Stress path and instability around mine openings

Cheminement de contraintes et instabilité d'effort autour des ouvertures de mine
Spannungsweg und -instabilität um Grube Hohlräume

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ABSTRACT: Stope design in Canadian hard rock mines is often carried out using the Stability Graph Method. However, this method does not account for the low confinement measured around stopes or quantify the amount of dilution associated with stope caving. A new approach to stope hangingwall stability is presented which consider the stress path the stope experiences and the effect of low confinement. A case study is used to illustrate the methodology.

RÉSUMÉ: La création de chantier parmi les mines de roche dure au Canada est souvent effectuée en utilisant la méthode de graphique de stabilité. Cependant, cette méthode n'explique pas le bas quantité de confinement mesurée entre les chanters ou attribuer à la quantité de dilution associée avec l'abattement de chantier. Dans ce papier est présente une nouvelle façon d'évaluer la stabilité d'éponts supérieure qui prend en contexte le réseau de contraintes, l'expérience acquise et l'effet d'une base quantité de confinement. Une étude de cas réel est utilisée pour expliques la méthode.

ZUSAMMENFASSUNG: Die Bemessung von grossen Erzabbaukavernen (stopes) in kanadischen Hartgesteinsgruben wird häufig mit der Stabilitätsdiagrammethode durchgefüht. Diese Methode erklärt jedoch nicht den wichtigen Einfluss niedriger radial Spannungen, die in der Nähe der Kavernenwände gemessen werden, noch bestimmt sie mit genügender Genauigkeit das Volumen von Felseinbrüchen. Eine neue Methode zur Stabilitätsbestimmung, die den Spannungsweg in Betracht nimmt, wird dargestellt. Diese Methode wird hier an Hand einer Fallstudie erklärt.

1 INTRODUCTION

Canada's hardrock mining industry is confronted with the need to mine at ever increasing depths in a safe and efficient manner. Yet, traditionally, mining at depth incurs rock mass failures that impact the economic and safety aspects of a mine. Today many mines are attempting to conduct risk analyses to help them assess the risks of mining at depth. In order for these attempts to be successful, a thorough knowledge of the geomechanics failure process and the factors that control or alleviate it are required. Hence, a fundamental understanding of the factors that affect rock mass failure is needed to develop technology that can be used to quantify the geomechanics risk of mining at depth.

Geomechanics instability and associated risk is traditionally quantified by comparing the stress to strength, i.e., a safety factor against failure. However, around mining stopes a single safety factor seldom meets the needs of the operator (Fig. 1). For example when excavating the topsill and bottomsill drifts the safety factor must be adequate to protect the workers, while during the non-entry stoping operations between these sill drifts the operator needs to rely on the safety factor against unplanned dilution. Today there are many two and three dimensional numerical programs that allow the potential instability of any shaped underground opening to be assessed. However, determining the input parameters for these numerical programs, i.e., the rock mass strength, is still a challenging task. A far greater challenge, however, is the interpretation of the results from such analyses in terms of operators' issues, e.g., depth of failure, depth of dilution, stope sequencing or stope dimensions. The interpretation of these results is particularly difficult in mining where the geometry and sequencing of the stopes creates complex stress paths around the mine openings.

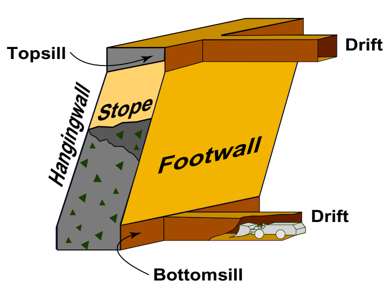

Figure 1: Illustration of a typical bulk-mining stope

The potential stress paths that a rock mass around an underground opening experiences are illustrated in Figure 2. Inspection of Figure 2 illustrates that for most stress paths the rock mass is fundamentally unloaded. Martin (1997) and Kaiser *et al.* (in press) showed that in addition to the increase and decrease of principal stresses, the direction of these stresses is also rotating. Martin (1997) proposed that this was one of the fundamental differences between laboratory strength determined under monotonic loading conditions and the in-situ strength. In this paper the importance of stress path on rock mass strength is examined. In particular, the issue of dilution potential, its relationship to stress path and the interpretation of numerical results to evaluate dilution potential is considered.

