Incorporation of Stress Induced Damage into Mathew’s Stability Graph Method

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Abstract

The Mathews Stability Graph method has proven to be an effective planning tool in bulk mining. In this method, the factors of rockmass quality (Q’), stresses (A), joint orientation (B), and gravity effects (C) are combined into the stability number (N’) which is then plotted against the hydraulic radius for the free surface in question. The stress factor “A” relates to the boundary stress on the surface of an open stope, and it decreases with increasing stresses but remains constant at 0.14 after the rockmass strength is exceeded.

For certain stopes at the Golden Giant Mine, because of depth and stope extraction, the stresses are sufficiently high that the stope rockmass strength is exceeded prior to extraction. This stress induced rockmass damage is represented by the stress damage factor “D” which is empirically related to the Extra Stress Deviator obtained from stress modelling. This type of damage is expressed as fracturing and cracking in hard rocks and leads to a reduction of the ratio (RQD/Jn) in the expression, Q’ = (RQD/Jn) D (Jr/Ja). With the damage factor “D”, the Mathews Stability Graph method has been used to assess stope stability and support requirements. This paper presents the back analysis of an area where the existing Mathews analysis did not give accurate predictions, and several case studies of how the ‘D’ factor has been successfully applied to stope planning at the Golden Giant Mine.

1. INTRODUCTION

The Golden Giant Mine is 100% owned and operated by Battle Mountain Gold Company. The mine is located at Hemlo, Ontario, 40 km east of the town of Marathon, on the north shore of Lake Superior. The mine produces gold bullion from the Hemlo deposit that was discovered in 1981. The first gold bar was poured in 1985. Annual production is 1,092,000 tonnes of ore yielding 340,000 ounces of gold.

The Hemlo orebody is mined by three separate operations. The Golden Giant Mine that is situated in the center is bounded on the west by the Williams Mine and on the east by the David Bell Mine. The Golden Giant orebody occurs in two zones, the Main zone and a footwall zone, which trend at 110 degrees and dip 65 degrees to the north. The Main zone contains the bulk of the ore and averages 20 m in width. Mineralization is associated with the Moose Lake Porphory intrusive, a highly strained and metamorphosed multiphase felsic intrusive body. The current mineral reserve is estimated at 7 million tonnes grading 9.3 grams per tonne (0.3 oz/t). The ore zone may consist of feldspathic, sericitic and baritic ore units. The immediate hanging wall of a few meters thickness is usually a sericitic metasediment and the hanging wall rockmass is a competent metasediment. The footwall rockmass is usually Quartz Eye schist.

The Mathews Stability Graph method has been used at the Golden Giant Mine by the engineers on site since 1990 for the purposes of stope planning and cablebolt design. This has resulted in successful stope planning and production. With increasing extraction ratio and depth of mining, the rockmass strength in the stopes is often exceeded at the Hemlo Gold Camp. It became clear that stress modelling should be integrated into the Mathews Stability Graph to become a useful engineering tool. The current study was intended to meet the challenge of using elastic stress modelling for such mining applications where non-linear rockmass deformation occurs on a large scale.
2. AN INNOVATIVE METHOD FOR STRESS MODELLING

2.1. In Situ Stresses

The *in situ* or far field stresses were measured by Golder Associates in both the Golden Giant Mine and the Williams Mine on the 4600 level (720 m depth) and 10100 level (220 m depth) respectively. The stress magnitude and orientation are shown in Table 1.

The elastic modulus of the rockmass was taken as 56 GPa, the Poisson’s ratio 0.25, and the mine surface elevation 5321 m (321 m above sea level). The above model parameters were successfully used in a shaft pillar modelling study at Golden Giant Mine by Noranda Technology Centre (Nickson and Rizkalla 1996).

<table>
<thead>
<tr>
<th>Stress component</th>
<th>Magnitude (MPa)</th>
<th>Orientation (trend/plunge)</th>
</tr>
</thead>
<tbody>
<tr>
<td>$P_3$</td>
<td>0.0214 x depth (m)</td>
<td>250/60 (nearly vertical)</td>
</tr>
<tr>
<td>$P_1/P_3$ ratio</td>
<td>2.0421</td>
<td>358/10 (nearly N-S)</td>
</tr>
<tr>
<td>$P_2/P_3$ ratio</td>
<td>1.3972</td>
<td>093/28 (nearly E-W)</td>
</tr>
</tbody>
</table>

2.2. Model Construction and Simplifications

The 3-dimensional boundary element modelling software MAP3D® was used to compute the induced elastic stresses. AutoCAD® R14 and RocCAD® v14 software were used to construct the three-dimensional geometry model of the stopes and drifts which were six-sided, box-like 3-dimensional objects defined by four points on both upper and lower levels. A model of the mined-out stopes for the Hemlo Gold Camp as of September 1998 is shown in Fig. 1.

