The Effect of Stress Damage on Dilution in Narrow Vein Mines

P Stewart¹, J Slade² and R Trueman³

ABSTRACT

Unplanned dilution has the potential to seriously undermine the economic viability of a mine, and in some cases results in mine closure. The effect of stress damage on dilution has become increasingly relevant as mining depths increase. In the case of narrow vein mining, the incremental extraction of long-hole rings has the potential to result in a moving high stressed zone at the stope brow. This leads the hanging wall and footwall to experience a spike in the stress to strength ratios as the brow passes. In some cases, the stress to strength ratio may be high enough to result in fracturing or damage to the rock mass.

The aim of the study described in this paper was to investigate whether stress damage results in a significant increase in dilution. The study involved analysis of overbreak from 410 case studies from the Kundana Gold operations in Western Australia. Site personnel had already undertaken calibration studies of the stress levels that result in rock mass damage. This calibration, in conjunction with numerical modelling showed that stress damaged stope walls at this mine had on average 50 per cent more overbreak than stope walls where stresses had not exceeded the damage criterion. For a design mining width of 1.5 metres, and with both walls impacted, this represents 36 per cent increase in dilution. After adjustment for possible sources of bias the difference reduced to an average 0.10 metres per stope wall, representing 13 per cent dilution for the mining width under consideration. The potential for stress damage related overbreak should therefore be considered as part of any assessment of narrow vein dilution.

INTRODUCTION

Narrow vein mines are particularly susceptible to high dilution levels. This is because narrow vein dilution is more sensitive to overbreak. For example, one metre of overbreak beyond the stope design limit represents 50 per cent unplanned dilution in a two metre wide orebody (or 33 per cent dilution if dilution is calculated as a percentage of material taken to the Run of Mine pad. In contrast, one metre of overbreak represents only ten per cent dilution in a ten metre wide orebody (or nine per cent as a proportion of material taken to ROM pad). While the impact of dilution on NPV is site-specific, dilution increases mining and milling costs, as well as potentially decreasing mill recovery and proportion of material taken to ROM pad. While the impact of dilution on NPV is site-specific, dilution increases mining and milling costs, as well as potentially decreasing mill recovery and proportion of material taken to ROM pad.

It has been hypothesised that stress damage affects narrow vein dilution. Understanding the circumstances in which stress damage affects dilution enables operators to minimise dilution within the limits of economically practicable mining. Many of the causes of unplanned dilution involve significant cost to reduce their impact. In contrast, the potential to avoid stress damage related dilution through combined geotechnical and mining engineering teamwork is high. In addition, the strategies employed to reduce stress concentration may also improve mineability and reduce ore losses as discussed by (Beck and Sandy, 2002; Beck and Sandy, 2003).

Post-stopping stresses are taken into account in all variants of the Stability Graph approach (Mathews et al, 1981; Potvin, 1988; Clark and Pakalnis, 1997; Mawdsley et al, 2001) but the stability graph approach does not take into account the stresses experienced by stope walls prior to stoping. However, Sprott et al (1999) propose an adjustment to the stability graph approach to account for pre-mining or virgin stresses. Sprott et al (1999) stress damage adjustments were successfully applied at the three large open-stopping Hemlo operations in Canada to predict stope stability and evaluate alternative extraction sequences. However, this approach does not consider the full stress history experienced by the stope wall, only the pre-mining stress. Therefore, this method does not take into account spikes in the stress history which may exceed the pre-mining stress.

It has been hypothesised that stope walls adjacent to the brow experience a spike in stress as the brow passes, which in the case of a shrinking central pillar extraction sequence potentially results in stress damage to the adjacent hanging wall and footwall. Narrow vein stope extraction involves relatively small mining increments along strike and therefore brow stresses have the potential to affect large sections of hanging wall and footwall. Figure 1 illustrates how brow stresses can affect large sections of hanging wall and footwall.

The aim of this study was to examine the effect of stress damage on dilution, as well as investigate how extraction sequence impacts on brow stresses in narrow vein mines. This has been achieved by back-analysing the stress path of 410 stope walls from the Barkers orebody. Barkers mine is one of two underground mines at the Kundana Gold operations in Western Australia. The stress path was obtained from linear elastic boundary element software. Each stage in the sequence corresponds to one month.

1. MAusIMM, Currently: Senior Geotechnical Consultant, AMC Consultants, Level 8, 135 Wickham Terrace, Brisbane Qld 4000. E-mail: pstewart@amcconsultants.com.au Formerly: JKMR, The University of Queensland, Ipsen Road, Indooroopilly Qld 4068.
2. MAusIMM, Senior Geological Engineer, Coffey Geosciences Pty Ltd, 14B Henley Beach Road, Mile End SA 5031.
3. JKMR, The University of Queensland, Ipsen Road, Indooroopilly Qld 4068.

FIG 1 - Long section: Incremental extraction exposes large areas of hanging wall and footwall to brow stress levels.

The Effect of Stress Damage on Dilution in Narrow Vein Mines

P Stewart¹, J Slade² and R Trueman³

ABSTRACT

Unplanned dilution has the potential to seriously undermine the economic viability of a mine, and in some cases results in mine closure. The effect of stress damage on dilution has become increasingly relevant as mining depths increase. In the case of narrow vein mining, the incremental extraction of long-hole rings has the potential to result in a moving high stressed zone at the stope brow. This leads the hanging wall and footwall to experience a spike in the stress to strength ratios as the brow passes. In some cases, the stress to strength ratio may be high enough to result in fracturing or damage to the rock mass.

