumes that the materials and equipment required to conduct each activity will be brought to the site. Although the operator may have equipment and materials available to conduct such activities while the facility İs operational, there is no guarantee that the same equipment or materials will be available to a regulatory autfiority once a facility has been abandoned. Therefore, use of third-party costs is essential to the development of a cost estimate that reflects a "reasonable worst-case scenario" and to ensure that financial assurance mechanisms can be counted upon to provide regulatory authorities with sufficient funds to conduct the necessary mine drainage treatment activities.

The methodology provides a variety of worksheets that can be used to estimate the costs of specific activities that are known to be conducted in treating mine drainage at coal mines. It is unlikely, however, that all of the worksheets provided in the methodology will be needed to estimate the costs for treating mine drainage for any particular site. In applying the methodology, die user must select from among the available worksheets only those that pertain to activities mat will be conducted to address mine drainage at the particular site of concern. The decision trees presented in figures 1 and 2 will assist the user in making that selection. Once the worksheets for each specific activity have been completed, the costs estimated for those activities are combined on unit summary worksheets to derive a cost estimate for each unit, then combined on a site summary worksheet to derive a comprehensive cost estimate for the site.

Because the types of activities that may be necessary to address mine drainage may vary significantly from one site to the next, and because it might be necessary to conduct unusual or uncommon activities at some sites to address the specific circumstances at those sites, the methodology may not include worksheets that address all the activities that may be necessary at any given site. In such cases, cost estimates for unusual or uncommon activities should be developed through the use of alternative approaches.

The methodology as currently designed is a general approach to estimating costs. The issue of treatment cost versus time and reduction of acidity loading over time can be addressed by using different time frames, flow rates, and acidity to run the model, according to site-specific conditions.

Worksheets are the primary tools the methodology provides to help estimate the costs of treating the mine drainage. The methodology presents seven categories of cost worksheets, each of which corresponds to one of various activities that might be undertaken to address discharges of mine drainage at coal mines. The categories of worksheets are: 1) source control, 2) active treatment, 3) passive treatment, 4) general treatment and polishing units, 5) discharge memods, 6) system operations, and 7) support activities. A site summary worksheet also is provided to sum all the costs of treatment of mine drainage that are associated with a particular site.

Many activities performed in treating mine drainage involve basic field construction work. Therefore, in the worksheets, typical construction costs Üıat most closely resemble field construction activities are used. Although several sources of information provide estimates of typical construction costs, hourly rates for labor and equipment set forth in *Mean's Cost Guides* are used frequently in the worksheets. *Mean's Cost Guides* are recognized industry standards in the United States for cost estimating. Because the guides are updated annually, the cost components of the worksheets can be updated readily, as well. When certain costs, such as those for laboratory analysis of water or soil samples, could not be found in *Mean's,* representative costs provided by a number of vendors were obtained and averaged. Costs of activities not found in *Mean's* can be established by obtaining current quotes from vendors or by interpolation of costs of similar activities obtained from the mining industries.

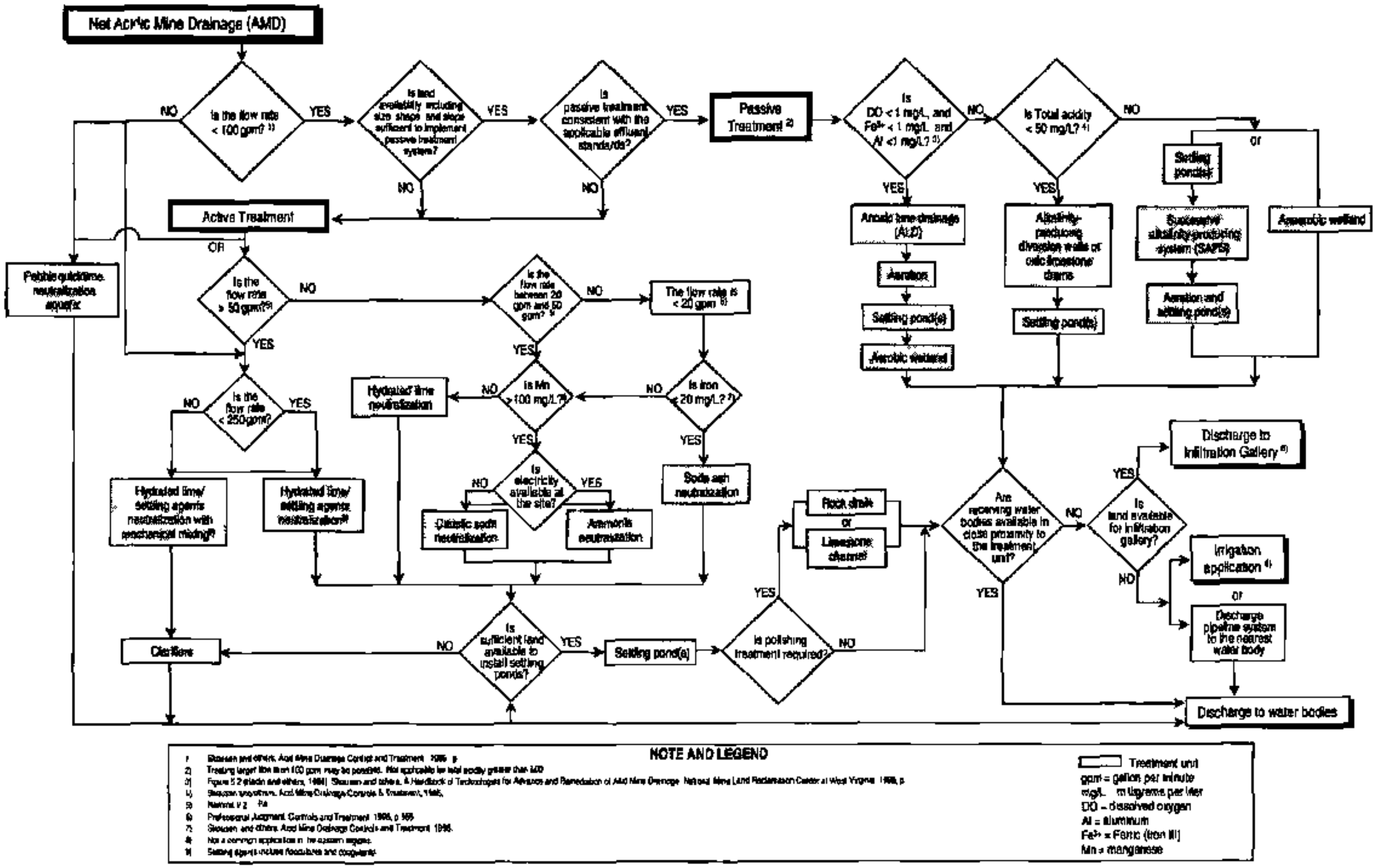

Figure 1 Decision tree for the selection of alternatives for the treatment of net acidic mine drainage

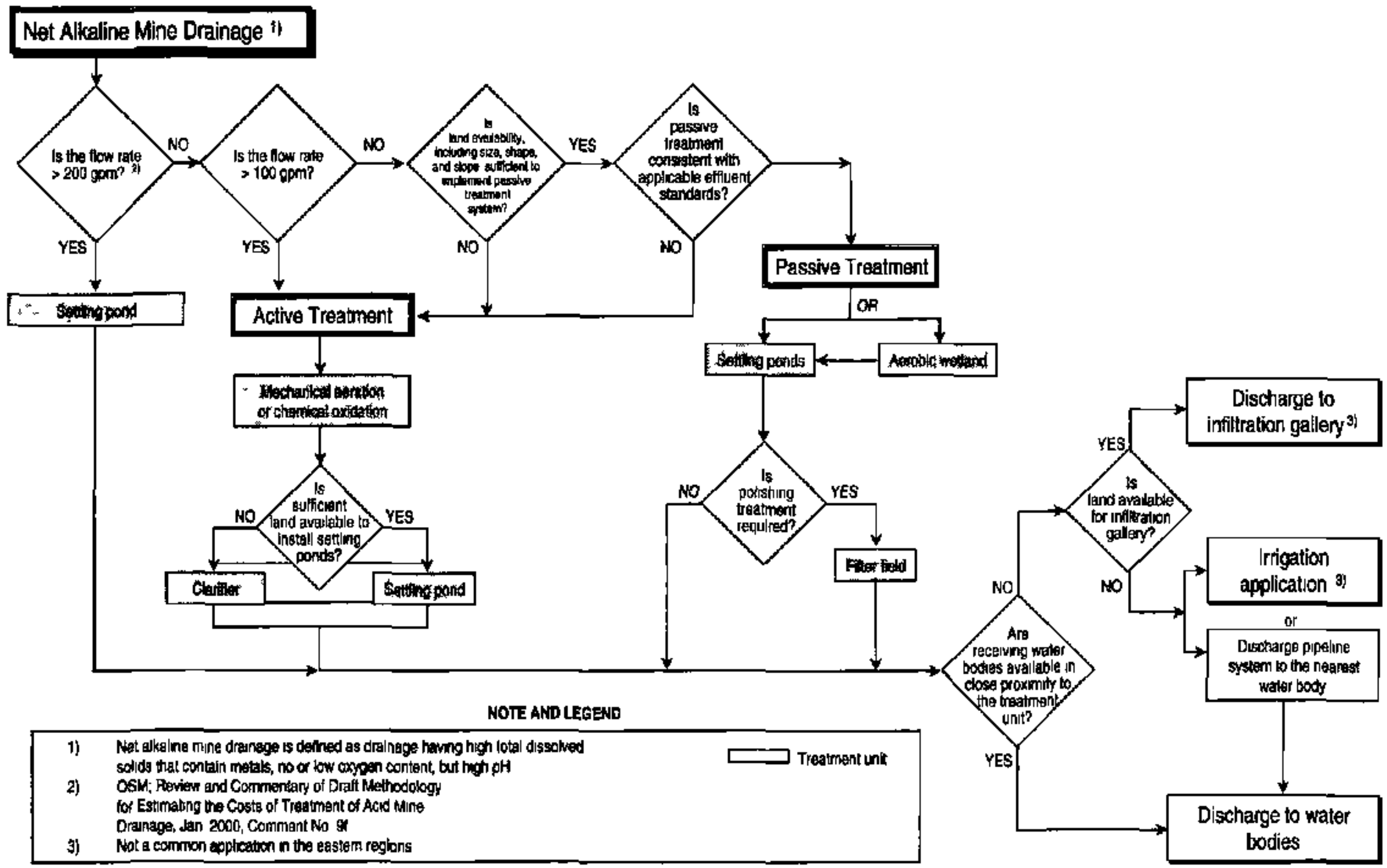

Figure 2. Decision tree for the selection of alternatives for the treatment of net alkaline mine drainage based on industry practices

In the methodology, the source control, active treatment, and passive treatment unit summary worksheets apply a factor of 16 percent to the subtotal of all capital costs to account for the costs of management and engineering design. That percentage is based on factors derived from *Mean's.* Further, in accordance with standard engineering practices, the source control, active treatment, and passive treatment unit summary worksheets apply an additional factor of 20 percent to all capital and annual operating costs to account for contingencies and unforeseen expenses that could occur during the treatment process.

The total calculated costs of source control, treatment, and system operations are adjusted to net present value to determine the long-term costs of treatment in current dollars. This method is used widely to evaluate the value of long-term investments and is useful in this analysis for determining the amount of bonding that is necessary for reclamation activities at a particular site. The three main components used m completing the net present value adjustment are:

- The number of years during which the system is expected to operate
- The annual inflation rate
- The estimated annual discount rate, which represents the cost to the owner of the facility of borrowing money

The use of different figures for years of treatment, die inflation rate, or the discount rate could have a significant effect on the cost estimate generated for an individual facility. The cost estimates are highly sensitive to small variations İn those parameters, and the user should select values carefully for the calculation of net present value to obtain the most realistic cost estimate.

4 USING THE WORKSHEETS TO ESTIMATE COSTS

The methodology is designed to offer a flexible and rapid means of generating reasonable and accurate estimates of the costs of treating mine drainage. The methodology prescribes the following four basic steps in using the worksheets to develop cost estimates.

1, Using information in the reclamation plan or other information available about the site, the user must determine the specific treatment processes that are to be conducted to address mine drainage at the site and the specific source control techniques (if any) that will be implemented. The user must also identify any conditions at the site (for example, difficult access to the site) that might require the conduct of additional activities or require capital expenditures that will add to the expense of implementing the selected treatment processes. In addition, the user must identify related data that are lacking and must be generated or assumed if a reasonable cost estimate is to be developed.

2. The user must identify and assemble all the worksheets needed to calculate the cost estimate. Table 1 identifies the worksheets for specific activities that are addressed in the methodology and that might be appropriate for the treatment of mine drainage at surface mines, underground mines, and coal refuse facilities, respectively. The user should review the applicable worksheets to become familiar with the data inputs necessary to use them and the assumptions upon which the cost data incorporated into the worksheets are based.

3. Using information in the reclamation plan or other information available about the site, the user must obtain the data that are required to use the worksheets and enter those data. The user should review all cost data that are incorporated into the worksheets to ensure that those data accurately reflect me potential tasks to be performed at the site. If necessary, the user can adjust the costs incorporated into the worksheets or replace them with other cost data that are more accurate for the site. Once all appropriate data have been entered, the user can estimate the costs of each activity by applying the method prescribed İn the worksheets.

4. The user must transfer the estimated cost of each activity to the source control, active treatment, or passive treatment summary worksheet, as appropriate, to derive cost estimates for each unit; apply allowances to those estimates to account for engineering expenses and contingencies; and adjust those estimates to net present value. The user should review the default factors applied on the source control, active treatment, or passive treatment summary worksheet to ensure that those factors are appropriate. If necessary, the user can adjust the default factors or replace them with more appropriate factors. Finally, the user must transfer the costs for each unit to the site summary worksheet to derive a comprehensive cost estimate for the site. Figure 3 provides example worksheets for active treatment; figure 4 provides example worksheets for passive treatment. Details of other worksheets are available in *Methodology for Estimating the Costs of Treatment of Mine Drainage,* prepared by Teta Tech.

Table 1. Worksheets applicable to the treatment of mine drainage at surface mines, underground mines, and coal refuse piles.

Worksheet	Code	Surface Mines			Underground Mines			Coal Refuse Piles		
		Source Control	Active Treatment	Passive Treatment	Source Control	Active Treatment	Passive Treatment	Source Control	Active Treatment	Passive Treatment
Capping of Acid-Producing Material	SC-2	•						•		
Regrading and Backfilling	SC-3	•						•		
Grouting and Mine Seals	SC-4				•					
Stormwater and Runoff	SC-5	•			•			•		
Alkaline Addition for Spoils	SC-6A/B	•			•			•		
Soda Ash Neutralization	AT-2A/B		•			•			•	
Caustic Soda Neutralization	AT-3A/B		•			•			•	
Hydrated Lime or Pebble Quicklime Neutralization	AT-4A/B		•			•			•	
Ammonia Neutralization	AT-5A/B		•			•			•	
Aeration Basins	AT-6A/B		•			•			•	
Pebble Quicklime Neutralization – Aquafix System	AT-7		•			•			•	
Alkalinity-Producing Diversion Wells	PT-2A/B			•			•			•
Anoxic Limestone Drains (ALD)	PT-3A-C			•			•			•
Successive Alkalinity-Producing Systems (SAPS)	PT-4A/B			•			•			•
Aerobic and Anaerobic Wetlands	PT-5A/B			•			•			•
Ponds	GTU-1A/B		•	•		•	•		•	•
Clarifiers	GTU-2A/B		•			•			•	
Rock Drains	GTU-3A/B		•	•		•	•		•	•
Filter Fields	GTU-4A/B		•	•		•	•		•	•
Open Limestone Channels (OLC)	GTU-5A-C		•	•		•	•		•	•
Infiltration Galleries	DM-1A/B		•	•		•	•		•	•
Irrigation Applications	DM-2A/B		•	•		•	•		•	•
Pipe Systems	DM-3		•	•		•	•		•	•
Chemical Consumption	OP-1		•	•		•	•		•	•
System Maintenance and Replacement	OP-2	•	•	•	•	•	•	•	•	•
Electricity	OP-3		•	•		•	•		•	•
Sludge Removal	OP-4		•	•		•	•		•	•
Sampling and Analysis	OP-5	•	•	•	•	•	•	•	•	•
Land Access	SW-1	•	•	•	•	•	•	•	•	•
Monitoring Wells	SW-2	•	•	•	•	•	•	•	•	•
Site Security	SW-3	•	•	•	•	•	•	•	•	•
Access Roads	SW-4		•	•		•	•		•	•

5 CONCLUSIONS

The methodology developed by Tetra Tech for OSM is user-friendly and allows an accurate determination of the amount of funds that should be provided to ameliorate AMD at coal mine sites. Recognizing that current mine operators may not be available to conduct treatment activities, the methodology reflects costs for an independent third-party to perform these activities. The methodology can be used for both new and exciting mining operations and for active and passive treatment processes. Because many treatment activities involve basic field construction work, *Means Cost Guides,* a recognized standard İn the construction industry, was selected as the basis for information for the worksheets. The worksheets also provide the flexibility for the user to modify the information for site-specific conditions.

