PROJECT NUMBER       SIM 02 03 02

TITLE OF PROJECT     Proactive approaches to rock mass stability and control

PRIMARY OUTPUTS

- A methodology for tap-testing as a means of quantification of seismic susceptibility of the rock mass close to excavations
- A study on the correlation between blasting configuration, system stiffness and seismicity
- Assessment of the effectiveness of Integration for Controlled Mining® projects being conducted by the mining industry (details subject to mine management approval)
- Booklet-ready form containing guidelines for Tap-testing
- Booklet-ready form containing guidelines for reduction of average short-term seismic risk by preferential scheduling of panel blasts

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Executive Summary

This project had three objectives. The first two objectives were to investigate further and, if possible, implement some findings of the fundamental research project GAP601a: tap-tests and panel sequencing. The third objective of the project was to investigate and report on the trials of Integration for Controlled Mining, a technique in which numerical modelling is combined with seismic information to provide short-term guides for mining.

The concept of **tap-tests** is that the immediate seismic response to blasting contains information about the general levels of stress and seismic hazard in the vicinity. Quantitative acoustic emission information was recorded within 2 hours after blasting at a variety of sites, and compared with the levels of seismic activity recorded by mine-wide networks over 3 month periods. What emerged is that the correlation between short- and long-term seismic responses does indeed exist, and that tap-tests can be easily implemented.

![Figure: Short-term seismic responses to blasting for two mines with different levels of seismic activity. The responses on the right come from the mine with lower levels of seismicity. Note the different vertical scales.](image)

The concept that the character of the short-term seismic responses to mining can be controlled to a degree by daily **panel sequencing** is appealing. It seems reasonable that blasting many panels close together in space would disturb a larger volume of rock and promote larger seismic events than distributed blasting, and this project attempted to prove this and develop some practical guidelines. A high resolution seismic micro-network was installed at Mponeng mine, and the seismic stiffness from data after blasting was compared with the blasting configuration. Two pillars were studied, and while the more complex pillar did not exhibit a consistent relationship, the simpler pillar showed a link between seismic stiffness (and therefore seismic hazard) and blasting configuration.
Figure: Blasting distributed panels tends to degrade the seismic stiffness less (and therefore leads to less seismic hazard) than blasting neighbouring panels at this pillar at Mponeng mine, even though the same amount of ore is extracted.

Integration for Controlled Mining is a numerical modelling technique in which recorded seismic data is combined with an elastic model for improved seismic hazard assessment in the short- and medium-term. The technique has been tried in various places, with mixed results. It would seem that while the technique is not ready for routine use now, it is at the implementation stage. The new generation of inelastic models currently under development and calibration should have even more enhanced accuracy and be able to model the evolution of inelastic deformation.

Figure: Integration of numerical models with recorded seismic data has the potential to offer improved seismic hazard assessments in the short- and medium-term, although the technique is only at the implementation stage now.
Acknowledgements

This research project would not have been possible without the support and assistance from several mine personnel at the Mponeng, ARM 2-shaft, Great Noligwa and Turfontein mines in South Africa. In particular, assistance from Johan Viljoen, Rob McGill, Wikus Mahne, Tony Ward, Shana Trollope, Johan Oelofse and Gordon Holder is gratefully acknowledged.
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1. Introduction

SIM020302 is an implementation project. The fundamental research project GAP601a† identified a number of possibilities for better understanding and even controlling the nature of the seismic rockmass response to mining. Two of these concepts, tap-tests and panel sequencing, were selected for further investigation and, if possible, implementation. In addition, a new development in numerical modeling – Integration for Controlled Mining – was been trialed at a few mines, and an objective of this project was to report on this to the wider industry.

The general principle behind Tap tests has been illuminated in GAP601a, and is summarised only briefly here. Condensed matter theories of statistical physics predicts that system susceptibility, however that may be measured, is a sensitive indicator of a systems approach to criticality (see GAP601a final report and the references therein). It has been shown that sensitive seismic monitoring can record such system responses to production blasts, and as such may allow susceptibility to be inferred. It was surmised that the greater the response to a standard production blast, the larger the likelihood of short-term seismic hazard. The purpose of the first part of the project was to establish the usefulness of this idea, calibrate the tap-test seismic response to subsequent seismic hazard and develop routine procedures and guidelines for these measurements. Section 2 details the work done on this part of the project.

![Figure 1.1: Schematic of most probable response of complex systems to a perturbation. The red (lower) line indicates the perturbation or tap, while blue (upper) line indicates the noisy response of the system. The left figure indicates a system close to equilibrium, while the right figure indicates a system closer to the critical point.](image)

In GAP601a it was observed that different blasting arrangements of the same number of panels could lead to different short-term seismic responses. It was surmised that a mode-switching comes into play when panels are blasted contiguously, i.e. neighbouring panel are blasted, leading to larger system instabilities. The second part of the project was directed towards confirming this observation and formulating a guideline for management of seismic hazard through intelligent scheduling of panel blasting. To accomplish this, quality seismic data from high resolution seismic micro-networks was compared to

† GAP601a, “Experimental and theoretical investigations of fundamental processes in mining induced fracturing and rock instability close to excavations”, SIMRAC/ISS International, 1999-2002
blasting configurations. Section 3 contains the work done on this topic during this project.

Numerical modeling is widely used for long-term (time scale of years) assessments of the stability of particular mining areas. However, the use of numerical modeling as a short-term (time scale of weeks) rock engineering tool has not been possible until recently. Two things have unlocked this: computer power advances allowing more detailed models, and work on the integration of recorded seismic data with these numerical models – see SIMRAC project GAP603. To determine whether enough work had been done on the modeling/seismic data combination, a few industry-supported projects were launched. These projects were called Integration for Controlled Mining, and the results of these practical trials are reported on in the third part of this project – Section 4.

Final discussions and future directions for research are contained in the last Section.
2. Tap-tests

The objective of this part of the project was to conduct a few tap-tests and correlate the results with seismic hazard in the area. If such a correlation could be found, then this would be a useful tool in assessing the seismic hazard of a mining region in the absence of a seismic monitoring network. One could imagine a situation where a number of isolated pillars in an old mine were being assessed for seismic hazard, and no reliable seismic data was available. In such a case, a quick test to indicate such hazard would be useful.

2.1 Tap-test sites

During the course of this project 7 South African sites were selected for tap-tests: 5 at ARM 2-shaft (West Wits region), 1 at Turfontein mine (Rustenburg Platinum Mines) and 1 at Mponeng mine (Western Levels). In addition, data collected during GAP601a at TauTona mine (Western Levels) was further analysed and compared.

The 5 ARM2# sites varied in terms of expected seismic hazard, due to differences in depth, geology and isolation from virgin rock. The tap-tests were conducted as these sites during September to December 2002.

50-40-RAW, ARM2#: This tiny pillar is at a depth of 1500m below surface, and forms part of the shaft pillar. Seismic activity is expected to be very low here, because of the shallow depth and relatively clean geology. The pillar is some 25m x 30m in extent.

Figure 2.1.1: Plan view of the 50-40-RAW site (outlined) at ARM 2-shaft.
70-SW-36, ARM2#: This pillar is quite long and thin (20m x 100m) and is situated at a depth of 2100m below surface. Relatively high seismicity was expected here from the nearby faults.

Figure 2.1.2: The 70-SW-36 pillar at ARM 2-shaft. The pillar is outlined, and the arrow indicates the direction of mining.

72-S-41, ARM2#: A rather large pillar (80m x 100m) at a depth of 2140m below surface. This pillar is not strictly speaking an isolated piece of ground since it lies adjacent to a large fault, around which no mining has taken place.

Figure 2.1.3: The 72-S-41 pillar at ARM 2-shaft. The area to be mined is outlined, and the arrows indicate the mining direction.
**72-NE-54a, ARM2#:** A deep (2140m) pillar that is relatively large at 40m x 90m. A small fault underneath was not expected to cause problems.

![Figure 2.1.4: The 72-NE-54A pillar at ARM 2-shaft. The pillar is outlined, and the mining direction indicated by the arrow.](image)

**74-NE-54A, ARM2#:** A very small pillar, 15m x 50m, at a depth of 2140m below surface. The depth and presence of complex geology caused expectations of appreciable seismicity.

![Figure 2.1.5: The 74-NE-54A pillar of ARM 2 shaft (outlined with the mining direction indicated by the arrow).](image)
The 6\textsuperscript{th} tap-test site was at the Turfontein shaft of Rustenburg Platinum Mines. The pillar was of approximate dimension 50m x 15m. The tap-test was conducted during April 2003.

