Towards Improved Criteria for Stabilising Pillar Design

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Abstract:
A critical evaluation of the weaknesses of current design practices has been carried out, pointing in particular to the inefficiency of using elastic ERR and absolute stress constrains, such as average pillar stress (APS), to assess deep longwall layouts protected by stabilising pillars. A new concept of "extended ERR" is proposed, consisting of limiting the on-reef stress to a limit stress value of 250 MPa, that might represent the effective strength of the unmined ground where work on the hanging- and footwalls of highly fractured rock was done. A strong correlation of peak stress chance with increased seismic emissions along pillars was obtained leading to the possible use of this parameter for deep pillar layout design. Variants of the parameter excess shear stress (ESS) were used for assessing geologically complex layouts through back analysis. As a result, it became clear that volume excess shear stress (VESS), as opposed to plane ESS or ubiquitous joint excess shear stress, is a better indicator of overall seismicity in geologically affected longwalls. Generic design considerations have been assembled in a new index termed "Rockmass Hazard Index" (or RHI), which takes account of the seismic hazard of geological discontinuities in deep mine layouts, by integrating the VESS parameter. In addition, RHI includes a factor that relates to the effect of face fracturing. The reliability of RHI has been tested, showing reasonable agreement with other design criteria.

Key Words: rockbursts, stabilising pillars, extended energy release rate, volume excess shear stress (VESS), rock mass hazard index (RHI).

Introduction

Deep gold mines of the Witwatersrand Basin in South Africa exploit pebbly conglomerate reefs at depths that extend over 3400 m below surface. Active mining areas, termed stopes, consist of narrow tabular excavations of roughly 1 to 1.5 metres high, depending on the type of reef exploited, extending over a larger dimension along the dip direction of up to 200 m wide when between strike stabilising pillars in longwall layouts. The geotechnical environment to which these stopes are subjected is complex and consists of:

- High mining induced stresses, which give rise to highly fractured rock conditions in the hanging- and footwall of excavations. Typically, mining-induced shear zones extend tens of metres from advancing faces (Ortlepp, 1978) and regional stability pillars (Lenhardt and Hagan, 1990).
• High levels of seismic emissions occur ahead of working faces once mining advances, further destabilising the rockmass. Seismic risk is greatly compounded in the vicinity of major geological structures such as dykes and faults.

• Rockbursts from mining-induced seismicity occur, which pose the most severe hazard in deep-level mining.

In view of these conditions, several strategies have been followed by the industry over the years to minimise the severity and frequency of rockbursts in active stopes (Gay et al., 1995). These include:

• Avoiding or limiting the formation "remnants" in order to keep stress levels on mining faces within tolerable limits.

• Modifying mining sequences near major geological structures (faults and dykes) so that these structures are stabilised or negotiated in such a way so as to reduce the potential for inducing seismicity.

• Designing regional mining layouts that integrate stiff support structures in the form of rock stabilising pillars, often in conjunction with backfill to increase lateral confinement.

Regional stabilising pillar systems have been in use in South African deep gold mines for nearly four decades, roughly since Cook et al. (1966) pointed out that by controlling the volumetric convergence of wide tabular excavations the energy dissipation due to mining could, consequently, be also controlled. To limit convergence in tabular excavations and, hence, to limit the rate of energy released, a "partial extraction" method was suggested, whereby reef pillars were to be left unmined as permanent regional support. It was hoped, that, by directly restricting the overall convergence in the back areas of longwalls, a reduction in the stresses and induced seismicity in the working faces would be obtained and, thus, the overall rockburst-related risk associated with mining deep stopes controlled.

Figure 1. Typical regional support system of a strike stabilising pillar layout in a deep level gold mine (layout depth from 2400 m to 3400 m below surface).

Figure 2. Typical regional support system of a dip stabilising pillar layout in a deep level gold mine (layout depth from 1500 m to 2700 m below surface).

