

Safety in Mines Research Advisory Committee

Final Project Report

Further assessment of seismic hazard/risk in the Bushveld Complex platinum mines and the implication for regional and local support design.

A.v.Z Brink, M.K.C. Roberts, S.M Spottiswoode

Research Agency: CSIR: Division of Mining Technology

Project Number: GAP 821

Internal Document 2002-0268

Date: March 2002

Executive Summary

The objective of this project was to assess the seismic risk in the Bushveld Complex platinum mines and to provide guidelines for regional and local support in areas where the seismic risk is considered to be high. In order to achieve this objective, seismic data from some recently installed seismic networks were analysed.

A significantly increased risk is observed with mining at depth (down to 2000 m below surface), and with larger mined-out areas. Potholes, a feature unique to the Bushveld Complex, do not show as increasing the potential for larger seismicity. The deteriorated ground condition experienced when approaching potholes, however, enhances the likelihood of 'shakedown' damage and therefore increases the probability of an undesirable consequence (an increased risk of experiencing seismic related damage/injuries).

Comparable mining areas were selected in the Bushveld and on the Far West Rand. Based on the analyses of the seismicity, and in particular the peak ground motion that an excavation is subjected to, the seismic risk associated with mining in the deepest areas of the Bushveld is about 6 times less than a similar area on the VCR.

An industry workshop on local support requirements in areas of higher seismic risk resulted in the specification of support requirements. A maximum design parameter for yielding support in terms of the ground motion velocity is 1 m/s. Subsequent seismic monitoring provided the confirmation of this specification. A stope support design methodology and parameters are given.

A single example of the need for better regional support was demonstrated. An abutment pillar greatly reduced the convergence rate and seismicity in an area of mining down-dip from an extensively mined-out area.

Table of Contents

Executive Summary	2
Table of Contents	3
Table of figures	5
1 Seismic Risk in the Bushveld Complex – a summary of earlier work	6
1.1 Introduction	6
1.2 Earlier seismic work in the Bushveld	9
1.3 Current status of seismic monitoring in the Bushveld and the perceived seismic risk.	10
1.4 Rock Related Risk Assessment Techniques on Platinum Mines	12
1.4.1 Shortcomings	12
1.4.2 Advantages:	12
2 A seismic risk evaluation on a Bushveld mine	14
2.1 Introduction	14
2.1.1 A deep Bushveld mine compared with the Far West Rand with reference to peak ground motion	15
2.1.2 Calculation of peak ground motion	16
2.1.3 Exposure	21
3 Stope support for rockburst conditions on Bushveld platinum mines.	23
3.1 Stope support design parameters	24
3.1.1 Dynamic convergence.	24
3.2 Velocity of dynamic convergence.	24
3.3 Ejection thickness.	25
3.4 The determination of an energy absorption criterion for the platinum reefs of the Bushveld Complex.	26
3.5 The type of support system that would satisfy the energy absorption criterion.	27
3.5.1 The permanent support.	28
3.5.2 The face support.	28
4 Conclusions and Recommendations	32
5 References	34
6 Appendix A – Results from a 12 week GMM monitoring exercise at Northam ..	36

6.1	Introduction	36
6.2	Results	36
6.3	Calibration	40
6.4	Summary.....	42

Table of figures

Figure 2.1	A comparison of the Energy Moment relation between Northam at 2100 m, the VCR at a similar depth and the Carbon Leader at 3300 m	15
Figure 2.2	The relation between the average stress drop and depth as observed at the Far West Rand and at Northam.	16
Figure 2.3	An area on the VCR used to demonstrate the application of a grid system to determine the number of times that the peak ground motion exceeds a preset threshold value.	19
Figure 2.4	A comparison of the observed number of times that a peak ground motion threshold are exceeded.....	20
Figure 2.5	As in Figure 2.4 but plotted on linear axes.	20
Figure 2.6	The seismic event distribution of all events larger than Mag 0 on the VCR Far West Rand at 2100 m. (30 min resolution).....	21
Figure 2.7	The seismic event distribution of all events larger than Mag 0 recorded at Northam at a depth of 2100 m. (30 min resolution)	22
Figure 3.1	Cumulative percentage of ejected block thickness for Ventersdorp Contact Reef.	25
Figure 3.2	Cumulative fallout thicknesses for the Ventersdorp Contact reef.	26
Figure 3.3	The variation of the energy absorption criterion for a range of ejection thicknesses.	27
Figure 3.4	The energy absorption capability of the yielding timber support system plotted as a function of the stope convergence and distance behind the stope face.	29
Figure 3.5	The determination of the distance behind the stope face where the energy absorption criterion is not satisfied.	31
Figure 6.1	Plan view of the seismic event location.....	37
Figure 6.2	Section view of the seismic event location	38
Figure 6.3	Seismic event distribution as a function of the day of the week	38
Figure 6.4	Seismic event distribution against time of day.....	39
Figure 6.5	The distribution of events versus date/time, showing the 5-week strike period.....	39
Figure 6.6	Frequency magnitude distribution for events recorded by Ground Motion Monitor ...	40