The aim of the study described in this paper was to investigate whether stress damage results in a significant increase in dilution. The study involved analysis of overbreak from 410 case studies from the Kundana Gold operations in Western Australia. Site personnel had already undertaken calibration studies of the stress levels that result in rock mass damage. This calibration, in conjunction with numerical modelling showed that stress damaged stope walls at this mine had on average 50 per cent more overbreak than stope walls where stresses had not exceeded the damage criterion. For a design mining width of 1.5 metres, and with both walls impacted, this represents 36 per cent increase in dilution. After adjustment for possible sources of bias the difference reduced to an average 0.10 metres per stope wall, representing 13 per cent dilution for the mining width under consideration. The potential for stress damage related overbreak should therefore be considered as part of any assessment of narrow vein dilution.

INTRODUCTION

Narrow vein mines are particularly susceptible to high dilution levels. This is because narrow vein dilution is more sensitive to overbreak. For example, one metre of overbreak beyond the stope design limit represents 50 per cent unplanned dilution in a two metre wide orebody (or 33 per cent dilution if dilution is calculated as a percentage of material taken to the Run of Mine pad. In contrast, one metre of overbreak represents only ten per cent dilution in a ten metre wide orebody (or nine per cent as a proportion of material taken to ROM pad). While the impact of dilution on NPV is site-specific, dilution increases mining and milling costs, as well as potentially decreasing mill recovery and proportion of material taken to ROM pad. While the impact of dilution on NPV is site-specific, dilution increases mining and milling costs, as well as potentially decreasing mill recovery and proportion of material taken to ROM pad.

It has been hypothesised that stress damage affects narrow vein dilution. Understanding the circumstances in which stress damage affects dilution enables operators to minimise dilution within the limits of economically practicable mining. Many of the causes of unplanned dilution involve significant cost to reduce their impact. In contrast, the potential to avoid stress damage related dilution through combined geotechnical and mining engineering teamwork is high. In addition, the strategies employed to reduce stress concentration may also improve mineability and reduce ore losses as discussed by (Beck and Sandy, 2002; Beck and Sandy, 2003).

Post-stopping stresses are taken into account in all variants of the Stability Graph approach (Mathews et al, 1981; Potvin, 1988; Clark and Pakalnis, 1997; Mawdsley et al, 2001) but the stability graph approach does not take into account the stresses experienced by stope walls prior to stoping. However, Sprott et al (1999) propose an adjustment to the stability graph approach to account for pre-mining or virgin stresses. Sprott et al (1999) stress damage adjustments were successfully applied at the three large open-stopping Hemlo operations in Canada to predict stope stability and evaluate alternative extraction sequences. However, this approach does not consider the full stress history experienced by the stope wall, only the pre-mining stress. Therefore, this method does not take into account spikes in the stress history which may exceed the pre-mining stress.

It has been hypothesised that stope walls adjacent to the brow experience a spike in stress as the brow passes, which in the case of a shrinking central pillar extraction sequence potentially results in stress damage to the adjacent hanging wall and footwall. Narrow vein stope extraction involves relatively small mining increments along strike and therefore brow stresses have the potential to affect large sections of hanging wall and footwall. Figure 1 illustrates how brow stresses can affect large sections of hanging wall and footwall.

The aim of this study was to examine the effect of stress damage on dilution, as well as investigate how extraction sequence impacts on brow stresses in narrow vein mines. This has been achieved by back-analysing the stress path of 410 stope walls from the Barkers orebody. Barkers mine is one of two underground mines at the Kundana Gold operations in Western Australia. The stress path was obtained from linear elastic boundary element software. Each stage in the sequence corresponds to one month.

1. MAusIMM, Currently: Senior Geotechnical Consultant, AMC Consultants, Level 8, 135 Wickham Terrace, Brisbane Qld 4000. E-mail: pstewart@amcconsultants.com.au Formerly: JKMR, The University of Queensland, Ipsen Road, Indooroopilly Qld 4068.
2. MAusIMM, Senior Geological Engineer, Coffey Geosciences Pty Ltd, 14B Henley Beach Road, Mile End SA 5031.
3. JKMR, The University of Queensland, Ipsen Road, Indooroopilly Qld 4068.

FIG 1 - Long section: Incremental extraction exposes large areas of hanging wall and footwall to brow stress levels.

The aim of this study was to examine the effect of stress damage on dilution, as well as investigate how extraction sequence impacts on brow stresses in narrow vein mines. This has been achieved by back-analysing the stress path of 410 stope walls from the Barkers orebody. Barkers mine is one of two underground mines at the Kundana Gold operations in Western Australia. The stress path was obtained from linear elastic boundary element modelling of a 32 step extraction sequence using Map3D boundary element software. Each stage in the sequence corresponds to one month.
STUDY AND CONDITIONS

Location
Kundana Gold Operations are located 25 km west-northwest of Coolgardie, within the Kundana Mine Lease as shown in Figure 2. Kundana Gold Operations ceased underground production in May 2004. Mining of the Barkers and Strzelecki orebodies formed one underground mining operation. All case studies referred to in this study come from the Barkers orebody.