Figure 2.1.6: A small pillar at Turfontein shaft that was used as a Tap-Test site. The temporary geophone was temporarily installed on the other side of the pillar, at the site denoted by the black dot. This panel (and others in the vicinity) was blasted and seismic data recorded for 2 hours afterwards.

The 7\textsuperscript{th} tap-test site was the 104/109 46-48 pillar at Mponeng mine. This pillar was substantially larger than the others, approximately 600m x 70m. Tap-test data was recorded here in February 2003.

Figure 2.1.7: The 104/109 46-48 pillar at Mponeng mine. The pillar width was approximately 70m, and the mining direction is indicated by the arrow. The mined-out areas are shaded grey. The position of the closest geophone to the pillar is indicated by the black dot.
**Section 336, TauTona:** This area was studied as part of GAP601a, and the data recorded there during 2001 was analysed further in this project. This mining area was highly stressed due to the depth (3100m below surface) and the pillar nature.

![Diagram of Section 336 of TauTona mine](image)

Figure 2.1.8: Section 336 of TauTona mine. The grid spacing is 80m.

At each of the ARM2#, Turfontein and TauTona sites, a high-frequency (30Hz) omni-directional uni-axial geophone was affixed to the sidewall with quick-dry cement close to where a production blast was to take place. The quick-dry cement takes approximately 5 minutes to dry, and was packed around the geophone to form something similar to a swallow’s nest. After the measurements, the geophone can be easily retrieved using a rock-hammer to chip away the cement.

Signals from the temporary uni-axial geophone were logged using a StandAlone QS (ISSI, 2001) unit. The SAQS unit uses approximately 3W of power, and thus a portable 12V battery provides sufficient power for the test.

Many thousands of micro-fractures can be generated in the first hours after production blasting, in response to the sudden removal of ore from the face. An example of a continuous seismogram recorded in this project is given in Figure 2.1.9.
Figure 2.1.9: Continuous seismic data recorded at 72 S 41 of ARM 2# on the 3 Oct 2002. The times of panel blasting ("B"), as well as distant blasting ("DB") are indicated in blue.

To quantify this data from a single sensor, the ISS Crack Counter (Seismic Triggered Parameters) mode of operation is useful. This technology was developed as part of the jointly-funded AngloGold/SIMRAC project GAP601a, and allows real-time extraction of quantitative acoustic emission information from the seismic signal. This data is stored on the SAQS hard disk for later retrieval and analysis. After download, plotting the cumulative number of STP seismic events as a function of time is then a simple matter.

While the ARM, Turfontein and TauTona tap-tests used a portable SAQS data logger for temporary recording, at Mponeng the uni-axial geophone installed 50m from the closest working area was connected to a QS unit, and STP information was sent to the seismic computer on surface in real-time. In this case, STP information was being recorded while the sensor was simultaneously being used for conventional mine seismic monitoring.

For all the tap-tests except Mponeng, the sensor was installed some 10-15m from the stope face. At Mponeng, the sensor was approximately 50m from the working face. The panel lengths at ARM2# were 15-20m, at TauTona it was 25-30m and at Mponeng 25-30m.
2.2 Tap-test results

It became apparent during these tests that STP information, while powerful, can be difficult to interpret if details of blasting are not well known. The STP algorithm triggers whenever a statistically significant seismic signal is recorded, and thus records the signatures of nearby and remote blasting. These signals are not easily filtered out of the dataset. This is a well-known problem in mine and tectonic seismology, and is usually dealt with by examining the P- to S-wave energy ratio, as well as the location and timing of the blasting. With STP data from a single uni-axial sensor, it is not possible to get source locations of seismic events, and so this cannot be used to filter out blasting signals. Furthermore, the bulk of the seismic signatures of micro-fracturing after blasting emanate from the rockmass immediately surrounding the excavation, and so the P- and S-wave arrivals are not well separated. Thus the only reliable way of separating the blasting signals from the seismic rockmass response is using knowledge of the blasting times. An example of this problem is given in Figure 2.2.1. Here, the closest panel being blasted generated 20mm/s at the temporary geophone. However, blasting vibrations from more distant panels obscured the small seismic signals of the rockmass response afterwards.

![Figure 2.2.1: A 2-hour continuous seismogram recorded at Turfontein shaft. This tap-test was inconclusive as other panel blasting obscured the small seismic signals containing information of the rockmass response to blasting.](image)

Of course, if the pillar being tested is far removed from any other blasting, then this problem does not exist. Alternatively, if the exact times of each hole being
blasted were known, for example with electronic detonators, then the STP data could be better interpreted.

Figure 2.2.2: Two hours of continuous seismic data from a 30Hz geophone 10m from one of the ARM2# stopes. The blast starts at 16h23, but lasts for about 12 minutes. A clear increase in micro-seismic activity is visible after the blasts.

Figure 2.2.3: Close-up of the figure 2.2.2. The production blast is probably the dense set of seismic events just before 16h25, while the individual shots are probably a nearby development end blasting.
Taking the most reliable STP data-sets for the ARM2# tests (site 70-SW-36 has to be discarded because the nearby blasting corrupted the STP data), Figure 2.2.4 is obtained. Typically some 100-700 STP events are recorded in the first hour after blasting.

![Graph](image1.png)

**Figure 2.2.4:** Cumulative number of STP seismic events as a function of time after production blasting at the ARM2# tests sites.

At section 336 of TauTona mine, STP data was recorded from a 30Hz geophone installed approximately 10m from the working stope in May 2001. Graphs of STP activity as a function of time are reproduced in Figure 2.2.5 (from GAP601a).

![Graph](image2.png)

**Figure 2.2.5:** Decays in STP activity after the same amount of blasting on 3 separate days in May 2001, recorded at section 336 of TauTona mine.
Using the parameters of the power-law fit, the cumulative number of STP events vs. time can be estimated. This is shown in Figure 2.2.6. What is immediately striking is that some 3000-15000 STP events are recorded in the 1st hour after blasting, an order of magnitude more than that recorded at ARM2# pillars.

![Figure 2.2.6: Estimates of the cumulative number of STP events as a function of the time after blasting, obtained from the power-law fit parameters in Figure 2.2.5.](image)

The tap-test data at Mponeng was analysed for the 2-5 February 2004. At this time, only 1 panel (109 46 E9) was being blasted within 300m of the STP geophone. A face length of 25m was blasted on each day, except for the 4th February when 30m was blasted. In this semi-controlled situation, finding the exact blasting time in the seismic data is not difficult (Figure 2.2.7).

Using this information, the cumulative number of STP events as a function of time after blasting may be plotted for these days – Figure 2.2.8. Some 5000 – 10000 STP events are recorded in the 1st hour after panel blasting.
Figure 2.2.7: Cumulative number of STP events and Peak Ground Motions of those events as a function of time. The blast at about time 20,000 sec is obvious.

Figure 2.2.8: Cumulative number of STP events as a function of time after blasting for the 2-5 February 2004, recorded at the Mponeng 109 46 E9 panel.
To test the postulated correlation between recorded STP events after blasting and the general levels of seismic activity in the mining areas, plots of seismic event local magnitude vs time were obtained, using data from the mine-wide seismic networks in 3-month periods roughly around the time of the tap-tests. This information is presented for the 5 ARM2# areas, as well as for the TauTona and Mponeng tap-test sites, in Figures 2.2.9-2.2.14.

Figure 2.2.9: Spikes of seismic event local magnitudes vs time for the time period around the tap-test (3 Oct 2002) at the 72-S-41 site at ARM 2#.
Figure 2.2.10: Spikes of seismic event local magnitudes vs time for the time period around the tap-test (20 Sept 2002) at the 72-NE-54a site at ARM 2#.

Figure 2.2.11: Spikes of seismic event local magnitudes vs time for the time period around the tap-test (19 Sept & 12 Nov 2002) at the 72-NE-56a site at ARM 2#. 
Figure 2.2.12: Spikes of seismic event local magnitudes vs time for the time period around the tap-test (11 Sept 2002) at the 50-40-RAW site at ARM 2#.

Figure 2.2.13: Spikes of seismic event local magnitudes vs time for a 6-month time period around the tap-tests at section 336 of TauTona mine. The Tap-Tests were carried out here during 3-7 May 2001.
Figure 2.2.14: Spikes of seismic event local magnitudes vs time for the 3-month time period around the tap-tests (2-5 Feb 2004) at the Mponeng tap-test site.