1 Seismic Risk in the Bushveld Complex – a summary of earlier work

1.1 Introduction

The primary output of this project is the evaluation of current and future seismic hazards/risks, and the effect on support strategies in the Bushveld Complex platinum mines. The project was addressed from a risk assessment point of view and in particular the assessment of seismic risk to the safety of the underground worker in platinum mines.

As in GAP 608, (Survey and assessment of techniques used to quantify the potential for rock mass instability SIMRAC 2000), it is important to qualify the term seismic risk as opposed to seismic hazard. The generic term, hazard, is defined by the Mine Health and Safety Act No 29 of 1996 and interpreted by the Tripartite Working Group (SIMRAC, 1998) as:

Hazard is a physical situation, object or condition, which, under specific circumstances has the potential to cause harm.

Risk is seen by the Act (and by the Tripartite Working Group) as a measure of the likelihood that some specific harm, arising from an incident, will occur.

If this general definition is applied to seismic events, the seismic hazard will then be those seismic events that have potential to cause harm. A greater hazard will imply the potential to cause greater harm. A study of what the maximum event magnitude in an area might be is typically a seismic hazard determination.

The generic equation:

$$Risk = Hazard * Vulnerability$$

was, in this case, extended to (Menoni, *et al*, 1999):

$$Risk = Seismic Hazard * Induced physical hazard * Systemic Vulnerability$$

(Induced physical hazard = triggered by the ground motion; support failure; fall of ground.

Systemic Vulnerability = Exposure of people; economic vulnerability; quality of information.)

A possible risk assessment methodology was outlined in the GAP 608 final report. This is shown in Table 1.1.1, in which the proposed procedure for evaluating seismic risk is summarised. The assessment is made in four categories, namely

- Level of Ground Motion
- Vulnerability of the Excavation to ground motion

- Exposure of people
- Quality of information

At present each category has effectively the same relative importance or weighting.

Rating	Risk Assessment Category	Parameter	Parameter rating
P ₁	Level of Ground Motion	M _{max}	p ₁
		Distance from source	p ₂
		Mean Return Time (Frequency) Seismic/Time Distribution	p ₃
P ₂	Vulnerability of Excavation to Ground Motion (or Falls of Ground)	ERR	p ₁
		Geology	p ₂
		Support	p ₃
		Ground condition	p ₄
		Escape ways (Site Effect Amplification)	p ₅ (p ₆)
P ₃	Exposure of people	People/Time distribution	p ₁
P ₄	Quality of Information	Mine plans/Structure/layout	p ₁
		Seismic Monitoring	p ₂
		Early Warning	p ₃
		Assessment interval and volume	p ₄
		Experience reference	p ₅
		Communication	p ₆

$$P_{\text{risk}} = f(P_1, P_2, P_3, P_4)$$

Table 1.1.1 A proposed risk assessment methodology as proposed in Gap 608.

Possible 'real time' inputs to the risk assessment process are:

- Quantitative seismic information

- Mine's panel/stope ratings
- Mine geometry (face position)
- Numerical analyses output
- GMSI's risk database (IRMS2000)

The GAP 714 report (Brink, et al, 2002) describes the implementation of the concept as outlined in Table 1.1.1. An expert system approach was adopted to manage the input of the above risk parameters and to provide the infrastructure for relative weighting of the respective inputs.

1.2 Earlier seismic work in the Bushveld

Van der Merwe in GAP 027 (Haile and Jager, 1995) and Durrheim, et al (1997) described the application of a PSS system at No 10 Shaft Wildebeestfontein (Impala). Closer inspection of the location of the seismic events in relation to the mining geometry shows that they were distributed as follows:

The majority (50%) of the events located on pillars in mined out areas, indicating that these yield pillars are to some extent failing. Most of the larger events recorded plotted at such locations, and some pillar damage and pillar "punching" into the footwall has been correlated with these events.