REFERENCES

Earle, J. and T, Callaghan. 1998. *Impacts of Mine Drainage on Aquatic Life, Water Uses, and Man-Made Structures, Coal Mine Drainage Prediction and Pollution Prevention in Pennsylvania.* Pages 4-1 through 4-16. Pennsylvania Department of Environmental Protection. Harrisburg, Pennsylvania. October.

Pennsylvania Department of Environmental Resources. 1998. *Coal Remitting - Best Professional Judgment Analysis.* Commonwealth of Pennsylvania. Harrisburg, Pennsylvania. November.

Skousen, J.G., and P.F. Ziemkiewicz *(editors).* 1996. *Acid Mine Drainage Control and Treatment,* Second Edition. West Virginia University and the National Mine Land Reclamation Center. Morgantown, West Virginia.

Skousen, J.G. and others (editors). 1998. *A Handbook of Technologies for Avoidance and Remediation of Acid Mine Drainage.* National Mine Land Reclamation Center. West Virginia University. Morgnatown, West Virginia. June.

U.S. Department of the Interior, Office of Surface Mining (OSM). 2000. *Methodology for Estimating the Costs of Treatment of Mine Drainage.* Final Report prepared by Tetra Tech EM Inc. under Contract No. 143868-CT99-12063. August.

SODA ASH (SODIUM CARBONATE) NEUTRALIZATION

1. [illegible]		
Assume that soda ash briquettes will be used		
a Design flow rate	0.0 gpm	
b Total acidity of mine drainage acidity	0.0 mg/L as $CaCO_3$	
c Molar acid equivalents loading (multiply line 1a times 3 785 L/gal, times line 1b, times 0 01 mole $CaCO_3$/g $CaCO_3$, times 0 001 g/mg, times 512,640 min/year)		0 equivalents of acid/year
d Theoretical amount of soda ash needed per equivalent of acid[a]		0 389 lb/equivalent
e Factor to account for excess alkalinity required in the effluent (20 to 30% - OSM)	0%	
f Actual annual amount of soda ash needed (add line 1e to 1, multiply the result by line 1c and by line 1d)		0 lbs
g Density of soda ash briquettes	157.05 lb/ft³	
h Annual volume of soda ash briquettes (divide line 1f by line 1g and y 27)		0 yd³
2. [illegible]		
Reaction pond is assumed to be a small circular pond with the bottom diameter equal to its depth. Alternate setups are possible		
a Porosity of soda ash briquettes (professional judgement)	40%	
b Actual volume of reaction pond required (divide line 1h by line 1a)		0 yd³
c Side wall slope (S)	0.5 ft rise/ft run	
d Depth of open pit (d), calculate as $\left[\sqrt{\pi\left(1+\frac{2}{S^2}\right)}\right]$		0 ft
e Surface area of open pit, calculate as $2\pi\left(0.5 + d\sqrt{\frac{1+S}{S^2}}\right)$		0 ft²
f Depth of excavation (add lines 2d 3a, and 3b)		0 ft
g Excavation volume, calculate as $\frac{\pi}{S^1}$ (then divide by 27 to yield cubic yard)		0 yd³
h Excavation footprint, calculate as $d\,\pi\left(\frac{S+}{S}\right)$		0 ft²

Figure 3. (page 1 of 2) Example worksheet for active treatment.

SODA ASH (SODIUM CARBONATE) NEUTRALIZATION

3. [illegible]		
a Thickness of clay liner (a minimum of 0 5 ft)	0 ft	
b Thickness of liner cover	0 ft	
c Surface area of reaction pond (line 2e)		0 ft²
d Volume of clay compacted (multiply line 3a by line 3c and divide by 27)		0 yd³
e Swelling factor[b]	40%	
f Volume of clay required (add 100% to the percentage in line 3e and multiply that percentage by line 3d)		0 yd³
g Volume of liner cover (multiply line 3b and 3c and divide by 27)		0 yd³
h Include synthetic liner? (Y or N)	Y	
i Surface area of synthetic liner (multiply line 2e by 1 25, a factor that accounts for liner anchor)		0 ft²
4. [illegible]		
a Volume to be excavated (Line 2g)		0 yd³
b Multiplier for clearing and grubbing (professional judgement. 200% for small site and 25% for regular site)	200%	
c Area to be cleared in ft² (multiply line 2h by line 4b)		0 ft²
d Area to be cleared in acres (divide line 4c by 43,560)		0 acres
e Area to be surveyed (equal to line 4c)		0.00 acres
f Survey rate	1 acre/day	
g Days required to conduct survey (multiply line 4e by line 4f)		0.00 days

a 106 g/mole*1 mole/mole / 6 efficiency / 1000g/kg * 2 2 lb/kg

b U S Environmental Protection Agency *Final Guidance Manual Cost Estimates for Closure and Post-Closure Plans (Subparts G and H)* January 1987 EPA/530-SW-87 009, Volume III, pg. 7-10. Compaction factor provided is for off-site clay

SODA ASH (SODIUM CARBONATE) NEUTRALIZATION

[illegible]				
a	Volume to be excavated (from AT-2A, line 4a)	0	yd^3	
b	Excavation footprint (from AT-2A, line 4c)	0	ft^2	
c	Unit cost of clearing and grubbing[a]	0.380	$\$/ft^2$	
d	**Total cost of clearing and grubbing (multiply line 1b by line 1c)**	0.00	$	
e	Unit cost of excavation[b]	4.330	$\$/yd^3$	
f	**Total cost of excavation (multiply line 1a by line 1e)**	0.00	$	
g	**Total Cost to Excavate Reaction Pit (add lines 1d and 1f)**			$0.00
[illegible]				
a	Volume of clay required (from AT-2a, line 3f)	0	yd^3	
b	Unit cost of purchase and placement of clay[c]	17.28	$\$/yd^3$	
c	**Total cost of clay liner (multiply line 2a by line 2b)**	0.00	$	
d	Volume of liner cover (from AT-2a, line 3g)	0	yd^3	
e	Unit cost of purchase and placement of liner cover[d]	6.57	$\$/yd^3$	
f	**Total cost for liner cover (multiply line 2d by line 2e)**	0.00	$	
g	Include synthetic liner? (Y or N)	Y		
h	Surface area of synthetic liner (from AT-2A, line 3i)	0	ft^2	
i	Unit cost of purchase and placement of synthetic liner[e]	1.73	$\$/ft^2$	
j	**Total cost of synthetic liner (multiply line 2h by line 2i)**	0.00	$	
k	**Total Cost to Line Reaction Pit (add lines 2c, 2f, and 2j)**			$0.00
[illegible]				
a	Quantity of soda ash for one year (from AT-2a, line 1f)	0	lbs	
b	Unit cost of purchase and delivery of soda ash briquettes[f]	0.140	\$/lb	
c	Unit cost of filling and spreading by dozer[g]	0.0003	\$/lb	
c	**Total Cost of Purchase and Delivery of a One Year Supply of Neutralization Chemical (add lines 3b and 3c, and multiply the result by line 3a)**			$0.00
TOTAL COST OF SODA ASH NEUTRALIZATION SYSTEM (add lines 1g, 2k, and 3c)				**$0.00**

[a] R.S. Means Company, Inc., *Environmental Remediation Unit Cost Data*, 1999, pg. 4-1 & 4-9, Item No. 17 01 0103 and 17 03 0101. The cost is that for medium brush with average grub and some trees, clearing and rough grading with a D6 dozer.

[b] R.S. Means Company, Inc., *Environmental Remediation Unit Cost Data*, 1999, pg. 4-15, Item No. 17 03 0276. The cost is that for excavation with a 1-yd^3 crawler-mounted, hydraulic excavator.

[c] R.S. Means Company, Inc., *Environmental Remediation Unit Cost Data*, 1999, pg. 9-78, Item No. 33 08 0507. The cost is that for construction of a clay liner of 10e-7 conductivity, with 6" lifts, and purchase and delivery of clay material from an off-site location.

[d] R.S. Means Company, Inc., *Environmental Remediation Unit Cost Data*, 1999, pg. 4-23, Item No. 17 03 0422. The cost is that for unclassified fill, 6" lifts, on-site with spreading and compaction.

[e] R.S. Means Company, Inc., *Environmental Remediation Unit Cost Data*, 1999, pg. 9-81, Item No. 33 08 0572. The cost is that for purchase, delivery, and installation of a 60 mil polymeric HDPE liner.

[f] Remine Version V. 1.21. Unit cost of caustic soda was obtained from Midstate Chemical, Pennsylvania. The cost is that for purchase and delivery within a 50-mile radius.

[g] R.S. Means Company, Inc., *Site Work and Landscape Cost Data*, 1999, pg. 53, Item No. 022 262 0010. The cost is that of spreading dumped material by dozer with no compaction.

Figure 3. (page 2 of 2) Example worksheet for active treatment

PASSIVE TREATMENT **PT-5A**

INVENTORY Page 1 of 2

AEROBIC AND ANAEROBIC WETLANDS

1 AMD CHARACTERISTICS			
a	Iron concentration of mine drainage	0 mg/L	
b	Manganese concentration of mine drainage	0 mg/L	
c	Design flow rate	0 gpm	
d	Design iron removal (grams of iron removed per square meter per day of wetland) 20 gmd for alkaline drainage and 5 gmd for acidic drainage[a]	20 gmd	
e	Design manganese removal (grams of iron removed per square meter per day of wetland)[a]	0.5 gmd	
f	Iron loading (multiply line 1a by line 1c 3 785 l/gal and 1 440 min/day and divide the product by 1 000 mg/g)		0 grams
g	Manganese loading (multiply line 1b by line 1c 3 785 l/gal and 1 440 min/day and divide the product by 1,000 mg/g)		0 grams
2. SURFACE AREA OF WETLAND			
a	Surface area of wetland required for iron removal (divide line 1f by line 1d and multiply the result by 10 76 square feet/square meter)		0 ft^2
b	Surface area of wetland required for manganese removal (divide line 1g by line 1e and multiply the result by 10 76 square feet/square meter)		0 ft^2
c	Total surface area of wetland (add lines 2a and 2b)		0 ft^2
3. HUMUS OR COMPOST LAYER REQUIREMENT			
a	Thickness of compost layer (0 5 to 1 ft Skousen 12 to 18 inch Brodie)	0.00 ft	
b	Surface area of compost layer in ft^2 (from line 2c)		0 ft^2
c	Surface area of compost layer (multiply line 3b by 0 111)		0 yd^2
d	Volume of compost required (multiply line 3a by line 3b and divide the product by 27)		0 yd^3
4 LIMESTONE LAYER (FOR ANAEROBIC WETLAND ONLY)			
	Thickness of limestone layer (0 5 to 1 ft Skousen)	0 0 ft	
b	*Estimated volume of limestone (multiply line 4a by line 2c)*		0 00 ft^3
c	Density of limestone	168.02 lbs/ft^3	
d	Efficiency[b]	0%	
e	Purity of limestone	0%	
	Estimated limestone weight of limestone (multiply line 4b by line 4c)		0 lbs

PASSIVE TREATMENT **PT-5A**

INVENTORY Page 2 of 2

AEROBIC AND ANAEROBIC WETLANDS

5. CLAY LINER			
a	Thickness of clay liner (a minimum of 0 5 ft)	0 ft	
b	Estimated volume of compacted clay (multiply line 2c by line 5a)		0 ft^3
c	Swelling factor[c]	40%	
d	Actual volume of clay required (add 100% to the percentage in line 5c and multiply line 5b by that percentage and divide the result by 27)		0 yd^3
6. DIMENSION OF WETLAND			
a	Free board (1 to 3 inch Skousen)	0 ft	
b	Effective depth (add lines 3a, 4a, 5a, and 6a)		0 ft
c	Total volume of wetland (multiply line 2c by line 6b)		0 ft^3
d	Total volume of wetland in cubic yard (divide line 6c by 27)		0 yd^3
e	Average width (depends on availability of land)	0 ft	
f	Average length (divide line 3c by the product of lines 6b and 6e)		0 ft
PIPING REQUIREMENT			
a	Distance from influent or previous treatment unit	0 ft	
b	Distance from wetland to discharge point or next treatment unit	0 ft	
c	Estimated length of influent and effluent piping required (50% safety factor)		0 ft
PLANT AND VEGETATION			
a	Density of cattails	1 plant/ft^2	
b	Number of cattails required (multiply line 7a by line 2c)		0 plants
c	Labor rate (default)	250 plants/hrs	
d	Labor hours required		0 hrs
SITE WORK			
a	Volume to be excavated in ft^3 (with 20% design factor)		0 ft^3
b	Volume to be excavated in yd^3 (divide line 8a by 27)		0 yd^3
c	*Area to be cleared in ft^2 (50% inflated) (multiply line 2c by 1.5)*		0 ft^2
d	Area to be cleared in areas (divide line 8c by 43 560)		0 00 acres
e	Area to be surveyed (same as Line 8d)		0 00 acres
f	Survey rate	1 acre/day	
g	Days required to complete survey (multiply line 8e by line 8f)		0 000 day

a Skousen and others 1996 Acid Mine Drainage Control and Treatment. Second Edition West Virginia University P 253 254

b Skousen and other 1996. Acid Mine Drainage Control and Treatment Second Edition West Virginia University Pg 238

c U S Environmental Protection Agency *Final Guidance Manual Cost Estimates for Closure and Post-Closure Plans (Subparts G and H)* January 1987 Volume II EPA/530-SW-87-009 Pg 7 10

Figure 4 (page 1 of 2) Example worksheet for passive treatment

PASSIVE TREATMENT **PT-5B**

INSTALLATION Page 1 of 2

AEROBIC AND ANAEROBIC WETLANDS

1. SURVEYING				
a	Unit cost of surveying[a]	648 36	\$/day	
b	Days required to conduct survey (from PT 5A, line 9g)	0 00	days	
c	**Cost of surveying (multiply line 1a by line 1b)**			\$0.00
2. CLEARING AND GRUBBING				
a	Unit cost of clearing and grubbing[b]	5 630 00	\$/acre	
b	Area to be cleared and grubbed (from PT 5A, line 9d)	0 00	acres	
c	**Total Cost of Clearing and Grubbing (multiply line 2a by line 2b)**			\$0 00
3. PURCHASE AND DELIVERY OF CLAY				
a	Volume of clay required (from PT 5A line 4f)	0	yd^3	
b	Unit cost of clay purchase[c]	5 00	$\$/yd^3$	
c	Unit cost of delivery of clay (20-mile radius)[d]	19 40	$\$/yd^3$	
d	**Total Cost of Purchase and Delivery of Clay (multiply line 3a by the sum of lines 3b and 3c)**			\$0 00
4. PURCHASE AND DELIVERY OF LIMESTONE (FOR ANAEROBIC WETLAND ONLY)				
a	Quantity of limestone (from PT 5A, line 4e)	0	lbs	
b	Unit cost of purchase and delivery of limestone (50 miles radius)[e]	0 040	\$/lbs	
c	**Total Cost of Purchase and Delivery of Limestone (multiply line 4a by line 4b)**			\$0 00
5. INSTALLATION OF HUMUS PEAT OR COMPOST LAYER				
a	Volume of humus peat or compost (from PT 5A line 3d)	0	yd^3	
b	Unit cost of purchase and spreading of the layer[f]	60 12	$\$/yd^3$	
c	Unit cost of delivery of humus peat or compost (20 miles radius)[g]	19 40	$\$/yd^3$	
d	**Cost of Purchase and Delivery Compost (multiply line 5a by the sum of lines 5b and 5c)**			\$0.00
6. EXCAVATION				
a	Volume to be excavated (from PT-5A line 9b)	0	yd^3	
b	Unit cost of excavation[h]	2 05	$\$/yd^3$	
c	**Total Cost of Excavation (multiply line 6a by line 6b)**			\$0 00
7. SPREADING AND COMPACTING OF CLAY LAYER				
a	Volume of clay layer (from PT 5A line 4f)	0	yd^3	
b	Unit cost of filling and spreading the clay by dozer[i]	1 39	$\$/yd^3$	
c	Unit cost of compacting the clay layer[j]	1 16	$\$/yd^3$	
d	Unit cost of compaction testing by nuclear method[k]	0 85	$\$/yd^3$	
e	Unit cost of compaction testing by sand cone method	0 23	$\$/yd^3$	
f	**Total Cost of Spreading and Compacting Clay Layer (multiply line 7a by the sum of lines 7b through 7e)**			\$0.00
8. INSTALLATION OF LIMESTONE LAYER (FOR ANAEROBIC WETLAND ONLY)				
a	Quantity of limestone (divide PT 5A line 4b by 27)	0	yd^3	
b	Unit cost of filling and spreading limestone[m]	1 39	$\$/yd^3$	
c	**Total Cost of Installation of Limestone Layer (multiply line 7a by line 7b)**			\$0.00