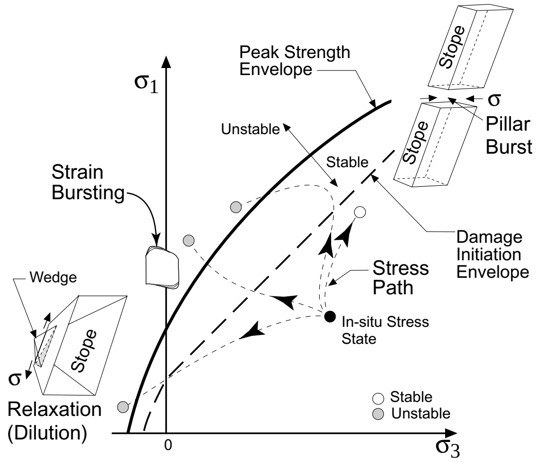

Figure 2: Illustration of possible stress paths near underground openings.

2 ROCK MASS STRENGTH

Stability analyses of underground openings require an estimate of the in-situ rock mass strength. The shear strength of a rock mass is usually described by a Coulomb criteria with two strength components: a constant cohesion and a normal-stress dependent friction component. In 1980, Hoek and Brown, proposed an empirical failure criterion which is now widely used in rock engineering and in the generalized form is given as:

$$\sigma_1^{'} = \sigma_3^{'} + \sigma_c \left(m \frac{\sigma_3^{'}}{\sigma_c} + s \right)^a \tag{1}$$

where $\sigma_1^{'}$ and $\sigma_3^{'}$ are the maximum and minimum effective stresses at failure, σ_c is the laboratory uniaxial compressive strength, and the empirical constants m and s are based on the rock mass quality. For most hard-rock masses the constant a is equal to 0.5 and Equation 1 is usually expressed in the following form:

$$\sigma_1 = \sigma_3 + \sqrt{m\sigma_c\sigma_3 + s{\sigma_c}^2} \tag{2}$$

where σ_1 and σ_3 are again the maximum and minimum effective stresses at failure. The empirical constants are related in a general sense to the angle of internal friction of the rock mass (m) and the rock mass cohesive strength (s) and Hoek and Brown (1998) provided a methodology for deriving the frictional and cohesive strength components for a given normal stress. The tensile strength, which reflects the interlocking blocks of the rock mass when they are not free to dilates, is given by:

$$\sigma_{tm} = \frac{\sigma_c}{2} \left(m - \sqrt{m^2 + 4s} \right) \tag{3}$$

Hoek and Brown (1998) suggested that m and s can be estimated by:

$$m = m_i \exp \left(\frac{GSI - 100}{28} \right) \tag{4}$$

and

$$s = \exp \left(\frac{GSI - 100}{9} \right) \tag{5}$$

where m_i is the value of m for intact rock and GSI is the Geological Strength Index developed by Hoek and Brown (1998). It can be seen from Equations 4 and 5 that as the rock mass quality improves, i.e., GSI approaches 100, the strength of the rock mass approaches the strength of the intact rock. For the boundary of a tunnel, where $\sigma_3 = 0$, Equation 2 reduces to:

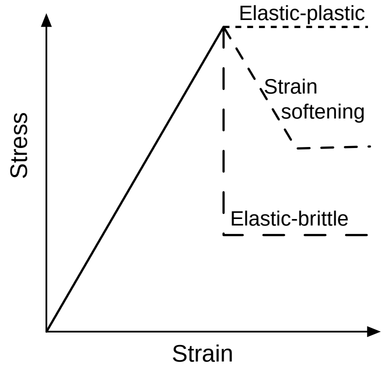

Figure 3: Post-peak failure characteristics: (a) Very good quality hard rock mass – Elastic brittle; (b) Average quality rock mass – Stain softening; (c) Very poor quality soft rock mass – Elastic-plastic.

$$\sigma_1 = \sqrt{s\sigma_c^2} \tag{6}$$

Myrvang (1991) carried out a series of back analyses of stress-induced rock mass failures around underground openings and showed that the rock mass strength was approximately $0.5\sigma_c$ suggesting that for stress induced failure $s = 0.25$.