Clearly the most seismically stable area is the pillar close to the ARM 2-shaft: 50/40RAW, with only small (magnitude < -0.5) events recorded in the 3 months centred around the tap-tests. The most active region at ARM2# was 72S41, with two magnitude 2.5+ in the month prior to the tests and a magnitude 1.8 during the test period. It is also clear that the Mponeng and TauTona tap-test areas experience higher levels of moderate seismicity. Mponeng’s 104/109 46-48 section experienced 24 seismic events of magnitude = 1.0 in the 3-month period centred around the tap-tests, while TauTona’s section 336 saw 19 similar events. Similar polygon volumes have been used to spatially filter seismic events around the tap-test sites. Table 2.2.1 and Figure 2.2.15 summarises this information along with the STP data.

<table>
<thead>
<tr>
<th>Tap-test site</th>
<th>Number of STP events within 1 hour of blasting</th>
<th>Number of events with m=1.0 in 3 months</th>
</tr>
</thead>
<tbody>
<tr>
<td>50 40 RAW, ARM2#</td>
<td>80</td>
<td>0</td>
</tr>
<tr>
<td>72 NE 56a, ARM2#</td>
<td>170-190</td>
<td>1</td>
</tr>
<tr>
<td>72 NE 54a, ARM2#</td>
<td>60</td>
<td>2</td>
</tr>
<tr>
<td>72 S 41, ARM2#</td>
<td>750</td>
<td>5</td>
</tr>
<tr>
<td>Section 336, TauTona</td>
<td>3,000-15,000</td>
<td>19</td>
</tr>
<tr>
<td>104/109 46-8, Mponeng</td>
<td>5,500-10,500</td>
<td>24</td>
</tr>
</tbody>
</table>

Table 2.2.1: Summary of the tap-test data and seismic event data for each of the 6 valid sites.
Any sensible measure for seismic hazard could be used for correlation against the STP Tap-Test responses. In this case, magnitude 1 was chosen as the threshold for damaging seismic events, and the number of such events was used for the correlation.

It is clear that there is a good correlation (with a correlation coefficient of 0.986) between the levels of STP activity in the first hour following a production blast and the levels of seismicity recorded in the area by mine-wide seismic network over a 3-month period around the time of the tests.

The advantage of this technique is that it does not need information about local geology and rock mass conditions, since the stable of the rockmass as a whole, including any unstable faults, etc. would be evaluated by a Tap-Test. Whether the cracking is emanating from one particularly large structure or many smaller structures cannot be determined with this approach, but whether this is required for general assessments of expected stability is a debatable point.
2.3 Tap-Test Conclusions

There appears to be a good (0.986) correlation between the tap-test results and the levels of seismicity recorded in the vicinity over a 3-month period roughly around the time of the Tap-Tests. The STP data is relatively easy to collect, since the equipment required – an omni-directional, uni-axial seismic sensor and data logger – is not onerous, and the tap-test takes 3 days to install, record, retrieve and analyse data.

The technique would be useful for pillars of a mine where adequate seismic network coverage is not possible, but where estimates of the seismic hazard associated with mining of such pillars is required.

High-sensitivity, high-frequency geophones or low-noise accelerometers are the recommended seismic sensors.

It was expected that tap-tests would be more accurate for smaller pillars. This is because the localization of resulting micro-seismic response to blasting is limited and the blast is a relatively large tap. However, pillars of up to 40,000 – 60,000 m$^2$ (the Mponeng and TauTona sites) still exhibit a good correlation, and so this appears to be a practical tool.

Practical guidelines for routine tap-tests:

- Try to schedule the tap-test for a time when only the area of interest is being blasted, or at least know the times of blasting of panels within 300m.
- If the area of interest could be blasted 1 hour before any other areas, this is good enough for a clear tap-test.
- Try to install the sensor within 20m of the working face. 10m is ideal.
- Quick-dry cement dries within about 5 minutes and is easy to use for temporary sensor installation. A rock-hammer can usually retrieve the sensor with no damage.
- A 24 Amp-Hr 12V battery (weight approximately 7kg, dimension approximately 150mm x 150mm x 150mm) is sufficient to power the SAQS unit for 2 days without deep discharging.
- High sampling rates, at least 6 kHz, are best for the uni-axial signal.
- Suggested STP parameters:
  - **Threshold** should be about 6 times the average LTA of the sensor (try 150 if not sure)
  - **Max number of noise points** = 10
  - **Min event length** = 50
3. Panel Sequencing

The objective of this part of the project was to test whether it is possible to affect the short-term character of the seismic rockmass response to mining by preferentially scheduling the blasting of non-contiguous (non-neighbouring) panels. If this is true, then small changes in production planning could decrease the short-term seismic hazard without affecting the amount of ore extracted.

3.1 Experimental Site

The 104/109 46-49 mining area of Mponeng mine was selected as the experimental site for this part of the project. Sequential grid mining is practiced at this site. The depth is approximately 3100m below surface.

![Figure 3.1.1: Plan views of Mponeng mine (left) with an enlargement of the 104/109 46-49 mining area (right).](image)

The high resolution extension to the Mponeng mine-wide seismic network consisted of 7 new geophone sites, augmenting the existing geophone in the area. Installation of this equipment took longer than expected, and the array was commissioned in August 2003. This delay was mainly attributed to the slow rate of drilling the geophone holes.

Prior to installation of the additional geophones, the mine-wide seismic system consistently recorded all events with log(Potency) above -1.4 (Hanks-Kanamori magnitude 0.0). About 4 events per day were being recorded.

After installation of the 7 additional tri-axial geophones, the 400m x 400m x 200m volume of interest records some 100 events per day, with a minimum consistent log(Potency) of -4.0 (Hanks-Kanamori magnitude -1.8).

‡ The potency of a seismic event is the scalar seismic moment divided by rigidity: \( P \ ? \frac{M}{\Omega} \). See Appendix 1 for details.
Figure 3.1.2: Plan view of the high resolution micro-network at Mponeng, with the installed tri-axial geophones indicated as circled triangles.

Figure 3.1.3: Seismic events recorded in the 2 pillars of the 104/109 46-49 area from September 2003 to January 2004. Only events locating within the 2 pillar-polygons have been shown.
Figure 3.1.4: Cumulative frequency - log(Potency) plot for seismic data recorded at 104/109 46-49 by the mine-wide seismic network prior to installation of the high resolution micro-network. All events with log(Potency) above -1.4 are being consistently recorded.

![Cumulative Frequency - Log(Potency) Plot](image1)

Figure 3.1.5: Cumulative frequency-log(Potency) plot for seismic data recorded at 104/109 46-49 by the high resolution micro-network. All seismic events with log(Potency) above -4.0 are now being consistently recorded.

![Cumulative Frequency - Log(Potency) Plot](image2)
### 3.2 Results

Between the 1\textsuperscript{st} September 2003 and 12\textsuperscript{th} January 2004, production in the 104/109 48-49 pillar at Mponeng mine was concentrated to a few panels. A plan view is given below (active panels in bold) in Figure 3.2.1.

![Figure 3.2.1: Mining activities (active panels in bold) at the 104/109 48-49 pillar of Mponeng mine during September 2003 – January 2004 (circled).](image)

During this period 1595 seismic events were recorded in the 104/109 48-49 pillar by the high resolution micro-network at Mponeng. A spatial plot (plan view) is given in Figure 3.2.2, showing that this activity is centred around the active panels, as we would expect.
Figure 3.2.2: Plot of micro-seismic events recorded between 1st September 2003 and 12th January 2004 at Mponeng’s 104/109 48-49 mining area. Only seismic events located in a 100m x 100m x 250m volume surrounding the working areas are shown.

The hypothesis is that distributed blasting is better for the seismic response than more concentrated blasting. The underlying reason that this is to be expected is that the stress fields of the rock mass become less correlated by more scattered blasting. Since large scale spatial correlations are necessary for large seismic events, it stands to reason that an uncorrelated system would lead to a more controlled seismic response, with mostly smaller seismic events.

Previous SIMRAC work (GAP303) has explored the link between seismic stiffness and the distribution of the seismic event sizes (b-value of the Gutenberg-Richter plot), and concluded that the higher the seismic stiffness, the higher the b-value and the lower the seismic hazard.

