A rock mass rating system for evaluating stope stability on the Bushveld Platinum mines

by B.P. Watson*

Synopsis

Several large falls of ground occur annually in the narrow, tabular stopes mined in the Bushveld Complex. These falls comprise safety risks and cause substantial production losses and compromise efforts to increase productivity.

Underground observations in conjunction with instrumentation sites were used to establish the strategic parameters required to describe the rock mass behaviour of shallow-dipping stopes on the Bushveld platinum mines. Existing rock mass rating systems were evaluated using the observations and instrumentation results. None of the current systems adequately described all the relevant geotechnical conditions. Thus a hybrid of several current systems was developed, and the proposed system is discussed in the paper.

The new rock mass rating system is termed the 'New Modified Stability Graph' system and follows the method originally described by Mathews *et al.*[1] and revised by Hutchinson and Diederichs[2]. Changes have been made to some of the tables provided by Barton[3] and to the stress analysis suggested by Hutchinson and Diederichs[2]. In addition, the logistical regression analysis has been applied to databases, formed using the suggested analyses, for three support resistances and risk levels established.

Introduction

The strategic minerals in the Bushveld Complex are located in shallow dipping, tabular reefs. The dips of the stopes generally vary between 9° and 25°, and the strength of the host rock masses invariably allows relatively large, stable stope spans to be developed. However, several large falls of ground (FOGs) occur annually, causing substantial production losses. These FOGs also comprise safety risks and compromise efforts to increase productivity. There is a need, therefore, for a methodology to design stable spans in combination with appropriate support systems that will ensure stability under a variety of rock mass conditions, for the duration of the mining period.

Potvin[4] suggested three fundamental aspects to be considered in an engineering rock mechanics design:

- Characterization of the rock mass to identify the relevant geotechnical properties and components affecting the rock mass behaviour
- Effects of stress
- Size, geometry and relative orientation of excavations with regard to the rock mass (including joint directions) and stress field.

Many of these aspects can be assessed via rock mass rating systems. This paper identifies important geotechnical parameters for the assessment of shallow-dipping stopes, and suggests a suitable rating system for span and support design of stopes on the Bushveld Complex platinum mines. The focus is on stopes not subjected to systematic stress induced fracturing, which limits the stoping depth to about 1 400 m.

Kirkaldie[5] identified a total of 28 parameters present in rock masses, which may influence their strength, deformability, permeability or stability behaviour. Of these parameters, 10 are related to rock material properties, 10 to properties of discontinuities, and 8 are hydro-geological properties. Because it is often difficult or impossible in a general characterization to include the many variables in such a complex natural material, it is necessary to develop suitable systems or models in which the complicated reality of the rock mass can be simplified through a selection of only a certain number of significant parameters.

Defining the most important geotechnical parameters for shallow-dipping stopes

The selection of appropriate geotechnical parameters was primarily determined from several SIMRAC research projects carried out by CSIR Miningtek. These projects included

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underground instrumentation and observations in platinum mines in the Bushveld Complex. The results of the investigations are detailed in Watson[6] and the parameters considered important are illustrated in photographs and summarized below:

- *Joint roughness*—If the critical joint set is planar with either slickensided, smooth or gouge filled characteristics, it is the type of joint set that could cause a panel collapse (see Figure 1)
- *Joint alteration*—Roberts[7] suggested that panel collapses often occur when J_a is greater than 3 (using the Q-rating system) (see Figure 1)
- *Joint dip angle*—With respect to the orientation of the stope (this is an important consideration for the evaluation of shallow dipping, tabular stopes) (see Figure 2)
- *Joint persistency*—This factor would be in multiples of 10 metres. Roberts[7] suggested that a factor of 2 or greater would be considered persistent enough to cause a panel collapse if pillar lines were sub-parallel to such a joint set (see Figure 3)
- *Stress condition*—Many of the observed collapses on the Bushveld platinum mines resulted from relatively high horizontal stress in the hangingwall (high *k*-ratios), creating curved fractures from which collapses occurred (see Figure 4). Very low stress conditions are also problematic, often resulting in blocks sliding out of the hangingwall even in otherwise good rock mass conditions (see Figure 5).

Ozbay and Roberts[8] suggested that faults or persistent joints striking parallel to pillar lines (see Figure 6) are considerably less stable than those perpendicular to the pillar lines. Thus, the issue of discontinuity orientation is raised.

Figure 1—Photograph showing a major FOG from a smooth discontinuity with highly altered surfaces

Literature search to find the most suitable rating system

Fourteen rock mass rating systems were considered as candidates for assessing geotechnical conditions in Bushveld mines. None of these methods adequately described all the observed geotechnical conditions and failure mechanisms. The final conclusion was that a hybrid system would provide the best results. Each of the rating systems was applied to