Figure 4 (page 2 of 2) Example worksheet for passive treatment

PASSIVE TREATMENT **PT-5B**

INSTALLATION Page 2 of 2

AEROBIC AND ANAEROBIC WETLANDS

9. PLANTS AND VEGETATION				
a	Labor hours required (from PT-5A line 8d)	0	hrs	
b	Unit cost of planting of cattails[n]	86 50	\$/hr	
c	**Total Cost of Plants and Vegetation (multiply line 9a by line 9b)**			\$0.00
10. INSTALLATION OF INFLUENT AND EFFLUENT PIPING				
a	Length of piping (from PT 5A line 7c)	0	ft	
b	Unit cost of purchase deliver and install 1" to 4" PVC	17 40	\$/ft	
c	Allowance factor for fittings and insulation (default)[o]	15%		
d	Unit cost for fittings and insulation (multiply line 10b by line 10c)	2 61	\$/ft	
e	**Total Cost of Installation of Piping (multiply line 10a by the sum of lines 10b and 10d)**			\$0.00
TOTAL COST OF WETLAND INSTALLATION (add lines 1c, 2c, 3d, 4d, 5d, 6d, 7d, 8c, and 10e)				\$0 00

a R.S. Means Company Inc *Environmental Remediation Data Unit Cost* 1998 pg 10-10 Item No 88 24 1201 The cost is that for surveying with a two-person crew

b R.S. Means Company Inc *Site Work and Landscape Cost Data* 1998 pg 39 Item No 021 104 0260 The cost is that for clearing and grubbing of dense brush including stumps

c R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 46 Item No 022 216 6000. The cost is that for purchasing clay fill or blasted rock, and loading onto a dump truck

d R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 54 Item No 022 216 6000. The cost is that for hauling with a 12 yd^3 dump truck for a 20-mile roundtrip

e Remine Version V 1 21 Average cost of pebble lime from Greyback Lime (Bellefonte PA) and Corsstone (Ertnaburg PA) The cost is that for purchase and delivery within a 50-mile radius

f R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 128, Item No 029 516 0400 The cost is that for purchase delivery and placement of 1 deep push spreader Convert the cost into per cubic yard unit by dividing 1 67 $\$/yd^2$ by 9 (ft^2/yd^2) times (1/12) then multiply the results by 27

g R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 54 Item No 022 216 6000 The cost is that for hauling with a 12 yd^3 dump truck for a 20-mile roundtrip

h R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 49 Item No 022 242 20200 The cost is that for bulk common earth excavation with a 75 HP 50-ft hauling dozer

i R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 53 Item No 022 262 0010 The cost is that for spreading dumped material by dozer no compaction

j R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 45 Item No 022 226 6220 The cost is that for compaction using towed vibrating roller with 6 lifts and 4 passes per lift

k R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 11 Item No 014 108 4735 The cost is that for soil density testing using nuclear method ASTM D2922 71 One test per 50 yd^3 compaction is assumed

R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 11 Item No 014 108 4740 Cost includes soil density testing using sand cone method ASTM D1556064 One test per 100 yd^3 compaction is assumed

m R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 pg 53 Item No 022 262 0010 The cost is that for spreading dumped material by dozer without compaction

n This cost is an engineering estimate based on the use of a crew consisting of two skilled workers The hourly rate for a skilled worker was taken from R.S. Means Company Inc *Site Work and Landscape Cost Data* 1999 back cover

o *R.S. Means Company Inc Mechanical Cost Data 1999 pg 123 Item No 151 550 1000 through 1150.*

p The cost is that for purchase and delivery of Sch 80 high impact/pressure PVC pipes with a range of 1" through 4

17* International Mining Congress and Exhibition of Turkey- IMCET'2001, ©2001, ISBN 975-395-417-4

Microbial Treatment of Cyanide and Heavy Metals Containing Waste Water from Gold Mining

K.Bosecker & P.Blumenroth
Federal Institute for Geosciences and Natural Resources (BGR), Stilleweg 2, 30655 Hannover, Germany

ABSTRACT: Cyanide-bearing waste water from'gold mining is usually decontaminated by oxidation so that the dissolved metals are precipitated as hydroxides. The disadvantages of this method are high consumption of chemicals and the formation and liberation of toxic compounds. Various bacterial and fungal strains capable of degrading cyanides or having a high potential for metal adsorption were isolated from water and sediment samples from a wastewater deposit in Romania. Some of the isolates were characterised and identified. For those strains showing the most rapid degradation or the greatest adsorption capacity, important parameters were analysed using synthetic media and process water and waste water from the cyanidation plant. Optimum pH and temperature levels, maximum tolerable cyanide and metal concentrations, as well as possibilities for the addition of required substances, such as hydrocarbon sources or phosphate, were determined. With a view towards the development of a pilot plant, the immobilisation of microorganisms was achieved on a natural zeolite. With a view to a technical application, further improvements are *to* follow *as* part of the development of a pilot plant.

1 INTRODUCTION

The cyanide spill in Baia Mare at the beginning of last year has been called the worst ecological disaster to hit Eastern Europe since the fallout from the Chernobyl nuclear plant disaster in the Ukraine in 1986. Regarding what happened, the question arises: "Was this catastrophe avoidable?"

For about one hundred years, cyanide solutions have been used to extract gold and silver from ores. This process is known as cyanidation and is summarised by the reaction shown In Equation (1):

$$4\,Au + 8\,NaCN + O_2 + 2\,H_2O \rightarrow 4\,Na[Au(CN)_2] + 4\,NaOH \quad (1)$$

The reaction with silver is similar. In the absence of other metals which form cyanide complexes, relatively weak cyanide solutions can be used because both noble metals form strong complexes with cyanide. However most gold-bearing ore deposits contain other metals which react with cyanide to form metallo-cyanide complexes, resulting in higher cyanide consumption rates and a residual solution which contains a wide range of cyanide species and complexes.

Gold is recovered from the cyariidation solution by cementation with zinc (Equation 2) or by adsorption on activated carbon, but there are many variations of each of these two procedures, leaving a solution which bears cyanide, metallo-cyanide complexes, cyanates, thiocyanates (SCN), and thiocyanate complexes along with other chemical species dissolved from the ore (Mudder 1997).

$$2\,Na[Au(CN)_2] + Zn \rightarrow 2\,Au + Na_2[Zn(CN)_4] \quad (2)$$

Depending on the constituents in the remaining solution and their respective concentrations, die process waste water is hazardous to the environment and needs special treatment and remediation. There are tfiree principal forms of cyanide present in the process solutions: free cyanide, weakly complexed cyanide, and strongly bound or complexed cyanide. The free form of cyanide, which exists as either molecular hydrogen cyanide HCN or the cyanide anion CN~, is considered the most toxic compound and is most readily removed from water through natural attenuation and chemical, physical, and biological treatment processes. The conventional treatment processes can reduce the levels of free cyanide in solution to eitiier non-detectable or environmentally acceptable concentrations.

The toxicity of the weakly metallo-cyanide complexes with copper, nickel or zinc, the second form of cyanide present In metallurgical processes, primarily arises from the free cyanide produced through dissociation of the complex and secondarily from the complex itself and any free metal yielded through breakdown of the complex.

The third form of cyanide includes the strongly bound metal cyanide complexes of iron and cobalt, which themselves are non-toxic and become toxic only after breakdown (Mudder 1997).

Analytically, cyanide is quantified either by the total cyanide or the WAD (weak acid dissociable) cyanide procedures. The total cyanide procedure reports all forms of free cyanide and metal-bound cyanides, including the non-toxic and stable iron cyanides. The WAD cyanide, the toxicologically important form, reports all forms of cyanide except cyanide-bound iron. For drinking water, the standard of total cyanide is 0.20 mg/1 (U.S.EPA). For aquatic life, the limits are in the range of 0.08-0.10 mg/1 WAD cyanide. In the case of aquatic life, fish, particularly die salmonids, are more sensitive to the effects of cyanide tfian other aquatic residents like insects, bacteria or algae.

Cyanide-bearing waste water from gold mining is usually decontaminated chemically by oxidation, e.g., with alkaline hypochlorite (Smith & Mudder 1991, Smith & Mudder 1995) so that dissolved metals precipitate as hydroxides. The disadvantages of this method are high consumption of chemicals and the formation and liberation of toxic compounds. Remediation is necessary because aquatic organisms in particular are affected by the toxic effects of the cyanide emitted, even in small concentrations (micrograms/litre).

Within the scope of scientific co-operation with the Mining Research and Design Institute (ICPM) at Baia Mare, Romania, a biotechnological process is being developed to replace the present chemical process (Blumenroth et al. 1999). In this process, microorganisms that can decompose cyanide and adsorb heavy metals are used. This ability has already been shown at several laboratories for different microbes (Boucabeille et al. 1994, Chaptawala et al. 1998, Dubey & Holmes 1995, Figueira et al. 1995, Gadd 1993, Raybuck 1992, Stoll & Duncan 1996, Tsezos 1990, Volesky 1994). The major advantages of biological treatment are that operating costs are low, metals are removed by adsorption, and the accumulation of toxic intermediate metabolites and final products is avoided. The only commercial plant currently known that treats waste water by cyanide biodegradation is at the Homestake Mine, USA (Whitlock & Mudder 1986, Whitlock 1990).

The pilot plant scheduled for Baia Mare has to be adapted to the local conditions. Therefore, several microorganisms were enriched and isolated from waste water and sediment samples, and degradation and sorption tests with local waste waters were performed (Blumenroth et al. 1997, Blumenroth & Bosecker 2000).

2 RESULTS

2.1 *Water analysis*

A pond near Baia Mare İs used as a basin for waste water from the central flotation and cyanidation plant and for the acid mine water from the Sasar mine. The water in this pond has an average total cyanide concentration of 10-15 mg/I and iron, copper and zinc concentrations of about 40 mg/1 each (waste water A). Contamination with organic compounds was found to be rather poor. The daily total cyanide output amounts to about 300 kg. The main components in the heavily contaminated water from the cyanidation plant (waste water B) are cyanide (approx. 1.500 mg/1) as well as copper and zinc. Both waste waters have a high level of calcium (> 200 mg/1) because during the cyanidation process the pH level İs controlled by the addition of lime. A list of some of the data İs given in Table 1.

Table I. Composition of water samples taken from the Bozinta pond (A) and the cyanide leaching plant in the Sasar Mine (B)

Parameters (all [mg/l] besides pH and cell numbers)	Waste water A (mean values)	Waste water B (mean values)
pH value	7 2	11.6
Cyanide, total	7.4	1160
Chloride	8.9	36.8
Nitrate	4.4	6.1
Sulphate	846	280
Phosphate	0.24	< 0.1
Copper	5.0	209
Iron	< 0.1	1.3
Zinc	2.5	269
Calcium	310	218
Cell numbers/ml (medium R2A/FP)	$1\,4\,10^4/3\,8\,10^4$.	0/0

2.2 *Isolates*

Several bacteria and fungi were selectively enriched and isolated from water and sediment samples from the Bozinta pond. The best cyanide-degrading bacteria and the strongest metal-adsorbing fungi were identified by die German Collection of Microorganisms and Cell Cultures (DSMZ, Braunschweig). The most important bactena were *Pseudomonas spec,* and *Burkholderia cepacia,* both of which are common aquatic organisms. The fungal isolates were identified as strains of the genera *Trichoderma, Epicoccum* and *Aspergillus.*

2.3 *Cyanide degradation*

Initial investigations on cyanide degradation by bacteria were performed using an artificially contaminated, standardised culture medium. The isolated bacteria were analysed for their effectiveness based on the use of cyanide as the sole nitrogen source. The quickest growing strains were used in further experiments. Two very good degraders were found, belonging to the species *Burkholderia cepacia*. Kinetic experiments were carried out at different cyanide concentrations (Fig. 1) and at 30°C or 15°C. Depending on the cyanide concentration, degradation was completed after 1-22 hours and was not influenced by a high sulfate concentration. The degradation was faster at 30°C than at 15°C.

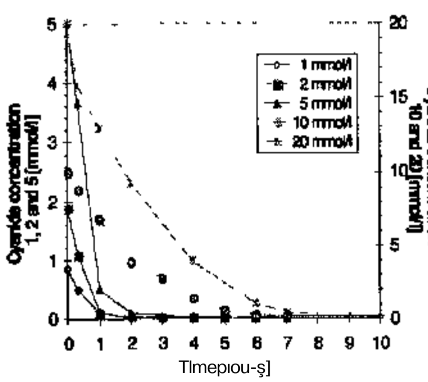

Figure 1 Degradation of different cyanide concentrations at 30°C by *Burkholdena cepacia*.

The degradation rates achieved at pH 7 and pH 9 were more or less the same. The maximum tolerated, degradable cyanide concentration was 520 mg/l CN~, which is several times me concentration in the pond. Therefore, treatment of highly contaminated waste water direct from the cyanidation plant might also be considered. Ammonia arising as a metabolite is assimilated by the microbes. For cyanide decomposition as well as for ammonia assimilation, the optimum pH value was found to be 8.

The degradation of cyanide did not require a carbon source, but for the basic metabolism of the organisms a carbon source was necessary. Besides glucose, the classic carbon source, cheap alternatives such as molasses, whey and liquid residues from beer and juice production were examined for their suitability. The addition of juice and beer residues (4%; v/v) led to successful growth results compared to the glucose concentration of 0.4 % used otherwise.

The bacteria present in the waste water from the Bozinta reservoir did not seriously contribute to the degradation of cyanide because of the small quantity of cells. As the cyanide concentration was very low, waste water A was artificially contaminated in most of the experiments (26-130 mg/l CN~). In both waste waters A and B, the addition of phosphate as a nutrient was vital for the degradation of cyanide. The rate of degradation depended on the level of phosphate present (Fig. 2). Calcium was bound by the addition of EDTA to prevent precipitation with phosphate. Without the addition of EDTA, the cyanide degradation slowed down considerably.

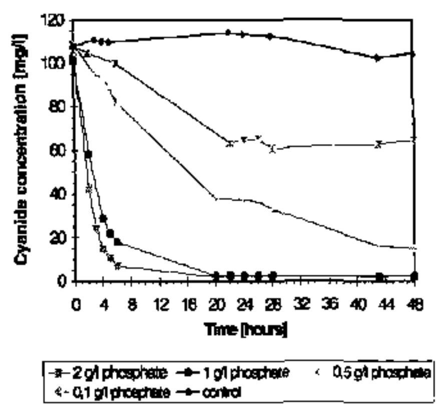

Figure 2 Cyanide degradation in waste water in relation to phosphate addition (5 *cepacia)*

The maximum metal tolerance was determined İn various dilutions of waste water B. According to the isolates being tested, the metal concentrations were in the range of 5-43 mg/l copper or 9-64 mg/l zinc (when present simultaneously). These results can only provide an approximate tolerance area because the metals mentioned are present not just as ions but also as different complexes, and for this reason they have a lower toxicity.