In order to use the Hoek-Brown criterion for estimating the rock mass strength the following properties of the rock mass are required: σ_c – the uniaxial compressive strength of the intact rock; m_i – the value of the Hoek-Brown constant for the intact rock; and GSI – the value of the Geological Strength Index for the rock mass. Once these properties are evaluated Equations 4 and 5 can be used to determine the m and s parameters that are required to carry out the stability analyses. In many situations the mine operator also needs to assess, in addition to the opening stability, the extent of the failed zone. This information is needed to design the support system and/or estimate the amount of dilution. In order to address these issues, an assessment must be made of the post-peak response of the rock mass (Figure 3). Hoek and Brown (1998) suggested that for a very good quality hard rock mass the post-peak behaviour can be descibed as elastic-brittle (Figure 3).

To illustrate the application of the Hoek-Brown criterion an example is taken from the Canadian hard rock mining industry in northern Ontario and described in the next section.

3 CASE STUDY: NARROW VEIN MINING

As with most narrow vein operations unplanned dilution disrupts the mine production and impacts directly on the mining costs. Therefore, decisions made by ground-control engineers, related to ground failure and dilution, must consider the economic effects on the production cycle as well as the operational safety issues. Unplanned dilution is illustrated in Figure 4 and can be defined as:

$$D = \frac{x \text{ m of F/W waste} + y \text{ m of H/W waste}}{\text{Planned Mining Width}} \tag{7}$$

In this case study the operator needed to know the amount of unplanned dilution that could be anticipated for the long hole stopes planned below a depth of 300 m. The operator had used long hole stoping at shallower depths and had noted that dilution became

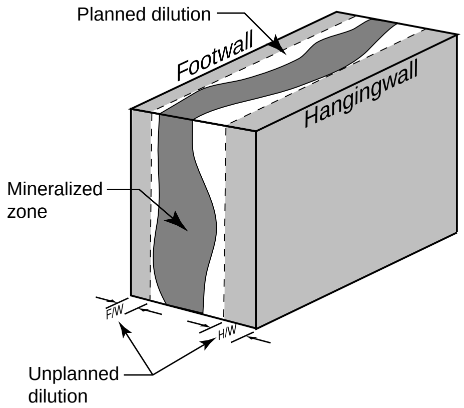

Figure 4: Illustration of planned and unplanned dilution.

excessive beyond a stope strike length of about 25 m with stope heights ranging from 20 to 40 m. In several cases, cavity surveys of individual stopes had been carried out to quantify the amount of dilution. Extensive field mapping and laboratory testing revealed that the 3-m-wide stopes were excavated in a rock mass with the following average properties:

Rock Type	Metavolcanics	
In-situ stress	$\sigma_1, \sigma_2, \sigma_3$	22, 13, 8 MPa
Intact rock strength	σ_c	120 MPa
Geological Strength Index	*GSI*	70
Hoek-Brown constant	m_i	16
	m	5.5
	s	0.035
Post-peak H-B parameters	m_r	3
	s_r	0
Dilation angle	α	10°
Deformation Modulus	E_m	35 GPa
Poisson's ratio	ν	0.2
Rock mass compressive strength	σ_{cm}	22.6 MPa
Rock mass tensile strength	σ_{tm}	-0.8 MPa

In the following sections several methods are explored to determine the most practical approach for quantifying the potential dilution.

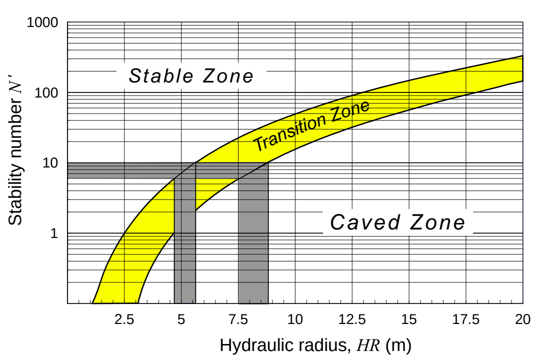

Figure 5: Stability graph showing the zones of stable ground and caving ground, i.e., dilution.