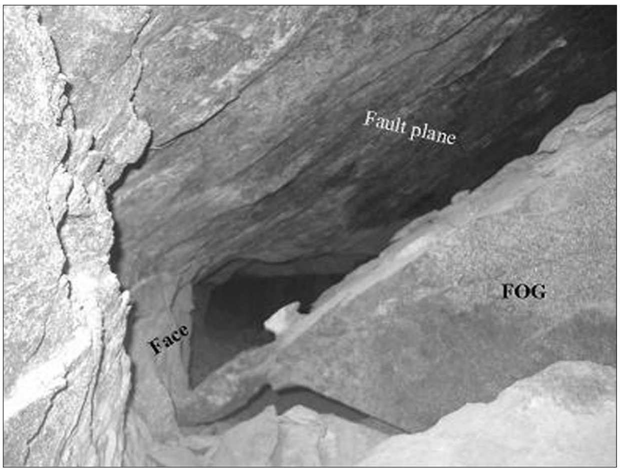


Figure 2—Photograph showing a major FOG from a shallow-dipping geological discontinuity

Figure 3—Photograph showing a collapse that occurred from a persistent, unfavourably orientated joint (Watson, 2003)

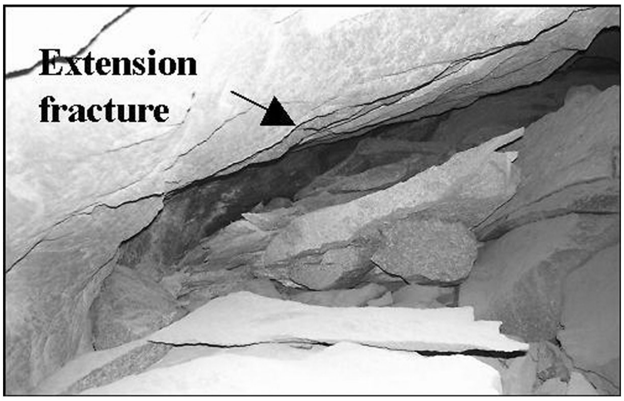


Figure 4—Photograph showing a FOG that occurred from shallow dipping extension fractures that formed at a face (Watson, 2003)

Figure 5—Photograph showing a FOG that occurred due to low confinement conditions

stopes that had collapsed and for which the failure mechanism was known. In this process a number of these systems could be immediately rejected as not being applicable. None of the remaining methods rated all of the parameters. On examination of the remaining systems, it was decided that a modified Q-system would best describe the joint properties, blockiness and stress conditions. Parameters from the stability graph method best evaluate joint orientation, relative to the stope hangingwall. These were then incorporated into the new method as described below. The system was required to provide appropriately weighted values to the parameters that defined the conditions

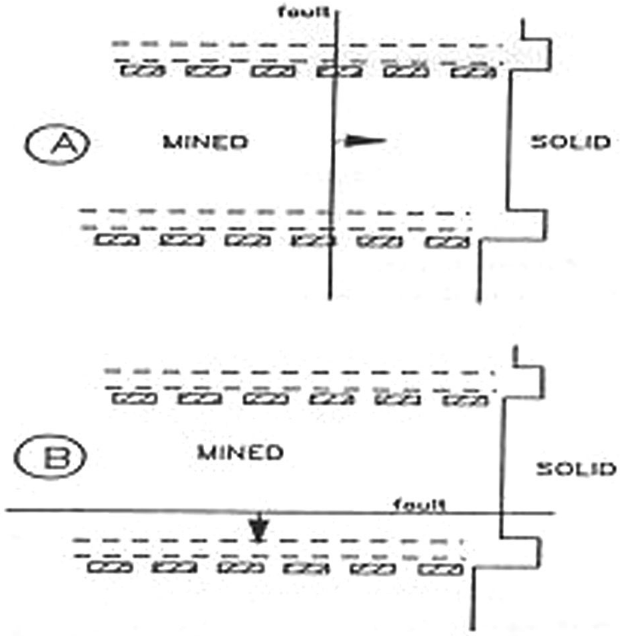


Figure 6—Favourable (A) and unfavourable (B) orientations of fault planes with respect to in-stope pillar layouts

experienced on the Bushveld mines, while remaining simple enough to be used by semi-skilled observers.

Description of the proposed hybrid rock mass rating system

The proposed method is termed the 'New Modified Stability Graph' (N") system. It is intended specifically for span and support design in stopes of the Bushveld mines and is simple, unambiguous, and capable of yielding repeatable results that conform to the physical conditions of the stopes. The system uses aspects from various rating systems, specifically a rating system designed and currently used by Impala Platinum and described by Watson and Noble[9], the Q-system (described by Barton *et al.*[3], Barton[10], and Barton[11]), and the Stability Graph method as revised by Hutchinson and Diederichs[2].

The system consists of five factors:

- A measure of block size for a jointed rock mass $(\frac{RQD}{J_n})$.
- A measure of joint surface strength and stiffness $(\frac{J_r}{J_a})$.
- A measure of the stress condition $(\frac{J_w}{SRF})$
- A measure of the joint orientation relative to the excavation hangingwall (B-factor)
- A measure of the influence of gravity on the hangingwall blocks (C-factor)

A detailed description of data collection and analyses is provided below.

