Final Project Report

Evaluation of the design criteria of Regularly Spaced Dip Pillars (RSDP) based on their in-situ performance

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Executive summary

We analysed seismicity in two deep gold mines in terms of strain energy release during mining. Mining of the Carbon Leader Reef (CLR) and the Ventersdorp Contact Reef (VCR) took place with regularly-spaced dip pillars and backfill for regional support. More than 500 000 m$^2$ of reef were mined over several years and induced more than 10 000 events of Local Magnitude greater than 0.0 per reef. Four approaches were taken to study the behaviour of these pillars, namely: comparison between modelling and observed seismicity; seismic source mechanisms; strong ground motion studies; and ground tilting. The best research results came from the studies of seismicity and numerical modelling and ground tilting.

It was necessary to develop automatic polygon (auto-poly) definitions to clearly separate areas of localised mining and seismicity. Here follow six conclusions based on work done in this project that are of direct rock engineering value:

1. **ERR is, on average, a direct measure of likely seismicity.** Seismicity per area mined is proportional to Energy release Rate (ERR) when seismicity is measured through summation of a range of parameters such as number of events with Magnitude (M) > 0, Seismic Moment and Apparent Volume.

2. **There was no sign of run-away pillar failure.** The seismicity rate per strain energy release appeared, if anything, to decrease at the highest levels of ERR.

3. **The effective pillar strengths are at least 300 MPa.** The modelling only showed an accelerated rate of strain energy release at high values of ERR if the pillar strengths were assumed to be less than 300 MPa.

4. **The benefit of backfill in reducing strain energy release is greater as pillar stresses approach their ultimate strengths.** In other words, backfill is most useful for regional support when regional rock support is at its most stressed.

5. **Seismicity was controlled far more by mining than by the presence of geological features (faults and dykes).** This was probably the result of good mining practice such as using bracket pillars within final pillars and mining through features at a good angle of incidence. Geological structures appeared to increase seismicity measured as Apparent Volume by about 9% on the VCR and 37% on the CLR. There was a greater influence for radiated energy. The greater influence on the CLR mining was at least partly attributable to the faults and dykes interrupting aseismic slip on bedding planes.

6. **Mining from both sides of a pillar at the same time does not generate any more seismicity than if only one side is mined at any time.** Modelling work suggests that reducing the amount of mining on each side of a pillar is likely to be less hazardous than mining each side in turn. This work places some doubt on the value of the so-called 70 m rule as a rockburst-control measure.

In addition to these six conclusions that are of rock engineering value, two new and fundamental insights into rock mechanics were obtained and should be explored further:

1. **Stopes and / or a yielding fracture zone ahead of the face created a soft loading system that influenced the character of seismic events at high levels of ERR.** Apparent stress was found to be lower for events associated with higher ERR mining where the face stresses should be higher. This contradiction is resolved when loading stiffness was calculated and it was found that P/S moment ratios, apparent stress and seismic stiffness all increased with increasing loading system stiffness. These effects were more strongly developed for mining in the Carbon Leader. The amount of seismicity per area mined increases with increasing span but, as the loading system softens, strong ground motions and therefore the damaging potential of each event decreases.

2. **A new mechanism is needed to explain time-dependent failure of brittle rock.** Several types of observation are incompatible with current rock mechanics theory, such as:
- The absence of measurable changes in tilt even at a level of 0.1 microradians following seismic events that caused co-seismic tilt changes up to 100 microradians as measured 90 m below reef;
- Tilt signals similar to those reported for “slow earthquakes”;
- More seismic events occurring far from active mining than was predicted from modelling; and
- A lower fall-off rate of aftershock density with distance, D, than expected, namely ~D^{-1.3} compared to the current theory of D^{-3}. (Data analysis part of SIM-05-03-02: Managing Rockburst Risk.)

One suggested mechanism is that the major mechanism of time-dependent deformations in deep-level mines is development of new cracks or growth of cracks rather than mobilisation of existing cracks.
Outcomes vs. Scope

There were four required outputs of this project, namely two Primary Outputs and two Other Outputs. They are listed here, followed by a brief description of how they were covered in the outcomes from this project.

**Evaluation of the design criteria of regularly spaced dip pillars based on their in situ performance**

The ERR was shown to provide a robust measure of seismicity and therefore the amount of seismicity was well managed by controlling the ERR through limiting spans between stability pillars and by applying backfill.

There were no signs of anomalous pillar failure, indicating that pillar stresses are below the design level of tolerable Average Pillar Stress.

**Identification of conditions under which the design of RSDP mining work best as a rockburst control method.**

Backfill is much more effective at reducing the total amount of strain energy release if there is substantial yield at the face than for the case of the generally assumed elastic behaviour of the rock mass.

The apparently softening response associated with the growing fracture zone around stoping as stope spans increased on the Carbon Leader Reef (CLR) led to lower values of Apparent Stress and therefore possibly less rockburst damage for each event of the same event Magnitude.

**Improved insight into the relationship between faults and dykes and pillars.**

Seismicity was much more strongly correlated with active stoping than with geological features marked on the 1:1000 plans. 38% more events on the CLR occurred within 40 m of faults and dykes than remote from these features while the number of events actually decreased on the Ventersdorp Contact Reef (VCR) when measured in terms of seismicity per strain energy release. These figures were greater for larger events, with the seismically radiated energy doubling due to dykes and pillars on the CLR and increasing by 29% on the VCR.

Unfortunately, attempts at obtaining moment tensors were unsuccessful due mostly to problems with the orientations and polarities of geophones of the mine network.

**Analysis and design methodologies that can be applied to other mine layout designs in tabular mines.**

Analysis methodologies were developed and written into software that can be applied to other mine layouts. These methodologies include a new model of face stiffness and new developments of interpreting seismicity in terms of geological features and variations in monthly hazard. The automatic grouping of seismicity and mining into polygons (auto-polys) has proved to be much more useful for analysis than the traditional fixed polygons (polys).

The difficulty of obtaining a digital map of what is mined when is a barrier to application of this or any software or analysis that needs to know the history of mining. At present, it appears that manual digitizing of mine plans is still needed in most, if not all, cases.

Software was also written to interpret records of ground tilting that provided new insights into the time-dependent behaviour of the rock mass.
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<td>CLR</td>
<td>Carbon Leader Reef</td>
</tr>
<tr>
<td>DME</td>
<td>Department of Minerals and Energy</td>
</tr>
<tr>
<td>ERR</td>
<td>Strain Energy Release Rate</td>
</tr>
<tr>
<td>ER</td>
<td>Strain Energy Release associated with mining</td>
</tr>
<tr>
<td>ESS</td>
<td>Excess Shear Stress</td>
</tr>
<tr>
<td>MHSC</td>
<td>Mine Health and Safety Council</td>
</tr>
<tr>
<td>MINF</td>
<td>MINing simulation using Fourier transforms</td>
</tr>
<tr>
<td>MINSINT</td>
<td>MINing Seismicity INTegrator</td>
</tr>
<tr>
<td>PPV</td>
<td>Peak Particle Velocity</td>
</tr>
<tr>
<td>PPA</td>
<td>Peak Particle Acceleration</td>
</tr>
<tr>
<td>SGMD</td>
<td>Strong Ground Motion Detector</td>
</tr>
<tr>
<td>RE</td>
<td>Rock Engineering</td>
</tr>
<tr>
<td>UCS</td>
<td>Uniaxial Compressive Strength</td>
</tr>
<tr>
<td>VCR</td>
<td>Ventersdorp Contact Reef</td>
</tr>
<tr>
<td>J</td>
<td>Joules</td>
</tr>
<tr>
<td>N-m</td>
<td>Newton-Metres, used for seismic moment</td>
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<tr>
<td>M; G; T</td>
<td>Mega ($10^6$); Giga ($10^9$); Tera ($10^{12}$)</td>
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Used to shorten numbers in Tables and Figures
1 Introduction

Deep-level mining of tabular reefs uses regional support to control seismicity. Many of the deep workings in the Carletonville gold mining district in South Africa have moved from strike stabilizing pillar systems to pillars and backfill for regional support (McGill, 2005, and Klokow et al, 2003).

Pillar mining is being practiced mostly in two forms, namely Sequential Grid Mining (Handley et al, 2000) and Closely Spaced Dip Pillar Mining (Klokow et al, 2003).

1.1 Structure of report

This report focuses on two areas of work from which the most useful results were obtained, namely an integrated analysis of seismicity and modelling, and interpretation of tilt and seismic data in terms of time-dependent behaviour of the rock mass. Short sections cover moment tensor inversions and strong ground motion recordings.

More details on this work can be found in a supplementary report that includes the project proposal and covers some areas of work in more detail, in particular the work on moment tensor inversions.

1.2 Energy Release Rate (ERR), Excess Shear Stress (ESS) and Average Pillar Stress (APS)

The concept of energy release associated with the incremental enlargement of tabular excavations was proposed in the landmark paper of Cook et al (1966). They also pointed out that the amount of energy release per area mined could be reduced through the application of regional pillars. This led to the use of stabilising pillars aligned along strike (van Antwerpen and Spengler, 1982; and McGarr and Wiebols, 1977) as a way of reducing the amount of seismicity and rockbursts. Stabilising pillars are now mostly aligned along dip (Handley et al, 2000; Klokow et al, 2003; and McGill, 2005).

In the case of ERR, the “seismic energy efficiency” ($\eta$) of converting strain energy release to seismicity has been measured (McGarr et al, 1979, Spottswoode, 1980). They showed that less than 1% of the strain energy released during two seismic events was radiated seismically, a figure similar to that proposed for earthquakes.

Although the number of rockbursts per area mined was found to be proportional to ERR (e.g. Jager and Ryder, 1999), Ryder and Jager (2002, p233) veered away from recommending the use of ERR as an indicator of seismicity, preferring to advocate its use as a measure of underground conditions. They were influenced by two factors in coming to this conclusion: recent unreferenced data with limited range of ERR values that showed very little or no correlation of seismicity with ERR level; and their own difficulties with reconciling the very low proportion of elastic strain energy that is released by seismically radiated energy. Instead Ryder and Jager (2002) focussed attention on the use of ESS as the most appropriate way to model seismicity, with particular focus on slip on pre-existing geological faults.