3D stress computation is time consuming. In order to reduce computing, a comparison study was carried out to determine the effects of removing the stopes far away from the area of interest (Yi 1997). Based on the results of comparisons and taking into account of the purposes of modelling, stopes that are far away from the area of interest are often removed from the model to speed up the computation.

Figure 1. Longitudinal for 3-dimensional stress model of mined-out stopes at Hemlo Gold Camp (as of September 1998).
An isotropic and homogeneous rockmass was assumed for the entire model for the Hemlo Gold Camp. The differences in rock units with respect to strength and elastic modulus values, the effects of foliation planes and the existence of distinct geological features were considered in the interpretation of results. This strategy was taken to reduce not only the computation time but also the uncertainties relating to the values of these parameters. It is believed that the inaccuracy introduced in the above assumptions is over-shadowed by the inaccuracy resulting from the following additional assumptions:

(a) linear elastic response was employed disregarding the non-linear nature of the failure process,  
(b) backfill was disregarded in stress computation, and  
(c) one single in situ stress regime was used disregarding the depth and the orientation of the ore zone.

In order to draw practical conclusions regardless of inaccurate stress values, an innovative approach in using modelling results was developed as described in the following section.

2.3. Modelling Approach

The conventional stress analysis involves computing stresses at different points in the rockmass and evaluating if rockmass strength is exceeded. An example would be to compute and plot the factor of safety contours for a volume of rockmass in the stope back. In such an analysis, a point is the basic rock unit for the factor of safety evaluation. In the present approach, the basic rockmass unit for stress computation is a stope prior to extraction. Only one stress value, the “Stope Extra Stress Deviator”, is computed for each stope. This stress parameter is computed immediately prior to the mining of the stope and it is used as an index for stress-induced rockmass damage.

The Stope Extra Stress Deviator was plotted against the hydraulic radius in a graph that was first used in case studies to define or calibrate both zones of stability conditions and scale for the damage factor. This graph was then used to predict stope stability. More details are presented below.

Definitions of Stope Stresses

The following two factors should be balanced if stress modelling is to be used as an effective planning tool at the mine site, (i) the required accuracy for the task, and (ii) the computing time. For the 3-dimensional boundary element method, the accuracy is determined firstly by the discretization parameters and secondly by the model size. Long range planning such as stope sequencing requires coarse discretization but a large model to ensure valid comparisons of proposed sequences, whereas detailed modelling for support design requires fine discretization to accurately assess support effectiveness around excavations, but, it requires a relatively small model.

Sharp corners of stopes often introduce computation errors in the vicinity. For a given discretization, the farther away the stress points are from the stope boundary, the better the accuracy. Another advantage of placing stress points at a distance from the free surfaces is that the computed stresses remain valid even if the non-elastic response of the backfill and the failing rockmass in the stope vicinity is not included in the modelling.

A typical stope is 20 m on strike, 4 ~ 40 m in thickness and 25 m in height. The typical element size is 10 m in the MAP3D® software if the computation is to be done in 12 hours. In the present study, the stress points were placed in the hanging wall of the stope and at a distance of 12 m from the hanging wall contact. This distance was somewhat arbitrarily chosen to ensure that it is more than one element length away from stope free surfaces. This choice of stress points remains valid for more accurate modelling schemes.

The “stope stresses” are defined as the average, in the three principal components, for three points located at stope mid-height and 12 m in the hanging wall. On the plan view, one of the three points is placed on the stope centre line and the other two at a distance from the two stope edges, see Fig. 2a.

Nature of Stope Stresses
The intact rock UCS for a typical ore rock at the Golden Giant Mine is about 180 MPa. The rockmass UCS is estimated to be about 60 MPa. This rockmass UCS can also be called the peak stress. After the peak stress is exceeded, the rockmass fails and transfers to the “post-peak regime”, see Fig. 2b. In this post-peak regime, the stresses in the rockmass decrease as it undergoes increasing damage and deformation, i.e., the stresses in the rockmass can never exceed the rockmass strength. The rockmass in the post-peak regime is therefore “destressed” due to damage. Fig. 2b shows that the computed stresses are the true stresses for linear elastic deformation, but they reflect in an empirical sense the non-linear rockmass deformation in the post-peak regime.