Procedure to determine N"

Data collection

A 5 m x 5 m window is marked on the hangingwall and all

Geotechnical mapping of rock structure

Mine:

Date: Depth below surface:

Panel no.: k-ratio:

Discontinuities									Infill		Rock				Stress		
Set Number	Type	Importance	No. in 5 m	Dip Angle	Dip Direction	Length (m)	Limits Visible	Roughness	Type	Thick	Type	Water	Weathering	RQD	Fracturing Y or N	Fracture Angle	comments

Figure 7—Data collection form

the joint sets within the block are geotechnically evaluated and the results entered into the form reproduced in Figure 7. The importance of each set is determined by the persistency and orientation, i.e. the set most likely to cause a large FOG or collapse is rated as the most critical. This set is used to define factors such as joint roughness and alteration.

The *RQD* is estimated from the number of discontinuities per unit volume as suggested by Palmström[12].

Joint roughness is determined using offsets, measured from a 0.5 m long ruler placed across the joint surface as described in Watson[6] (see Figure 8). The largest offset is related to the 'joint roughness number' (J_r) in Figure 9. This method was established to reduce the ambiguity of personal interpretation.

The *k*-ratio may be estimated by measuring the ratio of horizontal to vertical dimensions of the sockets in a nearby haulage, travelling way or raise. Although this method is not accurate, some idea of the stress conditions at the time of development can be achieved from a significant number of measurements. (This estimation is most accurate under isotropic, homogeneous conditions.)

Calculations

N" is defined as follows:

$$N'' = Q \times B \times C \qquad [1]$$

where:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF} \qquad [2]$$

B = A modified Mathew's factor for joint orientation.

C = A modified Mathew's factor for the effects of sliding and gravity.

Note that some of the tables used to define the original Q-system (as described by Barton *et al.*[3]) have been altered to cater for specific Bushveld platinum stoping problems. Some of the alterations were defined by the Impala Platinum System, as described by Watson and Noble[9], and further changes have been made to the J_r and SRF parameter as described in Watson[6].

Figure 8—Photograph showing an offset measurement made from a 0.5 m long ruler placed across a joint surface. (The offset measurement is used to determine joint roughness.)

RQD is calculated from the Palmström[12] equation (see Equation [3]).

$$RQD = 115 - 3.3J_v \qquad [3]$$

where J_v is the average number of joints in a cubic metre.

J_n is an assigned value for the number of joint sets (see Table II).

J_r is the joint roughness number determined from offset measurements made from a 0.5 m long ruler placed across the joint surface (see Figure 9). The largest offset is used to determine the factor.

J_a is the joint alteration number (see Table III)

J_w accounts for water inflow (see Table IV)

Table I

RQD value (Barton *et al.*[3])

Description	Value
Very poor	0–25
Poor	25–50
Fair	50–75
Good	75–90
Excellent	90–100

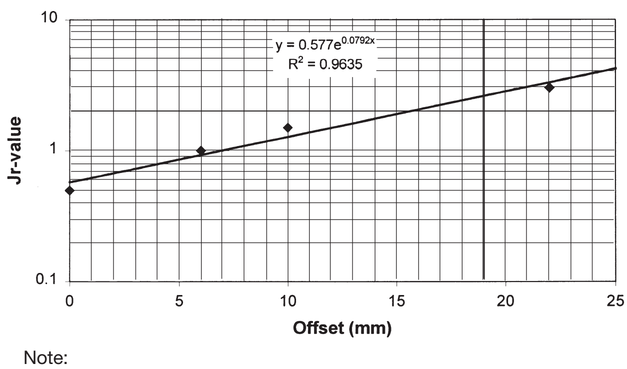
Note:
i. *RQD* ratings are assigned a value that is a multiple of 5.
ii. Where *RQD* is reported or measured as less than 10 (including 0), a nominal value of 10 is used.

Table II

Joint set number (J_n) (Barton[11])

Description	Value
No joints	0.5–1
One joint set	2
One joint set + random	3
Two joint sets	4
Two joint sets + random	6
Three joint sets	9
Three joint sets + random	12
Four or more joint sets, random, heavily jointed, 'sugar-cube', etc.	15
crushed, earth-like	20

Note: The stratification could be considered as a separate joint set where definite parting planes exist between lithologies. However, where it can be established that the support system caters for the highest potential parting plane, the stratification should be excluded from the analysis.



Note:
i. Add 1.0 to J_r if the mean spacing of the relevant joint set is greater than 3 m.
ii. J_r = 0.5 may be used if the lineations are orientated for minimum strength.
iii. J_r and J_a is applied to the joint set or discontinuity that is least favourable for stability both from a point of view of orientation and shear resistance.

Figure 9—Graph showing the relationship between J_r and offsets measured from a 0.5 m long ruler (Watson[6])

SRF accounts for the stress condition. Barton's[3] SRF values have been plotted against a strength/virgin or field stress ratio and graphs for high and low stress conditions (Figure 10 and Figure 11 respectively) have been established.