Geology
The Kundana Mining lease contains a sequence of rocks generally striking AMG 300° to 330° and dipping steeply west. Mineralisation is constrained within a deep crustal shear zone known as the Zuleika Shear (Slade, 2004). The general geology is comprised of a sequence of mafic to intermediate volcanics and derived sediments. The Kundana sequence is interpreted to form part of an upright isoclinal anticline between the east and west synclines (Hadlow, 1990). The dominant subvertical foliation associated with the Zuleika Shear Zone trends from 320° to 350°. The bulk of the mineralisation is in the form of thin planar laminated quartz veins which dip moderately to steeply to the west with strike lengths up to 600 m (Hadlow, 1990).

The Barkers orebody mineralisation occurs within a laminated quartz vein at the sheared contact between the footwall western facies of the Gabbro intrusion and a hanging wall felsic volcanic sediment (Reid and Bampton, 2001). The average vein dip is 70 degrees with width ranging from 0.2 to 0.7 m. Shearing extends up to 2.5 m either side of the vein.

Table 1 contains a summary of laboratory geomechanical properties of the Barkers ore and host rocks. The values shown are global estimates of each rock type based on sampling from limited exploration drill holes and data obtained from hollow inclusion cell stress measurements.

The stopes analysed for this study were limited to the panels for which detailed rockmass characterisation had been undertaken and where geotechnical domains showed high vertical consistency. Brunton and Trueman (2001) undertook detailed rockmass characterisation and scanline mapping in the 5990, 6055 and 6070 sill drives. Scanline mapping data was analysed using DIPS stereographic software and joint sets identified (Brunton and Trueman, 2001). Stereographic interpretation of joint sets resulted in identification of the five structural domains shown in Table 2. Table 2 also shows Q’ rock mass classification (Mathews et al., 2005).

<table>
<thead>
<tr>
<th>Location</th>
<th>UCS (MPa)</th>
<th>Static Young’s Modulus Av (GPa)</th>
<th>Static Young’s Modulus Av (GPa)</th>
<th>Static Poisson’s Ratio Av</th>
<th>Density (g/cm³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Barkers FW Gabbro</td>
<td>7</td>
<td>na</td>
<td>9</td>
<td>9</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>142</td>
<td>-</td>
<td>64</td>
<td>0.26</td>
<td>2.7</td>
</tr>
<tr>
<td></td>
<td>40</td>
<td>-</td>
<td>6</td>
<td>0.04</td>
<td>-</td>
</tr>
<tr>
<td>Quartz vein</td>
<td>3</td>
<td>3</td>
<td>na</td>
<td>na</td>
<td>75</td>
</tr>
<tr>
<td></td>
<td>130</td>
<td>8</td>
<td>-</td>
<td>-</td>
<td>2.71</td>
</tr>
<tr>
<td></td>
<td>6</td>
<td>1</td>
<td>-</td>
<td>-</td>
<td>0.15</td>
</tr>
</tbody>
</table>
1981). The high vertical consistency of geotechnical domains meant that it was reasonable to assume domain consistency up to two levels away (30 m) when necessary.

**In situ stresses**

Pascoe (2001) undertook hollow inclusion cell stress measurements at 319 m depth. Since this time there have been two additional series of stress measurements undertaken at Barkers, one at 505 m and one at 602 m depth. Using linear regression the three measurements were incorporated into a linear regression model of in situ stresses. However for this study, it was decided that the first measurement (319 m depth) would give a better indication of in situ stresses because the measurement was effectively in the middle of the panels considered in this study. Table 3 contains the in situ stress measurements at 319 m. The stopes analysed in this study range in depth from 205 m to 380 m below surface. While these depths seem quite moderate by international standards, the high horizontal to vertical stress ratio means that the maximum principal stress is over three times the weight of overburden. In addition, the shrinking central pillar extraction sequence results in even higher induced stresses.

### Table 2

<table>
<thead>
<tr>
<th>Domain</th>
<th>RQD</th>
<th>Jh</th>
<th>Jr</th>
<th>Js</th>
<th>Q'</th>
</tr>
</thead>
<tbody>
<tr>
<td>HWN</td>
<td>97</td>
<td>9</td>
<td>1.0</td>
<td>1.0</td>
<td>10.8</td>
</tr>
<tr>
<td>HWC</td>
<td>94</td>
<td>9</td>
<td>1.0</td>
<td>1.0</td>
<td>10.4</td>
</tr>
<tr>
<td>HWS</td>
<td>95</td>
<td>12</td>
<td>1.0</td>
<td>1.0</td>
<td>7.9</td>
</tr>
<tr>
<td>HWS1</td>
<td>86</td>
<td>6</td>
<td>1.25</td>
<td>1.0</td>
<td>17.9</td>
</tr>
<tr>
<td>HWS2</td>
<td>87</td>
<td>9</td>
<td>1.0</td>
<td>1.0</td>
<td>9.7</td>
</tr>
<tr>
<td>FWN</td>
<td>100</td>
<td>12</td>
<td>3.25</td>
<td>1.0</td>
<td>27.1</td>
</tr>
<tr>
<td>FWS</td>
<td>100</td>
<td>12</td>
<td>1.25</td>
<td>1.0</td>
<td>10.4</td>
</tr>
<tr>
<td>FZ1</td>
<td>100</td>
<td>12</td>
<td>1.25</td>
<td>1.0</td>
<td>10.4</td>
</tr>
</tbody>
</table>

Martin and Read (1996) suggest, based on the AECL Mine-by experiment and Hoek and Brown (1980) underground practical experience in massive brittle rocks that when the ratio of the far field maximum principal stress ($s_1$) to the short-term uniaxial compressive strength (UCS) exceeds approximately 0.2, crack induced damage weakens the rock. The ratio of far field in situ stress $s_1$ to the UCS on the top level of the panels analysed for this study is 0.14, while on the bottom level the ratio is 0.25. This implies that the case studies considered for this study are in the range at which stress damage would be expected.