Currently four major South African Gold mines continue using such a system in the form of reef pillars, which are left along the strike direction of the strata, a layout commonly known by "strike stabilising pillar layout" (e.g. Figure 1). Converting mine
layouts from strike to dip stabilising pillar systems has now become a reality in some South African deep gold mines. One mine (e.g. Figure 2) has been using dip pillars system for a number of years and two others are currently in the process of converting. The reasoning for such radical change is, among others, that the dip pillar system affords greater flexibility in mining geologically disturbed reefs and that lower ERR environments are expected in dip pillar systems than in longwall systems with strike pillars (Cockerill, 1996).

Weaknesses of current design criteria for layout design

The criteria used to design and assess strike stabilising pillar layouts vary slightly from mine to mine, but generally involve limiting the average stresses along the pillar length (APS) to a certain “safe” rockmass strength, or limiting the amount of elastic energy released due to mining ahead of faces (ERR), or both. Other parameters are also taken into consideration and include the width to height ratio of pillars, the limit foundation stress, production requirements in the form of minimal extraction ratios, and general mining layout limitations such as inter-level spacing and major geological discontinuities. Industry accepted limits of ERR have been reported to be around 30 MJ/m² and between 400 and 600 MPa for APS (Selie et al., 1996). Whether these figures represent real conditions of deep pillar layouts has been raised by some authors. Spottiswoode (1997) recently challenged the strength limit of rock around deep strike stabilizing pillars indicating that, subject to certain assumptions, a stress of 250 MPa was found to be appropriate. This value is considerably less than the earlier assumed strength of about 600 MPa. Vieira (1998) found that, on average, stresses along sections of actual strike stabilising pillars were below the traditional design strength of 600 MPa (Figure 3).

![Figure 3. Distribution of pillar stresses along 10 metres spaced section of 10 strike stabilising at depth between 3400 m to 2800 m below surface.](image)

![Figure 4. The number of damaging bursts vs. ERR on the VCR and CLR for the period Jan. 75 to Dec. 76 for the WDL mine; after Heunis, (1980).](image)

Traditional design criteria based on stress requirements to guarantee stability of hard rock pillars in deep level mines have always considered absolute values of stress. Indeed, absolute stress limits said to guarantee the integrity of regional stabilising pillars were first proposed in the work by Cook et al. (1973), which indicated that deformation in quartzitic footwall foundations of these pillars would, most probably, not occur if the average pillar stress was kept below 600 MPa.
Maccelari (1998) found that strike stabilising pillars, regardless of their width or associated dip span, hence absolute load distribution, will at some time generate seismic events of magnitude 2 or larger. As a result of extensive seismic studies of real mine layouts she went on to conclude that stress change, or the rate of stress change, as opposed to the absolute stress level given by APS, appeared to have a greater influence on the seismic behaviour of strike stabilising pillars in back areas. Victra (1997) further substantiated Maccelari’s conclusions in a numerical modelling study that showed pillar stress changes in the back area to correlate well with pillar related seismicity. The reliability of a design guideline for deep pillars based on absolute stress versus strength constraint has limited use in pillar stability design.

For the lack of a design criterion that allows for deformations, wherever they might occur, ERR is still the major mine design criterion, although it has a number of shortcomings, principal being the disadvantage of assuming an infinitely strong rockmass, except for a small region close to the mining faces. The elastic ERR at any point is usually calculated by:

$$ERR = \sigma D/2$$

where $\sigma$ is the stress before an element is mined and $D$ is the convergence between the hanging- and footwall after the element is mined. On-reef stresses and displacements are usually calculated using boundary element methods (e.g. Napier and Stephansen, 1987).

It may be convenient to recall that the ERR criterion became useful for estimating the propensity for the incidence of rockbursts in deep level mine layouts only once a significant, positive correlation (e.g. Figure 4) between the elastic spatial rate of energy release in faces of deep stopes and the incidence of seismicity with associated damage was established. Many authors (Cook et al., 1966; Ortlepp and Steele, 1972-73; Heumis, 1981 and Hagan, 1987) have further reported on such parametric agreement. In spite of practical confirmation that the reduction in ERR results in a consequent reduction of seismicity, some researchers argued that the ERR parameter may not have a casual relationship with seismic incidence (Spottiswoode, 1986).

