Innovative use of SMART cable bolt data through numerical back analysis for interpretation of post failure rock mass properties

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ABSTRACT Ground support is a significant component of underground mining costs. The rehabilitation of failed support, however, is often much more expensive than the initial installation. This paper investigates whether the right type and length of support can be designed correctly the first time, through forward modelling of the support and the mining process. Back analyses demonstrate how high-quality instrumentation and monitoring can help with the estimation of the post-peak (post-failure) rock mass parameters required for a comprehensive analysis. A hybrid modelling approach combines a three-dimensional mine-wide boundary element model with a non-linear finite element model to allow for the failure process in the rock mass. The back analyses demonstrate that, for the case study site, the post-peak Generalized Hoek-Brown value of $m_r$ must be reduced significantly in order to properly simulate displacements observed in instrumentation data. High-quality instrumentation data is the key to post-peak rock mass parameter calibration. The procedures outlined in the paper highlight the need for a new hybrid numerical modelling package that can use the influence of three-dimensional mining-induced stresses to predict displacements in, and potential failure of, rock masses surrounding mining infrastructure in order to radically alter strategic and tactical mine design.

KEYWORDS Post-peak rock mass parameters, Back analysis, Predictive modelling, Instrumentation data, SMART cable bolts, Mine-induced stresses, mine-wide modelling

INTRODUCTION

A significant component of underground mining costs stems from ground support in the form of rock bolts, mesh and screen, cable bolts, and shotcrete. Should any of this support approach or reach failure (stripped or broken cable bolts, severe bagging of screen, large numbers of broken rock bolts, etc.), the support needs to be rehabilitated. This rehabilitation is often much more expensive than the initial support installation, due not just to the cost of the support, but to loss of access, down-time of that area of the mine, potential for injuries, the slow nature of the work, and possible lost mining revenues. The question then becomes: can support design be made smarter? That is, can the initial design of mining support be done to a level such that the right type and length of support is installed the first time around, hence greatly reducing overall direct mining costs.

Most ground support elements are installed as dowels – un-tensioned tendons (e.g. passive cables, rebar, etc.). The issue here is that support is only provided when sufficient dilation or movement of the rock mass has occurred to generate resisting tensile forces in the support element. Significant stress-driven movement only occurs in a rock mass after the peak strength of the rock has been exceeded and it is well into a state of yield or failure. In other words, based on experience, stress-driven ground movements are not sufficient to load the support to any significant amount prior to yield of the rock mass. In DeGraaf, Hyett, Lausch, Bawden, and Yao (1999), instrumentation data indicated that displacements seen in instrumented cable bolts were not significant up to model-predicted failure and, hence, the cables never took any measurable load to that point. In other cases, support has been loaded to failure, as shown by broken cables. In such cases, in the authors’ experience, numerical models always indicate that elastic stresses in these regions significantly exceeded the peak values allowed by the constitutive model that was applied.

In order to improve underground mining support behaviour and reduce costs associated with over-design and rehabilitation due to under-design, there needs to be better modelling capability of support behaviour prior to design and installation. Hence, the need exists to model the rock mass behaviour well past the point of failure, into the post-peak regime, not just the typical elastic stress modelling that often occurs in mining today (Crowder and Bawden, 2004). Most importantly, models need to be able to calculate
displacements that occur due to failure with a high degree of confidence (something generally not possible today), as displacement up to the point of failure is almost negligible. To accomplish better modelling, improved instrumentation data is required for calibration of rock mass properties, particularly post-failure or post-peak parameters. In order to accomplish this, field case studies having adequate numbers of high-quality instruments and frequency of readings are required.

This paper examines the current state of modelling and instrumentation in mining. An assessment of some typical problems with how instrumentation is monitored and used, as well as how good data can be quite valuable, are shown. An overview of typical mine modelling procedures will be given, followed by a look at what types of models are required and what input is necessary for predictive forward modelling. A case study then demonstrates, through back analysis, how high-quality instrumentation and monitoring can help with the estimation of the post-peak (post-failure) rock mass parameters required for a comprehensive analysis.

THE ROCK MECHANICS PROBLEM

Rock mechanics is now a reasonably mature field of study, many aspects of which are well understood. To this end, the modelling capability for mining applications is also quite mature and sophisticated. In fact, the complexity and capability of models often significantly exceeds the ability of mining, rock mechanics, or geotechnical engineers to generate reliable rock mass parameters, particularly in the post peak regime (after failure). That is, once a rock mass has reached failure, it can still maintain some post peak or residual strength. If rock mass parameters for post-failure conditions were well developed for given types of rock and stress conditions, then explicit forward modelling of rock mass support could be performed.

