Economic benefits gained by rock mechanics: Three case studies

J.F.T. Agapito

Significant economic benefits can result when rock mechanics is applied within a practical framework and integrated within the other engineering functions of the mine organization. Such application and integration requires management support and understanding of rock mechanics as a practical tool to help attain high standards of safety, productivity and resource recovery. For such an approach, a long-term geotechnical program is needed in most operations to build an adequate data base and to ensure that design and ground control issues are handled in a cost-effective manner.

Best usage of rock mechanics in mining occurs when management recognizes the application as a long-term investment, where time is needed to realize good returns. The need for this approach is due to the difficulty in accurately predicting ground conditions ahead of mining.

The practical application of rock mechanics must follow an iterative process of design and verification based on site-specific experience to account for geologic and mining impacts. Ground conditions experienced during mining must be used to check the initial mine design and, if necessary, to provide a basis for design modification. Such a practical rock mechanics program includes comparative experience, in situ instrumentation and numerical modeling. The program begins with the geologic site investigation and continues throughout the life of the mine. It is an integral part of mine planning and operation.

This article presents a review of three case studies, where the long-term rock mechanics investment played a key role in the economic success of the operations. In each case, economic and rock mechanics issues are stated first, followed by a summary and brief discussion. Although the economic gains realized by these practical applications of rock mechanics were not computed, it is hoped that the economic benefits will be obvious from the results obtained.

Thethree case studies present the following examples:

- Wall stabilization of a large underground ore bin by tensioned, fully grouted bolts.
- Mining with total recovery and no subsidence by multi-lift, sublevel bench-and-fill stoping.
- Stability improvement in longwall gate roads by a two-entry system with yield pillars.

Case study 1 — Stabilization of an ore bin

Economic issue: Major delay of mine start-up if failure of ore bin were to occur.

Rock mechanics issue: Determination of adequacy of planned support after unexpected poor rock quality was encountered.

In 1982, at Molycorp's block caving molybdenum mine in northern New Mexico, the excavation of the underground ore bin encountered very poor rock quality in faulted and altered intrusive rocks. Despite use of substantial concrete and steel support, large wall movement caused long vertical cracks up to 6 mm (0.2 in.) wide in the concrete, buckling of steel sets and convergence deformation at rates of 150 mm (6 in.) per year. To control the rapid wall movement and stabilize the bin, 6-, 12- and 21-m-long (19-, 39- and 69-ft) tensioned, grouted rock bolts were installed. This allowed stability to be achieved without impacting the mine production schedule.

The use of long, grouted bolts to stabilize the bin proved practical and was accomplished in a cost-effective way. Alternatives such as heavier concrete or steel support would have been more expensive and would have affected the bin operation.

Discussion

Constructed at a depth of 810 m (2657 ft), the ore bin was located at the eastern end of the haulage level outside the caving area. A detailed description of its construction and rock engineering was given by Bratton and Agapito (1985).

The ore bin is T-shaped, stepping down to a rectangular cross section to accommodate the dumping mechanism (Fig. 1). The major dimensions are 16 m long x 4.7 m (52.5 x 15 ft) wide at the top, stepping down to 8.5 x 4.7 m (28 x 15 ft) at the lower part. The vertical depth is 24.5 m (80 ft), giving a live capacity of 1.27 kt (1400 st).

Design support consisted of 1.8-m (6-ft) split-set rock bolts and chain link mesh installed immediately after mining. Installation of W6x20 steel sets at 1.8-m (6-ft) vertical intervals followed. A double lattice of No. 8 reinforcing bars with spacings of 0.3 m (1 ft) was placed between the sets, and the space to the rock wall was filled with 20 MPa (2900 psi) minimum strength concrete to a minimum thickness of 0.8 m (2.6 ft).