Salamon (1993) reinforces Spottiswoode’s concerns, reiterating that ERR is of limited value in the process of combating the rockburst hazard because of its narrowness of scope, in that “…the magnitude of ERR depends only on the virgin stress field, the elastic properties of the rocks and the layout of the mining excavations”. He goes on to indicate that “… ERR is independent of the geological structures, the presence or otherwise of flaws (discontinuities) in the rock mass and the potential instability of these flaws”. The inability to recognise failure is an added negative characteristic of ERR, Salamon pointed out.

**Improving the use of energy release rate criterion for deep mine layout design**

*The concept of “extended ERR”*

Implicitly it has been indicated that the potential for seismicity on advancing longwall faces is controlled by limiting the ERR, the strain energy per area mined that, under certain idealised conditions, is released in the immediate vicinity of the face. In a purely elastic and infinitely strong rock mass, ERR is given by the strain energy of the rock removed. In practice, the rock at the face of narrow tabular stopes at depth is
severely fractured and holds only a small fraction of the stress predicted by elastic theory. If the failure is limited to a small volume of rock ahead of the face, then ERR is the work done during failure of this rock. If the rockmass is, as we know, not infinitely strong how then can the hanging- and footwall inelastic deformations of a given layout be accounted for by means of an elastic analysis? On addressing this challenge, Spottiswoode (1997) proposed to extend the concept of ERR by limiting, or capping, the on-reef stress to a certain value ($\sigma_F$) that might represent the effective strength of the unmined ground, at the face or at pillars or abutments. Figure 5 is a sketch showing the effect of $\sigma_F$ on ERR. The cross-hatched area is the work done on the element before it is mined and the stippled area is the work done when the element is mined. By comparison, the area under the dotted line would be used to calculate ERR, assuming purely elastic behaviour of the rock mass. The motivation for this approach is: 1) ERR is well understood and can be applied to very large mining problems with personal computers. 2) It allows for energy release in areas where there is no current mining (e.g. pillar edges, abutments etc). 3) Existing numerical models can be adapted to apply $\sigma_F$.

In the usual case of elastic rock mass, ERR is most conveniently viewed as the work done on the rock about to be mined at a level of stress much lower than that predicted using elastic theory. In deep level gold mines, however, the immediate hanging- and footwall of the stopes are as fractured as the rock to be mined. It is, therefore, likely that the vast majority of the energy associated with ERR is not expended in the rock actually removed, but in the hanging- and footwall strata. This would certainly be the case within a stabilising pillar that is unfractured in the centre, but has caused foundation failure and failure of their edges (Lenhardt and Hagan, 1990, Ozbay et al., 1993).

![Figure 5. Sketch to illustrate ERR on an element with a limit of $\sigma_F = 500$ MPa on the on-reef stress (solid outline) and "elastic" ERR (dashed line).](image)

Figure 6 shows ERR and seismicity ($M>2$) around part of a deep-level mine. Most seismic events are located close to the areas mined, with some events located on abutments or pillars. The area in the top right was mined with longwalls and strike stabilising pillars and was chosen for detailed analysis as well as to test the "extended ERR" method. In conventional analysis of seismicity, the entire block chosen would have been considered as a single seismogenic region.

In the calibration of the model, a range of limit stresses ($\sigma_F$) was considered, namely 200, 250, 300, 350, 400, 450, 500 and 600 MPa, as well as the conventional model with infinite strength. The respective results could be displayed as in Figure 6. The
lightly shaded areas in Figure 6 represent values of ERR <10 MJ/m² and they generated a low level of seismicity. This is borne out in Figure 7, which shows an increase in seismicity with increasing ERR. Smoothing functions were applied to reduce the effects of location errors and the finite number of seismic events recorded.

![Figure 6. Plot of ERR for σ_F = 250 MPa and seismicity with M>2 in part of a deep mine during 1994. The grid lines are approximately 1030 m apart. The very lowest values of ERR were lost in the grey-scaling.](image)

Figure 7. Plot of seismicity per grid block as a function of ERR, averaged into interval of 1.0 MJ/m² σ_c = 250 (MPa).