During the period of study, 1st September 2003 to 12th January 2004, there were 89 production blasts in the area of interest. Of these blasts, 75 consisted of one or more contiguous panels being blasted. The remaining 14 consisted of two spatially separated panels being blasted. In total, 2843 m$^2$ of rock was broken.
We would like to examine the seismic rock mass response to 1-segment blasting compared to 2-segment blasting. To quantify this, we take seismic events recorded in the 18 hours following blasting and compute seismic stiffness using the formula (Mendecki et al, 1997): 

\[ K_s = \frac{E_i}{M_i^2} \]

A graph presenting the results is shown below:

Figure 3.2.3: Seismic stiffness computed from seismic events recorded within 18 hours of blasting. The blue stars indicate seismic stiffness resulting from 1-segment blasting, while the red dots indicate seismic stiffness resulting from 2-segment blasting. The lines are best orthogonal fits to log(K_s) vs production. This graph indicates that, on average, the 2-segment blasts degrade the system stiffness less than the 1-segment blasts for the same production, and therefore result in less seismic hazard.
The graph has two interesting features. While there is a large degree of scatter, on average for the same type of blasting (1-segment or 2-segment) the larger the volume of rock blasted, the lower the resulting system stiffness and thus higher the seismic hazard. This is not surprising. However, what is interesting is that for the same amount of rock blasted, the 2-segment blasts degrade the system stiffness less than the 1-segment blasts do. Thus the 2-segment blasts result in lower seismic hazard than the 1-segment blasts do, for the same amount of ore extracted.

Of course, the large degree of scatter casts some doubt on this analysis. To compute the significance of difference in the intercepts of the two straight line fits (the gradients are very nearly the same), we resort to hypothesis testing. The null hypothesis here is that these two data sets result from the same underlying population. Typically this null hypothesis is then rejected if the probability of obtaining the observed is less than 0.05. After carrying this standard analysis out on our data (1-segment data: intercept of 6.30648, standard error of 0.22569, 54 samples; 2-segment data: intercept of 6.5626, standard error of 0.3949, 12 samples – see Figure 3.2.4), we find that there is a probability of observing this data of 0.18, and unfortunately the null hypothesis cannot be rejected.

Figure 3.2.4: Graphical representation of the obtained intercepts and standard errors for the 1-segment and 2-segment data. Unfortunately the null hypothesis that both data sets belong to the same population cannot be reliably ruled out.
Another analysis, on a larger time scale, is possible. The production blasts in this area of Mponeng are presented in figure 3.2.5.

Figure 3.2.5: Amounts and types of blasting in the 104/109 48-49 pillar at Mponeng. During time “A” the blasting was mostly of type 2-segments, while during time “B” the blasting was only of type 1-segment.

The times indicated in Figure 3.2.5 by “A” and “B” were time intervals of equal production: 450 m$^3$. These 2 time intervals are analysed in the table below.

<table>
<thead>
<tr>
<th></th>
<th>Duration</th>
<th>No. of 1-segment blasts</th>
<th>No. of 2-segment blasts</th>
<th>Total volume blasted</th>
<th>No. of seismic events</th>
<th>Seismic Stiffness (GPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Time A</td>
<td>13 days</td>
<td>3</td>
<td>7</td>
<td>450 m$^3$</td>
<td>275</td>
<td>48</td>
</tr>
<tr>
<td>Time B</td>
<td>17 days</td>
<td>13</td>
<td>0</td>
<td>450 m$^3$</td>
<td>241</td>
<td>17</td>
</tr>
</tbody>
</table>

Table 3.2.1: Comparison between periods “A” and “B” of Figure 3.2.4.

It is significant that, although the same amount of production has been carried out in a faster time, mining during time “A” has degraded the stiffness less, and therefore results in lower seismic hazard.

Data from these 2 periods have been used to generate the frequency-magnitude graphs of Figures 3.2.6 & 3.2.7.
Figure 3.2.6: Frequency-magnitude graph of seismic events recorded during time “A”.
The Moment-magnitude has been used here, since this is a more reliable measure of
magnitude with low-frequency geophones. A b-value of 0.62 is calculated with the bin-
corrected maximal-likelihood method.

Figure 3.2.7: Frequency-magnitude graph of seismic events recorded during time “B”.
The Moment-magnitude has been used here, since this is a more reliable measure of
magnitude with low-frequency geophones. A b-value of 0.59 is calculated with the bin-
corrected maximal-likelihood method.

The frequency-magnitude graphs show that the b-value from seismic events
recorded during time “A”, when the blasting was mainly in 2 segments, is higher
than the b-value from events recorded during time “B”, when blasting was
contiguous. The maximum magnitude is lower during time “A” than time “B”.
While the difference in b-values is not very large, the difference in seismic
stiffness (Table 3.2.1) is striking.
It would appear that for this pillar, the seismic stiffness of the rockmass is degraded less by blasting in multiple places compared with blasting the same volume of rock in one concentrated place.

A similar analysis can be conducted for the western pillar in Figure 3.1.3. Here the situation is more complicated, as many panels were being simultaneously worked. The blasting must be divided into 4 categories: 1-segment, 2-segment, 3-segment and 4-segment blasting. Seismic stiffness is estimated from seismic data recorded within 18 hours after blasting, and this analysis is presented in Figure 3.2.8.

Figure 3.2.8: Seismic stiffness estimated after different blasting configurations for the western pillar of Figure 3.1.3. Linear regressions have been applied (straight lines).

For the western pillar there is no consistent statistically significant relationship between seismic stiffness (and therefore seismic hazard) and blasting configuration. However, the slightly decreasing trend of seismic stiffness with more production is perhaps similar to that observed on the eastern pillar.
3.3 Conclusion

During GAP601a it was observed at section 336 of TauTona that blasting contiguous panels could degrade the seismic stiffness more than blasting spatially separated panels. This tentative observation was consistent with a theoretical physics complex-systems principle, in which system correlations allow larger events to occur. Blasting one large volume of rock perturbs the stress over a large volume, and thus promotes larger seismic events. Blasting panels separated in space (and ideally time) would create much smaller distributed disturbances, and therefore promote many smaller seismic events.

Two pillars at Mponeng were studied. The east pillar had 4 working panels, and thus only 1-segment or 2-segment blasting was possible. In this simple case, several indications of lower seismic stiffness (and thus increased seismic hazard) with contiguous blasting were observed, albeit with a large degree of scatter in the data. On the other hand, the west pillar was far more complicated, with 8 working panels and a wide variety of blasting configurations. In this complex case, no consistent relationship between seismic stiffness and blasting configuration was observed. The simpler pillar showed some indications of the expected behaviour of degraded seismic stiffness with increased production blasting, while this is debatable for the more complicated pillar.

This mixed result should be regarded as some weak support for the concept of seismic response control through daily panel scheduling. Certainly, if this is not difficult to implement, blasting neighbouring panels should be avoided wherever possible.
4. Assessment of Integration for Controlled Mining

This section describes a number of cases where integration has been implemented over the last couple of years. The cases are presented in chronological order, so that the path of development can be followed.

Integration was implemented per contract first at Hartebeestfontein Mine, #6 Shaft, integrating shear slip along a known structure in the shaft pillar. This required a hand-picked selection of events associated with the structure, which had its problems for routine application. A further application of shear slip integration was done at Great Noligwa Mine, BE54 area. Shear displacement of a single event along a known geological structure was done in order to assess the situation after the large event. The probability for more events was investigated, as well as the effect of the structure slipping on rock mass stress in surrounding areas.

The first effort of rock mass integration was through Differential Maps, and was implemented at West Driefontein #5 West Shaft. This provided a spatial picture of where events may be expected in the short term future. Together with other modelling techniques, a seismic hazard assessment of the full shaft pillar extraction was done on a quarterly base.

Then 3D rock mass integration was tested at TauTona Section 336 and Mponeng 109 level to the west. This showed great potential to achieve a calibrated model with realistic rock mass stresses, which can be used for successful assessment of rock mass stability as well as structure stability.

The above work lead to the formulation of a proposed methodology for integration, described in some detail in Section 4.6. The procedures and methodologies are currently in place and ready for implementation. It is believed that integration can provide improved results over conventional elastic modelling, and hence better seismic hazard assessment in the short- and medium-term.

4.1 Structure Stability Assessment – Harties 6 Shaft, Diagonal Dyke

4.1.1 Shear slip integration

The Diagonal Dyke posed a seismic hazard during the mining of the shaft pillar. This structure yielded large events historically, and was bound to yield more significant seismicity with mining. A back-analysis was performed, suggesting that it was required to model the two contact surfaces of the dyke distinctly. Back-analysis results appeared encouraging in terms of the level and position of modelled versus observed seismic moment.
The implementation of routine integration aimed at quantifying the seismic hazard posed by the structure on a fortnightly base. For this, the updated, fortnightly mining faces were required. Furthermore, it was required to attribute shear slip events to the dyke contacts. This was done by manually inspecting the events, and deciding whether the event was likely to be associated with the structure. The Seismic Displacement [Hofmann, 2001], as obtained from an assumed source model, was integrated as shear slip along planar structure boundary elements, hence modifying the prevailing shear stress on the structure contacts.