Early geologic analysis indicated that the ore bin would be constructed in good-quality, granitic rock; but as mine development progressed toward the bin area, it became apparent that the bin would be located instead in the lower-rock-quality intrusives. This rock was characterized by clay alteration and numerous small discontinuous faults filled with low-friction pyrite, clay and molybdenum. Several continuous

J.F.T. Agapito, member SME, is president, J.F.T. Agapito & Associates Inc., 715 Horizon Drive, Suite 340, Grand Junction, CO 81506.

gouge-filled faults and dripping water were also found.

The low rock quality encountered in the dump drift before the bin excavation prompted a re-evaluation of the design. A time series of cross-sectional profiles was analyzed by the finite element method to duplicate the excavation sequence. The results are shown in Fig. 2 as initial and final failure zones. A lateral zone of failure of up to 12.8 m (42 ft) was predicted after the bin was fully excavated.

It should be noted that a failure predicted from this analysis does not necessarily imply collapse. This illustrates a difficulty sometimes encountered in modeling. What do the results mean in practical terms?

At the time, it was believed that the planned support would probably be adequate to maintain stability if a large failure zone were to develop. It was also believed that the analysis results were a little pessimistic with regard to the size of the failure zone. However, the construction plan was changed to keep the permanent support as close to the working face as possible, and an instrumentation program was employed to provide warning of any trouble and to permit rapid response. Stability in the area was monitored by close visual inspections and by both tape and borehole extensometers.

The bin was excavated from the top down with permanent support for each lift installed before the next lift was mined. When the ore bin excavation was about one-third complete, with the fourth lift, 6 m (19 ft) below the track level, completed, signs of excessive rock movement began to appear. These signs included continuous vertical cracks in the concrete up to 6 mm wide (0.2 in.), a broken weld joint in the steel, buckling of W6x20 steel crossbraces, and convergence measurements at rates of 150 mm/a (6 in. per year).

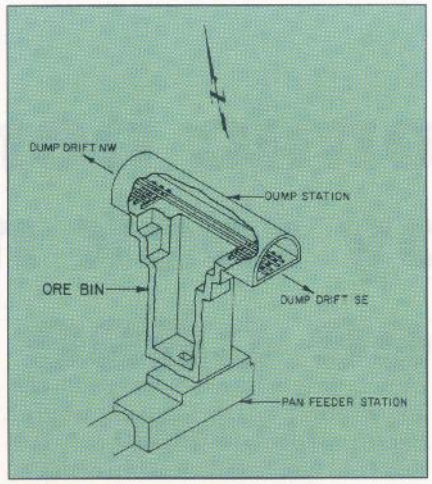

Fig. 1 — Ore bin.

To control the rapid wall movement, temporary 200- x 200-mm (8- x 8-in.) timber stulls were installed. Then a program of ground control analysis, monitoring and repair was begun. Reinforcement consisted of fourteen 12-m-long (39 ft) sectional threadbar bolts per side for the top four lifts, giving a total of 112 bolts. On the lower lifts, as yet unexcavated, the pattern would be a staggered array of five 6-m (19 ft) bolts on each side and three bolts on each end. Specifications called for each bolt to be anchored in resin, tensioned to 89 kN (20,000 lbf) and grouted full length.

Typical wall closure in the upper portion of the bin is shown in Fig. 3. Five months after the installation of the bolts, wall convergence had decreased significantly but still continued at a rate of 19 mm/a (0.7 in. per year). This long-term rate was judged too high for such a vital excavation. A second set of longer bolts was installed in the top four lifts to control movement and assure anchoring beyond the potential failure zone. This set consisted of fourteen 21-m (69-ft) tensioned grout bolts per lift, seven on each side. Three months after the installation of the 21-m (69-ft) bolts, convergence rates indicated stability. The monitoring program ended with the beginning of production and the filling of the ore bin.