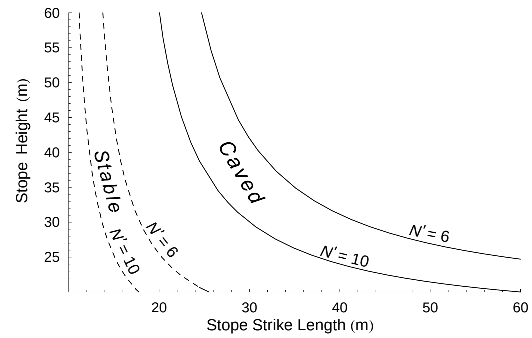

Figure 6: Combination of stope height and length that yield different expected stope hangingwall performances.

4 DILUTION AND STABILITY GRAPH

The Stability Graph Method for stope design was first introduced by Mathews *et al.* in 1981 and later modified by Potvin *et al.* (1989). The current version of the method, based on the analysis of more than 350 case histories collected from Canadian hard rock underground mines, is widely used as the first step in the stope design process (Fig. 5). The stability number N' is defined as:

$$N' = Q' \times A \times B \times C \tag{8}$$

where Q' is the modified Q Tunneling Quality Index; A is the rock stress factor; B is the joint orientation adjustment factor; and C is the gravity adjustment factor. The modified Tunneling Quality Index Q' is determined from the structural mapping of the rock mass and for the case study is equal to 18. The other factors A, B and C are determined for the stope faces being assessed using the guidelines outlines by Potvin *et al.* (1989). The stability number N' of the hangingwall for the case study ranged from 6 to 10.

Using the N' values for the hanginging wall the stability graph suggests that unsupported stopes will be stable if the hydraulic radius (HR, where HR is the area of the stope wall divided by the perimeter) of the stopes ranges between 4.5 and 6.5, while caving, i.e., dilution, will occur for unsupported stopes with a hydraulic radius between 7.5 and 8.5 (Fig. 5).

Figure 6 shows the stable and caved hydraulic radii expressed in terms of stope height and stope strike length. The stable region in Figure 6 suggests that 20-m high stope will be stable for strike length of approximately 20 m, which is in keeping with mining experience. Obviously this empirical approach must be used with some caution as predicted stope performance needs to be calibrated with actual performance. One of the shortcomings with this method is that the orientation of the stopes relative to the stress field has little influence on the predicted stope performance and the method does not quantify the amount of dilution. Both of these shortcoming are addressed in the next section. Nonetheless, the method provides an essential first step in the stope design process.

5 DILUTION AND CONFINING STRESS

It is well known in soil mechanics that stress-path is an important element in evaluating the stability of slopes, foundations, etc. Figure 2 suggests that the stress path associated with hangingwall dilution or relaxation is fundamentally an unloading stress path. This is illustrated in Figure 7 which shows the $\sigma_3 = 0$ isosurface for a stope excavated at depth and parallel to σ_1. Figure 7 was generated using the three-dimensional boundary element program

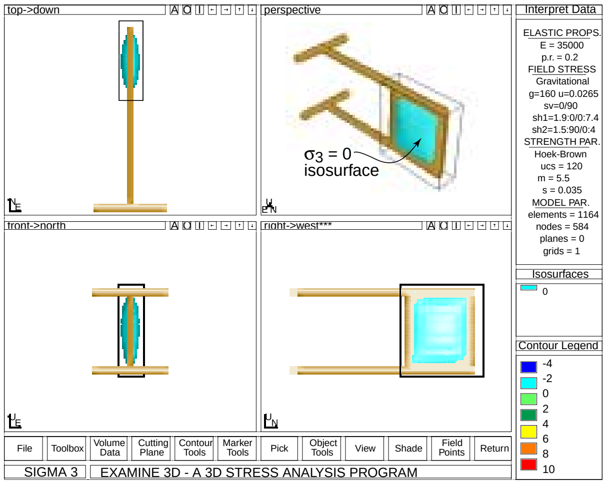

Figure 7: $\sigma_3 = 0$ isosurface around a stope at depth. The analysis was carried out using *Examine3D*.