Experiments with diluted effluent from the cyanidation plant (waste water B) resulted in lower degradation rates, which is explained by, among other factors, high contents of zinc and copper, which complex with cyanide and inhibit the growth of the microorganisms.

2.4 *Immobilization of bacteria*

If the bacteria tested in the laboratory are to be used on a technical scale, the immobilisation of the organisms on a suitable layer seems appropriate in order to obtain a larger biologically active surface area and to prevent the microorganisms being washed away by huge amounts of waste water, Apart from commercially available plastic layers, the growth on zeolite rock was examined. The use of zeolites has been reported for waste water treatment (Dictor et al. 1997, Klein & Ziehr 1987, White & Schnabel 1998, Marlon & Liebmann 1994, Suh et al. 1994) and provides a cheap alternative, as the zeolites can be obtained near the processing plant. Growth on the plastic materials was not successful, but the bacteria did attach themselves to zeolite. This was determined by cell staining with acridme orange. Investigations with an electron microscope are as follows.

There are various types of zeolite. When zeolite type 1 (mainly philippsite) was used, the degradation of cyanide followed at the same speed as with suspended cells (< 1 mg/1 after 4 hours). Growth on zeolite type 2 (mainly clinoptilolite) was less efficient and the reduction in cyanide correspondingly slower (< lmg/1 after 29 hours; Fig. 3).

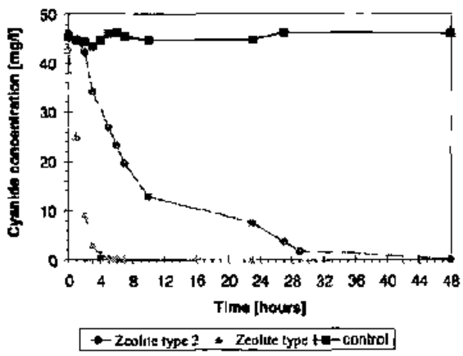

Figure 3. Degradation of cyanide by immobilised cells of *Burkhoidena cepacia.*

Recent long-term column experiments (3 weeks) with zeolite type 2 and repeated addition of glucose (0.2%) showed total degradation of all cyanides in waste water B (10-fold dilution, pH 8).

2.5 *Metal biosorption byfitngi*

Filamentous fungi with high cyanide resistance and metal tolerance, isolated from sediment samples from the Bozinta pond, were chosen for the biosorption experiments. Artificially contaminated synthetic medium or deionised water served as a matrix for the sorption tests. Metals were used separately or as a mixture of iron, copper and zinc in various concentrations (e.g. 25, 50 or 100 mg/I each) and sorption capacities were calculated under different conditions.

It turned out that different fungi had different preferences for the sorption of a particular metal (iron, copper or zinc), but the sorption rates depended on whether the metals were offered separately or in a mixture. Furthermore, the sorption capacity depended on the composition of the growth medium and the glucose concentration m the pre-culture. The fungal mycelium growing on the residue from juice production adsorbed the metals much better than biomass cultivated with molasses or glucose ("juice" 1% > molasses 1% > glucose 0.4% > glucose 4%; Fig. 4).

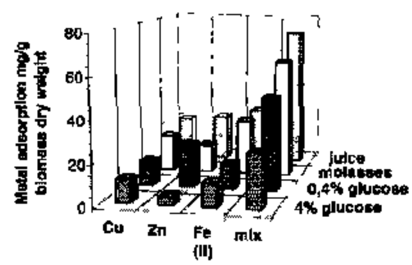

Figure 4. Adsorption levels depending on pre-cultunng conditions *(A fumigatus).*

The sorption rates of living biomass were greater than those of dead biomass. Dried mycelium showed the least effect. (Fig. 5). The incubation generally lasted for seven days, but after two to four days most of the metal was already adsorbed.

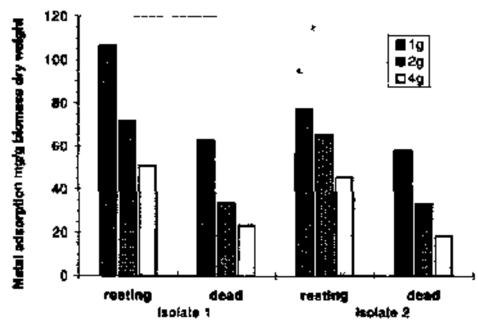

Figure 5 Comparison of metal sorption (mix) by different amounts of dead and resting biomass

3 CONCLUSIONS

Various bacteria and fungi capable of degrading cyanides or having a high potential for adsorbing heavy metals were isolated from water and sediment samples from a mining wastewater deposit in Baia Mare. Some of the isolates were characterised and identified. For those strains showing the most rapid degradation or the greatest adsorption capacity, important parameters were analysed in synthetic media and in process waters from the Baia Mare pond and the Sasar cyanidation plant. Optimal pH and temperature levels, maximum tolerable cyanide and metal concentrations, as well as possible additional-substances required, such as hydrocarbon sources or phosphate, were determined. With a view towards the development of a pilot plant, the immobilisation of microorganisms was tested on different support materials.

The results presented here show a high potential for developing a pilot plant on a technical scale for the biotreatment of cyanide and heavy metal containing waste water from mining and metallurgical processes. The chemistry, toxicology, and environmental fate of the various cyanide compounds are well understood. There are reliable and proven technologies for the destruction of cyanide to discharge treated mining and process" water into the environment. The application of appropriate technologies is no longer limited by science and engineering. More often it is a lack of knowledge, and mismanagement and inadequate legislation for environmental protection which cause environmental problems.

ACKNOWLEDGEMENTS

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REFERENCES

Blumenroth, P. & Bosecker K. 2000 Mikrobieller Abbau von Cyanid in Prozeß wässern der Goldgewinnung. In. Wippennann, Th. (Hrsg.). *Bergbau und Umweit,* 183-197, Springer Verlag Berlin.

Blumenroth, P., Bosecker, K., Michnea, A & Sasaran, N. 1999. Development of a bioremediation process for mining waste waters. In. Amils, R. & Ballester, A. (eds) • *Biokydrometallurgy and the environment towards the mining of the 21st century,* pp.547-558, Elsevier (Amsterdam).

Blumenroth, P, Dettmers, H. & Bosecker K. 1997. Isolation of microorganisms from mining wastewaters for cyanide degradation and biosorption of heavy metals *Proceed Internationa} Biohydrometallurgy Symposium IBS'97* - Biomine '97, 4-6 August, Sydney, Australia; Australian Mineral Foundation, Glenside, South Australia

Boucabeille, C, Bories, A., Ollivier, P. & Michel, G. 1994. Microbial degradation of metal complexed cyanides and thiocyanate from mining wastewaters. *Environ Pollut* 84 59-67

Chapatwala, K D, Babu, G R V , Vıjaya, O K., Kumar, K.P & Wolfram, J H. 1998 Biodegradation of cyanides, cyanates and thiocyanales lo ammonia and carbon dioxide by immobilized cells of Pseudomonas punda J Ind *Microbiol & Biotechno!* 20 28-33

Dıctor, M C, Battaglia-Brunet, F, Monn, D., Bones, A & Clarens, M 1997 Biological treatment of gold ore cyanidation wastewater tn Fixed bed reactors. *Environ Poll* 97(3). 287-294

Dubey, SK & Holmes, D.S 1995 Biological cyanide destruction mediated by microorganisms World *Journal of Microbiology & Biotechnology'* 11 * 257-265

Figueira, M M , Ciftiinelli, VS T & Lmardi, V R. 1995 Bacterial degradation of metal cyanide complexes In. C A Jerez, T. Vargas, H Toledo & J.V. Wiertz- *Biohvdrometallurgical Processing,* pp 333-339, University of Chile.

Gadd, G.M. 1993. Tansley Review No. 47 Interaclion of fungi wilh toxic metals *NewPhytol* 124-25-60

Klein, J. & Ziehr, H 1987 Immobilisierung von Mikroorganismen durch Adsorption *BioEngineering* 3 8-16

Marton, S I. & Liebmann, A. 1994 Possible uses of naiural zeolites in wastewater treatment procedure 2 Internat Symp and Exhibition on Environm Contam in Central and Eastern Europe *Proc Budapest* f20 -23 9 94)

Mudder, Terry 1997 Cyanide Monograph Mining Environmental Management, *Mining Journal Ltd,* London, UK.

Raybuck, S A 1992. Microbes and microbial enzymes for cyanide degradation *Biodegradation* 3: 3-18.

Smith, A & Mudder, T. 1991 Chemistry and Treatment of Cyanidation Wastes. *Mining Journal Books Ltd.,* London, England.

Smith, A & Mudder, T. 1995 Cyanide: dispelling the myths. *Mining Environmental Management,* June 1995,4-16.

Stall, A. & Duncan, J.R. 1996. Enhanced heavy metal removal from waste water by viable, glucose pretreated Saccharomyces cerevisiae cells *Biotechnol Lett* 18(10). 1209-1212.

Suh, Yeung-Jean, Park, Jong Mong & Yang, Ji-Won 1994 Biodegradation of cyanide compounds by Pseudomonas fluorescens immobilized on zeolite. *Enzvme Microb. Technol* (June) 16" 529-533

Tsezos, M 1990 Engineering aspects of metal binding by biomass. In: *Microbial Mineral Recovery.* Chapter 14 Eds. H.L Ehrlich *& C* L Brierley. McGraw Hill, Inc. USA

Volesky, B 1994. Advances m biosorption of metals- Selection ofbiomass types. *FEMS Microbiol* Rev. 14 291-302.

White, Daniel M. & Schnabel, William I998. Treatment of cyanide waste in a sequencing batch biofilm reactor *Wat Res* 32(1). 254-257.

Whitlock, J.L. 1990. Biological detoxification of precious metal processing wastewaters. *Geomicrobiol }* 8'241-249.

Whitlock, J.L. & Mudder, T.I. 1986. The Homestake wastewater treatment process Biological removal of toxic parameters from cyanidation wastewaters and bioassay effluent evaluation. In Lawrence, R.W , Bran ion, R.M R. & Ebner, H.G. (eds.): Fundamental and Applied Biohydrometallurgy, pp 327-339; Elsevier, Amsterdam, *Proceedings 6th Internat Symp Biohydrometallurgy,* Vancouver.

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Mining Engineering Education Using the Internet

G.N.Panagiotou, E.Tsiavos & T.N.Michalakopoulos
Department of Mining Engineering & Metallurgy, National Technical University of Athens, Greece

ABSTRACT: An Internet connection with e-mail and web-browsing services can provide both students and teachers with a wide variety of learning tools, which can be used at any time, in any place, ad lib. This paper presents and discusses various aspects of Internet-based education with particular emphasis on mining engineering examples.

1 INTRODUCTION

There is no doubt that computing and automation have led to great changes in the mining industry, and these changes are mirrored in the undergraduate and graduate curricula of mining engineering departments, increased emphasis has been placed on computer training and applications, and it İs considered impossible for a well-educated mining engineer to graduate today without significant understanding and skills in this area.

Continuing education and technology programs become increasingly important for upgrading the education of practising engineers in specialized fields, including computer applications. Several departments are developing continuing education courses in a wide variety of fields, but more commonly continuing education is handled in-house by companies, in the form of specialized training for their engineers.

Conferences on numerous mining engineering topics and mining magazines are also important means for both students and engineers to improve their understanding of the "body of knowledge" that characterizes mining engineering.

The familiarity of students and engineers with computing, the "globalisation" of knowledge, and the need for frequent access to training, career-advancement tools and technical references at affordable cost has led to the development of an educational technology which is based on the use of computers and networks that convey knowledge from the teacher-instructor to the student-trainee.

This paper presents and discusses various aspects of Internet-based education with particular emphasis on mining engineering examples.

2 INTERNET-BASED EDUCATION

The influence of the Internet on personal and corporate lives is so great that our society is being virtually recreated. There is no doubt that we already live in an "electronic society" or "e-society", which covers almost all aspects of human activity. This is illustrated by the title of the 14th Bled Electronic Commerce Conference: "E-Everything", Bled, Slovenia, June 25-26, 2001. Among the conference's research track topics one can see: e-business, e-commerce, e-marketing, e-trade, e-education, e-strategy, e-behavior, e-households, and e-democracy.

Long before the World Wide Web (WWW) reached wide acceptance, the Internet was being used for educational purposes, mostly via mailing lists and bulletin boards. The tremendous growth of the WWW, its attractive educational features and its ease of use made it very quickly the main platform for potential educational applications, especially for open and distance learning.

The educationally attractive WWW features include:

- ability to obtain multimedia documents;
- hypertext/hypermedia capability;
- WWW networked basis, allowing for distance learning, ad lib;
- interactivity between the student and teacher; ease of use and system-open;

The importance of the WWW as an educational tool can be realized by the number of web-based courses that become available on a daily basis. The complexity of these courses varies from the simple posting of lecture notes on a website to more sophisticated educational elements such as animated graphics, simulations, virtual laboratories, etc. The development of these sites using programming languages such as Java has made the courses more interactive, faster to execute and easily transportable to multiple platforms.

The Globewide Network Academy (GNA) is an educational non-profit organization in Texas, U.S.A., which develops distance-learning relationships and facilities for the public to use worldwide. The GNA's website <http://www.gnacademv.org> contains a comprehensive catalogue of distance-learning courses, as well as forums for distance learners and distance teachers. The catalogue lists (as of February 6, 2001) 23874 courses and 2508 programs, of various levels, of which 939 courses and 267 programs are on engineering topics.

According to JonesKnowledge.com <http://www.j onesknowledae. com>. a US-based company that provides end-to-end online learning solutions, students spent approximately $233 billion in direct costs on higher education in 1999, with $1 1 billion of that devoted to e-leaming. That figure is forecast to increase six-fold over the next ten years.

An article ("Taking over the world by degrees") published (January 5, 2001) on the Financial Time's website <http://www.ft.com>. describes some recent efforts to offer online education by British universities, and raises concerns about the future of distance learning. The U.K. government, through the Higher Education Funding Council, has established a joint e-university project with a budget of £400rn, and all U.K. universities are allowed to become members and hold shares.

Jones International University <http://www. jonesintemational .edu> is the first fully online accredited institution of higher learning whose campus is located in cyberspace and offers a variety of graduate, undergraduate, and certificate programs.

The apparent great interest in e-leaming projects has created the need for the development of adequate tools for the elaboration of web-based courses. These courseware development tools assist teachers with new course development and the move from traditional classroom pedagogy to an online format.

3 INTERNET-BASED MINING EDUCATION

Just browsing through the web is an educational experience in itself. As very often happens, one starts browsing with something specific in mind^nd ends up diverted from the initial goal for a while, because one has found something interesting on the way which was not explicitly looked for. This type of learning, which happens at an unexpected moment about an unexpected subject, is called "accidental learning". A mining engineering student can easily "mine" the Internet. Good starting points are the following sites:

- InfoMine (<http://www.infomine.com/>)
- MineNet (<http.//www .microserve.net/--doug/> 1
- Mining Technology (<http://www.mming-technology.com/>)
- Mining Educators' website (<http://www.uidaho.edu/mining school/>)

The last of these sites, developed and maintained by Prof. J.R. Sturgul at the University of Idaho, U.S.A., among other things, provides direct links to all the Mining Schools around the world

One of the unique features of engineering education is the combination of theoretical knowledge with practical experience. In conventional education, the former consists of lectures and exercises supported by handouts, lecture notes and textbooks, while the latter includes laboratory tests and field-work.

The web provides improved functionality in transmitting theoretical knowledge to students and possesses an effective mechanism to integrate multimedia tools into a single user interface.

Let's take as an example a mining engineering student located in Athens. Using his web browser he can attend a lecture on drill-and-blasting techniques, which is transmitted live from the Institute of Mining at the Technical University of Clausthal (TUC), Germany. A two-day event (26-27 January 2001) on drill-and-blasting was held at TUC and was audio-visually transmitted live via the Internet <http://www.bergbau.tu-clausthal.de/bus2001/BuS2O01.html>.

Then, in order to enhance his knowledge on these topics, he reads chapters: "9.1.3. Drilling" and "9.2.1. Blast Design" from the "Mining Engineering Handbook", published by the Society of Mining Engineers (SME), U.S.A. This comprehensive reference work, which distils the entire body of knowledge that characterizes minmg engineering as a disciplinary field, is available online at the SME's website <http://books.smenet.org/>.