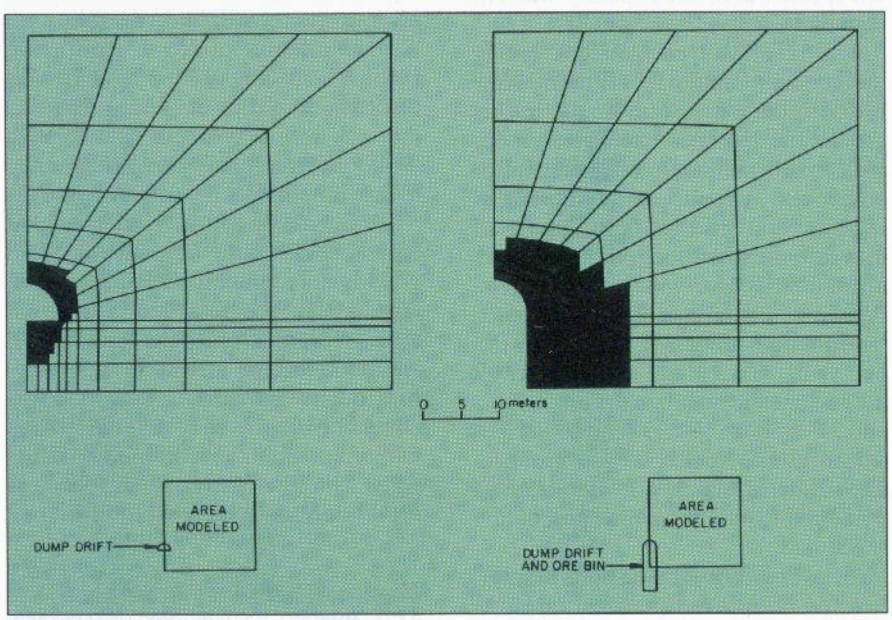

Fig. 2 — Results of finite element analysis showing extent of rock failure before and after excavation.

Case study 2—Development of a mining method

Economic issue: Development of profitable mining operations in a complex geologic rock mass within a city limit and monitoring of operations.

Rock mechanics issue: Balance of high extraction and productivity goals with a no-subsidence requirement.

The role of rock mechanics was central to the development of the mining method at Asamera's Cannon Mine, Wenatchee, WA. A multi-lift, sublevel bench-and-fill method was adopted to extract gold ores at depths of 60 to 200 m (197 to 656 ft). High strength backfill was used to provide long-term overburden support and safe working conditions during pillar recovery operations.

After geotechnical characterization, stope dimensions and layouts were evaluated by numerical modeling to predict global mine stability and determine backfill strength requirements. Numerical modeling was calibrated and refined against the ground conditions experienced, and mine personnel were trained in its use. This allowed practical and inexpensive use of numerical modeling as a routine planning tool.

The economic benefit gained by using rock mechanics to develop the mining method and establish the stope extraction sequence is reflected in the production costs and high ore reserve recovery achieved to date.

Discussion

The Cannon Mine is a joint venture between Asamera Minerals (US), Inc. (the operator) and Breakwater Resources Inc. Its several gold ore bodies occur within mineralization in the northwest-trending Chiwaukum Graben, a struc-

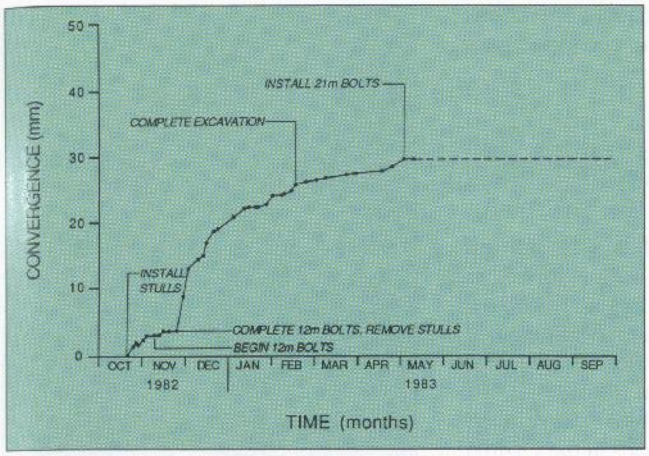

Fig. 3 — Typical wall closure in upper portion of the ore bin.