### Table 3

<table>
<thead>
<tr>
<th>Magnitude</th>
<th>Bearing</th>
<th>Plunge</th>
</tr>
</thead>
<tbody>
<tr>
<td>$s_1$</td>
<td>26.6</td>
<td>20</td>
</tr>
<tr>
<td>$s_2$</td>
<td>17.2</td>
<td>112</td>
</tr>
<tr>
<td>$s_3$</td>
<td>15.5</td>
<td>221</td>
</tr>
</tbody>
</table>

**MINING METHOD**

The stopes considered in this study come from the four panels located between the 5960 m sill drive and the 6135 m sill drive in the Barkers mine. All bar one of the panels has three sill drives. The mining method was a combination of the bottom-up modified Avoca method using development waste as fill and longhole open stoping with small rib pillars. Longhole open stoping with small rib pillars were always used on the top levels of the panels because there is no access for filling. Stoping proceeded from both ends of the orebody on multiple levels retreating to a central pillar. Typically, levels were spaced at 15 m intervals. Rib pillars were two metres along strike and the sill pillars separating panels were between one and four metres high. Figure 3 is a schematic representation of the Barkers mining method.

A central pillar extraction sequence has the advantage of a single access being sufficient to produce ore from two stopes. From a mining point of view it make sense to reduce the number of mining accesses.
of accesses not just to reduce capital costs but also for logistical reasons such as reduced tramming time for drilling equipment, less services to run and maintain, etc. However, a possible limitation of this extraction sequence is that a shrinking central pillar is created and that as mining retreats and the pillar becomes smaller, stress related problems increase. Figure 1 and Figure 3 illustrates how with each successive stope blast the brow retreats a small amount and a new section of stope wall experiences a spike in stress levels. This paper evaluates the potential for this stress spike to cause stress damage related overbreak.

**STRESS DAMAGE**

In generic terms, rock is considered ‘damaged’ when the strength of the rock is reduced. In fracture mechanics, damage or ‘crack damage threshold’ refers to the onset of irreversible volumetric strain (Bawden, 2002a). Volumetric strain is simply a measure of rock deformation. However, Wiles (2002) uses a post peak-strength damage definition. This approach is well suited to an empirical failure criterion based damage model. In practice it doesn’t really matter, provided the damage model parameters are calibrated to underground observations. As shown in the stress-strain curve shown in Figure 4, the onset of stress damage (or yield) marks a change from linear elastic deformation to non-linear plastic deformation (Bawden, 2002a). Up until the onset of stress damage, removal of the unloading stress-strain path follows the same linear path as loading. Prior to the onset of crack damage deformations are recoverable.

**Empirical failure criterion**

Ideally, empirical failure criterion should only be used when the opportunity to calibrate model parameters to underground observations exists. Wiles (2002) quantifies stress damage in terms of ‘excess stress’ and discusses how determining an appropriate stress path to failure is an important consideration when attempting to quantify the extent of stress damage. In the case of pillar failure, Wiles (2002) suggests an increasing load type stress path. While the brow can be considered a pillar at the early stages in the extraction sequence the edge of the very elongated pillar could be considered an abutment in which case Wiles (2002) suggests changes in shear stress would be an appropriate stress path.

**Deviatoric stress based damage criteria**

A deviatoric stress based damage criterion is expressed in Equation 1 where $s_1$ and $s_3$ are the maximum and minimum principal stresses, respectively and $\sigma_{ci}$ is the in situ crack initiation stress (Kaiser, 1994, Castro et al, 1996, Martin et al, 1996).

$$\sigma_1 - \sigma_3 \leq \sigma_{ci}$$

The advantage of the deviatoric stress approach to damage estimation is its simplicity and availability of input parameters. The input parameter, $\sigma_{ci}$ can be estimated from the short-term UCS, which is usually available. The AECL Mine-by-experiment indicates that the in situ crack initiation stress occurs at about 0.3 times the UCS (Martin et al, 1996). While the deviatoric stress approach has been calibrated to rock masses (Castro et al, 1996, Martin et al, 1996). The calibration process was limited to the massive or moderately jointed rock mass observations at the AECL’s Underground Research Laboratory experimental mine (Martin et al, 1996) and the Sudbury neutrino observatory cavern (Castro et al, 1996). This calibration also indicated that damage in massive and moderately jointed rock can be directly linked to the lab tested crack initiation threshold. Laboratory testing of rock specimens predicts crack initiation occurs when the deviatoric stress is between 0.25 and 0.5 times the UCS (Martin, 1994). The crack initiation threshold may not correspond to the onset of damage in all rock masses, especially those that are not ‘massive or moderately jointed’.