Examine3D[1]. It is well known that the behaviour of a jointed rock mass is significantly controlled by the confinement, e.g., a very blocky hangingwall or tunnel roof will simply collapse if the confinement is removed by a process often referred to as unravelling or slabbing. In a good quality rock mass with discontinuous joints this unravelling will not occur unless new fracture growth forms the blocky or slabby conditions. Hoek (1968) carried out a series of tests on plates of glass to examine the relationship between fracture growth and the confining stress expressed as the ratio of σ_3/σ_1 (Fig. 8). Figure 8 implies that as the confining stress (σ_3) approaches zero the potential for new crack growth increases significantly. Also, if σ_3 becomes tensile the potential for new fracture growth increases significantly. Hence, the hypothesis suggested here is that the confining stress expressed as σ_3, may be a good indicator for predicting the amount of dilution.

A series of three dimensional elastic analyses were carried out using *Examine3D* and the stress conditions for the case study. In Section 3 it was shown the rock mass tensile strength, based on the

[1]Available from RocScience Inc., 31 Balsam Avenue, Toronto, Ontario, Canada M4E 3B5; Fax: +1 416 698 0908; Internet: http://www.rocscience.com.

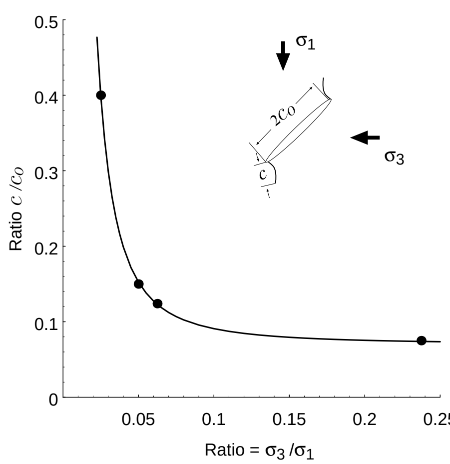

Figure 8: Relationship between fracture growth and the confining stress expressed as the ratio of σ_3/σ_1, data from Hoek (1968)

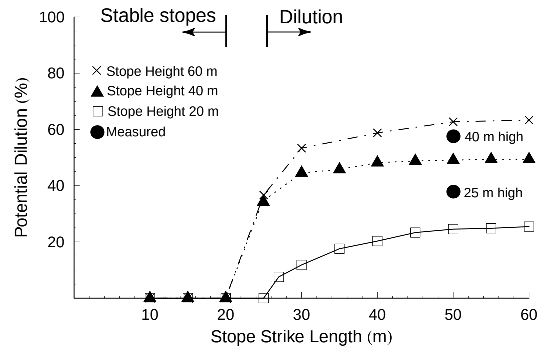

Figure 9: The predicted dilution from the *Examine3D* numerical analyses compared to measured results.

Hoek-Brown parameters, was -0.8 MPa. Hence when σ_3 exceeds this value, tensile failure will occur. In these analyses the depth of the $\sigma_3 = 0$ isosurface for the hangingwall at the mid height of the stope was recorded rather then the rock mass tensile strength. This approach seems more appropriate because once the confinement is reduced to zero the rock mass is free to dilate and unravel under gravity loading. The results from these analyses expressed as percent potential dilution, using Equation 7, are presented in Figure 9 along with the results from two cavity surveys. Examination of Figure 9 indicates the onset of dilution for stope heights ranging from 20 to 60 m occurs between 20 and 25 m. The predicted results are in good agreement with the field observation that dilution became excessive beyond stope strike lengths of 25 m, and the amount of dilution measured by the cavity surveys.