The area of positive ERR for the fully elastic solution (σ_F = infinity) was simply the area mined. Most events were not associated with elastic ERR, as they were located either too far ahead of or too far behind the face, or on abutments or pillars. Applying a limit stress of 250 MPa to the rockmass, as shown in Figure 6, Figure 7 and Figure 8, resulted in a good agreement between the seismicity data and ERR in the layout considered. The "extended ERR" value provided a linear fit in Figure 8 and included almost all the events within its spatial coverage. Similar plots for the other values of σ_F have shown compatible trends.

![Figure 8. Some values as in Figure 7, cumulated by total energy and by number of event. Plot of seismicity per grid block as a function of ERR, averaged into interval of 1.0 MJ/m². A line is drawn for an exact fit from zero to all the energy released by ERR.](image)

![Figure 9. Seismicity as a function of ERR for σ_F = 250 MPa and for the infinitely strong elastic solution, i.e. σ_F = ∞ MPa.](image)

One of implications of a finite limit to on-reef stress is an increase in the total energy released. It was found that the energy increased gradually for decreasing limit stress down to about 350 MPa, below this it increased rapidly.
Investigating stress influence on seismicity. Stress-change as a design parameter

Also by using MINFFT, Vieira (1997) numerically tested the possibility that pillar related seismicity is influenced by the rate of stress change on the stabilizing pillar itself. MINFFT has the capability of handling records of seismic parameters with reference to the space-time domains of the layout in analysis. Forty stabilizing pillars were analysed. Stress distributions along the longitudinal axis of each pillar were determined for each incremental numerical step, the equivalent to actual mining face advances.

![Cumulative stress change](image1)

![Seismic Moment](image2)

Figure 10. Cumulative stress changes along pillar length for seven mining steps (SdAPS-1 to SdAPS-8).

Figure 11. Cumulative seismic moment (cMo-n) for six mining steps (cMo-0 to cMo-7) for one back-analysed pillar.

Modelled stresses, as expected, were found to be higher in the back areas of pillars. These back areas, however, are subject to much reduced stress changes when compared to areas nearer to mining faces (e.g. Figure 10). As pillars get longer, the rate of stress change in the back area is reduced. Consequently, following Salamon and Wagner's (1979) rationale that seismic events are triggered by stress changes, a correspondent reduction in the seismic activity in the back should be observed. That tendency was corroborated in the back analysis performed and an example is shown in Figure 11. This figure indicates that the spatially distributed seismic moment in the back areas of one particular pillar is reduced when compared with the same activity nearest to the longwall mining faces, where stress changes are higher. On directly comparing the stress changes along pillar length (e.g. Figure 10) with spatially and temporally distributed seismic parameters (e.g. seismic moment in Figure 11), reasonable qualitative agreements were found for these two entities for various pillars (e.g. Figure 12).

![Seismic Moment and Core Stress Change](image3)

![Peak stress change](image4)

Figure 12 Cumulative stress changes along the length of a stabilising pillar length for same spatial and temporal domain.

Figure 13. Peak stress change along pillar length and incidence of along a stabilising pillar during three mining steps (i.e. three periods advance).
Most relevant in this analysis was the strong association between the peak value of stress change and the onset of the increase of seismicity along pillars (e.g. Figure 13). The above evidence led to the conclusion that, in general, greater values of stress changes in back areas of pillars, are associated with increased back area seismicity, measured as number of events or corresponding seismic moments. This consideration may strongly impact on mine layout design strategies.