Results analysis went beyond analysing shear stress or Excess Shear Stress (ESS) [Ryder, 1988] on the structure, but attempted to quantify structure stability by modelling the shear stiffness and potential energy release [Lachenicht, 2001] if the structure was to slip at that time. This was done by performing a so-called Stability Test, whereby the structure is allowed to slip in an artificial model step, and considering the change in shear stress and shear deformation. The gradient of this load-versus-deformation graph per structure boundary element defines shear stiffness, while the area under the curve defines potential energy release. Shear stiffness and potential energy release of the full structure are obtained by summing and averaging over all structure boundary elements.

Structure Stability Assessment results, provided on a routine basis as per the contracts with the mines involved, consisted of the following:

- **General information**
  - Past period's recorded seismicity, the mining periods incorporated into this model, the seismic events that had been integrated, some pictures of the mining geometry and steps, etc.

- **Excess Shear Stress picture**
  - This picture is composed of ESS contours on the structure, pointing out spatially where large events may occur.

- **Table of quantified modelling results**
  - Stability parameters extracted from the modelling results
    - **Stiffness and Energy graphs**
      - Structure Stiffness per mining step (Fig. 4.1.1.1a) – decreasing structure shear stiffness indicates a larger potential magnitude event.
      - Energy divided by stiffness per mining step (Fig. 4.1.1.1b) – high potential energy release and low stiffness indicates increased seismic hazard
      - Energy Versus Stiffness (Fig. 4.1.1.1c) – Recorded seismic release for the model steps are shown on the plot in terms of Moment Magnitude, making it a convenient plot to identify areas on the graph reminiscent of high seismic hazard.
The figures referred to above are examples from a typical report.

4.1.2 Shortcomings of the method

The above routine assessments looked very promising. However, a large event (over Mag 3.0) did occur just outside of the modelling area (west of the shaft pillar), not predicted by the modelling. This suggested that something was still lacking in the methodology.

The main problems were the following:

It was not easy to attribute seismic events clearly to shear slip mechanism on one of the two structure contacts, with this process probably fairly inaccurate.

Location accuracy was probably not good enough to do the above, and to extract sufficiently accurate seismic displacement along the structure contact.

Excess Shear Stress (from the numerical model) and Seismic Displacement (from recorded seismic events) did in general not coincide satisfactory, due to the inaccuracies noted above. The model was therefore in some cases forced to slip in areas of relative low ESS, which is unlikely in reality. Due to the progressive stepwise integration, this error probably accumulated, yielding increasingly unrealistic results.

In general the integration of shear slip along structures seemed to be too sensitive to inaccuracies, especially in the inferred seismic displacement, and therefore this approach is not well suited for routine analysis. Shear slip integration would be more suited when detailed information of event source mechanism is available.

The stability test, whereby a structure contact is allowed to slip artificially in the model, is still perceived to be a good indicator of structure shear stiffness and worst case potential seismic event. In order to quantify the probability to experience a large event, ESS and increase in ESS on structures should be used, assuming typically a Mohr-Coulomb strength criterion.
Fig. 4.1.1.1a: Graph of modelled structure shear stiffness per mining step. A decrease in stiffness indicates a larger potential magnitude event.

Fig. 4.1.1.1b: Graph of energy divided by stiffness per mining step. High potential energy release and low stiffness indicates increased seismic hazard.

Fig. 4.1.1.1c: Energy versus Stiffness, with historical seismic releases indicated. This allows for the identification of zones on the graph of increased seismic hazard.
4.2 Differential Maps – West Driefontein 5 West Shaft

4.2.1 Seismic Hazard spatially

The basic idea with Differential Maps is to quantify the difference between modelled and observed parameters, and consider the areas where observed ‘lags’ behind modelled as areas of increased seismic hazard. This can be contoured spatially and is referred to as a Differential Map (or Diffmap).

In the current method, the plane of maximum shear stress ahead of the face is taken as the plane of typical shear fracture events, with the Shear Stress on these planes, in theory, being $\frac{1}{2}(\frac{1}{\sigma} + \frac{1}{\sigma_3})$. It is assumed that this is the driving mechanism for the larger seismic events occurring in the rock mass ahead of the face. The modelled parameter is then compared with the seismicity parameter, Seismic Displacement, which is calculated from the recorded seismic data for a collection of events. Since seismic events are taken for a specific period (i.e. the mining step period), the differential modelled shear stress is used, in other words, the increase in shear stress from one model step to the next.

The procedure for calculating Differential Maps for tabular ore bodies is summarised as follows:

1. Calculate shear stress values on grids parallel to the grid plane in the area of active mining.
2. Project the maximum values of shear stress $T_{max}$ for all grids onto the reef plane. Use the direction orthogonal to the reef for the projection. This direction is associated with the direction of maximum error in the hypocentre location.
3. Calculate seismic displacement for the reef plane. In principle a similar projection is applied.

The differential map parameter is calculated as:

$$dmp = T_{max_{diff}} - u_{n}$$

where $T_{max_{diff}}$ is the difference (between mining steps) of the modelled shear stress normalised to unity, and $u_{n}$ is the normalised seismic displacement

Differential map values vary from $-1$ to $+1$, with $+1$ indicating a high hazard, and $-1$ a low hazard.
4.2.2 Differential maps on shaft pillar scale – West Driefontein 5 West Shaft

Integration through Differential Maps was implemented as part of a routine seismic hazard assessment program at West Driefontein 5 West Shaft. Whilst following an extraction sequence, a complete seismic hazard assessment was done on a quarterly basis for the shaft pillar as well as a pillar to the north. Diffmaps provided a spatial picture of seismic hazard for the following quarter. The expected level of rock mass seismic release was modelled through Volumetric Energy Release (VER – see e.g. [Lachenicht, 2001]). In addition to this, structures intersecting the pillars were also analysed in terms of the probability for large seismic events, by considering Excess Shear Stress.

Monthly mining steps were modelled, quantifying shear stress increase (as described in Section 4.3) due to the month’s mining. This, together with the total seismic displacement calculated for the month’s events in the whole shaft pillar, was used to calculate the Diffmap.

Fig. 4.2.2.1 is a plan view showing the area that was covered. Calculating Diffmaps is less computer intensive than actual integration in the boundary element package, so that such a large area, together with the regional mining of two mines, was easily modelled.

Two examples of Diffmaps are shown in Figs. 4.2.2.2.

In Fig. 4.2.2.2a the contour range has been chosen to indicate hazardous areas, viz. from 0.25 to 1. In the worst case, the normalised stress increase will be 1, while the normalised seismic displacement will be 0, hence giving a Diffmap value of 1. This will be contoured in red. The minimum contour was taken as 0.25, with this positive value reflecting a slight lag of actual deformation behind modelled stress.

In some cases large events not associated with intact rock mass failure occurred, but rather by shear failure along a geological discontinuity. In the Diffmap methodology all events are assumed to be shaped spherically according to the event Apparent Volume. In such cases, large deformation is inferred over the spherical volume, which is incorrect for shear slip events. This over-exaggerates rock mass de-stressing. Such large shear events should be taken into account when interpreting Differential Maps, or should be excluded using appropriate event filtering methods.
An advantage of a Diffmap (mitigating the problem mentioned above), is that the Diffmaps are not cumulative between consecutive steps. At each model step, the recent increase in stress is considered, together with the seismicity corresponding to that period. This means that the effect of a single large event will not be ‘dragged along’ for the mining steps following. At each step, only the latest stress increase and seismicity is considered as an indication of the seismic hazard of the immediate future.

The Diffmap shown in Fig. 4.2.2.2b was calculated in the same way, but the full range of values is shown. Diffmap values of zero are coloured green, and generally means zero stress increase as obtained from the numerical model, and zero seismic displacement as calculated from the recorded seismic events. Red means maximum stress increase coinciding with minimum seismic deformation, and hence indicates areas of increased seismic hazard.

In the plots the Diffmap is plotted together with the larger events over the three months following. This is a crude check of the effectiveness of the method and discrepancies can be expected. It is not as simple as assuming a Diffmap of the current step predicts the seismic events of the next step. In some cases there is correlation between Diffmap red areas and larger events following, while in other cases red areas are not followed by larger events. Events also occur in areas which are not red, which typically originate from shear slip along planes of weaknesses rather than high rock mass stress. However, the methodology should highlight areas of increased seismic hazard for certain mechanism events. High Diffmap areas overlapping with seismogenic geological structures should be taken as a definite seismic hazard.
Fig. 4.2.2.1: West Driefontein 5W Shaft - Plan view of the area, showing the pillar mining sequence (historical and planned) and geological structures.