Since the three stress points in Fig. 2a are not located within the stope, the computed stope stresses are not true stresses in the stope rockmass, rather, they are used as indices for

![Plan View and Looking North Diagram](image)

Figure 2. Definition of stope stresses where (a) shows the positions of stress points, and (b) demonstrates the meaning of stope stresses.

rockmass damage in the stope. This definition does not only have the advantage of improved modelling accuracy but also ensures that stope stresses remain in the elastic range to avoid the modelling of non-elasticity at the stope boundary. Since rockmass damage may not be uniform in the stope due to different rock units and localised stresses, the stope stresses reflect only the degree of damage and/or the extent of the damage zones.

**Definition of Extra Stress Deviator**

Large scale experiments in mechanically excavated (non-blast) circular tunnels at AECL, Pinawa, Manitouba demonstrated that the stress deviator criterion, $\sigma_1 - \sigma_3 = 0.3-0.4 \sigma_c$, can be used to estimate the depth of failure zone in the granitic hard rockmass (Martin 1997). Researchers at the Geomechanics Research Centre, Laurentian University applied this type of failure criterion to a study of damage initiation and damage depth around the Sudbury Neutrino Observatory cavern (Castro et al. 1996). The Stope Extra Stress Deviator criterion adopted for the present studies fall into the same category. However, it does not suffer from the same restrictions by avoiding the actual failure zone. The Stope Extra Stress Deviator is used for comparisons of ground conditions at different locations and under different mining-induced stresses.
From Table 1, the in situ stress deviator is, \( P_1 - P_3 = 1.0421 P_3 \), which increases linearly with depth. This means that if the stress deviator criterion is used to assess rockmass damage, then the damage increases with depth even if no underground excavations exist. This is contrary to the prior proposition that rockmass damage is caused by excavation induced stress changes. To eliminate this discrepancy, the “Extra Stress Deviator” is defined as the “stope stress deviator” minus the “in situ stress deviator” and expressed as follows:

\[
\text{Extra Stress Deviator} = (\sigma_1 - \sigma_3) - (P_1 - P_3)
\] (1)

Where \( \sigma_1 \) and \( \sigma_3 \) are computed major and minor principal stresses for a stope, and \( P_1 \) and \( P_3 \) are the major and minor in situ stress components. If the rock strength in different stopes is different from one another, this Extra Stress Deviator should be normalised to the intact rock strength. The Extra Stress Deviator parameter has the following innovative properties:

(a) it has meaningful elastic stress values even if the stope rockmass is severely damaged and despite our inability to model the backfill response, and

(b) it can be used to compare stress conditions for stopes at different depth.

**Stress Damage Graph**

The Extra Stress Deviator was plotted against the Hydraulic Radius (Fig. 3). This graph was calibrated using case studies to define different zones of stability, and it was then used to assess stope stability (Yi 1997 and 1998). An empirical linear relation between the damage factor, \( D \), and the Extra Stress Deviator is shown in the graph. This graph is called the Stress Damage Graph. Generally, it should be used together with the existing Mathews Stability Graph for stope stability assessment. More details are presented in the next section.

![Stress Damage Graph](image)

Figure 3. The Stress Damage Graph.
3. STOPE STABILITY METHODOLOGY

3.1. Mathews Stability Graph

In conventional terminology, stope stability depends on five most important factors, (i) rockmass quality, (ii) stope size, (iii) backfill, (iv) extraction ratio and depth of mining, and (v) artificial support. The Mathews Stope Stability Graph method captures all the important factors affecting the stope stability in relatively low stress conditions. The hydraulic radius defined as the area divided by the perimeter for a free surface is theoretically a measure of the freedom of movement for the rockmass. In other words, the hydraulic radius quantifies the effects of stope size and shape. The stability number is essentially a measure of rockmass quality, stress and gravity effects. The stress parameter quantifies the effect of extraction ratio and depth of mining. An associated Cablebolt Design Graph relates cablebolt density to block size and Hydraulic Radius. In other words, the Mathews Stability Graph quantified the conventional wisdom in stope design methodology and was a step forward from the art of mining to the engineering of mining.