**Improving the use of ESS criterion**

**Incremental excess shear stress along a plane:** In the normal application of the ESS criterion, rock engineers produce plots field points of the ESS along particular geological structures and attempt to interpret the results by visual inspection. The area of ESS is sometimes used as an indicator of the severity of the ESS, or the maximum value of ESS is used to indicate whether a shear type seismic event is possible. The effects of ESS change, as opposed to absolute, maximum ESS, however, needs be considered in mine layout assessment. ESS increments resulting from a particular mining geometry would be determined, therefore, by subtracting the ESS results of two consecutive mining steps. This would be termed “incremental ESS”. Obviously only positive increases in ESS are of significance. On a plan grid along a particular geological feature, field points of a “plane ESS” would be determined by:

$$\text{ESS}_{\text{plane}} = |r - \sigma_n \tan \phi|$$

(2)

Where $r$ is the maximum shear stress in the plane of the geological structure, $\sigma_n$ is the stress normal to the structure and $\phi$ is the dynamic friction angle, often taken at 30°. The impact of three variants of ESS were studied, namely the incremental “plane excess shear stress” (simply ESS), incremental “ubiquitous joint excess shear stress” (UJESS) and the incremental “volume excess shear stress” (VESS).

**Incremental ubiquitous joint excess shear stress:** In most deep mining environments, the fractured rockmass may have a dominant joint orientation. This condition can be modelled using “ubiquitous joint excess shear stress” (UJESS). In determining UJESS at any point the normal and shear stresses to the ubiquitous plane is required, followed by the determination of the ESS to such plane using (2). Incremental increases in the UJESS may be determined in a similar manner to that described above for the plane ESS.

**Incremental volume excess shear stress:** The “volume ESS” concept was introduced by Spottiswoode (1990) who found that this parameter provided better correlation to actual seismicity than the ERR parameter. The VESS is calculated at predefined grid points in a volume surrounding the actively mining excavations. The maximum excess shear stress value, along a critical plane oriented at 60 degrees to the maximum principal stress, is first calculated at each point following equations (3), (6), (7) and then (9):

$$\tau_{\text{crit}} = \frac{\sigma_1 - \sigma_3}{2} \times \sin 60^\circ$$

(3)

$$\sigma_{\text{crit}} = \frac{\sigma_1 + \sigma_3 - \sigma_1 - \sigma_3}{2} \times \cos 60^\circ$$

(4)

$$\text{ESS}_{\text{crit}} = |\tau_{\text{crit}}| - \sigma_{\text{crit}} \tan \phi$$

(5)
where $\sigma_1$ and $\sigma_2$ are the maximum and minimum principal stresses at a point. The dynamic friction angle $\phi$ is taken to be $30^\circ$. Considering $V_p$ as the volume associated with each point in the grid, the VESS is calculated as:

$$\text{VESS} = \text{ESS}_{cm} V_p$$  \hspace{1cm} (6)

Back analysis was carried out to assess the validity of all three variants of ESS. The results indicated that out of the three the VESS relates better to seismicity. VESS, however, is still insensitive to the effects of geological structures as no consideration is taken with regards to these in its determination.

**Development of a criterion that evaluates rock related hazard in mine layouts**

**The Rock burst Hazard Index (RHI):** An index was developed which combines two factors assumed to be responsible for hazard in working faces of deep level mines. The first factor is the likelihood of a seismic event to occur ($f_{seis}$) and the second relates to the conditions of the rock surrounding the excavation as characterised by a field stress condition factor at the face ($f_{stress}$). The resulting index is termed “Rockburst Hazard Index” (or RHI) and may be determined by:

$$\text{RHI} = f_{seis} \times f_{stress}$$  \hspace{1cm} (7)

**Likelihood of seismicity factor:** The component $f_{seis}$ of RHI relates to seismic incidence in the neighbouring rockmass and is determined from both the seismic moment, associated with the VESS of the rock surrounding the excavation, and the ESS along given geological structures present in the concerned area. The inclusion of VESS in the hazard factor stems from its favourable positive correlation with actual seismicity, as referred to in the previous paragraph. Although plane ESS correlates weakly with seismicity along geological structures, its choice to be included in the RHI calculations results from the overwhelming evidence that seismicity is indeed more severe near geological structures, and that the potential of these to slip is reasonably well assessed using ESS.