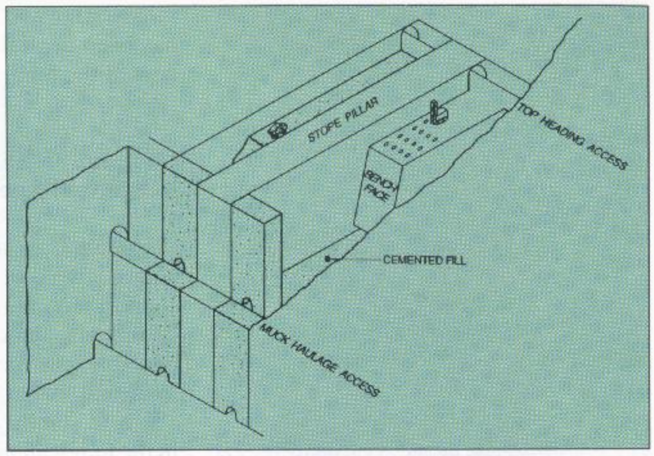

Fig. 4 — Multi-lift, sublevel bench-and-fill stoping.

ture that dominates the geology of central Washington.

Mining was initiated in the B-North Zone, a lens of silicified arkosic sandstones and siltstone of the Eocene Swauk Formation. Silicification is associated with the gold mineralization and has produced higher rock strengths in the ore zone than in the the surrounding country rock. The unsilicified rock consists of clay-altered sandstones and siltstone interbedded with very weak layers of sheared claystone.

A multi-lift, sublevel bench-and-fill mining method was selected and is illustrated in Fig. 4. This method was selected because it allowed high productivity and extraction while maintaining full overburden support by the use of cemented backfill. Previous publications have described the method (Kelly, 1986), the geotechnical design (Brechtel et al., 1987), and the backfill system (Baz-Dresch, 1987).

Stope and pillar widths of 7.3 m (24 ft) were used between 15- to 25-m-high (49- to 82-ft) levels (Fig. 4). Headings 4.6 m (15 ft) high were driven the full stope width and connected by a slot raise. The stope was then bench mined and backfilled with cemented fill with a laboratory strength of 8.3 MPa (1204 psi). Multi-lift overhead mining was done by working off cemented backfill with a maximum of four lifts in some areas. The pillars were mined in a similar manner except that to provide for better roof stability, the in-pillar drifts were driven 3.7 m (12 ft) wide and then slashed to full pillar width. The pillar stopes were backfilled with uncemented rock fill.

Geologic mapping during stope development showed that folded, sheared beds would form adversely oriented, wedge-like blocks. Subsequent analysis indicated a high probability of pillar failure as these structures were exposed by the full stope excavation. At this point, the effect of pillar failure on the global mine stability was evaluated by modeling the entire B-North Ore Zone. Results of the analysis indicated that failure would be localized and would not affect global stability.

Fig. 5 shows a cross section of vertical stresses with stress transfer and expected pillar failure. Experience to date has confirmed that pillar failures were localized and confined to claystone sheared areas.

A high backfill strength was selected to ensure long-term overburden support and stope wall stability. It was assumed that the large width-to-depth ratio and weak overburden strata would eventually cause full overburden load on the cemented-fill pillars. In addition, multi-lift mining of the pillars required the bond between the backfill and rock to be strong enough to support large undercut rock blocks. These blocks would be repeatedly shaken by blasting. Finally, some of the development access had to be driven through cemented backfill pillars because of the periphery shear zones.

Numerical modeling was used to assess the required backfill strength. An evaluation of overburden and arching loads, as well as shear stresses between fill and rock, was made by a series of analyses. The results indicated that a laboratory strength of 8.3 MPa (1204 psi) would provide a sufficient safety factor to account for variations in mixture control and stope placement segregation.