Currently, widespread ESS modelling is a time-consuming process requiring skilled interpretation. As will be seen below, the geological features intersecting the reef at the two areas studied increased the total amount of seismicity by a small amount on the VCR, and by a moderate amount on the CLR. As pointed out in this section, the small influence of geological features on the total amount of seismicity might have been attributed in part due to care being taken when mining in or near such features.
Ryder and Jager discuss three concepts that are followed in the analysis presented here. Two concepts have been applied in previous work using MINF modelling over many years, e.g. Spottiswoode (1997) and Spottiswoode et al (GAP612c / 722):

1. The Young’s modulus of the bulk of the rock mass was taken as 70 GPa;
2. Irreversible bulking of the immediate hanging-wall and foot-wall by 0.2 m was assumed. This concept is new to MINF modelling and was taken from Ryder and Jager (2002, p250, point 4); and
3. Weakening of the stress normal to reef.

ERR was calculated in two ways in this paper. For areas being actively mined, ERR was calculated from the ratio of strain energy release to area mined: for a single “elastic” element, \( \text{ERR} = \frac{1}{2} \times \text{stress before mining} \times \text{convergence after mining} \). For stationary faces, \( \text{ERR} = \frac{1}{2} \times \text{stress ahead of face} \times \text{convergence behind face} \).

Pillars are designed to hold up the entire weight of overburden and must therefore be assumed to have a certain strength, or load-carrying capability. The in-situ strength of these regional pillars is still not well known. Ryder and Jager (2002, p270) mentioned a range of possible strengths (from 500 MPa down to 240 MPa).

### 1.3 Interaction amongst mine events & other fractures

The stress field induced by deep-level mining is so high that fracturing and seismicity are common. Tabular stopes in brittle rock are surrounded by a fracture zone (Figure 1) that increases in size with increasing levels of ERR (Ryder and Jager, 2002). From a seismological perspective, the ensemble of seismic events plays a major role in relaxing stresses ahead of the face and is thought to occur through development and mobilisation of shear fractures, marked “S” in Figure 1a. These interact with joints “J” and bedding-plane slip “P”. For the purposes of this report, an equivalent fracture zone can be drawn (Figure 1b) to consist of seismic sources (S1 and S2) and a diffuse fracture zone that forms ahead of the face. As mining of the reef advances through this fracture zone, it forms the hanging- and foot-wall of the stopes. Because falls of ground and rockbursts occur when this region fails, its behaviour is of great importance to mining.

![Figure 1](image_url)

**Figure 1** (a) Cartoon of fracturing around an advancing face, from Adams et al. (1981); and (b) a simplified version of (a) showing two shear fractures, a fracture zone and force and displacement vectors driven by slip on shear zone S2.

Ortlepp (1997) mapped a series of interacting shear zones, some of which intersected one another and others that were closer together (<5m) than their spatial extent (>20m). Two shear zones that were most carefully studied formed ahead of the face, as in zone S2 (Figure 1b). Napier (1991) showed that a “daylighting” shear plane (S1) should slip about ten times more than a shear plane that intersects the reef ahead of the face (S2) in an elastic rock mass. In
more recent, unpublished work, Napier (2008, Pers. Comm.) showed that S2-type slip could force or mobilise a series of fractures into the stope if the vertical extent of slip was substantially more than its distance ahead of the face.

Malan and Napier (2003) studied convergence between points on the hanging- and foot-wall of stopes within our study area at Mponeng. Both blasting and seismicity appeared to have added very rapid stope convergence followed by an increased rate of convergence to a constant background rate. In contrast, Share (2007) reported on an apparently complete lack of post-seismic tilting, as observed 90 m off reef. This suggests that the fracture zone shows a delayed response to the loads imposed by mining and seismicity. Face preconditioning (Toper, 2007) is routinely applied on some mines to reduce rockburst damage. Its mechanism appears to be based on a highly generalised picture of the fracture zone and its behaviour. In summary, very little is known about the interaction between the fracture zone and seismic shear zones occurring ahead of the face.

In addition to studying seismicity as a possible function of ERR, we looked for changing character of seismicity in terms of apparent stress ($\tau_a = G \times E_s / M_0$; where $G$ the modulus of rigidity 30 GPa, $E_s$ is radiated energy and $M_0$ is the seismic moment; Table 3) and the ratio of seismic moments derived from P waves to moments derived from S waves (P/S moment ratio). The expectation was that apparent stress would increase with increasing ERR and elastic stresses ahead of the face. P/S moment ratio should be 1.0 on average for pure shear slip but Spottswoode et al. (2006) found values well in excess of 1.0 for events associated with failure of small pillars in a mine in the Bushveld Complex. Boettcher et al (2008) reported that M>2 events had P/S moment ratio of 1.0 while smaller events showed moderate to significant implosive components. A stiffness model was developed for this project to assist in analysing data for this study.

## 2 Case studies

During the course of this project underground observations to study pillar deformations and stresses at both a Carbon Leader Reef mine (Driefontein 5E) and a Ventersdorp Contact Reef mine (Mponeng) was carried out.

Both the Driefontein and the Mponeng dip pillar mining sites are accessed through haulages about 80 m off reef; a distance much greater than is used in follow-behind drives for longwall mining. Haulages at this distance will experience less mining-induced stresses and a lower probability of seismic damage.

The largest part of the work in this project was undertaken using seismicity data interpreted in terms of modelled mining history, from raise to mining up to final pillar positions, and geological discontinuities. Underground experiments were also undertaken at both mines in the form of recording of ground tilting at Mponeng and strong ground motion recording at Driefontein. Relevant details for the underground work will be provided in the sections of the report that discuss the work. General information on the study areas with particular reference to the seismic-modelling work is provided here for the two mines in parallel.

### Table 1 Mining and layout design for two case studies. Reference to (M) for McGill (2005) and (K) for Kloow et al (2002)

<table>
<thead>
<tr>
<th>Actual S/W</th>
<th>Driefontein 5E#</th>
<th>Mponeng</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shaft Collar Elevation</td>
<td>150 m</td>
<td>180 m</td>
</tr>
<tr>
<td>Area Mined</td>
<td>1 057 896 m²</td>
<td>543 636 m²</td>
</tr>
<tr>
<td>First Date</td>
<td>1999/11/15</td>
<td>2002/9/15</td>
</tr>
<tr>
<td>Last Date</td>
<td>2007/01/15</td>
<td>2006/7/15</td>
</tr>
<tr>
<td>Number of mining steps digitised</td>
<td>86</td>
<td>47</td>
</tr>
<tr>
<td>Reef type</td>
<td>CLR (K)</td>
<td>VCR (M)</td>
</tr>
<tr>
<td>Dip Angle</td>
<td>24°</td>
<td>24°</td>
</tr>
</tbody>
</table>
The basic data for this study were obtained from mine plans at a scale of 1:1000 and catalogues of seismic data. The reef was very planar as could be seen from the consistent orientation (along dip) and spacing (down dip) of crosscuts and their intersections to reef. We were then able to approximate the reef geometry as a plane in each case, simplifying the modelling of deformations and stresses. Plastic failure ahead of the face and abutments was modelled by using a cap, or limiting, stress on reef elements (Spottiswoode, 1997).

The face positions showing monthly face advance over several years were digitised and converted into arrays of square elements, each representing the amount of mining in each month using the MinSim 2000 software that was developed in-house and is now proprietary software of GijimaAst. Mining parameters are shown in Table 1 and mine outlines in Figure 2. The study areas were chosen to contain a considerable amount of mining as well as being as isolated as possible from surrounding mining. This was not entirely the case and Figure 2 also shows regions of mining and seismicity that were excluded from this study. Seismic data close to adjacent mining “O” and more than 100 m from any previous mining “F” were excluded from further analysis.

The seismic catalogues contained the date, time, location and seismic energy and seismic moment for each event. The energy was reported separately for P waves and S waves and the seismic moment was estimated separately for the P and S phases and assumed to be a pure shear source. Quality checks were performed and corrections made. About six weeks were missing from one data set, resulted in our rejection of two months of data. Some values of seismic energy were also unrealistically large, leading to values of apparent stress in excess of 50 MPa. These appeared to have been outliers caused by anomalous calculations of $E_S$. We reduced the impact of these anomalous values by limiting seismic energies to values derived from seismic moment and an upper estimate of apparent stress of 3 MPa using $E_S \leq 3 \text{ MPa } \times M_0 / G$.

The study areas are labelled “Driefontein” and “Mponeng” in this report and are also shortened to “Drie” and “Mpo”.

### Table 2 Modelling parameters for two case studies

<table>
<thead>
<tr>
<th></th>
<th>Driefontein 5E#</th>
<th>Mponeng</th>
</tr>
</thead>
<tbody>
<tr>
<td>Modelled stope width</td>
<td>0.8 m and 2.0 m</td>
<td>1.2 m</td>
</tr>
<tr>
<td>Elements used in MINF model</td>
<td>256×256</td>
<td>256×256</td>
</tr>
<tr>
<td>Element size</td>
<td>6 m×6 m</td>
<td>6 m×6 m</td>
</tr>
<tr>
<td>Youngs modulus</td>
<td>70000 MPa</td>
<td>70000 MPa</td>
</tr>
<tr>
<td>Poissons ratio</td>
<td>0.20</td>
<td>0.20</td>
</tr>
<tr>
<td>Stress gradient</td>
<td>0.030 MPa/m</td>
<td>0.030 MPa/m</td>
</tr>
<tr>
<td>$k$ - ratio</td>
<td>0.5 isotropic</td>
<td>0.5 isotropic</td>
</tr>
<tr>
<td>Backfill placement</td>
<td>Full height, only below 41 Level</td>
<td>Full height</td>
</tr>
<tr>
<td>Backfill “a”-value</td>
<td>10 MPa</td>
<td>10 MPa</td>
</tr>
<tr>
<td>Backfill “b” values</td>
<td>0.40</td>
<td>0.40</td>
</tr>
</tbody>
</table>

The seismic catalogues contained the date, time, location and seismic energy and seismic moment for each event. The energy was reported separately for P waves and S waves and the seismic moment was estimated separately for the P and S phases and assumed to be a pure shear source. Quality checks were performed and corrections made. About six weeks were missing from one data set, resulted in our rejection of two months of data. Some values of seismic energy were also unrealistically large, leading to values of apparent stress in excess of 50 MPa. These appeared to have been outliers caused by anomalous calculations of $E_S$. We reduced the impact of these anomalous values by limiting seismic energies to values derived from seismic moment and an upper estimate of apparent stress of 3 MPa using $E_S \leq 3 \text{ MPa } \times M_0 / G$.