Sprott et al’s (1999) adjustment for stress damage factor D, is based upon the difference between the in situ deviatoric stress and the post stoping deviatoric stress. The stress factor D formulation is based on the extra stress deviator. The extra stress deviator is the difference between the pre-mining deviatoric stress $(\sigma_1 - \sigma_3)$ and the post mining deviatoric stress $(P_1 - P_3)$, as calculated in Equation 2. The stress damage factor is then determined using Figure 5. Stress factor D does not consider the full stress history experienced by the stope wall, only the pre-mining stress. Therefore, this method does not take into account spikes in the stress history which may exceed the pre-mining stress.

$$\text{Extra stress deviator} = (\sigma_1 - \sigma_3) - (P_1 - P_3)$$

Figure 5 is the chart used to estimate the stress damage factor at the three Canadian Hemlo mines. Without additional knowledge of the case studies upon which the graph is based it is difficult to assess the range of conditions for which this graph is applicable.

**Empirical pillar yield charts**

The brow region is effectively at the edge of an elongated pillar. Therefore, empirical pillar yield charts based on the ratio of pillar strength to the pillar stress could be applied to predict the onset of brow stress damage. (Martin and Matbee, 2000) provide a comprehensive review and comparison of empirical pillar strength.
formula and charts. Brow stress damage would correspond to category three for pillar yield (fracturing in pillar walls) and the unstable region of the Confinement Formula Stability Graph (Lunde, 1994). The main advantage of this approach over the deviatoric stress approach is the very large database of rockmass conditions incorporated into the database means better prediction of stress damage when there is no opportunity to calibrate to underground observations. The confinement formula stability graph is widely used in Canada and has been shown to work quite well for pillar design (Bawden, 2002b). It should be noted that the case studies plotted on the graph are limited to pillar width to height ratios less than three. Due to the horizontal rather than vertical loading direction of vertical narrow vein stope pillars the ‘width’ corresponds to the bench height and the ‘height’ corresponds to the stope width. Therefore, a two metre wide stope with a 12 m high bench has a ‘width to height ratio’ of six.

**Barkers damage criterion**

The best method for predicting the location of areas likely to be affected by stress damage is a site calibrated stress damage model. This could be based on an empirical failure criterion, deviatoric stress, pillar stress or some other stress or strain parameter that correlates well with observed damage. At Barkers mine, geotechnical engineers in consultation with underground operators and shift supervisors calibrated the observations of strain (Figure 6 and Figure 7) and with linear elastic stress modelling (Slade, 2004). This calibration process indicated that when the stress normal to strike exceeded 125 MPa rockmass damage was observed. This damage criterion compares to the uniaxial compressive strength of 142 MPa. Using stress normal to strike, site geotechnical engineers utilised numerical models to predicted regions of high stress damage potential (Slade). Wiles (2002) suggests that rock mass damage is observable at approximately ten per cent strain. However, as shown in Figure 4, degradation of the rock mass through macro-cracking occurs when strain is only one per cent (Wiles, 2002). For this reason it can be expected that rockmass damage through macro-cracking would occur at linear elastic normal stress less than the observable stress damage limit of 125 MPa. This is because linear elastic models do not represent the decrease in stress associated with the onset of damage shown in Figure 4.

**STRESS PATH MODELLING**

Linear elastic estimates of normal and deviatoric stresses have been estimated using a Map 3D boundary element model of the mine. The estimates were then analysed with respect to the stress damage criterion and observed overbreak. The Barkers mine was modelled as a displacement discontinuity plane as shown in Figure 8. The 32 step extraction sequence was reconstituted from the dates recorded in the 205 Stope Report Sheets that were also the source of overbreak estimates for this study. Each step in the model represents one month. Stope Report Sheets were used by site geologists to record linear overbreak as well as ground conditions, generally. Stope Report Sheets covered approximately, 80 per cent of the stopes in the four panels examined in this study. It was necessary to estimate the extraction sequence for the remaining 20 per cent where dates were unknown.
Bolts can become trapped and guillotined in shear.

Shearing of footwall on foliation
Bolting pattern should cover this area to control loose slabs

Uplift/floor heave due to fracturing/bulking below the drive.

Fracturing caused by elevated stresses above backs
Shearing parallel to foliation to accommodate dilation/bulking.

LOC \( \sigma_1 \)

FIG 7 - Rock failure and reinforcement loading and failure indicators about an excavation under high stress conditions (Beck, 2003).

FIG 8 - Map 3D model of stress normal to strike: Barkers mine (January 2002).
The aim of the stress modelling was to capture the entire stress history for each of the 205 stope in the database. Stresses were logged at a point corresponding to the brow for each month in the sequence. The point selected corresponded approximately (±10 metres) with the mid-stope span post stoping and was located half an element width from the brow. Due to this limitation in accuracy, some stopes have the same stress history. For example within the accuracy of the monthly increments modelled, the 203 - 204 and 205 - 206 stopes have effectively the same stress path history. The stope numbers correspond to the number assigned when collating the database. Odd numbers correspond to hanging walls and even numbers correspond to footwalls. It is also worth noting that a stope was considered to be a different stope each time the stope was retreated more than five metres. Approximately, 40 per cent of the stopes have walls that partially overlap with other stopes. Selecting an element size of approximately, three to four metres meant that the point selected was approximately 1.5 to two metres from the edge of the brow at approximately the mid-stope height. The advantage of using a relatively large grid size meant that the stress was averaged over an along strike distance similar to a single blast. This had an averaging effect over the area of interest rather than selecting a small highly localised area of very high stress.