Table III

Joint alteration J_a(after Barton[3] and revised by Impala Platinum)

Description	Value
Tightly healed, hard rockwall joints, no filling	0.5
Slight infill, coating < 1 mm	1
1 mm < Joint filling < 3 mm	2
3 mm < Joint filling < 5 mm	4
Joint filling > 5 mm	6
Zones or bands of disintegrated or crushed filling.	8

Table IV

Joint water reduction factor (Jw) (Modified after Barton[11])

Condition of groundwater	Head of water (m)	Value
Dry excavation		1,0
Damp or dripping.	10–25	0,66
Large inflow	>25	0,5

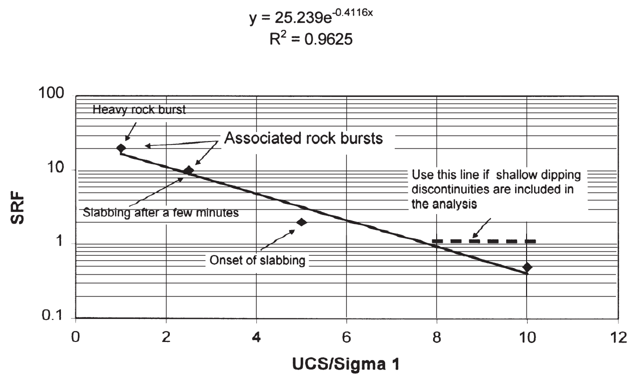


Figure 10—Stress reduction factor for high stress or low strength conditions (modified after Barton et al.[3])

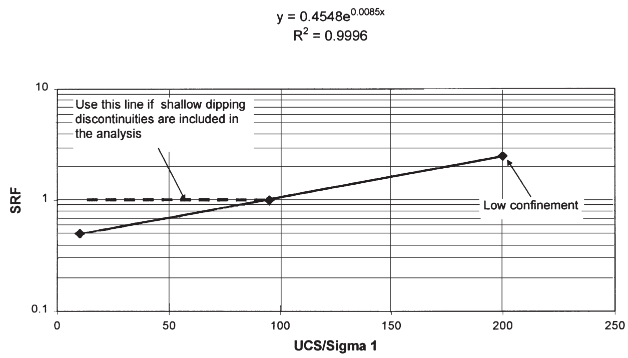


Figure 11—Stress reduction factor for low stress or high strength conditions (modified after Barton *et al.*[3])

Note that stress levels greater than an eighth or less than a hundredth of the UCS of the hangingwall rock would effect the overall rating negatively, i.e. lower the rock mass rating.

It should be noted that the geotechnical evaluation system was developed for shallow to intermediate depth mining, and the very high stress conditions shown in Figure

10 apply to pockets of high stress (usually shallow dipping) within a comparatively lower stress environment. At depth, systematic, steeply dipping fracturing will result in clamping stresses in the hangingwall and these stresses usually lead to increased stability.

Figure 11 applies to low stress environments, particularly where the *k*-ratio is low. Under these conditions there is little confinement on the steeply inclined discontinuities and blocks are able to slide out easily. Most of the shallow depth mining operations on the Bushveld platinum mines are affected by high *k*-ratios and can therefore be classified as favourable stress conditions, and therefore the less than unity provision to the right of Figure 10 and left of Figure 11 apply.

Note: where shallow dipping discontinuities are present, an SRF value of less than unity should not be used; i.e. in this case use SRF=1.

The stress conditions where stopes are located between an eighth and a hundredth of the UCS of the hangingwall rock are considered to be favourable for stability (by providing confinement to the steeply dipping discontinuities). In an environment of only vertical discontinuities, the negative effects of stress probably only manifest once fracturing occurs above the face. In the absence of shallow-dipping discontinuities or fractures, therefore, a value of unity or less seems applicable. However, where there are shallow dipping discontinuities, the negative effects of stress will be felt at much lower stresses (particularly at high *k*-ratios). (Shallow-dipping discontinuities are considered to be features inclined at less than 45° to the dip of the strata.) Where shallow-dipping discontinuities intersect the hangingwall or stress fracturing occurs above the stope face, the stress reduction, as shown in Figure 10, is probably applicable.

The A-factor for stress, originally proposed by Mathews *et al.*[1], was replaced by the J_w and *SRF* parameters, for the following reasons:

- It does not cater for the positive confining effects of moderate stress or the negative, destabilizing effects of very low stress, both of which play a significant role in stability on the Bushveld platinum mines
- It is determined from calculations made at the centre of a proposed excavation, whereas most of the stress related problems on the Bushveld platinum mines appear to emanate from the edge of the excavations
- Some of the Bushveld platinum mines experience water problems, which are not addressed by the stability graph method as revised by Hutchinson and Diederichs[2].

The B-factor describes the influence of discontinuity dip angle, with respect to the stope hangingwall (see Figure 12).

The C-factor describes the influence of gravity on the hangingwall blocks (see Figure 13).