The study areas are labelled “Driefontein” and “Mponeng” in this report and are also shortened to “Drie” and “Mpo”.
Figure 2 Mining and seismic events with Magnitude $M > 2.0$. Seismic events in areas marked "O" and "F" were excluded as they were considered to have been associated with mining outside the modelled area or were more than 100 m from any mining.

3 Integration of seismicity and modelling

We based our analysis on software and methods described by Spottiswoode (2005) to analyse the amount of seismicity in terms of modelled deformations. Significant extensions were developed during the current study. Deformations were modelled on a monthly basis using the custom-built MINF code. MINF generated output that described spatial distributions of area mined and strain energy released. Area mined, strain energy release and equivalent volume of convergence between the stope roof and floor were cumulated in defined polygons. The loading stiffness driving the deformations within each polygon was calculated. The MINSINT program allocated seismicity to active mining and stationary faces. MINSINT then cumulated and listed values within polygons to be used to compare seismicity to modelled deformations.

3.1 Software

A large amount of software was written as part of this project, principally as extensions to the MINF and MINSINT programs developed for GAP 612c and GAP 722. In addition two utility programs, EDIT_PATT and DIST_TIME were written.

The MINF suite of software now has over 33 000 words of internal comments to assist the author in maintaining the software. In comparison, this document has 16 000 words of text.

3.1.1 The modelling solution (MINF)

The “size” of the simulations was 256 by 256 square elements, each 6 m on a side, almost double the resolution used by Spottiswoode (2005) in a previous study of data from the Driefontein 5E shaft site. The MINF code solves for the elastic convergence and stress on each element within an elastic rock mass. Stope closure was limited by backfill that provided resistance ($\sigma$) according to the hyperbolic function $\sigma = \frac{a\epsilon}{b - \epsilon}$ where a and b are constants (Table 2) and $\epsilon$ is the strain on the backfill as the roof and floor converge towards one another.

The modelled stope width was reduced by 0.2 m from the actually mined width to account for bulking of the fracture zone, as recommended by Ryder and Jager (2002).
On the basis that pillars will start to yield at the assumed strength, MINF limits the on-reef stress component normal to the reef plane in unmined areas to given values. Initial results presented here are based on infinite strength, followed by a section in which the strength values are decreased. If the yield strength of a pillar is exceeded, considerable deformation and strain energy release is possible.

The loading stiffness was estimated with each polygon in an attempt to explain changes in the character of seismicity with increasing ERR (Figure 15).

![Stress vs Displacement](image)

**Figure 3** A sketch illustrating strain energy released by mining: (a) for elastic rock mass; (b) for a constant cap stress; and (c) for successive weakening of the fracture zone close to the face.

Figure 3a illustrates the energy released for a small step of mining (say a few metres) in an elastic environment. The rock immediately ahead of tabular deep-level stopes has been subjected to such high stresses that it has already failed or crushed and releases much less energy when mined. The lower the stress at the face, the more work has already been done (Figure 3b). If the face is heavily fractured, as is commonly observed in deep-level stopes, then the stress might have dropped substantially. In that case, practically all of the strain energy has already been released (Figure 3c). Ryder and Jager (2002, p239) refer to a “Salamon fracture zone” in which the stress at the face is almost zero, rising up to many hundreds of MPa about 3 m ahead of the face.

The MINF code provides for various types of plastic and brittle constitutive laws to approximate relaxation of the reef-normal stresses ahead of the face (Spottiswoode, 2001). The simplest of these will be applied in the report, namely limiting, or capping, the stress to a defined level (Spottiswoode, 1997). Similar results, not shown here, were also achieved when the strength increased linearly with distance from mined-out areas.

### 3.1.2 The integration solution (MINSINT)

Seismicity was attributed to mining by the program MINSINT in a manner similar that described by Spottiswoode (2004). The following factors were taken into consideration in planning the process of attributing, or distributing, seismicity to mining:

- Seismicity is caused by the high stresses in the vicinity of the edges of the stopes, either at the faces or abutments. In particular, seismicity is associated with stress changes caused by active mining.
- If seismicity is associated with faults or dykes, it will still occur under the influence of the mining, perhaps within a few tens of metres from the faces & abutments.
• Seismic locations are prone to errors of about 40 m in the plane of the reef. As the geophones are much closer to the plane of the reef than their spread across the plane of the reef, location errors are more at right angles to the reef.

• Seismic events have a finite dimension that increases with the event Magnitude and can be approximated by the source radius.

There is no clear rule to describe the potential of stationary faces for generating seismicity. Kgarume and Spottiswoode (2008) have recently shown that the probability of aftershocks of mine events falls off as distance $D^{-1.3}$ whereas quasi-static modelling predicts a fall-off rate of induced stresses of $D^{-3}$. Quasi-static stress changes due to mining are therefore not always optimal as a spatial predictor of seismicity. Nonetheless, we assumed that stationary faces retain a residual propensity for failure equal to a virtual 0.01 of an element size face advance on all faces using the definition of ERR for stationary face.

Based on these considerations, the hypocentre of each event was moved to the nearest likely mining, as follows:

• projecting it perpendicularly onto a grid point on the reef plane;
• erecting a 2-dimensional Gaussian error function around this grid point;
• multiplying the Gaussian error function by the energy release associated with mining (see Figure 3); and
• choosing the position of the maximum value of this product.

The strength of each event was then expanded according to the source size. In the example in Figure 4, event A was moved to the nearest area of active mining while event B was attributed to the nearest stationary face. An oblique view was used in Figure 4 and shows that event A located further from reef than event B.

Expand the influence according to the event size.

Figure 4 Diagram to illustrate how seismic events (A & B) were attributed to mining. (a) Gaussian function around the projection of each event to reef. (b) The events with contoured released strain energy. (c) Contours are the product of the values in (a) and (b). Event A is moved as shown.

3.1.3 Software utilities

The MINF code reads the pattern of mining in ASCII form as a “CXX” file written by MinGrid from digitized outlines (MLS files) generated using MinPlan. MINF and MINSINT were extended to read additional information, namely the placement of backfill, the stope width, east and west mining directions and the presence of dykes and faults. CXX files were also extended to provide information on areas of influence outside the study area that were to be excluded in the analysis, shown as “O” in Figure 2. Geological features were mapped using a utility program, EDIT_PATT.
A post-processor program, DIST_TIME, was written to study the distribution in space and time of seismic events that located away from active mining. This analysis has only been partially completed.

3.2 Grouping seismicity into polygons

At the start of this work, polygons were defined to cover the area between raises so that mining from either side towards each final pillar position would fall into a polygon. Klokow et al. (2003) used polygons between pillars, as the area mined was available to them in this form. Unfortunately this would have led to difficulties attributing individual seismic events to the correct polygon if they located within a pillar. In contrast, we have access to the spatial distribution of all areas mined.

![Figure 5 Depicts the mine plan of the area of interest at Driefontein 5E# and Mponeng showing the polygons in different colours, viewed in MinView3D.](image)

Table 3 Abbreviations used in Table 5 and Figure 6 and Figure 8 for model (M) and seismic (S) parameters

<table>
<thead>
<tr>
<th>Abbreviation</th>
<th>Data type</th>
<th>Description, units</th>
</tr>
</thead>
<tbody>
<tr>
<td>A_M</td>
<td>M</td>
<td>Area, m^2</td>
</tr>
<tr>
<td>E_M</td>
<td>M</td>
<td>Strain energy released, GJ</td>
</tr>
<tr>
<td>V_E</td>
<td>M</td>
<td>Volume of elastic convergence, m^3</td>
</tr>
<tr>
<td>ERR</td>
<td>M</td>
<td>E_M/A_M</td>
</tr>
<tr>
<td>E_S</td>
<td>S</td>
<td>Seismic energy radiated, GJ</td>
</tr>
<tr>
<td>M_0^P</td>
<td>S</td>
<td>Seismic moment estimated from P waves, MN-m</td>
</tr>
<tr>
<td>M_0^S</td>
<td>S</td>
<td>Seismic moment estimated from S waves, MN-m</td>
</tr>
<tr>
<td>M_0</td>
<td>S</td>
<td>Seismic moment estimated from P and S waves, MN-m</td>
</tr>
<tr>
<td>P</td>
<td>S</td>
<td>Potency = M_0 / G, m^3</td>
</tr>
<tr>
<td>r_S^a</td>
<td>S</td>
<td>Source radius, derived using Brune’s (1970) model from corner frequencies, m</td>
</tr>
<tr>
<td>V_a</td>
<td>S</td>
<td>Apparent volume, calculated as r^3, 1000 × m^3</td>
</tr>
<tr>
<td>N_0</td>
<td>S</td>
<td>Number of events when cumulated</td>
</tr>
<tr>
<td>r_a^s</td>
<td>S</td>
<td>Apparent stress = G×E_S/M_0, MPa with G = 70 GPa</td>
</tr>
<tr>
<td>A_S</td>
<td>S</td>
<td>Area of seismic slip taken as the largest circular shape</td>
</tr>
</tbody>
</table>
inscribed within a spherical shape for the Apparent Volume

\[ \eta \] M & S

\[ \gamma_E \] M & S

Seismic efficiency = \( E_S / E_M \)

Normalised seismic deformation = \( P / V_E, m^3 \)

### 3.3 Analysis with fixed polygons

Cumulated seismicity is plotted for both study areas for the fixed polygons in Figure 6. Apparent volume (\( V_a \)) was used as a measure of seismicity because it is commonly used in analyses of gold mine seismicity (van Aswegen, 2005) and because cumulated totals of \( V_a \) are less dominated by the few largest events (e.g. Figure 8a). Modelling and seismicity parameters used in Figure 6 and Figure 8 are explained in Table 2 and Table 3.