The ability to accurately predict the post-peak parameters of in situ rock masses is very much an art at this time. Engineers often rely on the suggestions of colleagues and empirical rules of thumb. There has been considerable debate as to how to alter the post-peak parameters of rock masses, as was discussed among several industry leaders and disseminated by Crowder and Bawden (2004). In that paper, the Generalized Hoek Brown failure criterion (GHB; Hoek, Carranza-Torres, and Corkum, 2002) was exclusively discussed as it is currently the most often used criterion for underground rock mechanics studies. The consensus was that the Geological Strength Index (GSI; Hoek, Kaiser, and Bawden, 1995) and the Disturbance factor (D; Hoek et al., 2002) are index parameters that are used to determine reasonable estimates of the peak strength parameters and should not enter into the determination of post-peak strength. Similarly, the unconfined compressive strength of intact rock samples, UCS or $\sigma_u$, and Hoek-Brown parameter $m_1$ are both properties of intact specimens determined from laboratory testing and, hence, should not be altered for determination of post-peak rock mass parameters. Thus, the consensus was that to effectively model the behaviour of a rock mass, any numerical model should have the option to enter peak and post-peak (elastic and plastic) values for $m_1$, $s$, and $\alpha$.

Although a consensus was reached, there still was considerable debate among these industry leaders, inferring that much research remains to be completed toward the reliable estimation of post-peak rock mass parameters. One methodology that can be utilized to look at this problem is discussed later in this paper.

PRESENT STATE OF MODELLING AND INSTRUMENTATION IN MINING

Most mines today utilize numerical modelling as part of their strategic and/or tactical mine design efforts. These models, normally linear elastic, are often not up to date and modelling is often done only in reaction to a problem. There are some instances in which non-linear models are used, but the input parameters for the post-peak characteristics of these rock masses are generally guessed at, or some rule of thumb is followed. The most common form of analysis is to examine the results of stresses calculated by the models. If the model is elastic, then the model may calculate stresses that are much higher than a rock mass can sustain, as determined by the chosen failure criterion. The strength factor, the ratio of the maximum stress allowed by the failure criterion to the current state of stress (eg. a strength factor less than one can be interpreted as ‘failed’), is one of the key results examined. The region of strength factor that shows ‘failure’—often referred to as the yield zone—may be used as a basis for how deep rock mass support should extend and, to a lesser degree, how much reinforcement to add. However, this yield zone is not necessarily the best representation of where failure of the rock mass occurs. Using a non-linear model with perfectly-plastic post-peak parameters, the extent of the failure zone may be quite similar to what is predicted by the yield zone from elastic models. However, under strongly strain softening conditions, non-linear models can exhibit different shapes and extents of failure zones than those from elastic models (Crowder, Coulson, and Bawden, 2006). Furthermore, displacements from elastic models are not considered valid and are hence generally not used.

However, accurate estimates of displacements are exactly what is required for support design. Mining openings are often designed to be only as large as necessary for equipment and services because any larger openings cost more and are more difficult to support. Excessive deformations can destroy support that is costly to rehabilitate, may cause falls of ground resulting in delays or injuries, and ultimately may impact equipment access.

Most mines use instrumentation in critical infrastructure, such as haulage drifts, crusher stations, intersections, etc., to monitor the displacement of the backs
and/or walls, support loads, etc. These instruments, however, are often installed long after the initial access excavation has been created—and often just soon before the mining front progresses through a particular area. The trouble with this approach is that the total accumulation of displacements and hence damage cannot be known to any degree of certainty. As well, data from the instruments are generally collected by hand, from time-to-time and entered into a database. The frequency of readings depends on whether or not a problem has been spotted, by how much time the engineer or technician has to collect data or by whether the mining front is approaching. The best way to collect data, however, is on a regular schedule, whether or not movement is anticipated. The other problem with instrumentation data today is that it is analysed only if there is a problem (i.e. reactive engineering). In summary, if instrumentation is to be used to aid in the design process at a mine for prediction purposes, high-quality and consistent data are required.

DERIVING PARAMETERS FOR MODELLING FROM INSTRUMENTATION

Research at the University of Toronto has looked into the problem of what post-peak rock mass parameters should be used to reasonably represent the rock mass. This was accomplished using quality instrumentation data from one case study mine, in a back analysis mode using numerical modelling. The details of the methodology and the verification can be found in Crowder and Bawden (2005). A short summary of the method follows.