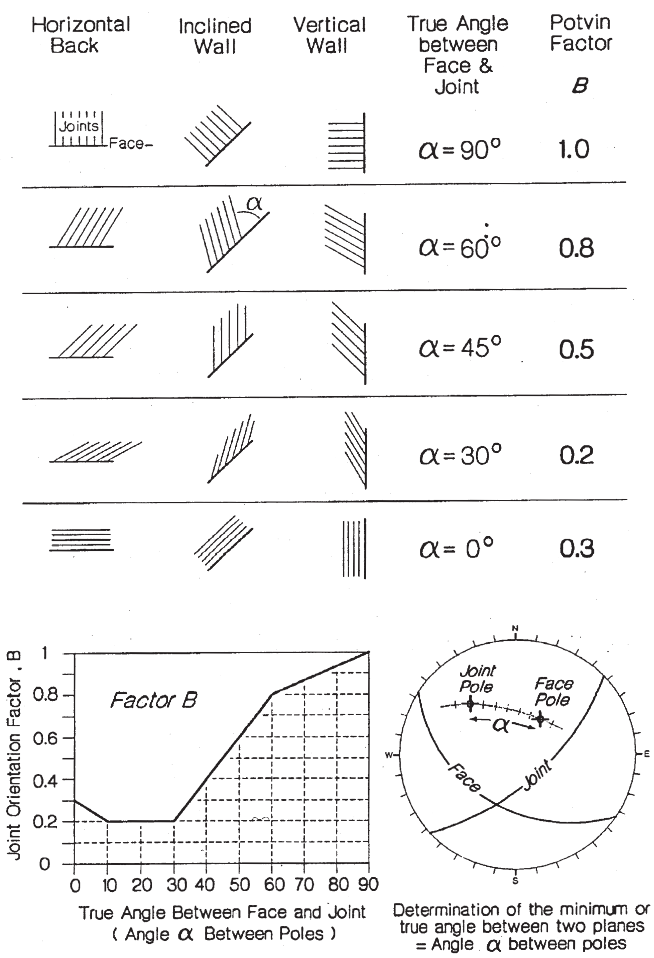


Figure 12—Determination of Joint Orientation Factor B, for stability graph analysis (after Mathews *et al.*[1] and revised by Hutchinson and Diederichs[2])

Development methodology of the N" system

Potvin[4] and Nickson[13] collected case histories of supported and unsupported open stopes. The databases were plotted separately and the effects of the cablebolt support could clearly be seen, i.e. geotechnical conditions being equal, larger stable spans were possible in the supported stopes.

Three of the more common support systems, in terms of their support resistances, used on the Bushveld platinum mines were evaluated to establish the effect of support resistance on maximum stable span:

- 48 kN/m^2 = 160 mm diameter mine poles spaced 1.5 m x 2 m
- 82 kN/m^2 = 200 mm diameter mine poles spaced 2 m x 2 m
- 128 kN/m^2 = Grout packs spaced 6 m x 6 m and 160 mm diameter mine poles spaced 1.5 m x 2 m or end-grain mat packs spaced 3.75 m x 4.35 m with four 200 mm diameter mine poles in 11m^2.

The support resistances of the three configurations were determined from underground and laboratory tests performed on individual elements, which were analysed to determine effective system resistances (the results are shown in Watson[6]). Databases of stable, unstable and collapsed stopes were collected. (Unstable refers to panels with FOGs, where a collapse appeared immanent.)

Most classification methods define stability with respect to smallest plan dimension of span. This is because these methods were designed to evaluate tunnels where the long span can be assumed to be infinite and therefore the short span is the critical dimension. If the long span is less than about five times the shorter span, stability increases as a result of the increased confinement and rigidity provided by the extra two abutments. Most of the collapses in the Bushveld platinum databases occurred at span ratios of less than 5:1 and therefore an analysis that includes the effects of all abutments was required for the study. The hydraulic radius (HR) is a function representing the size and shape of excavations (see Equation [4]) and according to Hutchinson and Diederichs[2] 'more accurately accounts for the combined influence of size and shape on excavation stability'. (For example, HR predicts similar conditions in 20 m x 20 m and 15 m x 30 m panels.) Values of N" were plotted against HR, where N" and HR are plotted on the y-axis and x-axis respectively. The HR is defined as the area divided by the perimeter of the excavation.

$$HR = \frac{w \times h}{2w + 2h} \qquad [4]$$

The logistical regression analysis was used by Trueman *et al.*[14] to analyse databases of stable and collapsed stopes from Australian mines. This type of analysis is appropriate to data sets with a binary dependant variable and a number of