Since the units of the ESS and VESS are not compatible, the ESS is converted into an equivalent seismic moment, with units of Nm, using Ryder (1988) procedure by which excess shear stress is assumed to occur in one single circular lobe that has a maximum value of $\tau_c$ and is obtained from:

$$\tau_c = 1.5 \bar{\tau}$$  \hspace{1cm} (8)

where $\bar{\tau}$ is the average increase in ESS. The seismic moment is then determined from:

$$M_p = 1.5 \tau_c a^2$$  \hspace{1cm} (9)

where $a$ is the radius of the ESS lobe. The contribution of both the VESS and ESS along geological structures may be summarised as an indication of the potential for seismic occurrence associated with each mining increment. In order to allow the comparison of mining increments of different sizes, the potential for seismicity is normalised by the area mined in the increment. The factor that characterises the likelihood of a seismic event to occur ($f_{stress}$) in a given mining geometry is then obtained as:

$$f_{stress} = (\Sigma \text{VESS}, + \Sigma \text{ESS}) / A$$  \hspace{1cm} (10)
**Rock conditions factor**: Stress fracturing influence on the rock related hazard index is accounted for in the value of ERR. Note that ERR, and not a stress component, is used. This is because ERR, as opposed to stress, is less sensitive to boundary conditions such as the element size of the numerical model used. Equation (1) reminds us that ERR may indeed be used as an indirect indicator of face stress. The condition factor for face stress in the RHI (f\textsubscript{stress}) is obtained, therefore, by:

\[
f_{\text{stress}} = (0.09 \times \text{ERR}^{0.488} - 1)/10
\]  

(11)

**Verification of RHI index and case studies**

The RHI was developed as an aid to rock engineers who needed to evaluate mining layouts. It would, therefore, be required that the RHI should relate to the risk of rockbursts associated with mining at great depths. Since there is no single parameter that indicates the risk of rockbursts, the RHI was evaluated against parameters normally employed in the evaluation of deep level layouts such as: elastic ERR (e.g. Figure 14), actual rock fall related data (e.g. Figure 15), seismic moment (e.g. Figure 16), and expert opinion (e.g. Figure 17).

![Figure 14. Relationship between RHI and ERR in a layout devoid of hazardous geological structures.](image)

![Figure 15. Relationship between panel shifts lost due to rockfalls and RHI for a longwall system in an East Rand mine.](image)

![Figure 16. Relationship between RHI and seismic moment per mining increment in a longwall with stabilising pillars (East Rand).](image)

![Figure 17. Correlation between RHI and expert opinion expressed as relative hazard for a number various mining situations.](image)

A comprehensive study undertaken by Esterhuizen (1997) has shown that the RHI concept can be used to assess deep mine layouts with a certain degree of confidence. RHI agrees well, for instance, with expert opinion on the actual hazard associated with mining. In some cases of longwalls with stabilising pillars, agreement was also obtained between the RHI and the seismic moment. When compared to fall of ground
data, the RHI provided good correlation for most of the time periods considered in a given case study.

Conclusions

A revised model of ERR, with a limit of 250 MPa to the on-reef stress, provided an improved predictor of the spatial distribution of seismicity compared to the normal ERR that assumes elastic behaviour of the entire rock mass. The standard energy calculation for ERR has been extended by adding the component of work done on the hanging- and footwalls when limiting the on-reef stress. Further work is suggested, including more back analyses and the possibility of lower limit stresses being appropriate for different geological settings and different field stresses. A strong correlation between peak stress chance, with the onset of increased seismic emissions in these pillar, lends itself to the view that this parameter has potential in deep pillar layout design. It was verified that, in many instances, a plane ESS correlates poorly with seismicity. VESS on the other hand showed better agreement. VESS is, therefore, integrated in the proposed rockmass hazard index (RHI) as an indicator of seismic hazard. The plane ESS along geological structures is also, however, included in the determination of RHI, for better estimating the potential for slip of a geological feature. The effect of face fracturing has been accounted in RHI as weighting factor. Relative success in the application of RHI has been noted. There are a number of potential problems, however, which were identified from discussions with experts in rock engineering and seismicity (Vieira et al., 1998). Additional research is suggested to consolidate RHI type assessments.

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References