Results of the backfill program have been very positive to date both in terms

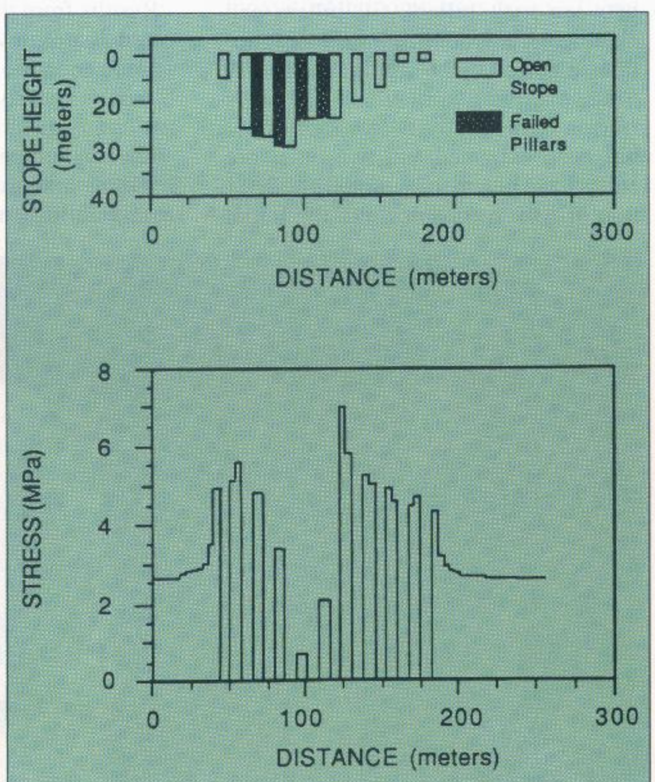

Fig. 5 — Cross section with vertical stress magnitudes showing transfer of stress due to predicted pillar failure.

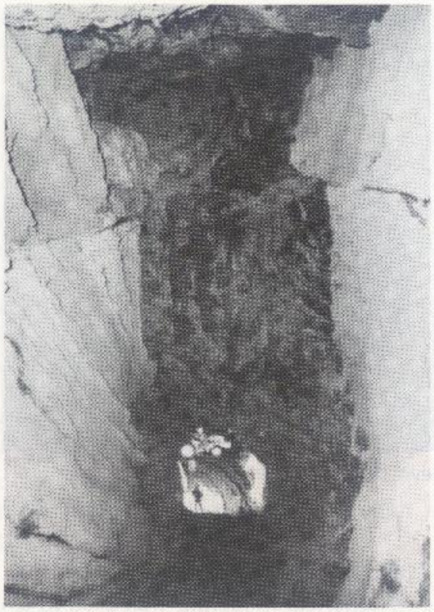

Fig. 6 — Backfill walls in stope D48.

of overburden support and stope wall stability. Fig. 6 shows a pillar being mined between two backfilled stopes and gives a general idea of the backfill stability.

An instrumentation program has been used to monitor stability and provide data for design verification. This program consists of stress and displacement measurements in both the rock and backfill and has been especially successful in tracking the transfer of load between rock pillars, fill pillars and abutments, thus confirming the assumptions made as to the arching capability of the overlying formations.

To date, most of the B-North Ore Zone has been extracted and two other ore zones have begun to be mined. Integration of rock mechanics into the initial mine design and subsequent operations has been central to the economic success of the mine.

Case study 3—Stability improvement in longwall gate roads

Economic issue: Delays in start-up and operation of longwalls with heavy support requirements in gate roads.

Rock mechanics issue: Maintenance of stable gate roads under adverse geologic and loading conditions.

Severe stability problems developed in the longwall gate roads at the Cyprus Star Point No. 1 Mine of Plateau Mining Co. near Price, UT. These problems consisted of excessive floor heave, roof falls and rib spalling. Floor heave was especially severe, causing tilting of cribs and significant reduction of roof support. Roof falls occurred, particularly near sandstone channels, due to crib tilt and high stresses.