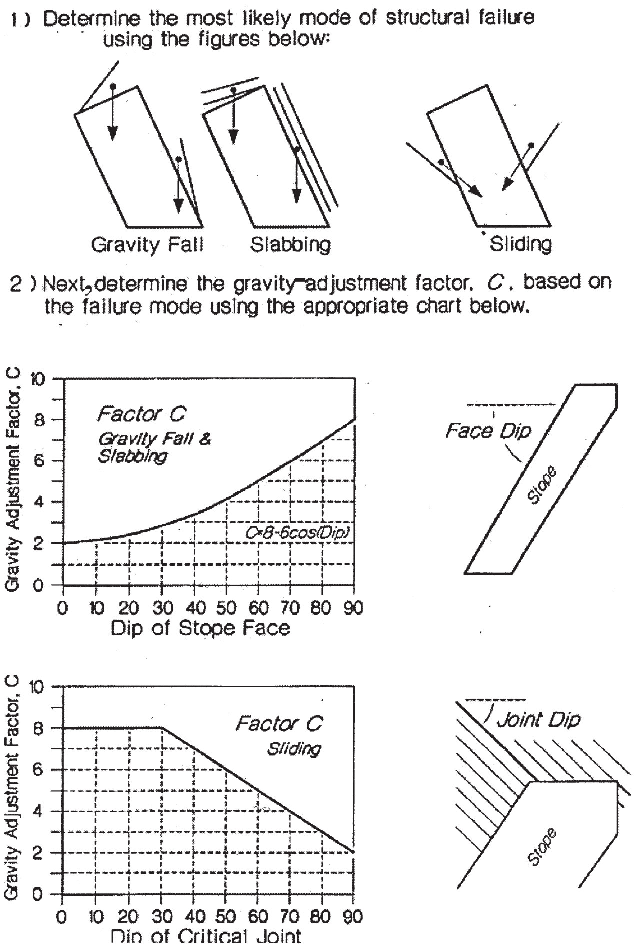


Figure 13—Determination of gravity adjustment factor, C, for stability graph analysis (after Mathews *et al.*[1] and revised by Hutchinson and Diederichs[2])

numerical independent variables. An advantage of this method of analysis in this application is that it is driven mainly by the data for the points that plot in the overlap area between stable and collapsed cases, while reducing the effect of data from points that plot far from the overlap area. Thus stopes that were not mined to the maximum stable span have little bearing on the location or orientation of the regression line. The general form of the logistical regression described by Charles[15] is shown in Equation [5].

$$Probability_{(stability)} = \frac{1}{\left(1+e^{-z}\right)} \qquad [5]$$

where: $z = k_o + k_1x_1 + ... + k_nx_n$

x_i are the independent variables, which in this analysis were the logarithms of N" and HR.

k_i values were estimated by an iterative process to maximize the likelihood function presented in Equation [6]. The likelihood function evaluates the probability of observing the given data. Its value is determined as the sum of:

- the departure of probability of stability from one for stable cases
- the departure of probability of stability from zero for 'not stable' cases.

It should be noted that the 'not stable' data points include both unstable and collapsed sites.

$$Likelihood = \sum_{Not\ stable} \log\left(1 - Probability_{(stability)}\right) + \sum_{stable} \log\left(Probability_{(stability)}\right) \qquad [6]$$

The result of maximizing the likelihood function is to set the k_i's such that, on average over the entire database, the probability of stability of stable panels is as close as possible to one, while the probability of stability of 'not stable' panels is as close as possible to zero.

The logistical regression method was applied to the data collected from the Bushveld platinum mines and levels of risk were assessed. Thus stability graphs were generated as shown in Figure 14 (48 kN/m2), Figure 15 (82 kN/m2) and Figure 16 (182 kN/m2). The k_i values used in the three curves are listed below:

$$Probability_{(stability)} = \frac{1}{\left(1+e^{-\left(\begin{smallmatrix}9.0234-3.0826\ln(HR)\\+1.2249\ln(N")\end{smallmatrix}\right)}\right)}$$

for Figure 14

$$Probability_{(stability)} = \frac{1}{\left(1+e^{-\left(\begin{smallmatrix}8.3168-3.4554\ln(HR)+\\2.6163\ln(N")\end{smallmatrix}\right)}\right)}$$

for Figure 15

$$Probability_{(stability)} = \frac{1}{\left(1+e^{-\left(\begin{smallmatrix}8.0668-2.1542\ln(HR)+\\2.6802\ln(N")\end{smallmatrix}\right)}\right)}$$

for Figure 16

Note the poor separation of the collapsed and stable stopes shown in Figure 14. An explanation for the poorly

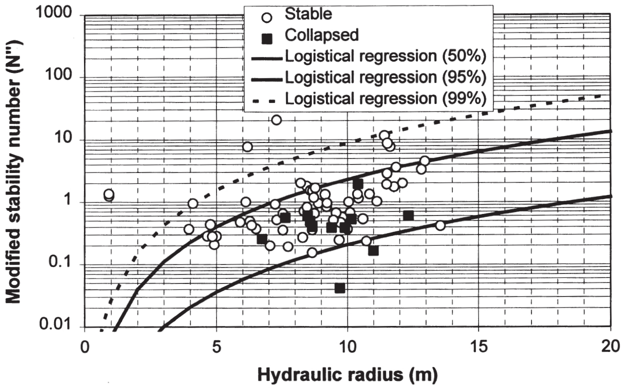


Figure 14—New modified stability graph plot of stopes supported on mine poles with a support resistance of 48 kN/m2

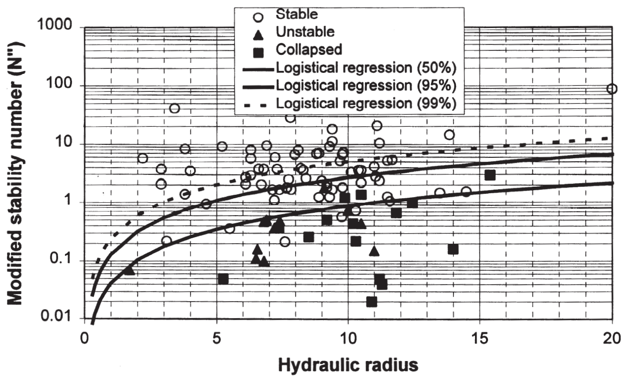


Figure 15—New modified stability graph plot of stopes supported on mine poles with a support resistance of 82 kN/m2