Differential Map for the period 2000/12 to 2001/02.
Contours: Max shear stress increase minus displacement (both normalised).
Events: > Mag 1.0 for the period 2001/03 to 2001/05.

Fig. 4.2.2.2: Differential Maps with larger events for the period following. In (a) only the Diffmap range between 0.25 and 1 is plotted, with this indicating the areas of increased seismic hazard. In (b) the full range is plotted as smooth contours.

Differential Map for the period 2003/03 to 2003/06.
Contours: Max shear stress increase minus displacement (both normalised).
Events: > Mag 1.0 for the period 2003/07 to 2003/09.
4.3 Assessing effect of a single event – Great Noligwa Mine

4.3.1 Shear stress on the fault after an event

It was required to do a stability assessment of the Noupoort fault in the BE54 area of Great Noligwa Mine, after a Mag 3.6 event occurred. A boundary element model was set up incorporating the mining adjacent to the fault, and the fault component which was most likely to slip.

The modelling back-analysis suggested that the part of the fault, adjacent to pillar which was being mined against the fault, acted as an asperity due to cohesion. It was possible to calibrate the fault strength (cohesion and friction angle), causing the fault area to slip in the model at the time of the event. This slip was initiated due to the static cohesion being ‘broken’ by the increasing shear stress – the asperity failed.

The back-analysis firstly modelled the sudden ride on the structure when the asperity failed. At the particular mining step corresponding to the event time, the asperity yielded, causing a decrease in shear stress, and increase in ride. This is shown in Figs. 4.3.1.1 as pictures of shear stress and ride before and after. This analysis suggested that the event source mechanism was modelled correctly as shear slip on the structure, and hence that integration of the actual event should be feasible.

Seismic Displacement was calculated from the event source parameters, using a source model and assuming an elliptical source shape. Although this is an assumption, it correlates well with the induced modelled ride, and is probably the best guess one can make of such an event slipping in a down-dip direction. Fig. 4.3.1.2a shows the seismic displacement inferred for the event, and a picture of the shear stress after integration. The centre point of the elliptical source was taken as the point of modelled maximum shear stress. Fig. 4.3.1.2b shows the shear stress after integration. Shear stress reduced mainly, but also increases in some areas.

Integration in this case therefore allows one to assess the situation after such a large event occurred, whether the seismic hazard was ‘taken away’ by an event or whether some areas may still exist where more events may follow.
4.3.2 Impact of large slip events on excavations

A further application is to assess the impact of slip along a structure on the rock mass stress, e.g. how it affects development in the structure footwall. By considering rock mass stresses (normally sigma 1), it can be established if excavations like tunnels or stopes are under higher stress due to the event that occurred, and hence if support requirements need to be revised. Whether stress increase or decrease is determined by the position and direction of shear slip relative to the excavation.

Fig. 4.3.2.1 shows an example of the impact of integrating shear slip inferred for a Mag 2.3 event on footwall development. In this example the stress level increased statically due to the event.
Fig. 4.3.1.1: View onto a fault surface, before and after an asperity failed. (a) and (b) are pictures of shear stress before and after, showing the shear stress decrease in general. (c) and (d) are pictures of ride before and after.
**Fig. 4.3.1.2**: Pictures showing the effect of integration. (a) is the Seismic Displacement inferred for the large event that occurred, using a source model and assuming an elliptical source shape. (b) shows the shear stress after integration of the seismic displacement centred at the position indicated by shear stress before (Fig. 4.3.1.1a). Shear stress is mainly reduced, but also increases in some areas.

**Fig. 4.3.2.1**: Pictures of sigma 1 on a grid intersecting the tunnels in the footwall of a structure. The left- and right-hand picture is before and after integration respectively, showing how the stress increased due to integration of an event that occurred in the vicinity of the tunnels.
4.4 3D Integration – TauTona Section 336

A project was funded by DEEPMINE to set up a Controlled Mining integration model at TauTona Mine, Section 336. The route followed was to set up a 3D integration model, where recorded seismic strain was integrated in 3D, by specifying the strain of a small 3D cubical volume. These small volumes comprised a grid covering the larger volume of interest. Fortnightly mining steps were incorporated, and it was required to adjust the seismic event location to account for Z location error.

Regular Controlled Mining reports were issued, providing information on the following:

* Rock mass stress – Mainly sigma 1 was analysed, and it was attempted to quantify rock mass stress levels and identifying in space where stress was building up due to insufficient relaxation by seismicity. Fig. 4.4.1 is an example of an area that was building up stress, which was subsequently removed by a large seismic event.

* Structure stability – Seismic displacement was not integrated as shear slip along the structures (due to the difficulties listed in Section 4.1.2), but instead structures were subjected to the stability test explained in Section 4.1.1. The idea was that the 3D rock mass stress, modified by 3D integration, would provide a more accurate stress regime so that the structure stability test yields improved results.

4.4.1 Fortnightly Controlled Mining reports

The model was updated fortnightly. Mining faces was fixed at measuring day each month, but the face in the middle of the month was estimated, due to the lack of that information.

An update report could be issued within 2 days, and consisted of the following:

* Picture of modelled Sigma 1 - stressed areas are shown as high (red) contours, as obtained after the previous mine step and integration of corresponding seismicity. An example is shown in Fig. 4.4.1.1.

* Quantitative rock mass stress analysis – For the analysis polygons indicated in Fig. 4.4.1.1, stress level was quantified from a Cumulative Frequency Distribution of sigma 1 values. The area under the curve was calculated and referred to as Sig1-CFD_area. The correlation of this with recorded seismic release (seismic moment) was very good in some cases, with an example shown in Fig. 4.4.1.2. It was also attempted to calibrate Sig1-CFD_area against recorded seismic
moment, and in this way an anticipated Mmax for the analysis polygon could be extracted from the modelling results.

Structure Stability Assessment – Potential Energy Release and Shear Stiffness was calculated for the structures incorporated into the integration model. An increase in Energy and decrease in Stiffness was used to indicate potential structure instability, with an example shown in Fig. 4.4.1.3. Although these results looked promising, a large event did occur on a structure that was not anticipated. This necessitated further investigation, and is mentioned below.

4.4.2 Excess Shear Stress analysis on structures

A Mag 3.2 event occurred on a structure within the TauTona Section 336 integration area, which was not readily anticipated. An increased seismic hazard (in particular the structure shear stiffness) was evident around two months earlier, but not shortly before the event occurred.

The procedure was revised, and modelled Excess Shear Stress (ESS) was used as an indicator of the probability to trigger a large seismic event. At this site, ESS may be more suited as a short term predictor, while structure stiffness is more relevant to the medium term. The integration model was re-analysed, and structure ESS was plotted for the model steps. In addition to ESS, increase in ESS (ESSdiff – differential) was considered. A large jump in ESS due to adjacent mining is likely to trigger a seismic event.

An important improvement introduced by integration, is that ESS increase caused by seismicity can also be analysed. It is possible that the occurrence of seismicity may cause a structure to be loaded, and subsequently yield a large seismic event. This seemed to be the case for the Mag 3.2 event, and this case study is described in Section 4.4.3 ([Van Aswegen, 2004]).
Fig. 4.4.1: Plan view pictures of sigma 1 on a grid in the foot. The left-hand picture indicates an area of stress build-up, which was subsequently removed by a large event, shown on the right-hand picture.

Fig. 4.4.1.1: Plan picture of modelled sigma 1 after integration, together with the seismic events. Polygons subjected to quantitative analysis are also indicated.
4.4.3 ESS analysis of a Mag 3.2 event

A series of monthly mining steps were modelled as the longwall approached the dyke. In each step, the ESS was calculated on the structure and then the seismic events associated with that step introduced in the model and the loading...
equilibrated. Details of these procedures are given in Hofmann et al., (2001) and Wiles et al. (2001).

The results for each step were that the seismic events ‘wiped out’ most of the ESS. During the last step prior to the mag. 3.2 event, however, the ESS on part of the structure actually increased after the integration step. This remarkable increase in ESS was exactly in the hypocentral area of the large event that followed.

The last three mining steps prior to the large tremor are illustrated in a series of snapshots of the appropriate part of the model (Fig. 4.4.3.1). For each mining step the ESS induced by the mining is shown, then the distribution of seismic events along the structure and then the new ESS after integration.