Some 35% of the recorded events located at active stope faces. These were generally quite small events, but some exceptions occurred, as in panel 5S at raise W1564 where an event of magnitude +0,8 was correlated with a substantial fall-of-ground, which was associated with a major joint surface.

Some 15% of the events located in the back areas of active and older panels. A number of these could be correlated with falls-of-ground.

Some of his conclusions are also listed as follows:

Although the largest number of events occurred during the blasting period and directly thereafter, the larger magnitude events tend to occur outside this time period.

The majority of the larger events occurred on the pillars and remnants, probably due to the (partial) failure and / or punching into the footwall of these pillars.

Some of the larger events occurred in the back areas of stopes, and generally coincided spatially with falls-of-ground.

Interpretation of seismic source parameters such as stress drop and energy index, demonstrates the build-up of conditions of higher stress at pillars and at the edges of remnants and regional/stabilising or barrier pillars.

The relevance of these conclusions in terms of seismic risk can be summarised as follows:

The probability of having events larger than $M_L=1,1$ is limited at the mining depth monitored. Extrapolating the observed frequency/magnitude distribution indicates that the probability of having an event in the order of, or larger than, $M_L=2,0$ is very small. Purely from a simplistic event magnitude criterion, an $M_L=2,0$ is significant because this is typically the magnitude of event that causes most of the accidents and fatalities in the deeper level RSA gold mines.

Geological structures seldom pose a significant seismic risk in the Bushveld and the majority of events are associated with pillars and remnants, therefore only personnel in the vicinity of these pillars/remnants are exposed to the associated risk.

A possible reason for the few large events on geological structures may be the clamping effect of relative high horizon stresses as indicated by a high k-ratio on most of the Bushveld mines. (Specifically not true at Northam)

1.3 Current status of seismic monitoring in the Bushveld and the perceived seismic risk.

Seismic monitoring in the Bushveld was initially only provided by the national seismic system run by the Council for Geosciences in Pretoria. A number of events were recorded during the years, but it is debatable how many of these could be classified as mining induced events. It is well known that events do occur in this region outside the immediate mining districts. Table 1.3.1 lists the larger events recorded by the Council for Geosciences.

ID	Date/time	Latitude	Longitude	Region_and_Description	MI
19129	6/29/90 3:23:48	-25,9	27,4	Rustenburg_Region	2,5
19504	10/22/90 2:40:06	-25,6	27,2	Rustenburg_Region	3,2
21012	1/13/92 5:50:47	-25,7	27,4	Rustenburg_Region	2,8
22338	2/3/93 1:20:24	-24,1	27,3	THABAZIMBI_REGION	2
23575	3/24/94 5:38:56	-25,6	27,2	Rustenburg_Region	2,7
24073	11/2/94 2:43:04	-25,7	27,2	Rustenburg_Region	3,4
24332	4/18/95 23:55:05	-25,3	27,5	THABAZIMBI_REGION	1,6
24558	8/8/95 7:46:05	-25,4	27,1	Rustenburg_Region	2,9
26029	7/25/97 23:34:00	-25,7	27,4	Rustenburg_Region	3,1
26543	3/10/98 0:10:11	-25,511	27,274	Rustenburg_Region	2,2
27080	10/8/98 22:44:07	-25,86	27,174	Rustenburg_Region	3,2

Table 1.3.1 A summary of events recorded by Council for Geosciences for the last ten years.

In the 1980's a number of GENTEL systems were installed on various mines. These systems were single site surface installations, but were still effective in recording mining-induced events. Accurate locations (say within 50 m) were not possible. Three GENTEL systems were originally installed at Impala (Bafokeng–North #12 Shaft, Wildebeestfontein South #9 Shaft, Wildebeestfontein South #10 Shaft) and three at Angloplats (Turffontein, Townlands and Frank shafts). Most of these systems are reported still to be in operation.

Towards the end of 2000 two PRISM systems (at Amandelbult and at Impala) were in operation. Another PRISM system has been installed (at Union section), while two ISS systems were being installed (at Northam and Rustenburg, Frank Shafts).