In the analyses used to generate Figure 9 the orientation of the stopes was parallel to σ_1. This stress orientation was based on stress measurements made in the local region but not at the mine site. Hence to determine if this assumption impacted on the results presented in Figure 9, additional analyses were carried out with the stope oriented perpendicular to σ_1. These results are presented in Figure 10 and shows that this stope orientation, relative to the stress field, would create far more dilution and the stable stope strike strike length would be reduce to 10 m. This finding is not in keeping with the case study experience and supports the notion that at this mine σ_1 is parallel to the stopes.

The analyses above support the field observation that stope stability is sensitive to the orientation of the stopes relative to the stress field. As mentioned previously the Stability Graph Method does not consider the orientation of the stope relative to the stress field, or more importantly the effect of confining stress.

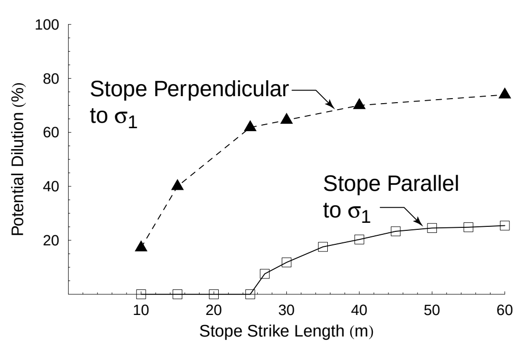

Figure 10: Comparison of the predicted dilution for stopes oriented parallel and perpendicular to σ_1.

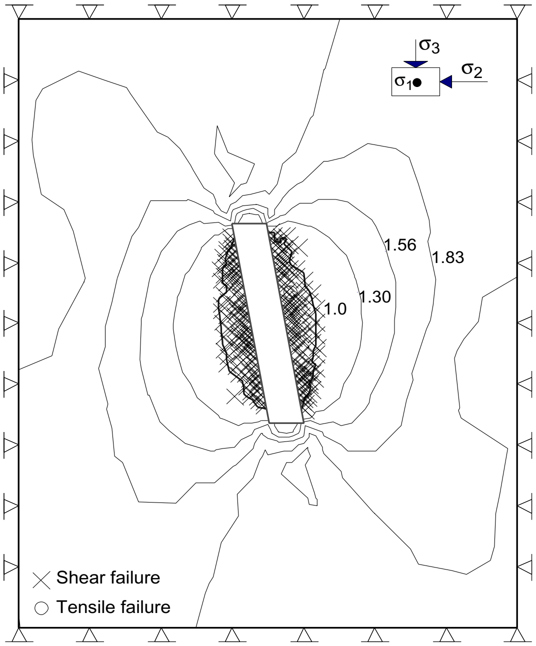

Figure 11: Strength factor results from *Phase2* for longhole stope oriented parallel to far-field σ_1.

6 POST-PEAK STRENGTH AND DILUTION

Two dimensional analyses cannot be used to determine the stable stope strike length because in these analyses the stopes have infinite length. Nonetheless, two dimensional nonlinear analyses can be used to quantify the extent of yielding that occurs around the stopes. This approach was used to estimate the amount of dilution around the 20-m-high stopes. These analyses were carried out using *Phase2*[1] with the properties provided in Section 3 and the elastic-brittle post-peak failure characteristic in Figure 3. Figure 11 shows that the maximum amount of yielding, i.e., hangingwall dilution, was approximately 48%. This amount is far greater than the amount predicted with the $\sigma_3 = 0$ isosurface approach and the measured dilution from the cavity survey (see Figure 9). Nonetheless, this approach does indicate that 20-m high stopes with long strike lengths will be unstable and the resulting dilution will be excessive.

7 CONCLUSIONS

Stope design in Canadian hardrock mines is usually carried out using the Stability Graph Method. However, this method does not account for the low confinement measured around stopes or quantify the amount of dilution associated with stope instability, i.e., caving. The method also does not consider the effect of the orientation of the stope relative to the far-field stress tensor.

Investigation of stress paths around stopes showed that stope hangingwalls are subjected to a stress path that is fundamentally unloading the hangingwall rock mass. A new approach to stope hangingwall stability is presented which considered the stress path the stope experiences and the effect of low confinement. The stope stability and dilution results, using this approach, agreed with field observations from the narrow vein case study.

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