The cumulative seismicity graphs in Figure 6 show that the seismic response for different areas within each mine are similar. The data follow trends that are quite linear for both graphs in Figure 6a, except for polygon P02 for Driefontein and P03 for Mponeng. Those two polygons are curved upwards when seismicity is plotted against area mined and become linear when compared to strain energy. These two polygons were therefore actually the only ones that were subject to continually increased levels of ERR.

**Figure 6** Cumulated apparent volume as a function of area mined (a) and strain energy released (b) for fixed polygons

---

1 The summation symbol (\( \Sigma \)) on the axes of Figure 7 and Figure 8 may print out as (S) on some printers.
### 3.4 Development of automatically generated polygons

The use of fixed polygons to group data was then changed to automatically generated polygons ("auto-polys") for better isolation of groups of seismicity and mining. This also allowed for separation of seismic events into those associated with active mining from events remote from active mining.

Auto-polys were created by erecting circular regions with radii of 50 m around each mined element and then grouping overlapping regions together into polygons. mined elements within each polygon were therefore at least 100 m from mined elements in other polygons and seismic events were unlikely to be attributed to the incorrect polygon, given the location error of about 40 m. Figure 7 illustrates the generation of auto-polys for one month for each of the two mines.

**Table 4 Number of mining steps and auto-polygons for each study area**

<table>
<thead>
<tr>
<th>Mine</th>
<th>Driefontein</th>
<th>Mponeng</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining steps (months)</td>
<td>53</td>
<td>42</td>
</tr>
<tr>
<td>Total number of polygons</td>
<td>593</td>
<td>374</td>
</tr>
</tbody>
</table>

**Figure 7** Sample showing auto polygons drawn around one month’s mining and seismicity during the same month.

**Table 5 Values used for graphs in Figure 6 and their interpretation.** In each case values from each polygon are sorted by increasing ERR and cumulated for model (X) and seismic data (Y).

<table>
<thead>
<tr>
<th>Label</th>
<th>X</th>
<th>Y</th>
<th>Phenomenon</th>
<th>Values</th>
</tr>
</thead>
<tbody>
<tr>
<td>(a)</td>
<td>$\sum A_M$</td>
<td>Several seismic measures, normalised to total of 100%</td>
<td>Seismicity rate increases with increasing span</td>
<td>n/a</td>
</tr>
<tr>
<td>(b)</td>
<td>$\sum E_M$</td>
<td>$\sum E_S$</td>
<td>Seismicity /area mined $\alpha$ ERR</td>
<td>n/a</td>
</tr>
<tr>
<td>(c)</td>
<td>$\sum E_M$</td>
<td>$\sum E_S$</td>
<td>$\eta$ (seismic energy efficiency)</td>
<td>0.0042</td>
</tr>
</tbody>
</table>
3.5 Interpretation of cumulative seismicity plots

Each graph in Figure 6a to e can be interpreted in terms of the response of the rock mass to mining.
The concave upwards shape of the graphs in Figure 8a indicate that the rate of seismicity per area mined (the slope of the graphs) increased with increasing levels of ERR. The least pronounced increase was for the event rate for Driefontein, perhaps indicating a steady growth of the fracture zone as mining advanced.

Figure 8b contrasts much more from Figure 8a than was the case for Figure 6a and Figure 6b. The increase in contrast when auto-polys are applied and the data sorted by ERR shows that seismicity per area mind is sensitive to changes in ERR. More particularly, the clustering of the graphs in Figure 8b about the constant rate of seismicity given by the dot-dashed lines is compatible with seismicity per area mined being proportional to ERR.

Data for seismic energy was extracted from Figure 8a and plotted in Figure 8c to show the estimates of the seismic efficiency, or the proportion of released energy that was radiated seismically. The derived values of 0.4% and 0.2% for the two mines were compatible with earlier work by Spottiswoode (1980).

The estimates of $\gamma_E$ of 0.25 and 0.19 derived from Figure 8d are higher than most of the values reported by Milev and Spottiswoode (1997), but substantially lower than the original work of McGarr and Wiebols (1977) who suggested that $\gamma_E$ should be equal to 1.0. As most of the values of $\gamma_E$ reported by Milev and Spottiswoode (1997) were less than 0.1 and were based on seismic moments inferred from local mine magnitudes, we suggest that the current estimates are more accurate than those of Milev and Spottiswoode (1997) and that our values better represent the seismic response of mining on the CLR and VCR. The factor of four or five difference between our work and that of McGarr and Wiebols (1977) should be considered in any future studies of $\gamma_E$.

Figure 8e was introduced here to indicate the degree to which seismic events occur in the near field of one another. The degree of near-field influence amongst the suite of events is calculated by dividing the area of inferred slip by the corresponding area of mining and is called the “overlap factor” here and has values of 73 and 67 for the two data sets (Table 5). The high degree of overlap could occur either by shear planes slipping many times or by shear planes forming close to previously formed shear planes. For comparison, the overlap factor of the shear fractures over face advance of Figure 1 from Adams et al. (2002) is about 3.2. Inspection of the figures of Ortlepp (1997, p46) yield similarly low values of overlap factor.

The difference in shape of the graphs of seismicity as a function of area mined and of strain energy released is dramatically illustrated in Figure 9 where it can be seen that cumulated apparent volume is a linear function of cumulated strain energy release for data from Mponeng. The proportionality of seismicity and strain energy released can also be tested by binning data points from polygons with similar values of ERR. Figure 10 shows excellent correlations between apparent stress per area mined and ERR for both mines.
Figure 9  Cumulative apparent volume as a function of cumulative area mined (a) and cumulative strain energy released (b) for Mponeng.

Figure 10 Correlation between apparent stress per area mined as a function of ERR with data from auto-polys binned or grouped in bins of approximately equal amounts of strain energy release.

3.6 Pillar strength

The dip pillars were left to reduce ERR and thereby resulting seismicity without themselves failing and generating excess seismicity. How would we know if they were indeed failing given the fact that a high rate of seismicity occurs for the last stages of mining towards the pillars? If they were indeed failing, then the rate of seismicity would be even higher due to merging of the failure zones associated with each of the two the faces as they approach one another. Our approach to answering the question was to reduce the cap stress in our models until so much strain energy was released that the linear relationships between seismicity and strain energy release apparent in Figure 8b and Figure 10 was clearly lost. This appears to be the case for values of cap stress of less than 300 MPa in Figure 11 and Figure 12 for both mines. The best estimates for pillar strength is therefore 300 MPa or more. The assumed value of pillar strength of 400 MPa for Driefontein (Klokow et al, 2003) falls within this range.
Figure 11 Effect of cap stress on strain energy release

Figure 12 shows that Apparent Volume (Va) correlates best with ERR for CLR data from Drie 5# and correlates well for VCR data from Mponeng. The six measures of total seismicity shown in Figure 12 were proposed by Spottiswoode (2004). The poorer correlation of other measures of seismicity with ERR is a reflection of the changing character of seismicity with ERR previously reported for Driefontein. The decreasing correlation factors at cap stress values of 300 MPa complement the results shown in Figure 11.

One of the major concerns about pillars is whether they could fail in the manner of a pillar run as happened in the Coalbrook disaster (e.g. van der Merwe, 2006). A full and complete pillar run of the type as was experienced at Coalbrook has, to our knowledge, never been experienced when squat (width: height >> 5) pillars have used. Pillars are at their most vulnerable to failure when they are at their narrowest and the ERR levels are at their highest. Seismicity per area mined is proportional to ERR without any sign of increased levels of seismicity at the highest values of ERR (Figure 10 and Figure 11).

Figure 12. Correlation of different measures of total seismicity with ERR for ranges of cap stress. Polygon data were grouped into ten data points containing increasing values of ERR and of approximately equal amounts of strain energy release.
3.7 Loading system

Allocation of seismicity to mining faces provided the opportunity of attributing an ERR value to each of the tens of thousand of seismic events. Seismic parameters for individual events have widely scattered values and it can be difficult to identify trends in large data sets unless the trend is very clear. One commonly used approach is to bin the data and to compute statistical parameters such as the mean and median to describe the population in each bin. The type of binning applied depends on the distribution of the entire population. In the case of log-normal distributions, it is prudent to use equal numbers of samples in each bin, rather than equal intervals. The resultant data points will then have equal confidence.

A program was written to perform the required binning and allowed the multi-parameter data sets to be explored quickly. The data for both mines was binned into 24 bins, each having an equal number of samples. For the Driefontein data, each bin contained 729 samples, whereas each bin for Mponeng contained 571 samples. We found some clear and unexpected variations of two parameters in particular, namely the P/S moment ratio and the Apparent Stress (Figure 8). Each data point represents the mean (circle) or median (solid dot) value calculated for each bin for either the P/S moment ratio or Apparent stress, plotted versus the face ERR of each of the 24 bins. A purely shearing source would have a P/S moment ratio equal to one, with P/S moment ratios greater than one indicating volume changes in the source region.

![Graph of P/S moment ratio versus face ERR](a)

![Graph of Apparent stress versus face ERR](b)

**Figure 13** Dependence of ratios of moments estimated from P waves & S waves (a) and apparent stress (b) as a function of the ERR at the closest active face.

Mponeng mine data behaved as expected. The P/S moment ratio is slightly more than one, suggesting a small influence of co-seismic stope convergence. Apparent stress versus face ERR is positively correlated, indicating that the driving stresses are higher, in agreement with increasing values of ERR.

The trends observed for the CLR (at Driefontein) data are completely different from those plotted for the VCR data from Mponeng and show a steady decrease in P/S moment ratio with increasing face ERR and a decreasing value of Apparent Stress with increasing ERR on the CLR.
We propose a conceptual model based on the behaviour of the fracture zone to explain the seismic characteristics for the Driefontein data in Figure 13 using an interpretation of Figure 1b. The system response of the diffuse fracture zone in Figure 1b to slip on shear zone S2 depends strongly on the behaviour of the fracture zone. For example, the case of the fracture zone being almost completely rigid probably explains the Mponeng data. On the other hand, a completely plastic or pliable fracture zone that provides the same strength during a seismic event as it did before the event would be no different from the case of the shear zone S1 in Figure 1b. If on the other hand, the fracture zone strain hardens during an event, providing more resistance, then a fracture zone that expands with increasing ERR would affect both P/S moment ratios and the Apparent Stress. The P/S moment ratios would increase as the shear zone becomes more isolated from the stope. Similarly, the Apparent Stress will decrease because the yielding fracture zone will reduce the rate of slip on the shear zone through providing a softer loading system.