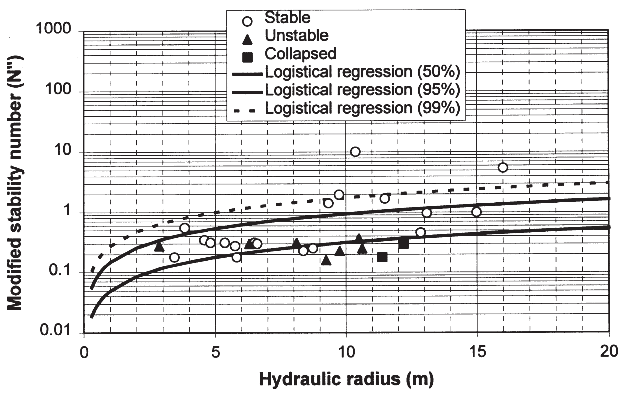


Figure 16—New modified stability graph plot of stopes supported on mine poles with a support resistance of 128 kN/m2

defined region of failure in Figure 14 is the susceptibility of the 160 mm diameter mine poles to blast and scraper damage, and there is a higher propensity for premature failure due to poor installation than with the 200 mm diameter mine poles. Some miners counter the adverse affects of premature failure by installing a higher-than-required density of mine poles. Thus, the effective support resistance is higher than recorded. In addition, other miners neglect to replace poles that have been blasted out or damaged, thus these stopes are under-supported and have a lower support resistance than assumed. These factors provide some uncertainty to the results shown in Figure 14. Previous work performed by Daehnke *et al.*[16] on 160 mm diameter mine poles has shown a large scatter in the strength properties of these elements underground, mainly due to blast damage and installation technique. Thus the average load bearing capacity of the system, as a whole, was relatively low, i.e. the effective support resistance was lower than expected.

The results appear unrealistic at the lower and upper ends of the three N" curves due to the limited data in these areas. However, in the region of high confidence (HRs between 6 and 15) the analyses seem reasonable.

The N" analysis may be used to determine risk levels of panel stability by plotting the relevant HR against the N"-value on the graph with the appropriate support resistance, i.e. Figure 14 (48 kN/m^2), Figure 15 (82 kN/m^2) or Figure 16 (182 kN/m^2). Panel spans, which result in HR values that plot below or to the right of the regression lines have a higher risk of collapse than those that plot above or to the left of these lines. Risk levels can be reduced by:

- Decreasing spans i.e. the plot moves to the left (HR is reduced)
- Increasing the support resistance i.e. the panel is plotted on a graph representing a higher support resistance.

Stability analysis with stratification

The N" system does not differentiate between the heights of potential parting planes above a stope, and some engineering judgement is required to determine whether higher planes should be included in the analysis. The problem can be dealt with in one of two ways using the N" method:

- Employing an adequate support resistance to cater for the height of the highest potential parting plane. If this is the case, then the span is determined using N" in conjunction with the appropriate N"-curve (see above), without including the parting planes in the analysis
- In the case where one parting plane exists at a height where the support resistance is unable to carry the deadweight, then the plane should be analysed in terms of the 'B'-factor. However, if a set of planes exist (e.g. stratification) the discontinuities should also be included as an additional joint set in the J_n parameter.

It should be noted that the relationship between span and the maximum height at which parting is likely to occur, is not known. This is a problem not only to the N' method but to all methods that rely on beam theory.

Limitations

The relatively small amount of collapsed data in the N"-graphs shown above (particularly for the data representing a support resistance of 128 kN/m^2) means that the exact positions of the probability lines are subject to uncertainty.

An apparent limitation could be that the N" methodology does not cater for domes of diameters less than about three-quarters of the panel span. It is believed that no system is capable or appropriate to deal with their random occurrence problem by reducing spans. Spans would be made too small for the general case. Other means of stabilizing small domes, such as cutting additional pillars or installing strong support under the dome, would appear to be a better solution. Large domes (greater than three-quarters of the panel span) should be catered for as per faults.

Conclusions

A methodology for designing stable panel spans on the Bushveld platinum mines has been developed using a modified version of the rock mass rating method originally described by Mathews *et al.*[1] and revised by Hutchinson and Diederichs[2]. The new rock mass rating system is termed the 'New Modified Stability Graph'. The logistical regression analysis has been applied to databases, formed using the suggested analyses, for three support resistances and risk levels established.

The 'New Modified Stability Graphs' may be used to determine risk levels of panel stability by plotting the relevant HR against the N"-value on the graph with the appropriate support resistance. Panel spans that result in HR the values plot below or to the right of the regression lines have a higher risk of collapse than those that plot above or to the left of these lines. Risk levels can be reduced by decreasing spans or increasing support resistance.

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References

1. MATHEWS, K.E., HOEK, E., WYLLIE, D.C., and STEWART, S.B.V. Prediction of stable excavations for mining at depths below 1000 m in hard rock. *CANMET Report. DSS Serial No. OSQ80-00081, DSS File No. 17SQ. 23440-0-9020*. Ottawa: Dept. Energy, Mines and Resources. 1981.
2. HUTCHINSON, D.J. and DIEDERICHS, M.S. *Cablebolting in underground mines*. BiTech Publishers Ltd, Canada, 1996. pp. 406.
3. BARTON, N., LIEN, R., and LUNDE, J. Engineering classification of rock masses for the design of tunnel support, *Rock Mech.*, vol. 6, no. 4, (1974). pp. 189–236.