![Diagram of ESS and mining steps](image1)

**Figs. 4.4.3.1:** Numbered snapshots of part of a Map3Di® model, showing the geometry of the last 3 of a series of mining steps (light grey) of a longwall stope along the shallowly dipping Carbon Leader Reef at TauTona mine, approaching a steeply dipping fault (dark grey mesh). Image 1 shows contours of ESS after the 3rd last mining step. Image 2 shows seismic events (sphere volumes equal to \(V_a\)). Seismic displacements derived from these events were integrated to yield a lowering of ESS (image3).
ESS after 3rd last integration step

ESS after 2nd last mining step

Seismic events for 2nd last integration step
The next mining step increased the ESS again (image4). Image 5 shows seismic events integrated after modelling the effect of the 2nd last mining step and image 6 shows the lowering of ESS following this integration.
Figs. 4.4.3.1: (continued from previous page) Image 7 shows the ESS after the last mining step, image 8 the events for the last integration step and image 9 the ESS after the last integration step. Here the ESS significantly increased after integration. A local magnitude 3.2 event followed this anomalous phenomenon (largest sphere in image 10).

Figs. 4.4.3.1: (continued) A local magnitude 3.2 event followed this anomalous phenomenon (largest sphere in image 10).
4.5 3D Integration – Mponeng 109-46 area

A similar procedure to TauTona Section 336 was tested at Mponeng Mine. The area was on 109 level to the west, around lines 44 to 49; this is where a high resolution seismic network is installed. The work is however in research stage and has not been reported formally.

A boundary element model was built, including the full mine in fair detail, as well as the backfilled areas. Structures have not been included to date.

A so-called Integration Volume was selected covering the area of interest. Within this volume, the mining geometry should be accurate, as well as seismic data (i.e. locations and source parameters). Monthly mining steps were incorporated with seismic data selected for the corresponding periods. Care was taken to take actual measuring day into account, so that seismic events correspond with mining face positions. The integration volume, seismic events, and 3D strain per 3D grid element, as inferred from seismic displacement from all events, are shown in Fig. 4.5.1.

Model steps consist of a monthly mining step followed by an integration step for that month’s seismic events, providing the normal boundary element method stresses for each model step. Fig. 4.5.2 shows an example of sigma 1 contoured on two perpendicular grids, before and after integration. The strain integrated is that shown in Fig. 4.5.1. This is a very useful picture to point out areas of increased seismic hazard as caused by a combination of progressive mining and occurrence of seismicity. The latter is a function of rock mass strength differences, in addition to residing stress.

4.5.1 Calibration through energy balance

The main purpose of the integration research at Mponeng Mine is to establish advanced procedures for calibrating the integration model.

It is attempted to use Seismic Displacement to estimate the observed Work Done. This is a different approach from before, where event source parameters, e.g. Seismic Moment, for a spatial selection of events was used to correlate with modelled. Test blocks are used and Seismic Displacement is calculated for a grid of cubical blocks, typically of dimension 5x5x5 m, within the test block. Displacement caused by all events at the particular grid points (centre points of 5
m blocks) is cumulated to give the total displacement associated with the seismic events over the specific time period (or mining step).

The next step is to estimate the energy released (or observed work done) associated with deformation of the blocks. For this a shear slip mechanism, driven by \( \frac{1}{2} \sigma_1^2 \), is assumed. It therefore requires principal stresses and these are obtained from the elastic numerical modelled, calculated for the centre of the 5m blocks. Only blocks within the solid un-mined rock mass are hence used, where the elastic modelled stresses should be reasonably accurate. The build-up of elastic stress is therefore considered, with this possibly culminating in a seismic event ahead of the face, also in the solid rock mass. Work Done is calculated from the change in shear stress (from modelled elastic stresses) and change in strain (from seismic displacement).
Fig. 4.5.1: The Integration Volume. The left-hand picture shows a typical month's seismic events, while the right-hand picture shows the strain to be integrated within the integration volume, as inferred from seismic displacement for all events.

Fig. 4.5.2: Contours of sigma 1 before and after integration. It can be seen where the abutment stress decreased, while stress increased for some areas ahead of the face.

Fig. 4.5.1.1: Correlation in time of modelled Potential Energy and observed Work Done.
The above observed Work Done is calibrated against modelled elastic Potential Energy within the same test blocks. This is extracted from the numerical model using a special feature of the boundary element package used, whereby a 3D element is allowed to deform artificially and calculating the energy release associated with that deformation (akin to the stability test for structures).

Calibration entails finding appropriate numerical model input parameters as well as seismic displacement parameters in order to give an energy balance between modelled and observed energy. Fig. 4.5.1.1 shows modelled and observed energy for a test block as an example. The picture where observed lags behind modelled makes sense, due to seismic release occurring intermittently in space and time. From time to time observed should ‘catch up’ with modelled, as is the case in the graph.

The concept is to achieve a calibrated integration model, which can then be used to extract the usual modelling information, regarding rock mass and structures. A proposed procedure for this is described in the next section.

4.6 Proposed methodology for implementing integration in practice

From the integration work reported on here, it is possible to put together a proposal for implementing integration in practice. All methodologies and procedures have been developed, and it is a matter of calibrating the integration for a new site. Although that may sound simple enough, the calibration is often a difficult process, and requires expert input from mine rock engineers. A further requirement is accurate seismic data.

Below is a generic proposal for integration that can be used in seismically active underground mines.

4.6.1 Seismic Hazard Assessment in underground mines

Seismic Hazard in an active mining scenario can be divided in two categories, viz. (i) rock mass related seismicity driven by stress, and (ii) shear slippage on geological structures, possibly triggered by mining close-by. Seismicity driven by rock mass stress is more suitable for modelling since such stress results are expected to be relatively accurate. However, the bigger seismic hazard, viz. large events on geological structures (e.g. faults or dykes), is inherently more
unpredictable. The strength distribution across the structure surface determines when a rupture will initiate and how large the slip area would be under the prevailing shear stress. This strength distribution, however, is unknown to a large degree, except for a guess of the average friction angle. This makes it difficult to quantify the probability of triggering a large event without specific input about rock mass and structure surface heterogeneities.

Numerical modelling can be used to quantify seismic hazard indicators, i.e. for rock mass and structures. Methodologies are sought to provide medium- and short-term seismic hazard assessment. These are based typically on the following modelling results:

- Rock mass stress level
- Elastic energy stored in the rock mass
- Excess Shear Stress on geological structures
- Shear Stiffness of geological structures

Improved results should be possible if information on rock mass and structure surface heterogeneities can be incorporated in some way.

4.6.2 Why integration is needed

Conventional modelling normally assumes homogenous, elastic rock mass and a constant in-situ stress state. In an attempt to improve results derived from a numerical model based on these assumptions, seismic deformation can be integrated into the elastic model, hence indirectly incorporating rock mass heterogeneities.

For example, a strong, competent volume within the large rock mass will yield limited seismicity under moderate stress levels. Seismic deformation will be lacking for that volume, so that the rock mass stress will not be relieved when the deformation is integrated into the integration volume of the model. Stress will therefore build up in that volume, indicated in the numerical model e.g. through contours of sigma 1. The danger in this example, is that as stress increases by progressing mining, the strength may eventually be exceeded, possibly causing a large and violent seismic event.

Modelling with integration therefore highlights areas where stress builds up due to the lack of sufficient seismic deformation. In general it is attempted to model
stress levels over the volume of interest more accurately than can be achieved by conventional elastic modelling.

4.6.3 Integration methodology

Non-linear modelling of the rock mass is not a simple matter. Such modelling packages are available, but require a major effort. The alternative approach is to use an elastic, boundary element numerical modelling package, and incorporate the seismic deformation as field loading effects.

The idea is therefore not to try and model the plastic strain, but rather revert to elastic stresses, with these modified by specifying 3D strain inferred from seismicity. For this a parameter called seismic displacement is calculated assuming a seismic source model and using recorded seismic source parameters as input. From seismic displacement, strain is calculated for small 3D rectangular volumes covering the volume of interest (the Integration Volume), typically cubic with side length 10 metres. It is usually assumed that the volume strains in the direction of the major principal stress modelled at that position.

The steps to be followed are briefly:

? Set up the mine-wide numerical model according the boundary element software package.

? Delineate the Integration Volume, i.e. the volume of interest. Both the mining geometry and seismic data should be accurate to a degree within this volume, and a too large volume might yield unacceptable model run times.

? Grid the seismic data onto the small volumes covering the Integration Volume. The end result is an estimate of the strain for each of the small volumes touched by seismic events.

? Integrate the seismic deformation into the elastic rock mass.

? Calibrate the integration model, hence achieving an energy balance between rock mass potential elastic energy and energy released co-seismically. Verify that modelling results make sense.