The cap model in MINF can be used to estimate face stiffness, as illustrated in Figure 14. In this hypothetical example, deformations in the fracture zone add to those on the element being mined to reduce the loading stiffness that advances the fracture zone.

Figure 14  Diagram showing calculations involved when estimating face stiffness measured associated with face advance of an infinitely long longwall. C1 and C2 are stope convergence for two stages of mining and S1 and S2 are the equivalent stress values ahead of the face.

Table 6 Calculations of loading system stiffness illustrated in Figure 9. A is the area of a single element, or 36 m² in this study.

<table>
<thead>
<tr>
<th>Distance from face</th>
<th>No cap: Elastic</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total</td>
<td>245.5</td>
</tr>
<tr>
<td>1</td>
<td>0</td>
</tr>
<tr>
<td>2</td>
<td>-54.5</td>
</tr>
<tr>
<td>Total</td>
<td>480.3</td>
</tr>
<tr>
<td>A×Δσ, MPa</td>
<td>300</td>
</tr>
<tr>
<td>0.086</td>
<td>0.038</td>
</tr>
<tr>
<td>0.012</td>
<td>0.136</td>
</tr>
<tr>
<td>Stiffness = (A×Δσ)/ΔD</td>
<td>65 MPa/m</td>
</tr>
<tr>
<td></td>
<td>169 MPa/m</td>
</tr>
</tbody>
</table>

Stiffness was calculated for each auto-poly by dividing the sum of all force changes in each poly by the sum of all the deformations on mined elements and ahead of the face for a face advance of one element. The forces and deformations are illustrated in Figure 14. Figure 15 shows results for Driefontein data, suggesting that a loading stiffness model might explain the behaviour of the seismicity associated with increasing fracturing ahead of CLR stopes at bigger spans and increasing ERR.
3.8 Influence of geological features

Seismicity was controlled far more by mining than by the presence of geological features (faults and dykes). This was probably the result of good mining practice such as using bracket pillars within final pillars and mining through features at a good angle of incidence. Geological structures appeared to increase seismicity by about 9% on the VCR and 37% on the CLR. The greater influence on the CLR mining was at least partly attributable to the faults and dykes interrupting aseismic slip on bedding planes.
Figure 16 Geological features as used by MINSINT (a) Mining in green, faults in yellow and dykes in red. (b) Faults in green, fading to blue to show 40 m drop-off in influence. Dykes in red, fading to green to show 40 m drop-off in influence.

Figure 17 shows that seismicity is far more closely associated spatially with mining than with geological features. In these figures, the distribution of perfectly correlated data follows the vertical dashed line as shown in Figure 17a while uncorrelated data would follow the “random” line. The deviation by which seismicity departs from perfect correlation with active mining (Figure 17a) can be explained mostly by location error. In contrast, seismicity correlates only slightly with geological features (Figure 17b).

Figure 17  (a) Cumulative distributions of seismic locations from active mining faces at Driefontein as well as the distribution of the mid-points of all elements in the MINF model from active mining. (b) Cumulative distributions of seismic locations from the larger faults and dykes as shown in the 1:1000 mine plan as well as the distribution of the mid-points of all elements in the MINF model from the geological features

In Figure 18, the rate of seismicity per area mined can be seen to increase significantly in the vicinity of geological features for Driefontein data and to show much less of an increase at Mponeng. Results for other measures of seismicity are shown in Table 7. The greater increase in seismicity over that extrapolated from areas further than 40 m from geological features shows that larger events are preferentially attracted to geological features.
3.9 Double-sided mining

The 70 metre rule was developed by Handley et al (2000, p163). Their motivation for this rule was based on the following:

“…. A study by Handley (1993) revealed that mining-induced stress in the planned pillar area increased suddenly when the panels were 70 metres apart on strike, i.e. when both sets of panels approaching the pillar position were approximately 20 metres from their stopping positions.

The model showed that if one set of panels was stopped, the rate of stress increase in the pillar area was reduced, and by observation, so was the seismicity. The other set of panels could then mine in relative safety to the pillar stopping position. Once this had been achieved, the second set of panels could resume mining, and also reach the pillar stopping position in relative safety. This became known as the 70 metre rule, and is still applied at Elandsrand today.”

Apparently, this rule was made as part of a conservative and preliminary assessment of seismic potential that has unfortunately not, to our knowledge, been further tested to date. This rule has not been applied at Driefontein 5# (F J Castelyn, 2007). At Mponeng, the rule is applied but without much conviction of its value (R. Carstens, 2007).
In this section we will test whether mining from both sides of a final pillar generates any more or less seismicity per unit of strain energy release than mining on one side only. We will also assess the hazard associated with final extraction against a pillar as a function of different order of mining.

### 3.9.1 Seismicity

In this section the effect of double-sided mining is studied by grouping polygon data according to directions of mining, towards the East, the West or from directions.

As can be seen in Figure 19, reducing the face length does not reduce the amount of seismicity per strain energy release.

- The best-fit line for double-sided mining is shown to illustrate the lack of curvature that would have followed if there was strong interaction between faces converging towards one another.
- Similarly, the lack of noticeable curvature in the graphs for mining West only or East only suggest that the seismicity per elastic strain energy release is independent of the face length and/or face advance rate.
- The commonly-adopted policy of reducing the number of panels and/or face advance rate when mining at high levels of ERR does not reduce the total amount of seismicity. However, because face advance on mined panels transfers stress onto adjacent panels, increasing the likelihood of seismicity on those panels, reducing the number of adjacent panels mined reduces the exposure of people to possible rockbursts. This qualitative statement will be supported by quantitative modelling in the next section.

![Graphs showing cumulative seismicity](testing_70m_rule.png)

**Figure 19** Cumulative seismicity as a function of cumulative strain energy release. Before cumulating values, the data for mining steps and within polygons were sorted into increasing distance between faces mining towards one another for double-sided mining (“Both” in figure) and for increasing area mined for single-sided mining.

### 3.9.2 Modelling of double-sided mining

The last stages of mining towards a simplified pillar geometry are sketched out in Figure 20. Four options for the order of mining four panels, labelled A, B, C and D in Figure 20 were modelled and results presented in Table 8.

Table 8 summarises the proportion of strain energy that is:

1. released on each panel as it is mined;
2. released on adjacent panels and those on the opposite side of the pillar; and

3. on other pillars when a cap stress is applied.

As total amount of strain energy release is the same regardless of the order of mining, as was shown in GAP612c (Spottswoode et al., 2000), the proportion of energy that is released on the mined panels is indicative of the relative hazard on these panels while they are being mined.

Figure 20 Plan view of hypothetical mining of the last four panels adjacent to a pillar. (a) is a view of the entire mining layout and sequence. (b) shows the position of the last four panels (A, B, C & D) in detail.

Table 8 (a) Entire modelled mining layout showing order of mining and (b) with details showing four final panels, A, B, C and D that are mining in four different sequences, as shown in Table xx. Cap stress = 400 MPa.
Modelling work suggests that reducing the amount of mining on each side of a pillar is likely to be less hazardous than mining each side in turn. This is in contrast to the 70 m rule and places some doubt on the value of the so-called 70 m rule as a rockburst-control measure. On the other hand, total energy requirements for ventilation and cooling are reduced for the more rapid completion of mining achieved when single-sided mining is practiced.

### 3.10 Backfill

Backfill has been applied at both areas to provide for regional and local support and to assist with ventilation. This section contains results of modelling to obtain the strain energy release in the same way as shown in the previous section, with and without backfill. The influence of backfill and of allowing yield at a cap stress of 300 MPa is shown in Figure 21. 300 MPa was chosen as the cap stress as this was shown to be a conservative value of the pillar strength. In Figure 21a, the absolute values of cumulated strain energy are shown while in Figure 21b, these values are normalised by the strain energy released when no backfill has been inserted. Polygon data are sorted into increasing value of ERR for the elastic case with no backfill. The benefit of backfill increases linearly with increasing area mined, from 0% until a total reduction of 8.8% is achieved for all mined areas when a linear rock mass is assumed. If a cap stress of 300 MPa is applied in the models, the backfill still reduces the total amount of strain energy released, although only by 1.9%. However, if backfill was not installed and a cap stress of 300 MPa was applied, then not only was the total amount of strain energy released 13.4% more, but there was a high rate of increase in strain energy over the last 10 000 m² or so of stoping, suggesting that seismicity would have been severe without backfill if the pillar strengths were as low as 300 MPa.

![Effect of backfill and cap stress on strain energy released, Mpo](image)

(a)

![Change in strain energy released when backfill is inserted and cap stress applied](image)

(b)

Figure 21 Effect of backfill and cap stress on modelled strain energy release at Mponeng.
Table 9 Effect of backfill and cap stress on the total amount of modelled strain energy release at Mponeng.

<table>
<thead>
<tr>
<th>Model</th>
<th>Strain energy released, GJ</th>
<th>Percent increase from elastic, no backfill</th>
</tr>
</thead>
<tbody>
<tr>
<td>Elastic, no backfill (el)</td>
<td>4006</td>
<td>0.0</td>
</tr>
<tr>
<td>Elastic with backfill (el bf)</td>
<td>3643</td>
<td>-8.8</td>
</tr>
<tr>
<td>Cap stress 300 MPa, no backfill (no bf_300)</td>
<td>4542</td>
<td>+13.4</td>
</tr>
<tr>
<td>Cap stress 300 MPa with backfill (bf_300)</td>
<td>3931</td>
<td>-1.9</td>
</tr>
</tbody>
</table>

The maximum benefit of backfill as regional support for ERR reduction is achieved when no pillars are left and the gravitational energy that is the source of strain energy release is reduced in proportion to the reduction in maximum stope convergence. To assess mining up to the extreme of 100% mining, a MINF model was set up to simulate mining from an infinite series of raises through to 100% mining (Figure 22). Energy release as a function of percent mined with and without for backfill and cap stress is shown in Figure 23.

Figure 22 Portion of an infinite series of raises and pillars, with mining taking place simultaneously from blue towards red.