3.2. Nature of Stress Damage in Stope

The failed rockmass in the post-peak regime retains lower and lower stresses with increasing deformation and hanging wall to foot wall convergence (Fig. 2b). Such a rockmass may be called “destressed” and may result in decreased clamping stresses on cablebolts. This may lead to decreasing support effectiveness for certain types of cablebolts.

Rockmass damage is usually localised on weak geological rock units and/or features. An example of this localisation is that blast hole squeezing usually occurs in the weak baritic or sericitic ore unit, not in the feldspathic ore unit.

Extreme fracturing in the form of small fragments often occurs near open and backfilled stopes with both reduced RQD and increased number of discontinuities. The stress-induced discontinuity surfaces are approximately parallel to compressive stresses and excavation surfaces, and act locally as if they were additional joint sets. Moderate fracturing may involve creation of distinct fractures spaced more than 100 mm apart with little decrease in RQD but these stress-induced discontinuities act locally as additional “joint sets”.

3.3. Integration of Stress Damage into Mathews Stability Graph

The stress factor “A” in Mathews Stability Graph is a linear function of boundary stress and intact rock strength. This factor remains constant at 0.14 after the rockmass strength is exceeded. In other words, “A” factor is related to stope boundary stresses after stope mining with the assumption that the rockmass unravels once the strength is exceeded. On the other hand, the Extra Stress Deviator in the Stress Damage Graph (Fig. 3) is related to the stresses in the solid stope prior to mining. The rockmass is considered damaged or destressed after the strength is exceeded, and the degree of damage is increased as the rockmass transfers to the post-peak regime (Fig. 2b). In terms of rockmass quality, the RQD is decreased and the number of discontinuity sets are increased with stress induced damage. Therefore, stress induced rockmass damage contributes to the decrease of the term, RQD/Jn, in the NGI rockmass classification system. The integration of stress damage into the Mathews Stability Graph results in the following expressions:

\[
Q' = \left( \frac{RQD}{Jn} \right) D \left( \frac{jr}{Ja} \right) \\
N' = Q' A B C
\]

Where \(Q'\) and \(N'\) are the modified rockmass quality index and stability number respectively, \(A\) is the stress factor for stope boundaries, \(B\) the joint orientation factor, \(C\) the gravity factor, and \(D\) the stress damage factor.

The evaluation of the stress damage factor “D” as described in section 2 involves more science than art and was made possible at the mine site only by the availability of powerful personal computers at low cost. The integration of stress damage into Mathews Stability Graph may mark an additional step forward in stope design methodology from the engineering to the science of mining.
3.4. Stope Stability Database and Calibration of Stress Damage Graph

Case studies employing modelling and observations of ground conditions helped define stability zones in the Stress Damage Graph (Fig. 3). The calibration of the Stress Damage Graph was aided by the Ground Database contained in RocCAD© software. Each point in the graph was linked to one or more rows of data in the database which included the following columns or fields: ID, Status, Extra Stress Deviator, Hydraulic Radius, Ground Conditions, Geology, Support, and Instrumentation. Based on case studies in the three mines in the Hemlo Gold Camp, namely Golden Giant Mine, Williams Mine and David Bell Mine, preliminary lines were drawn to define different zones of ground conditions. Based on the zones of stability and an understanding of the range of values for the “A”, “B” and “C” factors in equation (2b), a linear relation in the range of 1.0 – 0.1 between the damage factor “D” and the Extra Stress Deviator was chosen (Fig. 3).

In addition to the Stress Damage Graph, the mine longitudinal can also be linked to the same database to give visual access to the database information. The RocCAD© software has proven a very useful tool for tracking and predicting ground conditions for long and short range planning.

3.5. Limitations

The effects of extraordinary geological features such as shear zones and dykes could only be qualitatively discussed at the present time. It should also be realised that the Extra Stress Deviator is an indicator of rockmass damage in the stope in the general sense, not an accurate measure of damage.