Three-dimensional mine-wide elastic stresses have been obtained at the case study mine for incremental mining steps over several years using the elastic boundary element numerical analysis program Examine<sup>3D</sup>. Because this type of model can only represent one time step, several models were created to represent relevant time steps in the mine’s history, corresponding to certain times where the instruments were recording data. For each mining step (i.e. each Examine<sup>3D</sup> model), the stress state over a two-dimensional slice through the three-dimensional model at the location of an appropriate instrument was exported into the two-dimensional finite element analysis program, Phase<sup>2</sup>. This program allows for the rock mass to fail (i.e. it is non-linear) when stresses exceed the chosen failure criterion. Also, unlike Examine<sup>3D</sup>, it allows for the integration of rock mass support (such as cable bolts, rock bolts and shotcrete) explicitly in the model. After the model was created, the peak rock mass parameters were defined using the Generalized Hoek Brown (GHB) failure criterion, and the post-peak rock mass parameters were adjusted, through a parametric back analysis, until displacements above the back closely approximated those observed from the field instrumentation. Refer to Crowder and Bawden (2004) for a discussion of the importance of post-peak parameters and some guidelines on related rules of thumb.

The case study involves the Williams Mine, which is located near Marathon in the Hemlo region of northern Ontario (Fig. 1). The orebody dips to the north at about 70 degrees and ranges in thickness from 3 to 50 m over the full strike length of the orebody (~1.5 km). The majority of infrastructure development occurs in the footwall, which is for the most part composed of quartz-eye muscovite schist (Bawden, Dennison, and Lausch, 2000). More details of the ore extraction methods, of drift support, and other information about the mine can be found in Crowder and Bawden (2005) and LeBlanc and Murdoch (2000).

The Williams Mine primarily uses two types of instrumentation to monitor vertical movement in the backs of mining drifts. The Stretch Measurement for Assessment of Reinforcement Tension (SMART) cable bolts are instrumented to accurately assess the deformations that occur in the cable support elements (Bawden et al., 2000). The Multi-Point Borehole Extensometer (MPBX) is a flexible borehole extensometer that passively measures the deformation of the rock mass itself. That is, it provides no reinforcement in contrast to the SMART cable bolt. Both instruments are manufactured by Mine Design Technologies Inc. in Kingston, Ontario.

The instrumentation database for the Williams Mine, at the time of this study, had just over 200 SMART cable bolts and MPBXs combined. The case study described in this paper, however, only deals with two specific areas of the mine in which a total of about 95 instruments are located. Although there are seemingly plenty of instruments that could be used in the back analysis, only about 10 were really usable. A few of the instruments were installed in areas well away from the stopes and hence show little movement. Other instruments were installed after a majority of the mining in the area was already completed. Many of the instruments were installed at different times throughout the study period (1999–2003) and, hence, a complete record over time at a given loca-
tion did not exist. As well, other instruments exceeded the displacement capacity (of the instruments) and replacement instruments were installed after a time gap. One of the biggest problems with the instrumentation was an inadequate frequency of readings (for the purposes of this type of analysis). This is because all of the instruments are read by hand whenever mining personnel have the time. Hence, large gaps occur in the data, as shown in Figure 2. An example of one of the most reasonable, continuous data sets from an instrument at the Williams Mine is shown in Figure 3. All of these instrumentation problems cite the need for real-time acquisition of data from instrumentation. As well, instruments should be installed early on in the mining process, not just when the mining front passes a given area, so that a continuous, reliable set of data exist for monitoring and analysis. An example of a continuously monitored instrument that captures exactly when displacements occur in reaction to mining in the vicinity of the instrument is shown in Figure 4.

CASE STUDY ANALYSIS

This section details the results of the back analysis procedure in two separate locations at the Williams Mine. The first location is a large shrinking pillar of high-grade ore located in the upper east corner of the mine (Cells 1 and 2) at a depth ranging from about 400 to 600 m and is about 200 m in strike length. Mining has occurred above, below, to the west, and to the east of this area. The specific study area within this zone occurs on two levels, at about 550 m depth. The other area of interest occurs in what is known as the Sill Pillar (in Cell 5), which is about 80 m in height and was created as the mine originally extracted ore from two main blocks above and below this region. This pillar is located at an aver-
age depth of about 950 m. A long section of the mine showing the general locations of analysis is shown in Figure 5. All of the instruments in the study are located in the main haulage drifts that parallel the ore in the footwall rock unit. This unit is thought to be fairly consistent over the depth of the mine, except in the upper west regions of the mine, far from these study areas.