Figure 23 shows the benefit of backfill and the effect of cap stress on pillar stability very clearly. For the first 50% of mining, neither backfill nor cap stress have much effect on the amount of strain energy released. When the area is 75% mined, the stress on the pillars reaches their strength of 400 MPa and they deform (collapse) until the back areas hold the load that the pillars cannot hold. The highest values of ERR (proportional to the slope of the curves in Figure 23) for the elastic model without cap stress occur when the pillars are at their narrowest, as expected from theory.
The choice of stope width, Young’s modulus (70 GPa) and pillar spacing are such that backfill becomes a very effective stability factor and, by taking load off the shrinking pillars, reduce the amount of strain energy release so effectively that a much smaller increase in ERR occurs, and only at 98% mining. At this stage, most of the backfilled area is loaded almost to the virgin stress.

![Cumulative strain energy as a function of percent mined for an infinite series of pillars with infinite strength (“el”) and a cap strength of 400 MPa and with (“bf”) and without (“1m”) backfill.](image)

**Figure 23** Cumulative strain energy as a function of percent mined for an infinite series of pillars with infinite strength (“el”) and a cap strength of 400 MPa and with (“bf”) and without (“1m”) backfill.

In conclusion, backfill has the potential to dramatically reduce the ERR and likely seismicity even if pillars start to fail extensively.

### 3.11 Monthly hazard assessments

One of the steps in the project plan was to “Consider possible recommended changes to the mining method. This might include providing a rational design based on the on-going rock mass (deformations and seismicity) response to mining.” Spottiswoode (2005) reported that increasing seismicity in areas was more likely to be followed by a further increasing rate of seismicity. In this section, the persistence of an increased or decreased rate of seismicity is investigated in greater detail. As will be seen, a change in seismicity rate was a poor predictor of the amount of seismicity in the area next month.

Seismicity per strain energy release varies widely from month to month. It is probably not acceptable to follow a philosophy of “design as you mine” based on the rate of seismicity, as has been suggested previously.

When the rate of seismicity per strain energy release was high compared to the average over the entire mine in the current month it persisted into the next month 55% to 65% of the time. For 35% to 45% of the time, high seismicity rates are followed by low seismicity rates. Low seismicity rates also persisted somewhat more frequently than changing to rates higher than average. Figure 24 shows data for Mponeng.
Figure 24  Ratio of cumulative apparent volume (Va) to strain energy released (CENS) measured in polygons that have persisted for two months (VCR). The amount of seismicity has been normalised by the average rate of seismicity per strain energy release on both axes.

Currently, mines assess seismic hazard in terms of changes in seismicity within polygons. Figure 25 can be used to study whether an increase, or decrease, from one month to the next would persist into the following month. As can be seen in Figure 25, the reverse is true, namely changes are more likely to be reversed in the following month as would be expected if the total amount of seismicity is controlled by the amount of strain energy release and not by development of instabilities on the scale of a month.

Figure 25  A graph that helps answer the question: “Do changes in the seismicity rate from last month to this month persist to next month?”
4 Interpretation of tilt data

Site selection at Mponeng gold mine was done together with the Rock Engineering staff at the mine. A visit to the 104 Level haulage at Mponeng showed that this tunnel is located in very competent rock and very little damage was visible along its entire length. A number of sites were suitable for installation of the planned instruments, namely tiltmeters, strain-change meters and a reference trigger geophone. Subsequent underground visits indicated that the 113 Level Haulage West at Mponeng provided more suitable sites than the previously selected 104 Level site.

4.1.1 General description of the site

Mponeng Mine is part of the old Western Deep Levels gold mine, situated near Carletonville in South Africa. The mining method was changed from longwalling to Sequential Grid mining from 1996 onwards (McGill, 2005). Stopes are mined to an average stoping width of 1.4 m. Mining takes place exclusively on the Ventersdorp Contact Reef (VCR), which dips at about 24° to the South. Stopes are stabilized with backfill for both local and regional support and the faces are preconditioned as part of the blast cycle.

The major part of the area currently being mined in the southwest part of the mine is shown in Figure 26. The mining pattern related to the seven raises is shown. The planned mining consists of 180 m spans separated by 30 m pillars (McGill, 2005). The additional raise to the west of the site was constructed to avoid mining through a dyke.

![Figure 26 Mining outline in June 2005 and contours showing mining from August to November 2005 with seismic events from September 2005 to February 2006. The area in the box is shown in more detail in Figure 27.](image-url)
4.1.2 Dynamic tilt analyses

In this project, the tilt data were used to study the behaviour of deep-level mine layouts, particularly the performance of dip pillar systems. The tilt data were recorded by a triggered seismic recording system. The triggered data showed velocity and tilt data that correlated well in time, allowing for separation of the tilt data into coseismic and aseismic tilting.

4.1.2.1 Tilt and velocity recordings

Samples of recordings from the in-hole geophones and the tiltmeters recorded by the microseismic system installed at the site are shown in Figure 28. Two seismic events were chosen: a near-field seismic event Figure 28(a) and a far-field seismic event Figure 28 (b). The velocity traces from the geophones show the typical pattern familiar to seismologists with the further event showing longer apparent duration of ground velocity than the closer event. The tilt records differ in two major aspects:

- well-defined offset, or "permanent" deformation, particularly for the near-field event in Figure 28(a); and
- low-frequency response caused by the use of a fluid as a measuring transducer.

Overall, the decision to use the geophone signal to trigger the system was effective as well-defined tilt traces were recorded in most cases. We will also show the power-law behaviour of tilt steps suggest that all tilt steps above about 0.2 μradians have been recorded.

Figure 29 shows the tilt records settling to their final value together with our automatic fit to the data. The characteristic parameters of the low-pass response were obtained through trial and error fitting to selected tilt records. As is evident in Figure 29, the fit indicates a background tilt value prior to the seismic waves arriving at the tiltmeter (tilt$_{before}$) and a final tilt value after the waves have passed (tilt$_{after}$). The tilt values are set to zero at the end of the traces in Figure 29 to show the similarity in the damped response to a tilt change.

*Figure 27 Monitoring site at Mponeng Gold Mine.*
We refer to the tilt change during the event \((\text{tilt}_{\text{after}} - \text{tilt}_{\text{before}})\) as “coseismic” tilting and the tilt between successive events as “aseismic” tilting. The use of event triggering is possibly the best way of uniquely isolating the coseismic from the background aseismic tilting.

Figure 28 Example of coseismic tilt recorded from: (a) near-field seismic event with hypocentral distance 53 m; (b) far-field seismic event with hypocentral distance 259 m. The tilt traces are presented with the calculated final values plotting at zero tilt.
4.1.2.2 Analysis of tilt data

Figure 30 is a graph of estimated tilt values at the beginning and end of each tilt trace for tiltmeter 1. Three type of characteristic behaviour are marked in this figure:

1. Coseismic tilt steps during seismic events;
2. Aseismic tilting between events; and
3. Anomalous tilt changes that occur between events. These changes last some hours and are then corrected through a tilt change in the opposite direction. It is unlikely that genuine aseismic deformations could generate these offsets. Most probably these changes are artefacts generated by the measuring instrument.
In addition, some records did not have the flat portion at the start of the recording as indicated in Figure 28 and Figure 29 or end as well-behaved as in Figure 29. This was either due to early or late triggering or due to multiple events appearing in the same window. About 1% of the beginning levels and 10% of the ending levels were considered to be unreliable (RMS > 0.4 µradians) and the following adjustments were applied automatically to event pairs in which event B follows event A:

1. If the end of A and the beginning of B are both noisy, combine A and B into a single event;
2. If only the end of A is noisy, copy the value from the start of B; and
3. If only the start of B is noisy, copy the value from the end of A.

After removing the anomalous temporary tilt changes, making the adjustments mentioned above, separating the aseismic tilt from the coseismic tilt and then summing each of the tilt changes with time, the tilt history could be expressed as in Figure 31. Coseismic steps and aseismic deformations are now much clearer and exhibit some consistent features, such as:

4. Coseismic tilt steps often switch directions and generally cancel one another out on a scale of days;
5. Coseismic tilt steps show power-law behaviour over two or more orders of magnitude, as shown in Figure 32;
6. Aseismic tilting occurs in both directions, persisting in either direction over many weeks. There are some stages of rapid tilt that will require careful investigation; and
7. The aseismic X and Y records are similar in appearance, indicating overall tilt values at 45° to the X and Y components. There are also stages of correlation in the coseismic tilting, but these are not as coherent as in the aseismic data.

![Figure 31 Cumulative tilt, processed to separate coseismic (sx & sy) from aseismic (ax and ay) tilting](image)

Figure 31 is a “Frequency-Magnitude” plot similar to those used to study b-values of populations of seismic events. The scaling law shown in Figure 32 can be explained in terms of a size distribution of seismic events as follows, using the normally observed b = 1.0:

Tilt change \( \sim M_0 \)

\( M_0 \sim 10^{1.5 \cdot M} \)

\( N \sim M^b \)
where $M_0$ is the seismic moment and $N$ is the number of events of Magnitude $M$.

Assuming $b = 1$, and rearranging the terms gives:

$$N \sim (2/3) \times \log(M_0),$$ as illustrated.

![Diagram showing power law behavior of coseismic tilt changes](image)

**Figure 32** Power law behaviour of coseismic tilt changes

The power-law behavior shown in Figure 32 is as clear as any obtained from mine seismic data. As with the seismic data, the slope is flatter than -1 and indicates that only a minute portion of the total amount of tilt change is missing. If the data continues at the same rate down to zero values of tilt steps, then only about 1% of all tilt steps will be lost to tilt values smaller than 0.1 µradian, as estimated from direct and numerical integration.

### 4.1.2.3 Coseismic and aseismic rock deformations

Referring to the data shown in Figure 31, there were many instances of coseismic slip being reversed during a later event. We show possible mechanisms of tilt reversals in Figure 33. Four suggested sources of tilt are shown, marked as A to D, with A, B and C being shear slip ahead of the face and D being stope closure associated with face advance. All of these sources result in stope closure. Sources A and C are expected to result in counter-clockwise rotation or tilt at the site marked in the footwall, whereas sources B and D are expected to result in clockwise rotation or tilt. If source A or C follows source B or D, or vice versa, then opposite tilting would occur, as illustrated in Figure 31.
Studies of time of day distributions of deformation provide a useful way to study the time-dependent behaviour of the rock mass because stope blasting occurs at approximately the same time every day. In a perfectly elastic world, all deformations would take place at or very soon after the faces are blasted. In the real world, however, the change of the stope geometry is a continuous process.