Finally, he can read case studies, view state-of-the-art equipment, products and techniques and download the relevant literature from particular manufacturers' websites (e.g., from Atlas Copco's rock drilling equipment site <http://www.boomer-rig.com/> or for blasting products from Austin Powder's site <http://www.austinpowder.com/>).

R. Ganguli in his article "Online presentations aid classroom teaching" (Mining Engineering, Nov. 2000 <http://me. smenet. org/>) discusses elements of online pedagogy and information technology that are being used in the University of Alaska, Fairbanks, and the difficulties faced in the preparation and delivery of online mining engineering courses.

EduMine <http://www.edumine.com/> offers online courses on mining and geosciences. Its course catalogue includes online courses on: "Underground

Mining Methods and Equipment", "Risk Management İn Mining", "Geotechnical Data Collection for Excavation in Rock", "Practical Geostatistics, Modeling and Spatial Analysis", etc.

EduMine, which has partnering agreements with the Canadian Institute of Mining, Metallurgy and Petroleum and Svedala Process Technology, provides course authors with an authoring environment and tools for the conversion of existing course material into an online format or the development of new online material. EduMine's courses are based on the XML Internet standard, whose advantages include reduced bandwidth, intelligent processing, efficient management of content and greater client-side interactivity.

Practical experience in traditional mining engineering courses is gained through laboratory courses, and more often through visits and fieldwork in mines (the mining engineer's "Big Lab").

Mcintosh Redpath Engineering <http://www.mcintoshengineering.com/> provides online the "Rules of Thumb for the Hard Rock Mining Industry", which are practical guidelines and norms related more to the "art" and "praxis" of mining than to the "science" of mining.

Virtual Reality (VR) techniques have made it possible to establish shared "virtual laboratories" which are accessible via the Internet. The Cooperative Systems Engineering Group, Lancaster University, U.K., investigates the use of virtual reality in learning environments. The "DEVRL - Distributed Extensible Virtual Reality" project <http://www.comp.lancs.ac.Uk/computinp/research/cseg/proi ects/devri/> intends to provide distributed users with a number of cooperative virtual reality-based applications in order to experiment with a simulated set of physical properties.

The Virtual Reality Applications Research Team (VIRART), University of Nottingham, U.K., <http://www.virart.nottingham.ac.uk/> is involved in the specification, design, development and evaluation of virtual environments in both industrial and educational applications development,

The AIMS Research Unit, School of Chemical, Environmental and Mining Engineering, University of Nottingham, U.K., <http://www.nottingharn.ac.uk/ aims/> has done pioneering work in the development of mining and environmental management-related virtual reality applications. The user of these applications can interact with 3D computer-generated models of real-life systems and operate objects that respond to their actions in real time.

With these virtual reality models, the user can visualise new designs, techniques and methods, handle hazardous situations without actually putting himself at risk, and take training courses on how to use complex equipment in a safe and cost effective environment.

Information collection and analysis activities. which involve students collecting, compiling and comparing different types of information on a topic of interest, are an integral part of education, particularly at postgraduate level. Numerous papers are available on websites maintained by mining engineering departments (<http://www.uidaho.edu/mining school/>) or in electronic libraries (e-Iibraries), e.g., the ScienceDirect® of Elsevier Science B.V, (<http:// www.sciencedirect.com/>). Leading mining journals are available over the Internet, including:

- Mining Engineering (<http://me.smenet.org/>)
- Engineering & Mining Journal (<http://www.e-mj.com/>)
- Coal Age (<http://industryclick.com/>)
- World Mining Equipment (<http://www.wme.com>)
- Mining Environmental Management (<http://www. mining-ioumaI.com/osindex.htm>)
- Mining Journal (<http://www.mining-iournal.com/osindex. htm>)
- Mining Magazine (<hrtp://www.mining-journal. com/os İndex.htm>)

Finally, regarding student guidance, Question-and-Answer, and Ask-the-Expert services, these can be easily implemented via the Internet through e-mailing, mailing lists and forums. Internet-connected specialists from universities, businesses, organizations, etc. can serve as electronic mentors to students who want to explore specific study topics in an interactive format (e-mentoring). In the Mining Educators' website <http://www.uidaho.edu/mining school/> there is a list of Email Experts on various mining topics and a Mine Simulation Discussion List. Gibbs Associates maintains the MINGEOL mailing list for a discussion group on mining, geology, and earth science software.

Mining forums are maintained by MineNet <http://www.niicroserve.net/~doug/forum.html> and Trans Tech Publications <http://www.bulk-online.com/>.

4 WEB-BASED MINING SYMPOSIA

MineSim '96, the First International Symposium on Mine Simulation via the Internet, was organized by the Department of Mining Engineering and Metallurgy of the National Technical University of Athens (NTUA), Greece and the Department of Metallurgy and Mining of the University of Idaho (Ul), U.S.A. It was held in "cyberspace" on 2-13 December 1996. (<http://www.metal.ntua.gr/msslab/MineSim96/>)

The symposium was run using a web pages arrangement, set up and maintained by the Mining

Systems Simulation Unit (MSSLab) at NTUA. İt was the first such symposium for any area of mining and mining-related topics, and it appears to be the first such full-scale international symposium on any topic.

The success of any symposium is, primarily, based on the quality of the papers presented, but in the case of web-based symposia, and in addition to the papers' quality, the success depends heavily on issues not common in conventional symposia, such as hardware and software, and network availability, capability and reliability, not to mention programming skills. MSSLab was responsible for the planning, development and management of this "virtual" symposium.

The basic concept in designing MineSim '96 was to "model'" a conventional-type symposium, so that a participant that had ever been to a conference would find no difficulty in participating in MineSim '96. The design principles for the web pages were: easy navigation, functionality and speed.

Registration was non-compulsory and a registration fee was not required. However, a short registration form was provided for the symposium's site visitors for the purpose of creating an automatically updated list of participants, allowing purchase of the proceedings at a reduced price and the distribution of personalized symposium badges. At the end of the symposium, 406 participants from 35 countries were listed, while the symposium's home page had 2188 "hits'*.

The symposium's Call for Papers attracted 111 abstracts and 82 papers, representing 24 countries, which were arranged and presented in ten sessions, namely (the number of papers is given in parentheses): Simulation - Open Pit Mining Operations (7); Simulation - Underground Mining Operations (6); Simulation - General Topics (10); Virtual Reality (4); Expert Systems - Genetic Algorithms (9); Neural Networks (4); Mine Safety - Training (3); Modeling, Planning and Production Scheduling (24); Rock Mechanics (7); Mine Equipment (8).

By clicking on a paper's title, the participant was ready to read the presentation of this particular paper. Following the presentation, as in a regular conference, there was a "Q&A time". A navigation bar at the top of each paper facilitated dialogue between participants, asking questions or making comments, and the paper's authors, replying to questions and defending their theses. Each paper had a Discussion List associated with it and an e-mail address in the form: PaperCode@www.metal.ntua.gr. Both participants and authors communicated by sending e-mail messages to the paper's Discussion List. For each paper, messages could be viewed sorted by thread, date, subject and author.

MineSim '96 was supported by a pool of sponsors, 13 organizations and 22 industrial sponsors, and through links to their respective websites, they presented their products, services and brochures.

The "sightseeing" program of the symposium included a "virtual tour" to the Acropolis of Athens and the monasteries of Mount Athos.

MineSim '96 proposed a new way of organizing symposia on specialized topics in which the Limited interest does not justify the organization of a regular symposium due to the high costs involved on tine part of both the organizers and the participants.

From 1-12 December 1997, a similar event, the "First International Conference on Information Technologies in the Minerals Industry - via the Internet" (MinelT '97), was organized by the Department of Mining Engineering and Metallurgy of NTUA, and run using a web pages arrangement set-up maintained by MSSLab. (http://www.metal.ntua.gr/ msslab/MineiT97/). Following a number of requests by participants and members of the International Organizing Committee, the conference period was extended till the end of the year. During the course of the conference mere were 1683 "hits" and 296 registrants, from 35 countries, submitted a non-compulsory registration form.

During this period, the conference's website provided a forum for the presentation, discussion and criticism of state-of-the-art and emerging information technologies applied in the minerals industry, covering a wide spectrum of applications from orebody modeling to training and reclamation.

Forty-six papers, from North and South America, Europe, Australia, Africa and Asia, were accepted for presentation and arranged in six technical sessions: Exploration - Orebody Modeling; Mine Planning - Mining Operations; Rock Mechanics - Excavation Engineering; Mine Equipment; Mine Safety - Training; Reclamation - Environmental Issues. Several interesting case histories were offered, while high quality 3D colour graphics and animations supported many papers.

MinelT '97 was sponsored by O&K Orenstein and Koppel Inc., USA.

The proceedings of both symposia, the permanent record of the "virtual conferences", were published by A.A. Balkema Publishers, Rotterdam, Netherlands (<http://www.balkema.nl>) in hardback with a companion CD-ROM.

5 EDUCATING WITH RITAS

RITAS (Remote Interactive Training Ad-lib System) is a web-based environment developed by the Mining Systems Simulation Unit (MSSLab), NTUA, to facilitate remote learning and training using the Internet.

In its present state of development, RITAS aims to offer an on-line course for final year students of the Department attending the class "Mining Systems Simulation". It is expected that in the near future RITAS will include additional mining classes.

RITAS is accessible 24 hours a day via the Internet, by directing a web browser to: <http://kimolos.metal .ntua.gr/ritas/>. Instructions are provided both in Greek and English.

The Mining Systems Simulation course is based on the GPSS/H discrete-event simulation language, developed by Wolverine Software Corp., U.S.A., and therefore RITAS provides students with the ability to run simulations on the web browser, using the GPSS/H Student Version Release 3.0. The student version of GPSS/H İs a full functioning version of GPSS/H, but has certain limitations regarding the size of the model one can run.

RITAS includes four main sections.

The *Help Section* provides students with online tutorial material, which describes the philosophy, the structure and the way that GPSS/H functions. In addition, the syntax of all GPSS/H language code-statements is available online, m tables that present the code name, operands, and description with examples. Using this section, students are able to retrieve quickly information regarding the language syntax when practising on simulation examples or working with exercises.

The *Example Section* instructs students how to simulate an underground mine transportation system using GPSS/H. The conversion of the real system into a GPSS/H model is analysed step by step using both text and animated pictures (Figure 1).

The aim of the *Model Section* is to familiarize students with the "fundamental nature" of a simulation study, i.e., the ability that a simulation has to answer "what if?" questions. The student can simulate different operating scenarios of a "loader-truck" mining system, by changing the values of the tabulated operation and cost input data and running the simulation model. The simulation output is provided both in the form of tables and charts, from which the student can realise the effect of each scenario on the system's output, operating cost, etc.

In the *Source Section,* which is the "heart" of RITAS, the student can develop, using GPSS/H, his own simulation models of the systems he studies. This can be done by either writing the GPSS/H code in a text window provided, or by uploading the GPSS/H source file written with a text editor. The simulation-run results are provided either in the GPSS/H Standard Output format or a more simplified and ergonomie "Quick-View" format.

The development of this final section, which is based on Client/Server architecture, was implemented by using web-based programming techniques (HTML, JavaScript, Perl, CGI, ISAPI). Figure 2 shows the task-flow to run a GPSS/H simulation model via the RITAS website.

RITAS has been used as the sole student-course interface for teaching Mining Systems Simulation in the Department of Mining Engineering & Metallurgy for the last two academic years (1999-2001). More than 30 students each year attended the class and worked with RITAS either from the Department's PC Lab or by using PCs at home. The feedback received from the students was very positive, while their constructive criticism led to certain "ergonomie" modifications of the RITAS web page layout

6 CONCLUSIONS

At a time when people will redirect their careers 5-7 times before they retire, and jobs requinng higher education have increased by 86% f<http//www.jonesknowledge com/higher/about.html>). society is paying more and more attention to education, including continuing education, career-advancement short courses and training m specialized topics.

Computing and network technology have been introduced into the educational community in order to enhance learning, communication and teacher-student interaction. Web-based services and tools are widely available, allowing educators to easily develop and implement classroom-instructional material online to supplement classroom-teaching methods, or to deliver distance education courses.

Online learning (e-leaming) is a powerful medium that gives educational establishments the opportunity to deliver lifelong learning to millions, with no time zone or distance barriers, and offers the potential for long-term revenues.

However, education, apart form being a process of information transfer and problem solving, is primarily a process of enlightening students so that they absorb ideas and ways of thinking, preparing Üıem for life. This requires extensive social interaction, tuition and mentonng.

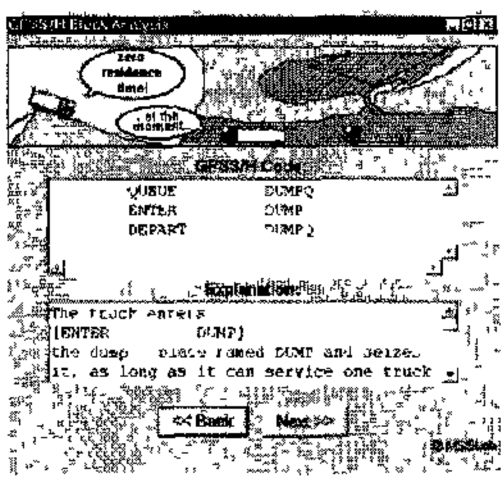

Figure 1 Analysis of conversion of the real system into a GPSS/H modei ustng both text and animated pictures

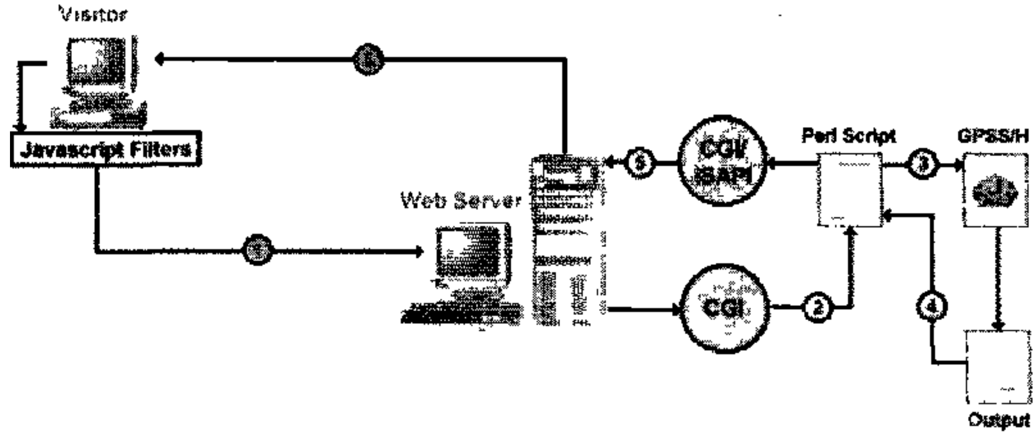

Figure 2. Task flow to run a GPSS/H simulation model via the RITAS website.

A good educational scheme should take into account the background of each student and tailor its material to the student's capabilities. In addition, it should provide appropriate remediation to students who experience difficulties with some concepts.

Certainly, the Internet, thanks to its attractive educational features, is an excellent vehicle for the delivery of online learning materials to the international community, including the widely dispersed mining communities, but it is unlikely to replace the human education element provided by classroom teachers.

REFERENCES

All references are available online URLs are provided within the paper's text

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Mining and Society: No Mining, No Future

P.N.Martens & L.Rattmann
Institute of Mining Engineering I, Aachen University of Technology, Aachen, Germany

ABSTRACT: At present, mining as a supplier of raw materials is an important foundation of our society. In order to give an initial overview of the world mining industry, some facts and figures are presented, such as the annual production of mining products compared to that of other commodities, the share of mining products in world trade, and the contribution of the mining industry to gross domestic product. How important mining is for society is illustrated by means of the dependence of the world's primary energy production on mineral fuels and by means of the average consumption of mining products per person in the world. To demonstrate the global presence and the actual role of mining products in everyday life, a car and the raw materials required to build it is taken as an example. By extension from the present, an outlook for the future role of mining in society is given. Although it is incontestable that renewable energy sources and recyclmg will play an increasingly large role in society, mining will remain indispensable in the next few decades and beyond İn satisfying society's demands for energy and raw materials.