The reason for examining the full stress history was to consider whether extended periods of high stress or multiple stress spikes impacted on observed overbreak. Stress path histories were plotted for all stopes. Figure 9 is a typical stress path history. The peak normal stress shown in Figure 9 occurs at the brow. The typical stress path shown in Figure 9 produced a relatively slow increase in stress then spikes as the brow passes before dropping off to zero following mining. Over 80 per cent of the 205 stopes analysed had stress paths similar to the typical stress path shown in Figure 9. Modelled normal stresses at the brow were up to five times the \textit{in situ} stress levels. As shown in Figure 10, modelling suggests some stopes experienced moderate increases in stress up to six months prior to the brow reaching that point. These increases can be attributed to the effect of adjacent mining. Figure 11 illustrates a stope that experienced sustained high stress when production from this level was delayed for several months.

![Figure 9](image9.png)

Figure 9 - Stress path history for the 131-132 stope (5975 to 5990 level) illustrating typical stress path for Barkers case studies.

![Figure 10](image10.png)

Figure 10 - Stress path history for the 163-164 stope (6020 to 6040 level) illustrating moderately increasing stress associated with adjacent mining.
ANALYSIS OF RESULTS

Effect of normal stress
As discussed earlier, strain damage was observed in ore drives at sites where the modelled normal stress exceeds 125 MPa stress. In the panels studied none of the 205 stopes modelled had peak normal stresses greater than 125 MPa. Besides this correlation, back-analysis of case studies with peak normal stresses greater than 100 MPa were found to have on average 0.27 m more overbreak than the 400 stope surfaces with peak normal stresses less than 100 MPa. Peak normal stress below 100 MPa did not affect overbreak. As discussed earlier, it is probable damage (decrease in rock mass strength) would have occurred at peak normal stress less than 125 MPa. This explains why the 100 MPa cut off for stress damage related overbreak is less than the 125 MPa criterion for observable damage.

The average overbreak for the ten case studies with normal stresses greater than 100 MPa was 0.77 m, compared to 0.50 m average overbreak for the 400 case studies with peak normal stresses less than 100 MPa. The t-test result indicated 94 per cent confidence in this result. The validity of the t-test depends on equal variances and a normal distribution for both data sets. Figure 12 and Figure 13 are histograms of the two data sets. The distribution of the ten case studies with peak normal stresses greater than 100 MPa appears to be skewed slightly to the left. This may be a function of the small number of case studies with a peak stress greater than 100 MPa.

Due to these concerns about the normality assumption, a Monte Carlo simulation was conducted to manually evaluate the probability that the difference between the samples was a random event. Using the random number generator function in Excel, 100 samples of ten were selected from the 410 case studies and the average overbreak evaluated for each of the 100 samples. Six of the samples had average overbreak equal to or greater than 0.77 m. This result indicates that there is a six per cent chance that the difference in overbreak was a random event and therefore confirms the t-test result of 94 per cent confidence.

Other factors affecting overbreak at Barkers
An important consideration when reviewing a statistical result is to ensure that there is not another explanation for the result. In other words, is there some underlying effect that could have biased the result? This is especially important when there are only ten case studies upon which the result is based. To answer this question, comprehensive back-analyses of other stability factors was conducted.
**Effect of northern domain**

On average, the 83 northern domains case studies had 0.24 m more overbreak than the database average of 0.5 m even though Q values were similar. Slade (2004) notes that kinematic instability associated with the relief planes that formed due to the mobilisation/slip of healed quartz/carbonate veins following stoping was probably the cause of this. Many of these veins were not included in scanline mapping or Q classification that occurred prior to stoping.

**Effect of pillars on overbreak**

Leaving pillars resulted in 0.16 m more overbreak than stopes with filling along strike. One hypothesis for this result is that the longhole rising required to restart the stope after each pillar could be responsible for the increased overbreak. (Scoble and Moss, 1994) suggests based on their experience that damage initiated at the slot (or in this case rise) is likely to be greater due the higher powder factors and higher confinement, adding that this damage will then tends to unravel from the hanging wall as the stope is mined out. However, stopes at the ends of the orebody are also started by longhole rises and they had lower dilution than either the filled or pillar stopes.

**Effect of blasting pattern on overbreak**

Previous back-analysis of Barkers stope stability (Stewart and Trueman, 2001) demonstrated that blasting pattern significantly affected stability. The ‘in-line’ pattern had 0.19 m less overbreak on average than the ‘dice five pattern’ and 0.15 m less overbreak than the ‘staggered’ pattern.

**Factors that did not affect overbreak at Barkers**

Within the range of spans and rockmass conditions collated at Barkers, overbreak was not correlated to either the stability number N, nor the hydraulic radius, S. Mid-stope stresses, as quantified by the mid-stope maximum induced stress and used in the formulation of Factor A, did not affect overbreak. Neither stress relaxation (full, partial and tangential) nor stope height (ranging from 10 to 20 m) significantly affected overbreak at Barkers. As shown in Figure 14, the Extended Mathews Stability chart predicts that all case studies should be stable (less than 0.5 m overbreak). However, 144 out the 412 stope surfaces had overbreak exceeding 0.5 m. The effect of stress damage and leaving pillars accounts for 83 of the 144 case studies with overbreak exceeding 0.5 m.