The rock mechanics work consisted of a comparative evaluation of several gate road layouts and has been described by Maleki et al., 1986 and 1987. The program was based on in situ instrumentation and numerical analysis. Test mining was also performed to help evaluate the structural behavior of yield pillars before they were used in the gate roads.

Results from the above program indicated that a layout with a two-entry system and a yield pillar would provide the best stability for multi-seam longwall mining.

The two-entry system was adopted, resulting in significant improvements in roof stability for more than 5 km (3 miles) of two-entry development completed to date. Considerable economic benefits were gained by a reduction in production delays and ground support problems.

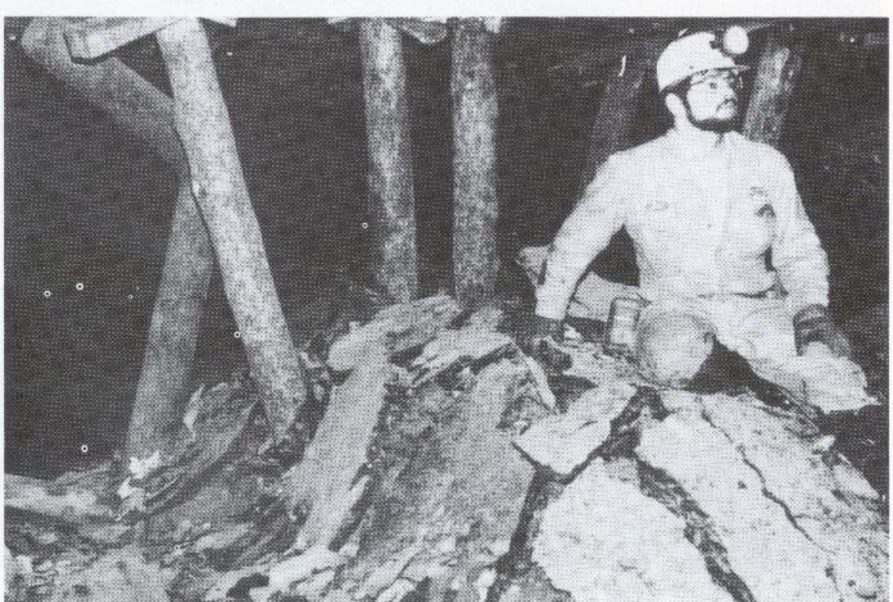

Fig. 7 — Typical floor heave behind the longwall face — Eighth left, room two.

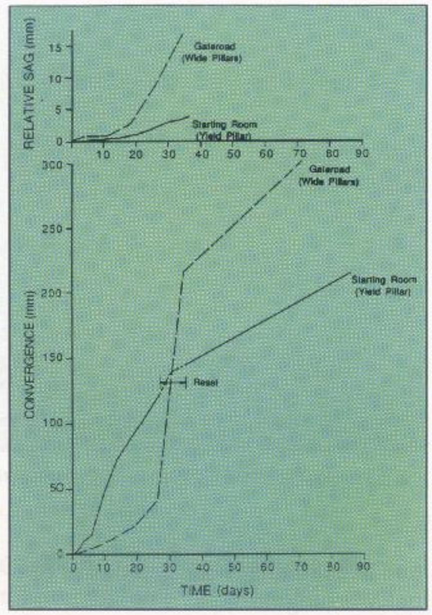

Fig. 8 — Comparison of entry convergence and roof sag for the starting room and gate roads.

Discussion

The Star Point No. 1 Mine works a three-seam coal deposit. Seam thickness varies from 1.5 to 3.6 m (5 to 12 ft). Numerous sandstone channels of large horizontal extent are often encountered during gate road development. At the time of this writing, longwall mining was being performed in the two upper seams, which are separated by 9 to 17 m (30 to 56 ft) of interburden.