The determination of GHB parameters of the footwall rock mass at the Williams Mine has been well researched and discussed (Kazakidis, 1990; Crowder and Bawden, 2005) and need not be repeated here. The rock properties used are: unconstrained compressive strength (UCS or $\sigma_{uc}$) = 175 MPa; Poisson’s Ratio = 0.25; Geological Strength Index (GSI) = 60; and $m_1$ (GHB material constant for intact rock) = 10. Using these values, and using the appropriate GHB equations, the rock mass peak input parameters to Phase$^2$ are: Young’s Modulus = 17800 MPa; $m_b = 2.397$; $s = 0.0117$; and $a = 0.503$. The GHB parameters $m_b$, $s$, and $a$ are all unitless.

Because the estimated peak rock mass parameters were well established for this rock by previous studies, the task then became to iterate in a back analysis fashion to determine what appropriate post-peak parameters were necessary for the model to closely simulate the displacements observed in the field from the instrumentation. The parameters to vary in the post-peak are the (so-called) ‘residual’ values of $m_b$ and $s$, which are denoted by Phase$^2$ as $m_r$ and $s_r$, respectively. The dilation parameter must also be varied.

In the first case study area, Cells 1 and 2, three different instruments were examined using the back analysis procedure. A parametric study was used to independently adjust each of the three post-peak parameters from the peak value to a value approaching zero that still allowed for the model to compute, i.e. not to the point of numerical instability. A series of time steps were chosen such that nearby mining had caused significant mine-induced stress redistribution and, hence, progressive failure and displacement within the rock mass. The observations from the instrument at this location are shown at the chosen time steps, as a plot of vertical displacement of the back with distance into the back, i.e. above the drift (Fig. 6). Each line represents a different time step.

In order to most closely simulate the observed displacements, the value of $m_b$ was reduced to a post-peak $m_r = 0.24$, or 10% of the original value. The value of $s$ was reduced to a post-peak $s_r = 0.002$, or about 20% of the initial peak value. The dilation parameter was set to 0.4, or about one-sixth of the value of $m_b$ (Crowder and Bawden, 2004). When these parameters were applied, the resulting calculated displacements are shown for the same time steps as was used for the instrument. The modelling results overlaid on the instrument data are shown in Figure 7. It must be noted that any displacements that occurred in the model prior to the step corresponding to the installation of the instrument have been zeroed out (Crowder et al., 2006). Hence, the modelling results shown in Figure 7 can be directly compared to the instrument data for the same time intervals.

For the second location, the Cell 5 sill pillar, a back analysis was performed in a similar fashion to the results presented previously. Again, the post-peak parameters that best simulated the observed field displacements were $m_r = 0.24$, or 10% of the original value, and the value of $s$ was reduced to a post-peak $s_r = 0.0059$, or about 50% of the initial peak value. The modelling results overlaid on the instrument data are shown in Figure 8. The dilation parameter was set to 0.8, or about one-third of the value of $m_b$. The
The key result is that both areas of the mine require the GHB post-peak value of \( m_r \) to be reduced significantly from the peak value. This suggests that after failure, the rock can only sustain a small fraction of its original shear strength. The difference in values of \( s_r \) is not considered of great importance, as small changes in \( s \) do not affect the depth of failure, but increase (only slightly) the magnitude of displacement in the models (Crowder et al., 2006). Similarly, the dilation parameter does not greatly affect the depth of failure, but does influence the curvature of the displacement versus depth plots. The greatest influence on the displacements is controlled by \( m_r \). The influence of slight variations in post-peak parameters can be best explained by examining Figure 9. This figure shows the GHB strength envelopes in \( \sigma_{1r}-\sigma_{3r} \) stress space. The peak curve applies to both areas of analysis in the mine. The variation of post-peak parameters that seemed to best fit the field data are shown as the band of strength envelopes well below the peak curve. This range is quite tight considering that, although both areas of analysis are in the same footwall rock formation, there are some variations in this unit over the 500 m (vertical) that separate the areas (Fig. 5).

DISCUSSION

The results shown in the previous section indicate that the potential exists for numerical modelling to predict actual displacements and hence support loads, observed in the field with reasonable confidence. However, at this point, an analysis must be completed for each mine, as the results shown in this paper are very specific to the rock mass at the Williams Mine. In fact, the post-peak parameters determined in this study may not be the ultimate best estimate for the post-peak parameters for this rock mass. However, these values provide very reasonable results, given the inherent uncertainty in many of the input parameters and the use of a continuum model for a discontinuous rock mass. Most importantly, these results demonstrate that the numerical modelling and back analysis processes have been shown to be effective.