Stewart (2005) observed that case studies plotting well inside the stable zone of the stability graph are not sensitive to the stability graph parameters N and S. This observation is based on consideration of 530 Barkers narrow vein case studies and 261 narrow vein case studies from Trout Lake mine and Callinan mine. Case studies plotting near the stable-failure boundary are sensitive to N and S, case studies plotting well above the line are not sensitive to N and S. While this analysis was undertaken with respect to the Extended Mathews version of the stability chart there is no apparent reason why these findings would not apply to other versions of the stability chart such as the original Mathews Method (Mathews, et al, 1981) and the Modified Stability Chart (Potvin, 1988).

Stewart (2005) separates dilution causes into two broad categories. Firstly, span dependent geotechnical parameters captured by the stability chart and secondly, blast related overbreak or localised effects on dilution. It was determined that in the case of narrow vein stope design explicit consideration of blasting related dilution is necessary. Stewart (2005) has developed a narrow vein stope design method that incorporates the stability graph approach for determining geotechnically stable spans while addressing blasting related overbreak prediction. Dilution prediction and mitigation strategies have been proposed for both the feasibility and operating stage in project life.

Because stress damage degrades the rockmass it could be considered a geotechnical parameter. However, because stress damage into the stope hanging wall and footwall would affect a limited depth of stope wall, it is uncertain whether stress damage at the brow would have the potential to affect the stability chart predicted stable hydraulic radius. It is suspected that the depth of stress damage would be unlikely to have the potential to affect the geotechnical stability of the stope wall. Stress damage is more likely to act locally causing localised dilution. Therefore, it has been proposed that stress damage related dilution potential be evaluated separately from the stability graph method using one of the methods discussed in this paper. Methods for assessing the potential for stress damage related dilution have been incorporated into the narrow vein stope design method proposed by Stewart (2005).
Potential impact of other factors on stress damage conclusion

Eight of the ten case studies (80 per cent) with peak normal stress greater than 100 MPa had a rib pillar abutment along strike. Lower stress damage or ore loss were associated with peak normal stresses less than 100 MPa. Therefore, the higher proportion of rib pillar case studies could be a source of bias and may account for some of the 0.27 m difference between the two groups. Recalling that the average effect of a rib pillar abutment on overbreak is 0.16 m, the effect of rib pillars can be calculated by subtracting 0.16 from four of the case studies and recalculating the average for the group. Using this averaging method, the effect of rib pillar abutments accounted for 0.06 m of the 0.27 m difference between the two groups. Taking into account the effect of rib pillars, the ten case studies with peak normal stresses greater than 100 MPa have an average 0.21 m more overbreak than case studies below 100 MPa.

Similarly, 50 per cent of the ten case studies with peak normal stresses exceeding 100 MPa were mined with the dice-five or the staggered patterns as blasting records were not available for the panels considered in this study. After adjusting for the effect of rib pillars and northern domain faults the effect of stress damage on overbreak reducing from an average 0.23 m to an average 0.10 m. However, while 0.10 m seems quite small it is important to recall that in a 2 m wide stope 0.1 m of overbreak from both the hanging wall and the footwall represents ten per cent dilution. Assuming that there are no other parameters which could have biased the stress damage conclusion, it is reasonable to conclude that stress damage was a significant cause of overbreak at Barkers.

Effect of deviatoric stress

Deviatoric stress is considered to be a good predictor of stress damage and has been used by consultants to evaluate stress damage potential at the feasibility stage when there is no possibility of stress damage calibration (Beck and Sandy, 2002). This approach is based on the AECL’s Mine-by-experiment (Martin and Read, 1996) and observations in the Sudbury neutrino observatory cavern (Castro et al., 1996) which indicate that rock mass damage occurs when $s_3 - s_1$ exceeds 0.3 times the uniaxial compressive strength. Only four of the 205 Barkers stopes experienced deviatoric stresses exceeding 0.3 times the UCS (37.5 MPa). The average overbreak for these four stopes was actually less than the average overbreak for the rest of the database. However, with only four stopes exceeding the damage criterion this result was not significant.

STRATEGIES TO MINIMISE STRESS DAMAGE AT THE BROW

In addition to the potential for stress damage related dilution quantified in this paper and by Sprott et al., (1999), a shrinking central pillar sequence has also been associated with ore loss related to impracticable mining conditions (Beck and Sandy, 2002; Beck and Sandy, 2003). Therefore, when there is a significant risk of either stress damage or seismicity, the consequences of selecting a shrinking central pillar sequence need to be evaluated carefully. Avoiding a shrinking central pillar sequence is likely to reduce stress concentration. However, in terms of maximising the NPV of an operation or project, the possible production difficulties and increased capital costs associated with alternative extraction sequences may outweigh the benefits of reduced ore loss and dilution. In such cases where a shrinking central pillar extraction is chosen to maximise overall NPV, the potential exists for production, planning and geotechnical personnel to work together to reduce stress concentration as far as practical by evaluating extraction sequences as part of the short-term mine planning process. Modelling a detailed extraction sequence facilitates attempts to analyse the effect of alternative extraction sequence on stress concentration.