1 INTRODUCTION

Mining affects the natural environment. The use of mineral fuels causes CO_2 emissions and global warming, and mining consumes non-renewable resources. These facts lead to the impression that were mining to be stopped immediately, everybody could drive home and look forward to a better future. But the situation with mining is not that simple. Without mining there would be no homes; there would be no fuel or cars to drive. Without mining there would be no future for anybody to look forward to. Mining has been the foundation of society since the earliest times. The beginning of mankind is often dated to the time of the first use of tools. These tools were made of flint mined from the earth. This aim of this paper is to point out the importance of mining for society now and in the future.

2 WORLD MINING - PRESENT FACTS AND FIGURES

In any discussion of world mining, it is normal to first look at world mining production. The global mining production of different commodities is shown in Figure 1. All in all, total production was around 31 billion t in 1998. If one relates the overall mining production to the '98 world population of 5.9 billion people, this means that every single inhabitant of the world needs at least 5 t of mining products per year For the year 1900, the same calculation results in a figure of 0.5 t per person per year. The expansion of world mining in this century has been encouraged by two factors; firstly, by the exceptional growth in the world population, and secondly, by a growing demand for mining products per inhabitant. Projecting the present figure of 5 t per person per year over an expected life of 60 years, every human being will roughly consume more than 3001 of mining products (Figure 2), with natural aggregates and mineral fuels accounting for a share of more than 90%. Of course, this average differs substantially between industrialised and developing countries. For example, in Germany the lifetime consumption of raw materials per person including natural aggregates has been calculated to be around 1,2301 (Wellmer & Stein, 1998).

A comparison of mining and the primary industry sector of agriculture, fishery and forestry is shown in Figure 3. For example, in 1998 production of cereals amounted to 2 billion t, production of wood to 3.5 billion t and the total production was 13.4 billion t. Thus, the tonnage of overall mining production is 2.3 times higher than the tonnage of the overall production of agriculture, fishery and forestry. It should also be mentioned that this level of agricultural production would be impossible without mineral fertilizers provided by the mining industry.

World Mine Production 1998 (1000 t)	
Diamonds	0 03
PGM	0 30
Gold	2 50
Electronic Metals	3 20
Silver	16
Cobalt	23
Niobium, Columbium	29
Tungsten	33
Uranium	35
Vanadium	45
Antimony	81
Molybdemum	135
Mica	195
Tin	219
Magnesium	380
Zirkonium	397
Dısthene	538
Graphite	590
Boron	776
Nickel	1,100
Asbestos	1,790
Dıatomıte	2,190
Titanium	2,770
Lead	3,180
Chromium	4,160
Fluorspar	4,750
Barytes	5,800
Zinc	7,400
Talcum	7,600
Manganese	8,500
Feldspar	8,600
Bentonite	9,600
Magnesite	12,000
Copper	12,300
Peat	26,000
Potash	26,000
Aluminium	28,000
Terra alba	38,000
Phosphate	45,000
Sulfur	56,000
Gypsum	102,000
Salt	190,000
• ndustoal sands	300,000
Clay	500,000
Iron	563,000
Lignite	848,000
Natural Gas (Oil Equivalent)	2,121,300
Crude Oil	3,578,000
Hard Coal	3,740,000
Quarry Stone	4,100,000
Sand & Gravel	15,000,000
TOTAL	*31,356 538*

Figure I World mining production, 1998 (WellmerA Wagner, 2000)

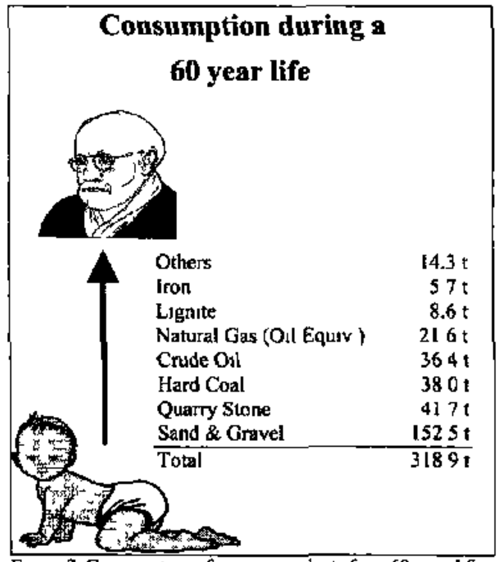

Fıgure 2 Consumptıon of mınıng products for a 60 year lıfe expectancy

The figures for global mining production only" show the tonnage of real mining products. The total world mass moved by mining is 17.8 km^3 of rock per year (Figure 4). This is four times the amount moved by rivers before the influence of mankind (Neumann-Mahlkau, 1996). A rock volume of 17 8 km^3 would be sufficient to cover the total land area of the Netherlands, which has an extension of 37,333 km^2, to a depth of 0.5 m. Thus, mining is expected to be by far the biggest mass mover in the world.

World Agriculture, Fishery and Forestry Production 1998 (10001)	
Cereals	2,079,928
Roots & tubers	648,132
Sugar crops & sweeteners	1,515,473
Pulses	56,123
Nuts	6,767
Oil-beanng crops	478,749
Vegetables	549,838
Fruits	510,217
Feedstuffs	3,144,275
Beverage crops & spices	18,139
Vegetable Fibres	18,435
Livestock products	777,300
Fishery products	113,100
Forestry products*	3,521,000
TOTAL	13,437,477

** Wood calculated at a density of 0 8 t/m'*

Figure 3 World agriculture, fishery and forestry production (FAOSTAT database)

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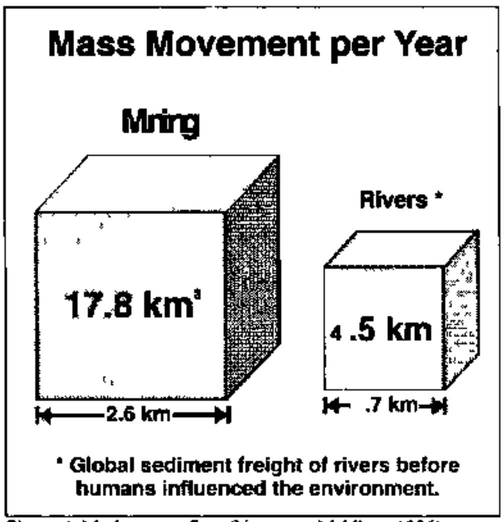


Figure 4. Mining mass flow (Neumann-Mahlkau, 1996).

So as to give an idea of the economic impact of world mining, the contribution of mining to the world gross domestic product (GDP) can be estimated. The total value of global mine production in 1998 was about US $1,000 billion. Compared with the world GDP in 1998 of a little less than US $30,000 billion, this means the contribution of mining is roughly 3% of world GDP.

Another interesting aspect of mining is its position in world trade (Figure 5). The total value of world exports of mining products in 1999 was US $556 billion or around 10% of total world exports. The export value of fuels was US $401 billion or 7% of total world exports (WTO, 2001).

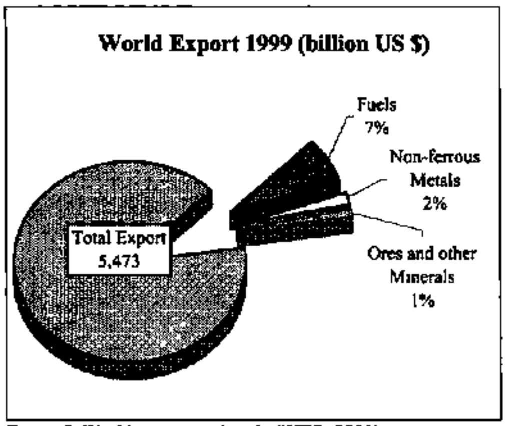


Figure 5. World mining and trade (WTO, 2001).

In this respect, the first impression of mining is confirmed. Mining produces incredible mass flows in connection with fundamental impacts on nature. The economic significance of mining related to its contribution to the GDP and world trade is rather small. So, what is the benefit of mining and what exactly does it signify for mankind?

One of the omnipresent outputs of mining is energy. The world production of primary energy in 1995 was 363.04 quadrillion Btu. This energy was provided almost entirely by coal (25%), oil (40%), natural gas (21%) and uranium (6%). Only 8% was supplied by renewable energy sources (Figure 7).

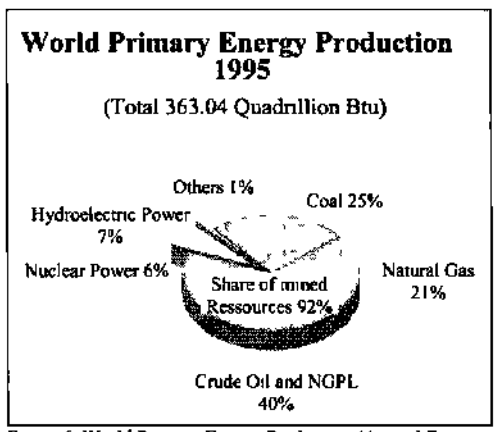


Figure 6. World Primary Energy Production (Annual Energy Review, 1996)

More than 92% of the world's primary energy production is based on mineral fuels. In other words, if mining and mineral fuels were to be taken away, immediately, 9 out of every 10 lights would go out, 9 out of every 10 cars would not run, 9 out of every 10 heaters would stop working, and so on.

All of these figures can only give a rough impression of the importance of mining for society. The best way to show how essential mining is for society is to have a look at everyday life and the things surrounding us. For instance, let's have a look at die car. Figure 7 shows how many mining products are needed to build a single car. All together, around 5,000 kg of ore and other mining products are required to produce a car, and this is only one example. Nothing that surrounds us would be possible without mining. In other words, mining products are a part of almost everything.

Coming back to society's attitude towards mining, which is often focused exclusively on the negative impacts on the natural environment, It must be said that mining transfers natural resources from nature to society, and therefore mining is impossible without minimum impact on nature. On the other hand, it ıs the aim of this article to clearly point out that we all need mining products to satisfy our needs. An additional aspect is the fact that unlike traffic, agriculture or urban development, the impact of mining on nature is often confined to a limited period of time. Reclamation often takes place after mining operations, restoring nature to its original state.

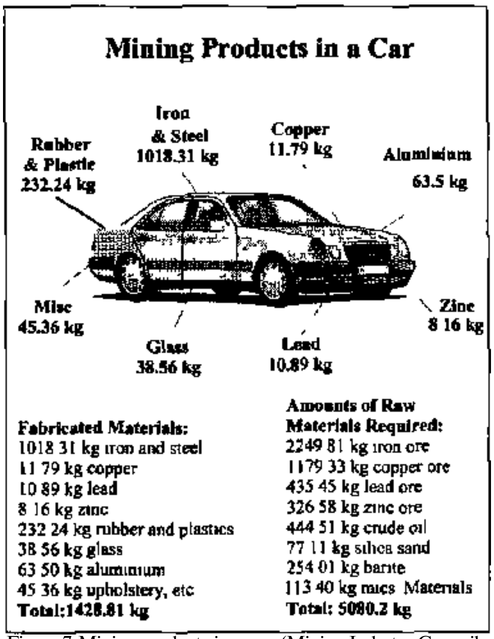


Figure 7 Mining products in a car (Mining Industry Council Missouri, 99)

At this point, it should also be mentioned that nowhere else is the depletion of natural resources more omnipresent than in mining, and that nobody else is more aware of the limitation of natural resources than a miner. However, a critical note is that the mining industry often regards natural resources simply from the aspect of the deposit itself.

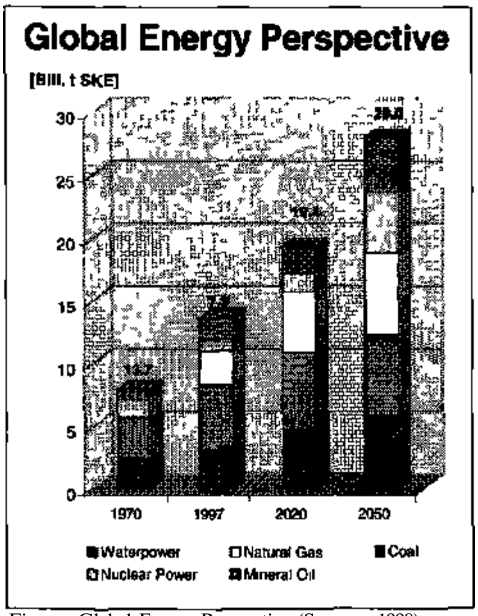
3 WORLD MINING - FUTURE DEVELOPMENTS

AU of this shows that mining is an important foundation of our present society. But that does not justify everything the mining industry does. Mining now and in the future has to take place in an ecologically, economically and socially justifiable way. Miners and non-miners have to come together to realise that aim. The goal must be to handle limited natural resources m a responsible manner Recycling and the use of renewable energy sources must be encouraged So, let us now take a closer look at these two aspects m the future.

The question of future developments m the world energy supply was discussed at the 17th World Energy Congress in Houston in 1998 (Semrau, 1998). With respect to the importance of primary energy supplies from mimng, the congress came to the following results: the restructuring of our energy system towards the use of regenerative energy supplies will be possible m the long term, approximately into the second half of the next century During the next decades, however, primary energy supplies from mining will shll dominate in meeting the increasing demand for energy, especially in developing countries and in threshold countries. For this purpose, however, efficient and ecofhendly technology for energy supply will be required.

A scenario of four steps was drafted

1 Improvement of technologies for the current energy system, i.e., technologies for fossil energy carriers until approximately the year 2015.
2. Development of new technologies to supply current conventional energies until approximately the year 2050.
3 Additional use of new energy systems on a small scale until approximately the year 2050
4 Use of regenerative energy sources on a large scale, from approximately the year 2050 onwards.



Figures Global Energy Perspective (Semrau, 1998) .

The development of world energy consumption could, for instance, look like a scenario in the study "Global Energy Perspectives", which was submitted by the International Institute for Applied Systems Analysis (IIASA) and the World Energy Council (Figure 8).
It can be said that at least until the year 2050, and probably longer, mining will have an outstanding role in the world energy supply.

The question of the future significance of mining for supplying mineral resources may be clarified by taking a look at the metals sector. In 1992, the consumption of the non-ferrous metals aluminium, copper, zinc, lead and tin amounted to around 46 Mio. t in the western world (Figure 9). The proportion of secondary resources in total resource consumption varied between 18 % for zinc and 52 % for lead. This means that a considerable part of the consumption *of* these metals is already covered by recycled materials. Incidentally, this is also true of natural aggregates, where hardly any improvement can be expected in recycling rates, at least in highly populated areas.

Consumption of Base Metals in the Western World 1992

	Overall-consumption (Mt)	Used material in smelters, refineries and other products (Mt)	Used material in %
Aluminium	22.24	6.27	28.20
Copper	12.26	4.60	37.50
Zinc	6.47	1.54	23.80
Lead	4.69	2.43	52.00
Tin	0.20	0.04	17.90
Total	**45.86**	**14.87**	**32.40**

Figure 9. Consumption of Base Metals in the Western World (Metalstatistics, 1993).

However, it remains doubtful whether there will be a further increase in the proportion of recycling during the next decades due to the growing demand for resources. Recycling as a source of resources has its technical, economical and also ecological limits, since a recycling quota of 100 % would only be possible with high inputs of energy and other resources, so that precisely from the point of view of resource-saving, a recycling quota of 100 % is not desirable. Consequently, mining will remain an important, maybe even the most important, supplier for the worldwide demand in resources in the long run. This is confirmed by a look at the statistical lifespans of some resources (Figure 10) (Wellmer, 1998).

Statistical lifespans could indeed give the impression that the world's important resources, such as petroleum, lead and zinc, will be exhausted sometime in the next decades. However, this is not the case, because if one considers the development of the statistical lifetime of supposedly scarce resources over the years, it can be seen that this has remained constant for decades. For instance, in 1955 the statistical lifetime of zinc was around 25 years, yet there are still no signs of a scarcity of this resource. On the one hand, this is due to the fact that new deposits are still being discovered today. On the other hand, technical progress enables mining to exploit deposits, the exploitation of which would have been unthinkable some years ago. The quality of mining operations will continue to be improved in the future in every respect so that they may meet growing requirements.