The procedures that were used in the process of this study are quite labour intensive and took several months to complete. With that said, if numerical tools could be developed to automate or improve the usability of this process, the modelling would not be a huge undertaking. As the models currently exist, they are easy to use and do an excellent job for the purposes for which they were designed, but each one on its own cannot handle the complex problems addressed in this paper. New tools need to be developed to handle all the issues in one numerical package. Such a program should integrate rock mass support using the correct support constitutive models (Moosavi, Bawden, and Hyett, 2002). In the authors’ opinion, the ideal numerical modelling tool would be a hybrid using the ability of three-dimensional elastic models to generate mine-wide elastic mine-induced stresses, combined with the ability of non-linear (e.g. finite element, finite difference, distinct element, etc.) models to use post-peak parameters to model failure and deformation while including explicit models of rock mass support.

It is necessary at this point to mention that the approach taken by the authors was in fact to illustrate the point of the power of numerical tools combined with excellent quality instrumentation data. There exist other brands of numerical tools that solve parts of the problems in much the same manner as the Rocscience tools. Three-dimensional finite element, finite difference, and distinct element packages that can run mine-wide models and incorporate rock mass support exist. For a number of reasons, these programs were not used. Because of the scale of the mine-wide model required to obtain appropriate mine induced global stress change, combined with the accuracy and detail required for the back analysis, the computation time would not be feasible. Additionally, the only numerical model that incorporates the bulb cable constitutive behaviour (one of the key instruments used in this study) is Phase². The authors’ believe that a hybrid approach is better suited to this type of problem.

![Fig. 9. Generalized Hoek-Brown peak strength envelope and a tight band representing the post-peak strength envelopes that produced the best back analysis results in Areas 1 and 2.](image-url)
However, even if the ideal numerical modelling tool existed, proper instrumentation, recording, and input of the field data needs to be done. Instrumentation data needs to be available in near real-time, entered automatically into databases, and transferred to numerical models. The ideal system would constantly record data so that no sudden changes in displacements were missed. This would require automated remote data collection that is promptly conveyed to computers on surface. Mining operations do not have sufficient personnel to provide manual reading of instruments of the quality required for the proposed analysis.

If such a numerical package existed and if instrumentation data were routinely updated and recorded, the goal of using such a tool would be predictive forward modelling of ground support. That is, once the model was calibrated using a procedure similar to that described in this paper (albeit with more directly relevant numerical tools) and the rock mass parameters up to failure and in the post-peak were deemed sufficiently reliable, forward modelling could be used to predict the behaviour of the rock mass and ground support prior to mining in a given area. This opens up a huge potential for cost savings for mine operations. Forward modelling can lead to better support design. Different types of support can be tried out in models at almost zero cost. Different timing of support installation, depth of support, etc. could be studied to minimize support cost, while minimizing the potential for support rehabilitation. Forward modelling of the type discussed would be expected to lead to improved overall mine design. For example, different sequencing strategies could be tested. The impact of mining strategies that result in unwanted pillars can be investigated and hopefully the foresight provided by such a model can minimize the chance that ore will be sterilized due to unexpected ground failure.

CONCLUSIONS

All of the goals that were discussed in the previous section are currently technically feasible. However, the combined use of tools that are currently available to mining professionals to achieve these goals are far too labour and/or computationally intense. Additionally, these techniques today require a very high level of technical expertise. Should appropriate numerical tools be developed, calibrated forward modelling of the mining process can lead to improved intelligent mine design. While more engineering time would have to be spent on monitoring and mine design, if the amount of rehabilitation of mining drifts could be reduced, then overall mining costs could be greatly diminished and the return on investment would be substantial.

With these proposed new tools and upgraded monitoring systems, real-time mine modelling could be performed, whether in-house or via contracts. The advent of wireless instruments and real-time to-surface monitoring, combined with internet technology, could mean that the mine instrumentation and modeling for all mines for a given company could be performed at a central site or by a third party. Microseismic monitoring provides an additional mine-wide monitoring system to assist in validation of the location of failure predicted by the numerical models.

The key to post-peak rock mass parameter calibration is to ensure high-quality instrumentation data. The frequency of readings, as already discussed, is of prime importance. It would also be imperative to install instrumentation as soon as the mining drifts are excavated. The models discussed in this paper only showed displacements relative to when the instruments were actually installed. The modelling results also showed, in some cases, large amounts of displacement prior to installation of the instruments. Perhaps the routine installation of wireless instrumentation should be part of the mine development process.

To summarize, the procedures outlined in this paper highlight the need for a new hybrid numerical modeling package that can use the influence of three-dimensional mining-induced stresses to predict displacements in, and potential failure of, rock masses surrounding mining infrastructure in order to radically alter strategic and tactical mine design. The early installation of high-quality, real-time automated instrumentation is required for forward modelling of the mining process. A critical problem is that there exists only a small market to drive this innovation. An industry-wide strategy would be required to fund such an ambitious project.

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