Extraction strategies which exacerbate stress concentration associated with a shrinking central pillar sequence were highlighted by ‘stepping’ through the 32 month sequence. At this point it is important to note that the principal stress at Barkers is parallel to strike and that the effect of a central pillar extraction sequence on stress concentration would be more severe if the maximum principal stress was perpendicular to strike.

The following is a summary of the observations made when stepping through the stress history for Barkers between September 2000 and August 2002:

- The brow stress can be up to five times the in situ maximum principal stress. The typical Barkers stress path at the brow starts with an in situ stress around 20 MPa which then ramps up to an average peak normal stress of 54 MPa.
- Maintaining a relatively even retreat between levels reduces brow stresses.

Figures 15 and 16 illustrates how the uneven profile created when the bottom level is retreated 28 m ahead of the middle level increases stress concentration at the brow on the middle level.

Typically, rib pillars reduce modelled brow stress by 10 MPa provided they are within 15 m of the brow. Underground observations indicate that within 15 m of the brow pillars had not failed. Therefore, it is reasonable to assume that pillars were in reality bearing load close to that predicted by linear elastic modelling. This may be a useful strategy if peak brow stresses are predicted to be close to the damage threshold. However, leaving a 2 m pillar would result in 12 per cent ore loss over a 17 m strike length.

(Beck and Sandy, 2002) refer to the potential of sequencing layered orebodies to manage stress concentration. While some operations extract the uppermost orebodies first to ‘shadow’ lower orebodies, there is the risk that shadowing of weak ground could result in increased dilution due to stress relaxation (Beck and Sandy, 2002). For cases of full and tangential relaxation (Stewart, 2004) propose that stress Factor A should be set to 0.7 when applying the various stability graphs. Therefore, alternative extraction sequences can be evaluated for both stress damage and stress relaxation potential.

By increasing the length of strike extracted in each production blast the amount of hanging wall and footwall exposed to high stresses would decrease, thereby decreasing stress damage potential. The extent to which production blasts can be lengthened along strike will depend on vein geometry, rockmass conditions and drill and blast parameters.

CONCLUSIONS

Narrow vein retreat stoping is conducted in relatively small increments and depending on the extraction sequence this may result in extensive areas of the stope walls adjacent to the brow experiencing high stress prior to stoping. Linear elastic modelling of 32 months of stope extraction at the Barker mine between 205 m and 380 m depth indicated that stresses normal to the brow can be up to five times the in situ stress. On average stresses normal to the brow were 54 MPa or approximately twice the in situ maximum principal stress. The stress concentration effect could be expected to be significantly higher if the maximum principal stress had been perpendicular to strike.
THE EFFECT OF STRESS DAMAGE ON DILUTION IN NARROW VEIN MINES

**Fig 15** - Uneven retreat 6040 level and 6055 level (July 2001).

**Fig 16** - Even retreat 6040 level and 6055 level (August 2001).
Stopes with peak normal stress exceeding 100 MPa had on average 0.27 m more overbreak per stope wall than stopes with peak normal stress less than 100 MPa. The effect of stress damage on overbreak was significant at a normal stress 25 MPa less than the underground observable stress and strain damage criterion of 125 MPa. After taking into account possible bias due to the effect of rib pillars and kinematic instability in the northern domain, the average difference in overbreak reduced to 0.10 m per stope wall. In a 1.5 m wide stope, 0.1 m overbreak from both the hanging wall and footwall corresponds to 13 per cent dilution. Peak stress normal to the brow has been demonstrated to be a useful parameter for assessing the potential for stress damage related overbreak. At Barkers, peak stress normal to brow greater than 100 MPa was associated with overbreak and corresponds to a magnitude equal to 0.7 times the UCS. It remains to be seen whether a criterion of peak normal stress greater than 0.7 times the UCS for stress damage related overbreak would be applicable at other narrow vein sites.

Four methods for evaluating stress damage potential at the brow have been proposed. The selection of a method for evaluating stress damage potential will depend on rock mass conditions as well as knowledge of the rock mass at that stage in the project’s life.

In situations where a shrinking central pillar sequence is unavoidable, the possibility of mitigating stress concentration through collaboration between production, planning and geotechnical personnel exists. Maintaining a relatively even profile has the potential to reduce stress concentration. In addition, a linear elastic model of the mine would enable alternative sequences to be routinely evaluated as part of the mine planning process.

ACKNOWLEDGEMENTS

This study would not have been possible without the collaboration and support of Placerdome personnel. This study was partly funded by the now completed second AMIRA Blasting and Reinforcement Technology Project. The Julius Kruttschnitt Mineral Research Centre provided funding for the extension of the work into the area of stress damage. We wish to acknowledge the support of Placerdome who granted permission for this study to be published.

REFERENCES


Haddow, H, 1990. The structural setting of the Kundana Gold Mine, MSc thesis (unpublished), Western Australian School of Mines, Curtin University of Technology, Kalgoorlie.


Slade, J, 2004. Seismic characteristics of faults at the Kundana Gold mine, Eastern Goldfields, Western Australia, Department of Mining Engineering and Mining Surveying, Western Australian School of Mines, pp 60 (Curtin University of Technology: Kalgoorlie).