Statistical Lifespan of Mineral Resources

raw material	unit	world resources 1996	world output 1996	statistical lifespan (years)
bauxite	1000 t	22983000	114000	202
lead	1000 t	63400	2912	22
copper	1000 t	311500	11006	28
zinc	1000 t	143200	7283	20
tin	1000 t	7190	196	37
crude oil	10^6 t	147700	3379	44
petroleum gas	10^9 m^3	149000	2330	64
hard coal	10^6 t	637000	3806	167
brown coal	10^6 t	221000	883	250
uranium	10^3 t	3220	36.2	89

Figure 10. Statistical lifespans of different mineral resources (Welimer, 1998).

Consequently, two things can be said about the future significance of mimng. One is that until the second half of the next century the world energy supply will depend largely on primary energy sources obtained by mining. The other is mat in the future, like today, the demand for mineral resources will essentially be met by mining.

4 CONCLUSIONS

The development of mankind and its standard of living have always essentially depended on the availability of mining products. Technical progress in the past and today is unthinkable without the raw materials provided by mining. Miners have always been aware of the fact that their work is an intervention in nature and that they remove non-renewable resources. As a mining university, we emphasise this responsibility and teach environment- and resource-friendly methods.

REFERENCES

Annual Energy Review 1996. *Energy Information Administration. July 1997.*

FAOSTAT Database. *Food and Agriculture Organisation of the United Nations, 2001.*

International Trade Statistics 2000. *World Trade Organisation (WTO), 2001.*

Metal statistics 1982-1992, 80th Edition, *Published m* Germany *by: Metallgesellschaft AG, Frankfurt* am *Main, 1993.*

Mining Industry Council of Missouri: Role of Mining in Society. http://Yrwvi.momic.cam/sociery.htm. *March 1999.*

Neumann-Mahikau, P.: Anthropogener Massen transport - em geologischer Faktor[1]? *Schriftenreihe Deutsche Geologische Gesellschaß No.l: Geologische Stqffkreisläufe und ihre Veränderung durch den Menschen. Hannover: 1996.*

Semrau, Gerhard: 17. Weltenergiekongreß 1998 in Houston. *Gluckauf134 (1998). pp. 648-654.*

Wellmer, F.-W.: Lebensdauer und Verfügbarkeit energetischer und mineralischer Rohstoffe. *Enmetall 51 (1998) No. W. pp. 663-675.*

Wellmer, F.-W. & Slein, V.- Mögliche Ziele nachhaltiger Entwicklungen bei mineralischen Rohstoffen. *Erzmetall 51 (1998) No. I, pp. 27-38.*

Wellmer, F.-W. & Wagner, M: Roh Stofftrends am Beginn des *l.* Jahrtausends. *Enmetall 53 {2000) No. 10, pp 569-582.*

1 f* International Mining Congress and Exhibition of Turkey- İMCET2001, © 2001, ISBN 975-395-417-4

Safety and Welfare of Mine Employees in Australian Black Coal Mines

K.B.Cram
Project Manager Workplace Environment, Joint Coal Board.Corrimal, Australia

N.I.Aziz
Associate Professor in Mining Engineering, Faculty of Engineering, University ofWollongong, Australia

ABSTRACT: This paper outlines the status of Australian coal mining industry with respect to safety and welfare of the mine employee. The impact of tie longer shift hours and compressed working week are discussed in relation to workers safety and employment levels. Longer shift hours and compressed works have shown to be a benefit to miners, both in safety and socially. The paper also examines the role of each of government organisations, the mining and manufacturing industries on the issue and goes on to descnbe the various environmental control measures introduced to the Australian coalmines to ensure high safety standards are maintained. Dust monitoring and control, noise pollution control, and diesel particulate control measures have been targeted vigorously and as a result there has a continual drop in the coal mine related diseases as well as a decline in the workers lost time injury claims.

1 INTRODUCTION

Worldwide, there has been an increasing awareness of the health and safety of mine employees İn the mining industry. This emphasis is now recognised and keenly pursued in Australia by all parties concerned including, the mining companies, mining machinery manufacturers, the miners union and, above all, the government organisations responsible for the safety and welfare of mine employees such as the Joint Coal Board and the Department of Mineral Resources in NSW and similar organizations in other states in the Australian federation.

Increased mechanisation, mining thicker seams, increased production, improvement in productivity, the continuing decline in the workforce and the high costs of legal litigations have provided a sound platform for all the parties concerned to deal effectively with the safety and welfare of the mine employees.

It is not uncommon in mines today to see the mining companies rules and guidelines appearing alongside the rules laid down by legislators of a particular state. This commitment to mine safety was further reinforced in a recent Joint Coal Safety Forum held at Wollongong, on 12 July, 2000, where the key representative from the coal mining industry, the mine workers union met both the NSW Minister of Mineral Resources and the Queensland Minster for Mines and Energy. Key issues discussed at the forum were regulatory reform, work practices, training and education, compliance and enforcement strategies, investigation procedures and protocols. Both Minsters agreed the forum provided the opportunity for both states to identify key safety issues and greatly assist the practical implementation of better safety practices.

The recent trend on the workforce downsizing, because of the increased efficiency of the modern mining machinery and technologies, would likely to dnve the miner away from the profession, with the ultimate consequences of the future shortages of the skilled miners. Thus, it is necessary to provide a climate that would maintain the skilled workforce in the industry by recognising the side effects of heavy mechanisation and increased production. Issues to be addressed include increased mine dust levels, increased diesel emissions, increased noise levels, possible mcreases in work related injuries of high severity, and worker's compensation.

In Australia the safety and welfare of the mine employee are protected by various laws and agreements under the powers constituted by the Australian Commonwealth and the states, and through various organisations such as The Joint Coal Board of NSW. This paper examines various aspects of mining operations which has a direct impact of the miners welfare and safety. Dust control, diesel emission, management practices, miners welfare and the role of governmental bodies such as the Joint coal board is discussed together with the consequences of increased work hours and compressed working week. The compressed working weeks is defined as reducing the number of days worked per week as a result of increased shift hours and adherence to the agreed total hours worked per week.

2 ROLE OF GOVERNMENT AND INDUSTRY ORGANISATIONS

In each state of the Australian Commonwealth there are a number of government organizations which are engaged in a variety of issues related to the mine workers safety and welfare. These range from setting legislations and implementation to research. These bodies include;

Government:

- Departments of Mines and Energy (Queensland)
- Safety in Mines Testing and Research Services- STMT ARS (Qld)
  Department of Mineral Resources (New south Wales),
- Joint Coal Board (NSW)
- Departments of Mineralsand Energy (WA)
- Department for Industry, Science, and Resources (Australia)

Industry:

- Queensland Mining Council
  New south Wales Mineral Council
  Western Australian Chamber of Mines
  Minerals Council of Australia
  Australian Coal Association Research Program (ACARP)

In the State of NSW the role and responsibilities of two such organizations are as follows:

2.1 *The Joint Coal Board*

In NSW the Joint Coal Board is constituted under an arrangement between the Governor- General of Australia and the Governor of the State of New South Wales made pursuant to the provisions of the Coal Industry Act 1946 (Commonwealth) and the Coal Industry Act 1946 (NSW).

The powers and functions of the board are stated in identical provisions of the Commonwealth Act, Sections 23 to 27 and the new South Wales Act Sections 24 to 28 and are as follows (JCB annual report 1999-2000):

- To provide occupational health and rehabilitation services for workers engaged in the coal industry including providing preventive medical services, monitoring workers* health and investigating related health matters,
- To collect, collate and disseminate accidents and other statistics related to die health and welfare of workers engaged in the industry,
- To refer matters related to the safety of workers engaged in the coal industry, as it thinks fit, to the Chief Inspector of Coal Mines or the Commonwealth Minister and the State Minister for consideration,
- To provide courses in the production and utilisation of coal under international development assistance programs sponsored or administered by the Commonwealth Government or approved by the Commonwealth Minister and the State Minister,
- To report to the Commonwealth Minister and the State Minister as it thinks fit, or when requested by either Minister, on matters related to the health or welfare of workers engaged in the coal mining industry, or on any other matter concerning or arising out of the Board's powers of functions,
- To publish reports or information of public interest concerning or arising out of the Board's powers and functions, and
- To promote the welfare of workers and former workers in the coal industry in the State, their dependants, and communities in coal mining areas.

Until such time as the Commonwealth Ministers and the State Minister direct, the Board has the following powers and functions:

- To monitor, promote and specify adequate training standards relating to health and safety for workers engaged in the coal industry;
- To monitor dust in coal mines; and
- To collect, collate and disseminate statistics related to the coal industry, other than statistics related to the health and welfare of workers.

The Australian Government has acknowledged the activities of the Joint Coal Board as being a State of NSW function. A draft bill by 'the NSW Government was released in 2000 detailing a new statutory corporation, comprising the Joint Coal Board, NSW Mines Rescue and NSW Coal Superannuation. This new organization İs expected to be enacted in 2001.

2.2 *NSW Department of Mineral Resources (DMR)*

The Occupational Health and Safety Act 1983 (OH&S Act) is the primary legislation for the regulation of occupational health and safety in all NSW industry, including mining. The OH&S Act 2000 is to be introduced in 2001, which is a restructure of the 1983 Act but containing the same duties.

Mining engineering, including mines inspection, environment and related issues are brought together in the Mine Safety & Environment Division. Coal mine safety is administered by the DMR through the OH&S Act and the Coal Mines Regulation Act 1982 (CMRA). New regulations were introduced İn September 1999. Coal Mines (General) Regulation

1999, the Coal Mines (Underground) Regulation 1999 and the Coal Mines (OpenCut) Regulation 1999. An additional regulation, the Coal Mines (Investigation) Regulation 1999 was also created to provide support for changes to the CMRA, relating to investigations and inspections.

The DMR is undertaking a review of the 1982 CMRA with an aim of providing the most effective regulatory framework. The new legislative framework is planned to have a mix of the revised Act(s), regulations, codes of practice and guidelines. It is proposed that the coal mining industry will operate under safety management systems and major hazard management plans. The DMR proposes monitoring industry through inspections and audits, and investigations and reviews. To improve industry safety practice, the DMR may issue reports, provide education, issue notices and, where appropriate, prosecute.

The DMR recognises that the workforce has a right to be involved in decisions which impact on health and safety in the workplace. At present, the generally accepted ways through which the workforce plays a part in decision making are OH&S committees, Check Inspectors (Workforce Representatives), and İn some instances consultation on the development of the systems under the new regulations. The mining industry must recognise that workforce involvement needs to expand where decisions affecting health and safety are concerned.

3 INJURIES AND FATALITIES

Table 1 shows coal production, employment, lost time injuries, claims and number of mines in NSW between 1990 and 2000. As can be seen from Table 1 and Figures 1 ,2, and 3, that there is a 40% increase in production, a 36 % reduction İn employment, 35% drop in workers compensation claim lodgement and 76% drop of in lost time injury frequency rate. Clearly, there is a general improvement in the industries overall operation both in productivity gains and in safety at the expense of reduced manpower.

4 EFFECT OF SHIFT WORK ON SAFETY PERFORMANCE

A recent study conducted by Cliff, Beach and Leveritt (2000)have shown that increasing working shift hours to 12 hours in conjunction with compressed work weeks can provide a significant benefit to the worker. Figure 4 shows the mean LTIFR for 8, 10 and 12-hour shift lengths. Currently the 12-hour shift is the most common shift length in Queensland mining industry as shown in Table 2. Cliff, Beach and Leveritt (2000) have also indicated that the shift length alone does-not provide the real answer to improvement to increased safety. Other factors such as ratio of days on to days off, number of shifts worked in succession, start time of the shift and shift rotation onto nights, can have a significant effect on safety performances as shown in Table 2. The average hours worked per compressed week was 40 hours. Other benefits of the compressed working week is die opportunity of the employee to spend longer quality time with family, which in rural locations can have significant impact on family life stability and cohesion.

Table 1. Production, employment, lost time injuries, claims and number of mine in NSW between 1990 and 2000 (JCB annual report 1999-2000).

Year Ended 30"[1] June	Utiils	1990	1991	1992	1993	1994	J 995	1996	1997	1998	1999	2000
Raw coal production	ml	93.9	96 7	101.2	102.9	1019	107.8	113 1	123.6	134	1314	132 9
Employment No	No	17200	17000	16600	15100	14700	14300	14473	14793	13522	11064	10150
Workers Compo Claim Lodged	No	7979	7510	6616	5903	5690	6231	5533	5095	4651	3552	3285
Lost Time Injury Fret). Rate	No	168	143	108	78	68	73	5fi	48	52	39	34
Severity rate	Days	N/A	2546	1878	1210	1128	1332	1050	999	1215	1079	954
Lost days/ employee due to WC	Days	S.3	4.7	4	34	3.6	4.1	3.7	3.3	3.2	3.5	3.3
Fatalities	No	1	9	7	4	1	2	2	6	2	3	3
No of mines	No	70	72	70	67	68	69	72	68	66	64	58

Fatalities: Those injuries which result in the death of the worker
Incident rate: The number of injuries occurring for each 100 persons employed.
Lost-time injury (LT1): A work injury which results in the loss of a day (7 hours) or more off work.
No of LT1 : Where the injured worker has less than 7 hours as a result of the injury. Includes medical expense claims.
Disease claims: where the worker has contracted or aggravated a disease in the course of employment and to which the employment was a contributing factor.
Occupational Disease: Diseases contracted or aggravated in the courses of employment and to which the employment was a contributing factor.

5 ENVIRONMENTAL ISSUES

5.1 *Dust Control and Monitoring*

The system of dust monitoring in mines is the responsibility of each state. In NSW, the Joint Coal Board carries out dust monitoring in all the coal mines, and in Queensland it is done by Department of Mines & Energy andSIMTARS.

In 1954 the NSW Joint Coal established an expert advisory body, the Standing Committee on Dust Research and Control, which comprised representative of the colliery proprietors, mining unions, government departments and Joint Coal Board medical and technical personnel. The main roles of the committee are to:

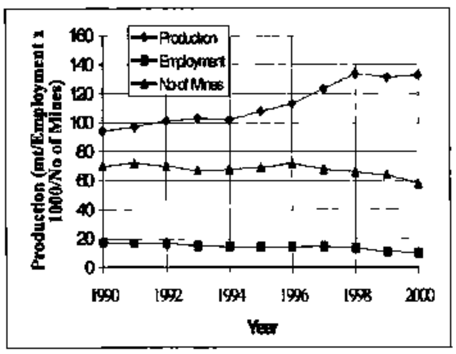


Figure 1 Production, mployment and number of mines m NSW between 1990 and 2000

- Monitoring dust sampling results,
- Evaluating dust hazards
- Encouraging improvement to dust control methods, and
- Disseminating information and educate mine personnel

The Standing Dust Committee meets bi-monthly and conducts the majority of its meetings at mine sites. Monitoring of the dust in the coal industry is carried out using two approved types of instruments. These include, the DuPont P2500, a constant flow sampler combined with a Cassella 25 mm cyclone sampling head for routine personal sampling tests and Hund Instantaneous Dust Monitor for general and mine environment dust concentration monitoring.

The specified limit for respirable dust in die underground mine environment is 3rng of respirable dust (i.e., < 5 microns, (im) per *m?* of air sampled. The specified limit for quartz containing dust is 0.15 mg of respirable quartz per m^ of air sampled. Normally five people are sampled from the production crew on all production shifts for continuous miners once per year; and on all production shifts for longwalls twice per year. All other locations in the mine, monitoring is carried out once per year. These include, other underground workers (eg drillers, transport, etc) and surface washaries, loading and crushing plants, and open cut mines (Cram and Glover, Feb. 1997, and Glover and Cram Dec. 1997, and Cram June 2000).

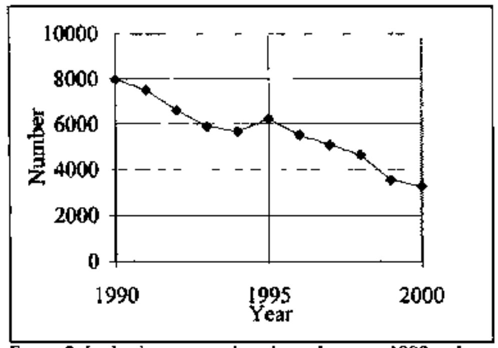


Figure 2. Lodged compensation claims between 1990 and 2000

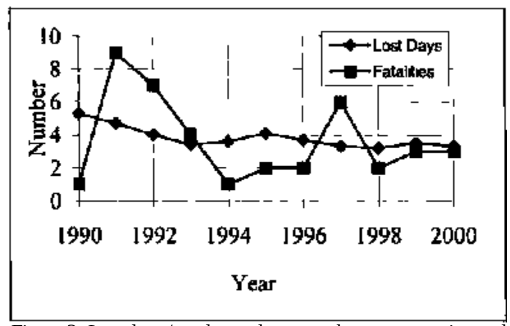


Figure 3. Lost days /employee due to worker compensation and fatalities between 1990 and 2000.