Severe floor heave began to occur in the gate roads behind the face at depths of 300 to 500 m (984 to 1640 ft) (Fig. 7). This occurred in the original three-entry layout, consisting of 15.2-m-wide x 24.4-m-long (50- x 80-ft) pillars and 5.5-m (18-ft) spans. The floor heave tilted the cribs and greatly reduced roof support. Roof falls occurred mostly by sandstone channel margins due to a combination of weak strata, high stresses and poor roof support. In addition, stress transfer to lower seams, capable of causing future stability problems, was very likely due to thin interburdens.

Reducing the number of entries and using yield pillars was an approach that offered the possibility of improving ground problems. This was because of two major factors:

• A change from a three- to a two-entry system significantly reduced the excavated area. Stability in areas with ad-

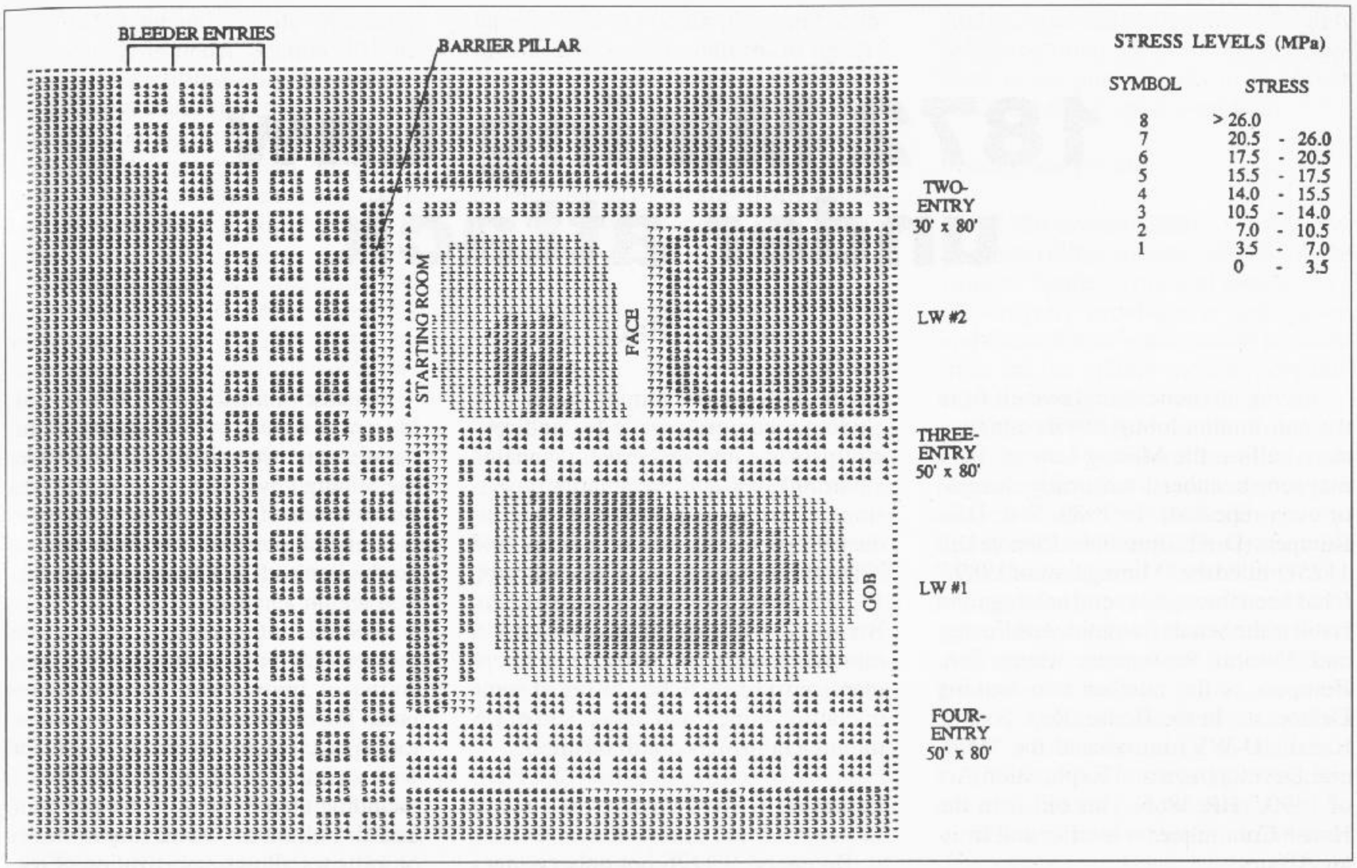

Fig. 9 — Vertical stress levels for two longwalls and bleeder entries.

verse geological stress conditions is obviously related to the amount of ground excavated, mostly in the intersections where the roof spans are wider.

• Yield pillars reduce the stresses in the gate roads and thus floor heave and the stresses transmitted to the lower seams.

A yield pillar test was conducted in the starting room pillar of a longwall because of inexperience with yield pillars. Results showed a five-fold reduction in floor heave and roof sag in the starting room as compared to the middle gate road entry (Fig. 8).

Numerical modeling also indicated a lowering of stresses with a two-entry yield pillar layout. Fig. 9 gives an example of one of the analyses performed. Vertical stresses are shown for an area covering two longwalls and four-, three- and two-entry gate roads.

A layout with a 9.1-m-wide by 24.4-m-long (30- x 80-ft) pillars with 5.5-m-wide (18-ft) spans was adopted in 1985. Experience to date shows that floor heave and roof falls have been greatly reduced, resulting in improved coal recovery and productivity.

The economic benefits are very obvious both in terms of improved safety and reduced mining cost.

Conclusions

Practical rock mechanics applied within a framework of site specific experience can produce considerable economic benefits to mine operations.