Malan (1999) has extensively studied the time-dependent behaviour of mine excavations, most particularly stopes, for example Figure 34.

To create a graph similar to those presented by Malan (1999), we stacked the data for all 109 days onto a single day, taking absolute values of tilt changes as indicators of the total amount of deformation, as shown in Figure 35. This figure also shows the cumulative number of events recorded by the mine network having moment-magnitudes greater than zero. The rate of coseismic and aseismic tilting and seismicity as recorded by the mine network are approximately constant until the daily blasting time which takes place from about 19:30 until shortly before 21:00 in this area. After this time, all parameters show a decreasing rate which is summarized in Table 10.
Figure 35 Cumulative absolute tilt changes stacked over 24 hours. The values of aseismic tilting were halved for easier visual comparison between coseismic and aseismic tilting.

The increased rates during the blast window as shown in Figure 35 and Table 10 are most pronounced for coseismic tilting and least pronounced for aseismic tilting. Similarly the aseismic tilting returns most closely to the background rate of aseismic tilting. Work prepared for the talk by Milev and Spottiswoode (2008b) led to the simplification of preparation of Figure 35 into Figure 36. Availability of blasting data from Mponeng as read from an HTML file by custom software was also added to Figure 36 and results summarise in Table 10.
Figure 36 Vector sum of coseismic and aseismic tilt, seismicity and blasting as a function of time of day. The time scale in (b) is enlarged from that in (a).

Table 10 Average rate of tilting and seismicity during blasting time and of post-blast tilting and seismicity as a function of rate of tilting and seismicity before the blast.

<table>
<thead>
<tr>
<th>Rate</th>
<th>Rate</th>
</tr>
</thead>
<tbody>
<tr>
<td>19:30 to 21:00</td>
<td>21:00 to 24:00</td>
</tr>
<tr>
<td>00:00 to 19:30</td>
<td>00:00 to 19:30</td>
</tr>
<tr>
<td>Coseismic tilting</td>
<td>X 19.7</td>
</tr>
<tr>
<td></td>
<td>Y 20.8</td>
</tr>
<tr>
<td>Aseismic tilting</td>
<td>X 4.4</td>
</tr>
<tr>
<td></td>
<td>Y 4.8</td>
</tr>
<tr>
<td>Seismicity; M&gt;0</td>
<td>5.8</td>
</tr>
<tr>
<td>Seismicity; M&gt;1</td>
<td>6.9</td>
</tr>
<tr>
<td>Blasting</td>
<td>35.6</td>
</tr>
<tr>
<td>6/2005 to 12/2007, lower VCR</td>
<td></td>
</tr>
</tbody>
</table>

4.1.2.4 Discussion

The object of this work has been specifically to assist in extracting insights into the loads carried by the dip pillars and their behaviour during their formation. However, the tilt data are illustrating coseismic and aseismic behaviour that has widespread implications for the time-dependent behaviour of the rock mass. This is particularly evident when the tilt data are summed, or stacked, onto one day and compared to the in-stope measurements of Malan (1999). Malan’s records show a very sharp increased rate at blasting time because his data were recorded on a single day and because his instrument was within metres of the stope face and the blast time was therefore over within minutes, compared to the time window of about one hour shown in Figure 16. In contrast, the tilt data shown here consists of discrete coseismic jumps superimposed on a background tilt rate that is significantly less influenced by the blast time.

The coseismic tilt steps occur when a sufficiently large event takes place. Figure 34 shows only one such step for closure, but still shows strong after-tilting with no recognizable steps either after the blasting or after the event. This after-tilting can therefore be called aseismic closure. In contrast, the aseismic tilting in Figure 35 is very much smaller than the coseismic tilting.

These observations beg the question: why does the in-stope closure consist of a greater proportion of aseismic closure than is the case for tilt measurements made off-reef? Our
preliminary interpretation is that the in-stope closure is dominated by deformations in the low-stress, fractured region around the mined-out area where seismicity does not take place. Due to the low stresses acting in this region, these deformations have limited influence on the generally elastic rock mass between the stopes and the tiltmeter site 87 m in the footwall. Our tilt data, on the other hand, are responding to stress changes in the highly stressed region ahead of the face.

We suggest that seismicity forms the major reliever of stress in this region through failure of previously unfractured rock with simultaneous remobilization of the fracture zone. Slow, time-dependent deformations of the fracture zone have a lesser influence on regional deformations, as measured by aseismic tilting.

4.1.2.5 Dynamic tilt conclusions

Although the tilt records themselves are generally very clean, analysis of the data has been difficult at times. Inevitably, questions have arisen in connection with quality and interpretation and these have been addressed, at least partially. In the process, methods have been developed that will simplify processing of more tilt, modeling and seismic data and will facilitate further analysis.

The following results have been obtained from analysis of data recorded over 109 days:

- Coseismic tilt jumps occur in either direction;
- The total amount of coseismic tilt in each direction is approximately the same;
- The time-of-day tilting shows the effect of blasting on the coseismic tilting but to a lesser degree on aseismic tilting;
- Tilt jumps show a power-law behaviour compatible with the Gutenberg-Richter relationship with \( b = 1 \) over more than two orders of magnitude; and
- Numerical tools have been developed to estimate tilting from mining, assuming elastic rock mass behaviour.

4.1.3 Quasi-static tilt analyses

The existing monitoring system, Impulse, was upgraded with a feature to provide independent continuous tilt readings. This enabled us to study aseismic deformations with greater detail. Two types of tilt were observed:

- Tilt associated with a seismic event - this type of tilt has a similar slope before and after the seismic event and jumps during the seismic event. We defined this as ‘fast’ seismic event type of behaviour;
- Tilt not associated with a seismic event – this type of tilt is shows an increase of the tilt without a seismic event being recorded. We defined this as ‘slow’ seismic event type of behaviour.

The tilt rate associated with the ‘fast’ seismic event is shown in Figure 37. In this figure, the jump in the tilt has been removed for better illustration of the tilt rate before and after the seismic event. Figure 38 illustrates the seismograms and dynamic tilt for the seismic event listed in Figure 37.
Figure 37 The tilt rate associated with the 'fast' seismic event (Example 1)

Figure 38 Dynamic tilt and seismograms for the seismic event listed in Figure 37

Second example of a tilt rate associated with a seismic event is shown in Figure 39. The dynamic tilt for this event is given in Figure 40.
Figure 39 The tilt rate associated with the ‘fast’ seismic event (Example 2)

Figure 40 Dynamic tilt and associated seismic event listed in Figure 39

Two examples of slow seismic events are given in Figure 41 and Figure 42.
Figure 41 The ‘slow’ seismic event shown in expanded time scale (Example 1)

Figure 42 The ‘slow’ seismic event shown in expanded time scale (Example 2)
5 Time-dependent failure of brittle rock

A new mechanism is needed to explain time-dependent failure of brittle rock. Several types of observation are incompatible with current rock mechanics theory, such as

5.1.1 Lack of after-tilting

Figure 43 Continuous tilt recording (one minute intervals) showing the large tilt steps associated with seismic events, measured at Mponeng.

Figure 44 Aftertilt associated with a “fast” seismic event on 2007/08/30. The co-seismic jump has been removed to highlight the lack of change in tilt rate that might have followed the seismic event.

In an attempt to identify changes in ground tilting following seismic events, pre- and post-seismic tilting was studied, as in Figure 43 and Figure 44. We were surprised to see a complete lack of after-tilting following seismic events, as shown in Figure 44. When considered against changes in stope convergence following seismic events, widespread continuous time-
dependent deformations are concentrated in the “ductile” low-stress region around stopes whereas seismicity responds to stress-weakening in the high-stress region ahead of stopes.

5.1.2 Seismicity distant from active mining

In Figure 9, cap stress values of 300 MPa or more were shown to adequately model the amount of back-area seismicity purely from elastic stress transfer over distances of up to 50m, the range to which the auto-polygons extended from areas of active mining.

At distance greater than 50m, the incidence of seismicity falls off slower than modeled using quasi-static modeling, as shown in Figure 45. At distances greater than about 300 m, a cap strength less than 300 MPa would be required to maintain the observed proportion of seismicity distant from active mining compared to seismicity close to active mining.

It is suggested that time-dependent weakening is the cause of the seismicity over and above that expected from quasi-static modeling. As can be seen in Figure 46, seismicity remote from active mining is widely spread across areas of earlier mining.

Figure 45 Comparison between strain energy release cumulated from far to near with seismicity similarly cumulated, Drie 5#.
5.1.3 Aftershocks

Figure 46 Seismic events locating more than 200m from active mining

Figure 47 Rate of background seismicity, aftershocks and ratio of aftershocks to background seismicity as a function of distance from M>2 seismic events. Background and aftershocks are M>0.
6 Strong ground motion measurements

During the course of this project a new device based on a triaxial accelerometer was developed and manufactured. The instrument, the Strong Ground Motion Detector (SGMD), is a portable battery-powered stand-alone device with backed-up memory capable of storing up to 163 triaxial accelerograms. Some of the parameters such as window length, pre-trigger and post-trigger length, sampling rate and the trigger criteria are user-selectable. A photograph illustrating the device installed in underground conditions is shown in Figure 48.

![Figure 48](image)

Figure 48 Photograph of the SGM installed at Mponeng mine Site 113 / 46

The device was tested on the surface using a drop test conducted at Savuka Testing facility on 17/11/2005. A 10 t mass was dropped from different heights on a Durapack support unit to simulate the effect of seismic loading. Two SGMDs were attached to the Durapack. The first SGMD was attached on the side of the Durapack close to the top, and the second one was attached beneath the support unit. However, due to malfunctioning of the trigger mechanism of the second unit, an accelerogram was recorded only from the top SGMD.