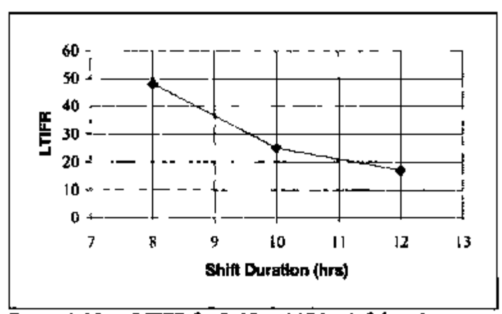


Figure 4 Mean LTIFR for 8, 10 and 12 hr shift length.

In NSW sample collection commences at the time of leaving the crib room at die start of the shift and ceases on arrival at the crib room at the end of the shift. The sampling period is normally not less than 5 hours. The integrity of results is guaranteed by the Board's dust samplers who are present in the workplace during the sampling shift recording such information as ventilation quantities, blocked sprays, operator location, water pressure or anything which may affect the results. The sample monitoring results are forwarded to the mine manager, and senior government inspector, district check inspector and included in the Joint Coal Board data base. If the results exceed the specified limit a resample İs generally taken within 7 days or when changes are made to reduce the dust exposure. Table 3 shows the results of dust monitoring surveys that have been conducted in NSW mines since March 1984. Up to May 2000, and after over 15 years of sampling, over 38 000 personal dust samples, excluding re-samples, have been collected from over 8 000 mining locations. Sampling locations were 30 % underground longwall faces, 65% other underground operation (mainly continuous miner operations in non longwalling mines and longwall roadway development), and 5 % open cuts and coal preparation plants. Out of the 10 800 (30% of total) samples monitored from longwall faces, 6.28% exceeded the limit, where only 1.4 % of 24 900 (65% of total) exceeded the limit from non-longwalling operations and only 0.7 % of 2 400 (5% of total) tests exceeded the limit from opencut /coal preparation plants. The results show that the area of main concern remains to be at the longwall mining operation.

Table 2. Frequency of Rosters with 8,10 and 12 hour shift length.

Shift length (hours)	rosters	Percent	Valtd %	Cum %
8	54	24 3	24 3	24 3
10	31	16 8	16 8	41 1
12	109	58 9	58.9	100
Total	185	100	100	

- Monitoring dust sampling results,
- Evaluating dust hazards
- Encouraging improvement to dust control methods, and
- Disseminating information and educate mine personnel

Figure 5 shows the dust monitoring survey along the longwall face by longwall occupation between 1984 and May 2 000. The percentages exceeding the limit of 3 mg/m3 is also shown in the graph. A significant improvement has been achieved in the % exceeding the limit. Peaked at over 18%. Post 1990 reduction in dust concentration was attributed by various initiatives by coal mining companies. These initiatives included the methods of coal cutting, better water sprays, the adoptions of good ventilation practices and the introduction of effective dust suppression practices.

Table 3. Respirable Dust Results (Excluding re-samples) 1984-May 2000 (Source)

Mining Methods	Number of Personnel Samples (excl Re-samples)	Number >3mg/m3	% Exceeding Limit
Longwall Faces	10 800	679	6 3
Other Underground	24 900	356	1.4
Open Cut / Washenes	2400	16	0.7

It should be noted that two significant changes has occurred in the past 15 years. Firstly, the number of longwall faces had doubled and the average daily longwall face output increased from 4 000 tonnes to over 8 000 tonnes / day.

5.2 *Occupational Noise*

Noise Induced Hearing Loss (NIHL) is the most prevalent compensable industrial disease in Australia and entails substantial economic costs. It is believed Ürat there is substantial under-reporting of this disease and that the data on compensation for NIHL therefore represents only a proportion of the actual problem Table 4 shows the number of claims for NIHL in NSW (Joint Coal Board Occupational Disease Statistics 1999/2000).

During 1995/1996 the NSW Government introduced a hearing loss threshold, below which no compensation is payable. As a result, there was a significant drop in claims lodged. Deafness claims account for most claims. In 1999-00 of the 300 occupational disease claims lodged, 184 or 61.3% were for deafness (NSW Joint Coal Board, Statistics 1999/00). Excessive exposure to noise have been found to contribute to increased absenteeism, lower performance and possible contribution to accidents, which can be considered as an additional and unrecognisable cost. NIHL is irreversible and leads to communication difficulties, impairment of interpersonal relationships, social isolation and a significant degradation in the quality of life for the employee.

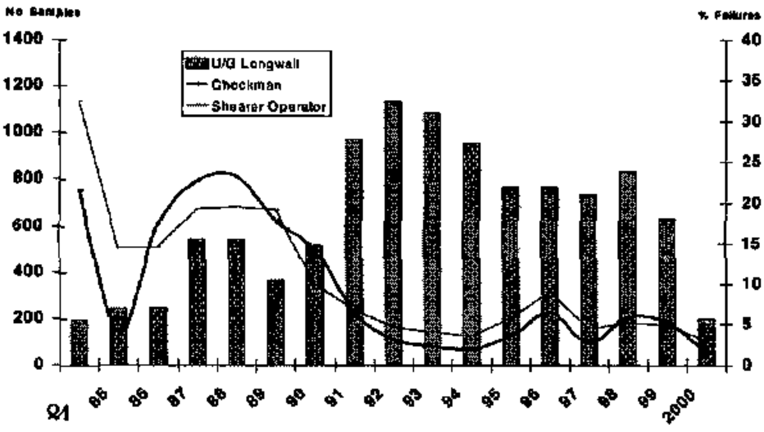


Figure 5 Dust monitoring survey along the longwall face by longwall occupation between 1984 and May 2000

Table 4. NSW Coal Industy-Number of claims.

Year (Ending June)	NSW Coal Industry Occupational Disease Deafness Claims
1991/92	541
1992/93	652
1993/94	806
1994/95	714
1995/96	514
1996/97	211
1997/98	259
1998/99	215
1999/00	184

The national standard for exposure to noise in the occupational environment is an average daily exposure level of 85dB(A) over an 8 hour period. The exposure to noise is taken to be that measured at the employee's ear position without taking into account any protection which may be offered by personal hearing protection.

In May 1996, the Work Cover Authority of NSW issued a Code of Practice-Noise Management and Protection of Hearing at Work. The Code of Practice applies to all places of work other than mines within the meaning of the Coal Mines Regulation Act 1982 and the Mines Inspection Act 1901 and to all persons in those workplaces with potential for exposure to excessive noise. The Code of Practice, which commenced May 1997, provides practical guidance on compliance with the Occupational Health and Safety (noise) Regulations 1996.

Since 1985, the Joint Coal Board has collected over 1 000 individual personal samples of noise levels and noise exposure in most NSW underground mines, selected open coal mines and coal preparation plants. The study showed that over 80% of the individual personal samples exceeded an average daily exposure level of 85 dB(A weighting).

5. 3 *Diesel Particulate*

The demand for increased power and increased mobility in recent years has seen a surge in the use of diesel engines in Australian coal mines. Presently, there are more than 3 000 diesel powered machines in Australian coal mines and İs increasing in a similar fashion as has been in the USA coal mines as reported (MSHA 1997).

While the diesel engine provides the workplace with an efficient and reliable source of power, İt İs normally noisy, smoky and can produce an unpleasant odour. More importantly though, is the concern over the possible health effects of using diesel equipment in confined space, such as in an underground mine.

Control of diesel exhaust levels in NSW and Queensland is partly determined by monitoring the gaseous components such as the NO, and CO in raw exhaust as per Coal Mines Regulation Act 1982 (CMRA 1982).

The American Conference of Governmental Industrial Hygienists (ACGIH) has recommended 0.15 mg/m3 as a workplace standard threshold limit for DP exposure. In general, many non-mining workplaces in which diesel equipment is used, the level of DP falls well below the recommended ACGIH exposure standard. In contrast, studies show that the DP levels in the mining environment can be significantly higher than exposures in the ambient air or in other workplaces as shown in Figure 6 (Davies, 1997)

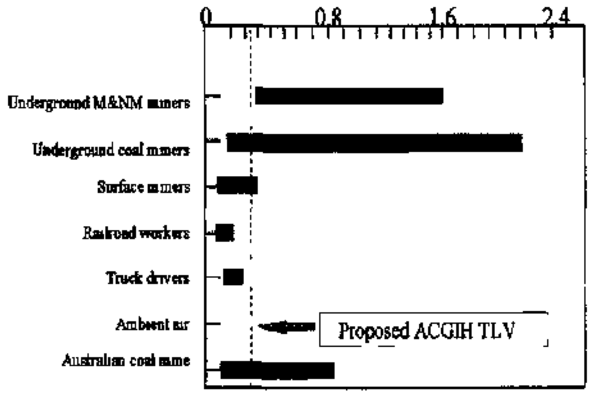


Figure 6 Personal DP exposure in mg/m^3 from USA mines and Australian coal mines (Davies, 1997)

Since 1990, there have been a significant increase in the concern of the effect of DP on the health of the mine employees, particularly in underground operations. Personal monitonng from the field survey conducted at nine underground mines in Australia by Pratt et al (1995) has mdicated that the exposure of the workforce ranges between 0.1 and 2.2 mg/m3 of DP, depending on job type and mining operation A detailed breakdown of the exposures from all nine underground mines are shown in Table 5. All samples collected were on a full-shift personal basis with a minimum of 4 hours sampling duration. A total of 134 personal samples were collected at the mines.

Currently mere is no engine emission standard for DP and no occupational exposure standards are available in Australian coal mining operations However, the ACGIH released notifications of a proposed diesel exhaust standard of 0.15 mg/m^3.

Table 5 DP exposure rates in nine mines (After Pratt et al, 1995)

Mine	Fuel type	Samples	DP exposure mg/m3
A	Standard	12	014-0 56
B	Standard	17	0 15-0 31
C	Low S	24	0 03-0 17
D	Standard	23	0 06-047*
E	Low S	22	0.04-165*
F	Standard	13	0 13-032
G	Standard	17	0 06-0 62
H	Low S	16	0 10-0 25

*(LW Move in progress requiring heavy equipment)

Table 6 shows a detailed breakdown of the exposures of various machine operators for each mine and the CMRA specified ventilation requirement per KW engine capacity, bearing in mind that the Act requires that, to ensure protection of the workforce, each piece of diesel equipment is required to operate with a minimum fresh air ventilation of 0 06 m3 /sec/kW of power to provide adequate dilution of exhaust gases.

Table 6 Diesel particulate exposure rates by different mine machine operators

Machine Operator	Engine Power (KW)	Vent Rqd/KW	DAP mg/m3
Shearer Transporter	84 00	50	I 7
Chock transporter	11200	67	0 3 - 0 70
Eimco	75.00	45	0 15-0 30
MPV	68 30	41	0 15-0 25
PJB Power tram	50 00	30	05 -02
Domino	42 00	25	0 05 - 0 1
Wagner	J12	67	0 2 - 0 6
Gardner	48 5	29	01-06
Myne Bus	65 0	39	03

The research work conducted by Pratt et al (1997) added significantly to the knowledge of the extent of employee exposure to DP and to me methods by which exposure can be controlled. The study has shown that the traditional control strategies such as engine and scrubber tank maintenance, regular gaseous emission testing, and the requirement of minimum ventilation rates provide considerable control of workforce exposure The study also showed that the use of correct fuel quality such as low sulphur fuel, regular engine tuning, and engine decoking provides further significant overall reductions in DP exposures and creates a more under heavy load conditions, additional controls in the form of disposable filters fitted to the exhaust outlet appear to provide the most effective control at this stage. Figure 7 shows the effect of sulphur in diesel fuel on engine emissions respectively as reported by Pratt et al (1997).

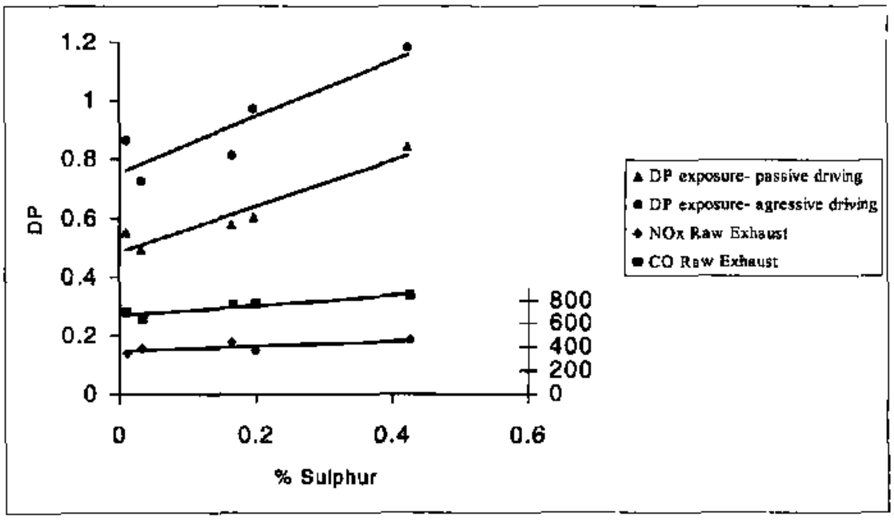


Figure 7. The effect of sulphur in diesel fuel on engine emissions.

6 CONCLUSIONS

The combined efforts of the industry, governments and miners union In Australia has seen a dramatic improvement in the health and wellbeing of coal mine personnel. Longer shift hours and compressed working weeks have shown benefits both to the employer and employee. The various safety measures outlined in the paper have clearly yielded benefits. Initiatives in workplace environment improvements, such as dust, noise and diesel emission control, are all positive step in the right direction in improving the health of the mine employee because of the hostile environment of the workplace. As a result, there has been 40% improvement in production at 36% reduction in manpower between 1990 and 2000. At the same time there was a drop of 35% in worker compensation, and 76% drop in LTIFR. Clearly, the continuing decline in the workforce employments may, in the long run, prove not to be in the best interest of the Australian coal industry to maintain the skilled workforce. For the industry to move forward and confront successfully the difficult challenges of die technological advancement and market economy, greater attention must be paid on better training of its skilled workforce. Trained mine personnel is an invaluable asset which no mine manager can afford to ignore.

REFERENCES

NSW and Queensland Governments Target Coal Mine safety, Coal Industry News, The Aus. IMM Bulletin, August / September 2000, p 41.

Cliff, D., Beach, R. and Leveritt, S., (2000), Safety performance related to shiftwork in the Queensland mining industry, Queensland Mining Council Report, April, 96p.

Coal Mines Regulation Act 1982 No 67 and associated Regulations.

Cram, K.., (2000), New south Wales resperiable dust levels in lingwalls. Paper presented at Standing Dust Committee, Belmont, NSW June, 7p

Cram, K., and Glover, D., (1997), Studies on respirable dust in coal mines, NSW, Australia, in proceedings, 27 International Conference on Safety in Mines Research Institutes, 20-22 February, New Delhi, India, pp 605-614.

Davies, B, (1997), Präsentation to NSW Joint Coal Board Sub-committee Diesel Particulate, Dec. 7P.

Glover, D., and Cram, K,.(1997), Respirable airborne dust exposure levels in the New South Wales coal mining industry, the second International Conference on the health of miners, in Applied Occupational and Environmental Hygiene, Vol 12, No.12, Dec., pp 980-987.

Joint Coal Board Annual Report 1999-2000.

Joint Coal Board Occupational Disease Statistics 1999-2000.

Mines Safety and Health Administration (MSHA) Publications^ 1997), Practical ways to reduce exposure to diesel exhaust in Mining-a toolbox, March.

Pratt, S.L, Grainger, A.P., Todd, J., Meena, G. G., Rogers, A.J., and Davis, B.,(1997), Evaluation and control of employee exposure to diesel particulate at several Australian coal Mines, the second International Conference on the health of miners, in Applied Occupational and Environmental Hygiene, Vol 12, No. 12, December, pp 1032-1040.