Best results are obtained when management views rock mechanics with a long-term perspective and integrates rock mechanics activity with the mine geologic and engineering functions. This requires trained personnel and a program directed to optimizing the structural design and the solution of ground control problems.

Reasonable, adequate tools—instrumentation and numerical modeling—have been in existence for some years. They have, however, been generally under-used by the mining industry. With the advent of microcomputers, the trend should be for more mines to develop and improve their geotechnical capabilities.

The case studies reviewed here have demonstrated that rock mechanics can help mine operations gain significant economic benefits. ♦

Acknowledgments

The author acknowledges contributions of the management and engineering staff of the companies involved here. The results obtained from the various programs have been due to a long-term team effort between these companies and our consulting firm.

References

Agapito, J.F.T., and Hunter, R.V., 1989, "Rock mechanics program helps improve trona recovery at Stauffer's Big Island Mine in Wyoming," Mining Engineering, January, pp. 24-28.

Baz-Dresch, J., 1987, "High strength backfilling at the Cannon Mine," Proceedings, Third Western Regional Conference on Precious Metals, Coal and Environment, Rapid City, SD, SME-AIME.

Bratton, G.P., and Agapito, J.F.T., 1985, "Construction and geologic engineering of an underground ore bin," Proceedings. RETC Conference, New York, pp. 177-191.

Brechtel, C.E., et al., 1987, "Mine design at the Cannon Mine: Integration of operational planning and geomechanical design," First International Conference on Gold Mining, Vancouver, BC, Canada.

Kelly, T.R., 1986, "Stoping practices at the Cannon Mine," SME preprint 88-332, SME Annual Meeting, Phoenix, AZ, Jan. 25-28.

Maleki, H.N., et al., 1987, "Longwall design improvements through a geotechnical program," American Mining Congress Coal Convention. Cincinnati, OH.

Maleki, H.N., et al., 1986, "Gate road layout design for two-seam longwall mining," International Journal of Mining and Geological Engineering. Vol. 4, pp. 11-127.