This should yield a rock mass stress distribution improved over what conventional modelling would have given. The usual analyses can then be performed towards quantifying Stability Criteria. Geological structures can also be incorporated in order to quantify their seismic hazard.
4.6.4 Stability Criteria from numerical modelling

Stability criteria are required for the rock mass and geological structures. Useful criteria obtainable from numerical modelling are described below.

**Rock mass Stability Criteria**

*Energy Release Rate (ERR)* quantifies hazardous stress distributions residing close to the mining face and hence indicates face burst potential. It is mainly a function of mining span, face shape and stress level.

*Volumetric Energy Release (VER)* represents the energy stored in the 'mined volume' of a delineated area at its worst case loading state. It is related to the conventional Energy Release Rate, but calculates the energy associated with the total mining step, and not only at the face. Modelled Volumetric Energy Release correlates well with recorded seismic release.

**Structure Stability Criteria**

*Excess Shear Stress (ESS)* quantifies the amount by how much the shear stress exceeds the shear strength along a structure contact surface. Shear stress is determined by the loading system and the structure orientation, while shear strength is determined by the contact property assumptions. If the shear strength is kept constant, ESS assesses the influence of the loading system on the structure, and as such reflects the probability of experiencing large events. A general high level of ESS suggests that the probability is high for a rupture to initiate, possibly yielding a large event of shear slip mechanism.

*ESS Differential* refers to a sudden increase in ESS on a structure surface due to mining in the vicinity, and is an indicator of the probability to trigger a large event.

*Potential Seismic Moment* reflects the potential size of a shear slip event on a structure. Modelled in-elastic shear deformation, or *ride*, can be used to calculate modelled seismic moment from the numerical model. Modelled seismic moment, however, is more suited as a design criterion (long-term seismic hazard indicator), e.g. to compare seismic hazard for different mining layouts. This is because structure deformation usually occurs intermittently in space and time due to varying strength across its surface, so that modelled seismic moment should correlate better over the longer term. However, at any time in the mining
sequence, potential seismic moment can be tested in the model, indicating the worst case seismic event possible at that stage.

*Structure Shear Stiffness* characterises the Loading System Stiffness imposed on the structure, and can be used as a short- and medium-term indicator of structure stability. A low stiffness means potentially large deformation and hence high seismic hazard. It can be estimated by performing a so-called stability test in the modelling package, whereby each structure boundary element is allowed to yield by dropping the strength across the structure. From the drop in shear stress and the accompanying shear deformation, the structure shear stiffness can be calculated.
5. Discussion and Future Directions

This implementation project covered 3 areas: tap-tests, panel sequencing and Integration for Controlled Mining. The first two topics investigated observations of the fundamental project GAP601a with a view to developing practical advice for the rock engineering practitioner.

Several tap-tests were conducted during the course of this project. Sufficiently clear data was obtained from 5 different sites, and compared with data recorded during GAP601a. It would appear that a clear relationship exists between levels of seismicity in an area and the quantitative acoustic emissions response after a single blast. This implies that a simple seismic test could be carried out on pillars to get some indication of the expected seismic hazard. This would be helpful for remote areas of a mine that are not well monitored by a seismic network. One could imagine this information being useful in planning which pillars to extract, and what kind of support may be required.

While these tap-tests were carried out using production blasting as the ‘taps’, there should be no reason why development blasts should not give similar results, provided sufficiently sensitive sensors are used. The technique is relatively simple to perform.

A further research topic would be to apply this technique after every production blast at a particular place, and obtain correlations between the details of the immediate seismic response (within 1 hour of blasting) and the seismic hazard experienced over the next 24 hours.

The idea that daily panel scheduling could to some degree control the character of the seismic response is appealing. It seems reasonable that blasting panels close together in space would disturb a large volume of rock and promote large seismic events. This project aimed to prove this by correlating levels of seismic stiffness with blasting configurations, since it has been previously established that seismic stiffness is inversely related to seismic hazard. While data from a complex mining environment showed no consistent relationship, data from a mini-longwall with 5 working panels did indeed show that seismic stiffness is degraded less (and therefore seismic hazard is reduced) by distributed blasting. Blasting of neighbouring panels should be avoided wherever convenient.

As the quality of recorded seismic data improves, and more accurate daily blasting information becomes available, this guideline should naturally be reviewed at each site by the mine rock engineers.

Over the past 3 years a wide range of Integration for Controlled Mining techniques were investigated. It cannot be stated that large events were successfully predicted at all the test sites, but results suggests that improved results over conventional elastic modelling can be achieved. The modification of
rock mass stresses by 3D integration seems satisfactory in general, and it remains a matter of calibration on-site to allow useful results to be extracted.

What is appealing is that the usual familiar modelling results, like Excess Shear Stress, potential Seismic Moment, Energy Release Rate, etc. can be extracted from an integration model, and hence results interpretation is not more complex.

Certain limitations are the same as for elastic boundary element code. Although stresses are modified by integration, it probably can not account for a completely non-linear rock mass. The research and development of a new generation of inelastic models, partially being funded by SIMRAC, has been motivated by this shortcoming. These new models should be able to handle the evolution of inelastic deformation.

A further limitation to Integration for Controlled Mining is the accuracy of seismic data, which should be reasonably accurate so that modelled stresses coincide with recorded seismic events. Trends towards higher resolution seismic networks in South African mines, and work on improved location of seismic events, will improve the quality of this seismic data in time.

All the work done highlighted pitfalls of integration, and steered the development of methodologies and procedures to do integration efficiently using available software and tools. It is a pity that funding was not readily available for implementation over the years. The feeling is probably that results are not guaranteed, being a new method. Hopefully this report highlights that improved results can be achieved over conventional elastic modelling, which is a step forward. The various applications discussed in this report can be readily implemented, and together with expert input from mine rock engineers, have the potential for improved short- and medium- term seismic hazard assessment through a better understanding of the rock mass response to mining.
Appendix A: Potency

An easy-to-determine parameter pertaining to the source is seismic potency $P \equiv \frac{\Delta \varepsilon}{V} [m^3]$, where $\Delta \varepsilon$ is the strain change at the source and $V$ the source volume or, for a planar source $P \equiv \bar{u}A, [m^2]$, where $\bar{u}$ is an average displacement and $A$ the source area (King (1978), Ben-Menahem (1981)). Potency is most frequently estimated from the amplitude of the low frequency displacement spectra $a$ of the recorded waveforms (Keilis-Borok (1960)).

$$P_{P, S} \equiv 4v_{P, S} R \frac{a_{P, S}^2}{\sigma_{P, S}^2}$$

where $v_{(P, S)}$ is P or S-wave velocity, $R$ is the distance from the source and $\sigma_{P, S}^2$ is the root-mean-square value for the radiation pattern of far-field amplitudes averaged over the focal sphere: $P \equiv 0.516$ and $S \equiv 0.632$. Seismic moment is the product of rigidity and potency, $M \equiv P$, and it is an ambiguous measure for sources occurring at rock interfaces with different rigidities or for sources in a thin layer of low rigidity, e.g. fault gouge, sandwiched in a stiff rock (Heaton (1989)). Since the linear elastic waves generated by shear dislocation have no information on material properties at the source, the potency is an objective observable scaling parameter for the size of an event (Ben-Zion (2001)).

Frequency-Potency Distribution

The well-known relation between $E$ and $M$ then becomes:

$$\log E = d \log M = c' \quad \log E = d \log P = c$$

and magnitude becomes:

$$m = \frac{2}{3}d \log P = \frac{2}{3}c = 3.2$$

Combining this with the Gutenberg-Richter frequency-magnitude relation, $\log N = \log m = \log a = bm$ gives the following frequency-potency distribution:

$$\log N \equiv P^{\gamma} \equiv \log ? \equiv \log P$$

or

$$N \equiv P^{\gamma} \equiv P^{\gamma}$$
where \( N \) is the number of events with seismic potency not smaller than \( P \), and \( 10^{-2.8 \cdot 2.48^{3/3}} \cdot 2^{\frac{2}{3}} \) dB. The parameter \( \frac{2}{3} \) measures the number of events with potency not smaller than one, \( N \). The one largest expected event would have a potency \( P_{\text{max}} \) that corresponds to \( \log 10 \) \( \log \frac{1}{P_{\text{max}}} \), thus \( P_{\text{max}} \) \( \frac{1}{P_{\text{min}}} \) \( N \), \( P_{\text{min}} \), where \( P_{\text{min}} \) is the minimum potency above which the data set is complete and it defines the sensitivity of the monitoring system. The mean recurrence time of events with potency not smaller than \( P \) is \( \frac{t}{N} \), \( \frac{1}{P_{\text{max}}} \) \( \frac{1}{P_{\text{min}}} \), where \( t \) is the period of observation.
References


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