4. APPLICATION OF INTEGRATED MATHEWS STABILITY GRAPHS TO STOPE PLANNING

The Integrated Mathews Stability Graphs have been applied to both long and short range planning. The selection of mining sequence may be classified as long range planning, whereas the decision for stope panelling and support design may be classified as short range planning. In long range planning, one Stress Damage Graph plot is made for each proposed mining sequence. Comparisons of stopes of different sequences are made in terms of stope stability. The best mining sequence was then chosen after further evaluating operational constraints. The decisions for stope panelling and cablebolt design are usually made immediately prior to stope extraction, where the Mathews Stability Graph and the Stress Damage Graph were used as an engineering tool. Examples for the application of the integrated Mathews Stability Method at the Golden Giant Mine, Battle Mountain Gold are presented in this section.

4.1. Planning of Block 4 East Stopes in Golden Giant Mine

The remaining stopes in Block 4 East are shown in Fig. 4. Two planning issues were to be resolved: (i) short range planning for the planned sequence 1 to 17 in terms of support and stope layout, and (ii) comparisons of two different sequences for the 6 stopes to the east of 11E stope.
Short range planning

The Stress Damage Graph for the planned sequence 1 to 17 was obtained and is shown in Fig. 5a. The plots showed that all the stope backs have hydraulic radii less than 4.5, but stress damage would occur in all stopes. For example, the damage factor for the 5E stope would decrease from 0.4 for 4325-5E stope to 0.1 for 4375-5E stope as mining advances. This suggested that increased cablebolt support would be required for the 4375-5E stope back. The damage factor for each stope obtained from Fig. 5 was used to lower the rockmass quality Q’ and stability number N’ in the Mathews Stability Graph (equations 2a and b) to assist in support design and other planning decisions.
Figure 5. Stress damage graphs for (a) step Nos. 1 to 17, (b) Sequence A (step Nos. 18 to 23) and (c) Sequence B (step Nos. 18 to 23) for sequence B.

Long range planning

The stopes 12E to 17E are to be mined after the rest of the Block 4 East stopes are mined. Two sequences, A and B, were proposed initially. Sequence A is a west to east sequence whereas sequence B is an east to west sequence. The stress damage plots for the two sequences are shown in Figs. 5b and c respectively. Fig. 5b shows that the stress damage is similar for all 6 stopes, whereas Fig. 5c shows that the damage increases with each mined stope. Sequence A is better in terms of ground control in that the same support design may be applied to all stopes.

4.2. Case Studies in Block 3 of Golden Giant Mine

The mining of the combined Q14/15 pillar stope between 4566 and 4600 sublevels provides an interesting case for the application of the Integrated Mathews Stability Graph method. Fig. 6 shows the stope geometry after the mining of the Q14/15 stope between 4566 and 4600 sublevels. The back and hanging wall for the stope plotted in the Support Required Zone if stress damage factor was not applied (Fig. 7).

The Extra Stress Deviator for the stope was 56 MPa and it plotted in the middle of the Destressed Zone (Fig. 3). Incorporating a stress damage factor, D = 0.2, the back point
Figure 6. Longitudinal after the mining of Q14/15 stope between 4566 and 4600 sublevels.

Figure 7. Mathews Stability Graph for the Q14/15 stope between 4566 and 4600 sublevels (cf. Potvin 1988, Nickson 1992 and Potvin and Milne 1992).
dropped to the “Caved Zone” (Fig. 7). Indeed, the back on the 4600 level (Fig. 6) caved to about 5 m height before the backfilling operation was completed despite cablebolting with double bulge cables.

When this analysis was performed, it was too late to avoid stope back failure on the 4600 level by modifying the planning for the Q14/15 stope between 4566 and 4600 sublevels. However, a planning decision was made to prevent back failure on 4533 sublevel by increasing the 2E stope width and decreasing Q14/15 stope width on 4533 sublevel (Fig. 6).

5. CONCLUSIONS

An innovative modelling approach was developed to use 3-dimensional elastic stress modelling for the purposes of stope planning in situations where non-elastic rockmass deformation occurs on a large scale. The Stress Damage Graph alone can be used for stope sequence comparisons.

Stress modelling was integrated with Mathews Stability Graph method to provide a useful engineering tool. This integrated method entails a complete rationale for stope planning and cablebolt design.

ACKNOWLEDGEMENTS

Case studies were also done at both Williams Mine and David Bell Mine, which helped the development of the methodology presented here. MAP3D is a registered trademark of Mine Modelling PTY Ltd., Australia. AutoCAD is a registered trademark of Autodesk Inc., U.S.A. RocCAD is AutoCAD-based engineering software, copyright 1998-1999, XY RocCAD Inc., Canada.

REFERENCES