The recorded accelerogram was generated by a drop performed from 0.48 m height which generated an impact velocity of about 3 m/s. This accelerogram is shown in Figure 49.
**Figure 49** Accelerogram recorded during a drop test at Savuka Testing Centre

It is clearly shown in Figure 49 that the maximum acceleration is along the vertical direction. The accelerogram is highly asymmetrical. The horizontal components show similar accelerograms. This indicates that the plastic deformations due to the impact are much larger than the elastic deformations and the Durapack sustained permanent damage. The accelerograms were integrated to velocities and a maximum velocity of 1.1 m/s was calculated at the side wall of the support oriented in the vertical direction. The velocities calculated from the horizontal components are 0.6 m/s along the support unit and 0.5 m/s in a direction normal to the support unit.

Similar behaviour is expected underground where the hangingwall has been accelerated by a strong seismic event beyond the threshold of elastic deformations. This is illustrated by the accelerogram shown in Figure 50. In the case of small seismic events the hangingwall behaviour is within the range of the elastic deformations, resulting in the accelerogram showing deflections in both positive and negative directions. This is illustrated in Figure 51 where the accelerogram is well developed in both directions. However, there are some discrepancies with the data that need to be resolved, namely:

- The presence of a great number of false triggers in the data set,
- The asymmetry shown in Figure 50 appears strongly on all three channels, and
- The three traces cross-correlated with one another show a correlation factor typically of greater than 0.9. This is in contrast to previous studies where geophones were used in which the cross-correlation factors rarely exceeded 0.3.

**Figure 50** Accelerogram of ‘large’ seismic events recorded underground at Site 113 / 46
An unexpected and unrealistic high degree of similarity amongst the X, Y and Z components was found during the testing phase. We suspected that the mechanical arrangement of the transducers, batteries and recording electronics resulted in vibration causing cross-axis effects. Further development was carried out to move the transducers out of the main body. This allows an independent installation, where the accelerometer is attached directly to the hanging wall. A schematic diagram of the latest SGMD design is shown in Figure 52.

Two seismic events were chosen, one closer to the instrument (Figure 53) and one further from the instrument (Figure 54) to illustrate the enhancement of the recorded signal.
Figure 53 The seismogram and the power spectrum of close seismic event: $f_0$ is the corner frequency; $f_C$ is the cut-off frequency; and $f_N$ is the Nyquist frequency.

The acceleration spectra shown in Figure 53 and Figure 54 are annotated to illustrate the following important features, most of which were described by Spottiswoode (1993). Frequency values are shown in Table 6.1. The double-dashed lines marked with “+2” show the characteristic low frequency behaviour for far-field seismograms, namely acceleration spectral values increasing as frequency $f_2$. If the corner frequency ($f_0$) is significantly less than $f_{\text{MAX}}$ (Hanks, 1982, Spottiswoode, 1993), then the acceleration spectrum should be flat between $f_0$ and $f_{\text{MAX}}$, as is the case for the closer event shown in Figure 54, but not for the more distant event shown in Figure 53.

Table 6.1 Description and values of frequencies marked in Figure 53 and Figure 54

<table>
<thead>
<tr>
<th>Marked frequency</th>
<th>Description</th>
<th>Value in Figure 53 (Hz)</th>
<th>Value in Figure 54</th>
</tr>
</thead>
<tbody>
<tr>
<td>$f_L$</td>
<td>Upper limit of noise or stope resonance</td>
<td>30</td>
<td>22</td>
</tr>
<tr>
<td>$f_0$</td>
<td>Corner frequency</td>
<td>$\leq 73$</td>
<td>52</td>
</tr>
<tr>
<td>$f_{\text{MAX}}$</td>
<td>Frequency above which spectral energy is severely reduced by inelastic attenuation</td>
<td>73</td>
<td>111</td>
</tr>
<tr>
<td>$f_C$</td>
<td>Antialiasing filter</td>
<td>250</td>
<td>250</td>
</tr>
<tr>
<td>$f_N$</td>
<td>Nyquist frequency</td>
<td>500</td>
<td>500</td>
</tr>
</tbody>
</table>

The increased spectral values at frequencies below $f_L$ are of concern and will be studied in more detail during the next quarter as will more interpretation of the seismograms. Suffice it to say
that at this stage that records of true strong ground motion are very likely to provide insights into hanging-wall behaviour that were alluded to in earlier work.

It can be seen from Figure 53 and Figure 54 that each component X, Y and Z has a specific character corresponding to the expected ground acceleration on the surface of the excavations.

Another point of concern was the lower sampling rate (1000 Hz) that the instrument is using. We calculated the power spectrum for both examples (shown in upper right part of Figure 53 and Figure 54). In both cases the corner frequencies are laying around 100 and 110 Hz. Most of the expected strong ground motion events have frequencies below that threshold. Therefore, there is a very little misrepresentation of the recorded acceleration in this frequency band. However, the spectral behaviour is not adequate in the low frequency range. An enhancement of the signal could be done by applying of low frequency filter.

As an additional step towards a better SGMD, Techware and M&M has been commissioned to use MEMS accelerometers instead of the current piezofilm accelerometers.

The Strong Ground Motion Detector (SGMD) device has been redesigned to use fourth generation MEMS accelerometers. Laboratory tests using a shake table were performed at the National Metrology Laboratory at the CSIR. Each component was tested independently and an excitation force of 1 g and 0.5 g was applied over a frequency range between 10 Hz and 400 Hz. During the calibration a ‘chirp’ signal was generated using a specially designed switch on the trigger component. During the first test difficulties were encountered in calibrating the heavy (~300 gm) boats of the SGMDs. Other difficulties were found with the DC supply of the SGMDs and the strong motion limitations of the calibration equipment.
7 Moment tensors

In this section, the nature of the seismic sources as described by parameters computed from its moment tensor was investigated in relation to the stress state associated with the formation of the pillars at Mponeng. The seismic moment tensor is a convenient quantity that describes the size of an event, its source geometry, and the nature of the source mechanism and can be calculated using recordings of the radiated wavefield through moment tensor inversion (MTI). The main objective of this study was to determine whether or not the nature of the seismicity changes as the pillars are formed.

7.1 Summary results of moment tensor inversions

Only a brief summary of this work is presented here as the results were subject to two major problems with the data that led to us not being sufficiently confident of the results to present them as correct interpretation in terms of moment tensors. A full report of the work is presented in a section of the supplementary report that was written before the full impact of the problems was recognised.

Summary results in Figure 55, Figure 56 and Figure 57 are shown and interpretations attempted before the problems with the data are explained.

Figure 55 Fracture map for the January 2005 - June 2006 mining steps. Fracture planes are identified using Doppler shift in S-wave corner frequency (Mponeng)
Figure 56 Nature of the source versus the face ERR closest to each event for Mponeng

The main findings of the MTI work that did not bear close scrutiny were:

- Not all the fault planes align with the fracture patterns expected from the mining geometry and known geological features (Figure 57). This is contrary to the many observation of shear slip, either on geological features or when mining-induced shear zones are formed.
Almost 50 per cent of the events studied showed implosive mechanisms indicating that co-seismic closure of the stope is a factor. Approximately 35 per cent are dominated by shearing, and 15 per cent are tensile in nature (Figure 56). There is no plausible mechanism for large-scale tensile failure in deep gold mines. Recent work using long-period carefully calibrated data from four stations has recently led to the conclusion that source mechanisms are predominantly shear with up to 60% implosive volume change (Boettcher et al, 2008).

7.2 Problems with interpretation of seismograms

Displacement seismograms in the far field of point sources in infinite homogeneous elastic solids should have the appearance of a single uni-directional pulse, with the P-wave first motions in the direction of the line from the source to the geophone site. In the next two sections, it will be shown that this ideal was not met and it is suggested that it is difficult to say whether any of the moment tensor inversions derived for this study are correct.

7.2.1 Geophone orientations

The alignment of geophones in the Mponeng network were given to be such that motions towards the South, West and down would show positive signals in channel numbers 1, 2 and 3 respectively. In Figure 58, the angle of maximum motion of the early P waves is compared to the angle expected from the event and geophone location and the given geophone orientations.

It can be seen in Figure 58a that most of the geophones showed consistent results for the horizontal components of motion, as indicated by the line of Y=X. Data from sites S14 and S15 are suggestive of errors in the wiring from the geophones to the recording equipment while the data for site S13 are compatible with the geophones misaligned by about 18°.

The comparisons shown in Figure 58b are unlikely to be attributable solely to misalignment of geophones as horizontal and vertical geophones normally only operate satisfactorily within 5° to 10° of their specified orientation. A more likely explanation is that the sub-horizontal layering and stoping will lead to ray paths bending more in a vertical sense than in a horizontal sense.

Any deviation from the lines Y=X in Figure 58 will lead to difficulties interpreting data in terms of moment tensors.

Figure 58 Comparison between horizontal (a) and vertical (b) angles of incidence inferred from P-wave amplitudes to those inferred from geophone orientations and event and geophone coordinates.
7.2.2 Seismogram shapes

Displacement seismograms in the far field of point sources in infinite homogeneous elastic solids should have the appearance of a single uni-directional pulse. One of the simpler seismograms that was analysed in this project is displayed in Figure 59 as displacement seismogram obtained by direct integration Figure 59a and then after rotation Figure 59b & c. The P-wave pulse is enlarged in the lower portion of Figure 59.

There are two important features in Figure 59:

1. The vertical component of motion (green trace in Figure 59a) is larger than the N-S component. This should suggest that the ray path approaches the geophone site at an angle of more than 45° from the horizontal. It can be seen in Figure 58 that none of the ray paths is more vertical than 45° from the horizontal, whereas many of the points of sites S21 and S27 showed an apparent incident angle greater than 45°, as shown also in Figure 59.

2. The P-wave source pulse, while it is linearly polarised (Figure 59b), is far from being uni-directional and moves a large amount in the opposite direction from the first motion. This factor was only investigated in light of the work of Spottiswoode et al (2006) who found that pillar-failure events at a platinum mine showed consistent bi-directional P-wave ground motions similar to those in Figure 59.

![Figure 59](image)

**Figure 59** Displacement seismograms at site 27 of event 1050928001. (a) represents the raw seismograms integrated once to represent ground displacements, (b) has been rotated towards the source based on P-wave amplitudes and (c) has been rotated towards the source based geophone directions and geophone and event locations.
8 References