4. POTVIN, Y. Empirical open stope design in Canada. Ph.D. Thesis, Dept. of Mining and Mineral Processing, University of British Columbia, 1988. pp. 343.
5. KIRKALDIE, L. Rock classification systems for engineering purposes. *STP 984,* Amer. Society for Testing Materials, 1988. pp. 167.
6. WATSON, B.P. The feasibility of using rock mass ratings for the design of panel spans and support in the shallow to intermediate depth Bushveld Platinum Mines, M.Sc. project, Dept. of mining engineering, University of Witwatersrand, Johannesburg. RSA. 2003.
7. ROBERTS, M.K.C. Addressing the problem of joints sub parallel to pillar lines. Contract report number 2002–0339, Johannesburg. RSA. 2002.
8. OZBAY, M.U. and ROBERTS, M.K.C. Yield pillars in stope support, *Proc. of the SANGORM Symposium on Rock Mechanics in Afric*a. RSA, November, 1988.
9. WATSON, B.P. and NOBLE, K.R. Comparison between geotechnical areas on the Bushveld Complex platinum mines, to identify critical spans and suitable in-panel support. *Proc. SARES97* Johannesburg. RSA, 1997. pp. 440–451.
10. BARTON, N. Rock mass characterisation workshop. Workshop notes. 1997.
11. BARTON, N. Some new Q-value correlations to assist in site characterisation and tunnel design, *International Journal of Rock Mechanics and Mining Sciences*, vol. 39, 2002. pp. 185–216.
12. PALMSTRÖM, A. The volumetric joint count – A useful and simple measure of the degree of rock jointing. *Proc. 4th Congr. Int. Assn. Engng*. Geol., Delhi, vol. 5, 1982. pp. 221–228.
13. NICKSON, S.D. Cable support guidelines for underground hard rock mine operations. M.Sc. Thesis, Dept. Mining and Mineral Processing, University of British Columbia, 1993. pp. 223.
14. TRUEMAN, R., MIKULA, P., MAWDESLEY, C., and HARRIES, N. Experience in Australia with the application of the Mathews' method for open stope design. *CIM Bulletin*, vol. 93, no. 1036. 2000. pp. 162–167.
15. CHARLES, M.F. Logistical regression (on line), www.shsu.edu/~icc_cmf/newCJ742/6LOGITnew.doc (accessed 04-11-2002).
16. DAEHNKE, A., WATSON, B.P., ROBERTS, D. ACHEAMPONG, E., and VAN ZYL, M. The impact of soft loading conditions on the performance of elongate support elements. *SIMRAC GAP 613 final report*. 2000. CSIR. Division of Miningtek, Johannesburg. RSA. ◆

Brisbane event backs university-based revival for minerals sector*

The future technical needs of the global minerals sector appear to be in good hands based on the amount of interest from student researchers from around the globe who will participate in a unique minerals industry conference to be held in Brisbane, Australia, in September 2004.

The programme for the JKMRC International Student Conference reflects the broadening research and technical interests of young professionals who will be leading the industry in a few years from now.

Although the breadth and depth of interest from young researchers in the field is encouraging, the dwindling numbers of students taking up professions in the mining industry remains a concern.

Conference convenor Mr David Goeldner said the event was timely in the context of the mining industry's determination to be responsive to broader community interests and to take another look at its traditional engineering disciplines to secure a sustainable future.

'As far as I am aware, this is the first time students from around the world engaged in minerals industry research will present their research on such a high profile and public platform in Australia,' he said.

'It's also encouraging to see a blend of research activities being undertaken by universities around the globe, which take into account the mining sector's desire to be socially responsible and to reinvent itself as a clean, green industry with a sustainable future.'

Mr Goeldner said meeting and sustaining these objectives is very much tied to the nature of research being done at universities and large research and development organizations around the globe.

'I think many people would be surprised to learn that much of what can be achieved to make the minerals sector safer, greener, and more socially aware, is based on technical and engineering know-how that has been around for some time,' he said.

'That means many of the traditional minerals engineering programmes that have started to dwindle around the globe should be looked at again and revived.'

PhD and Masters' students from the United Kingdom, Finland, Indonesia, Turkey, South Africa, Tanzania, USA, Canada, and around Australia will present their work over two days before an expected delegation of senior government and mining company representatives, leading academics, and fellow researchers.

Topics being presented include energy use reduction in mining and milling, intelligent clothing for health monitoring, the use of virtual reality in underground mining, and reducing dust emissions from processing plants.

Themes consistent through the conference will be the importance of nurturing technical know-how, and making the mining sector evermore efficient in its use of global mineral resources.

The JKMRC International Student Conference is supported by the University of Queensland's Sustainable Minerals Institute and Minerals Engineering International, and will be held at the Brisbane Convention and Exhibition Centre from 6–7 September 2004.

For further information go to www.jkconference2004.com.. ◆

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