Brink, et al, 2001 concluded in GAP 711 the following:

- The vast majority of the mining areas in the Bushveld do not experience seismicity that poses a significant risk to the safety of underground workers.
- In some areas seismic incidents do occur and a number of fatalities can be linked to seismicity. (Strain bursting on highly stressed pillars is a form of mining-induced seismicity or rock bursting).
- The primary issue is the ability to identify those areas with an associated high seismic risk and the implementation of effective control strategies to minimise such risk.
- Seismicity is assumed by most of the industry to only occur in or around highly stressed pillars/remnants. The researchers believed they found evidence of dynamic failure on a fault structure, although subsequent observation could not confirm this.
- The maximum amount of seismic energy released in the Bushveld mines at current mining depths is significantly less than in the deep level gold mines.
- Accepting the above observation, seismic risk in the Bushveld is due to being close to the source (for example in the case of strain bursting) and also the apparent inability of the support units to yield while maintaining a stable hangingwall.
- There are strong indications that the daily blasting has a less concentrating effect in terms of temporal distribution of seismicity. A larger percentage of the seismicity occurs during shift times.

It was recommended that:

- By accepting that high seismic risk does occur in some areas in the Bushveld mines, a methodology should be developed to pro-actively identify these areas.
- A comprehensive support strategy should be developed for the abovementioned areas in order to provide effective rockfall and rockburst control.

1.4 Rock Related Risk Assessment Techniques on Platinum Mines

In the SIMRAC report GAP 608 the assessment techniques for seismic risk in gold and platinum mines were described. It was realised that seismic risk comprises more than the probability of a large event occurring, but also the vulnerability of the excavation to seismic events. Such vulnerability is well defined in the standard rock engineering risk assessment techniques (other than seismic).

1.4.1 Shortcomings

Listed below are some of the more important shortcomings of the risk assessment and management systems in operation at present. These were highlighted during interviews on the mines, but it does not imply that every operations is experiencing all, even any, of these shortcomings.

- Subjectivity.

It is extremely difficult to ensure even reasonable consistency in the assessment of risk levels. (Rustenburg Section disagrees with this statement)

- The general lack of assessments done by personnel external to the mine.
- Lack of sufficient rock engineering resources, trained and certificated personnel.
- One of the most important factors, namely that of rockburst risk, has been very difficult to quantify.
- Lack of follow-up on recommendations and action plans is a problem.
- Resistance to change has been a problem i.e. after the introduction of new risk assessment and management systems. (Rustenburg Section did not experience resistance to change)
- In the risk rating procedure, uncertainty with regard to the attitude of major geological discontinuities may result in excessively high-risk ratings.
- A risk assessment system must not be seen as a 'black box' providing exact solutions. A degree of engineering experience and judgement is still necessary.

1.4.2 Advantages:

- Past problems associated with the declaration of special areas have largely been cleared up.
- Systems are seen as important management tools forming an integral part of the planning process.

- There has been a marked improvement in communication between production personnel and rock engineers since the introduction of such systems.
- The systems do not replace underground trips but do allow rock engineers, managers and supervisors to focus on problem areas and to be more pro-active.
- On some mines there has been a marked improvement in rock-related accident rates since the introduction of the systems. On other mines it is still too early to say whether there has been any direct effect. Significant improvement on two mines may have been as a result of the introduction of improved support systems that were brought in at the same time as the risk assessment system.
- On one mine, a risk assessment system has been very useful in determining whether certain remnants are safe to mine or not.
- The formal approach of assessing risk ensures that problem areas are more effectively highlighted and addressed.

2 A seismic risk evaluation on a Bushveld mine

2.1 Introduction

In order to achieve a practical evaluation of the risk imposed by seismicity in the Bushveld, it was decided to quantify the risk related aspects of the Bushveld seismicity with respect to the gold mining in the Far West Rand.

For comparison, areas were selected at the deepest mining at Northam mine (approximately 2100 m), on the VCR at the same depth and on the Carbon Leader reef at approximately 3300 m. All three areas had a face length of between 200 m and 250 m, a span of approximately 150 m and with extensively mined areas up dip. The production rate was also comparable. The area at Northam was selected because of its depth and given configuration, whereas the areas in the Far West Rand were purely selected on the basis of their similarity and therefore with implied normalisation to the Northam data.

Figure 2.1 shows the Energy Moment relation of 6 months' seismicity per mining area (+/- 800 events). ISS International (Van Aswegen, 2001) links the slope (d) of the Energy Moment relation, ($\log E = c + d \log M$) to the stiffness of the surrounding rock mass, i.e. its ability to resist deformation with increasing stresses. It is obvious from Figure 2.1 that as for the d -slope (and implied surrounding stiffness), there are little difference between the Northam and the VCR mining. The CL mining shows a slightly steeper slope, which may be a property of the specific layout and the characteristics of the surrounding quartzite. The higher energy levels are linked to the higher stress environment resulting from being approximately 1200 m deeper.

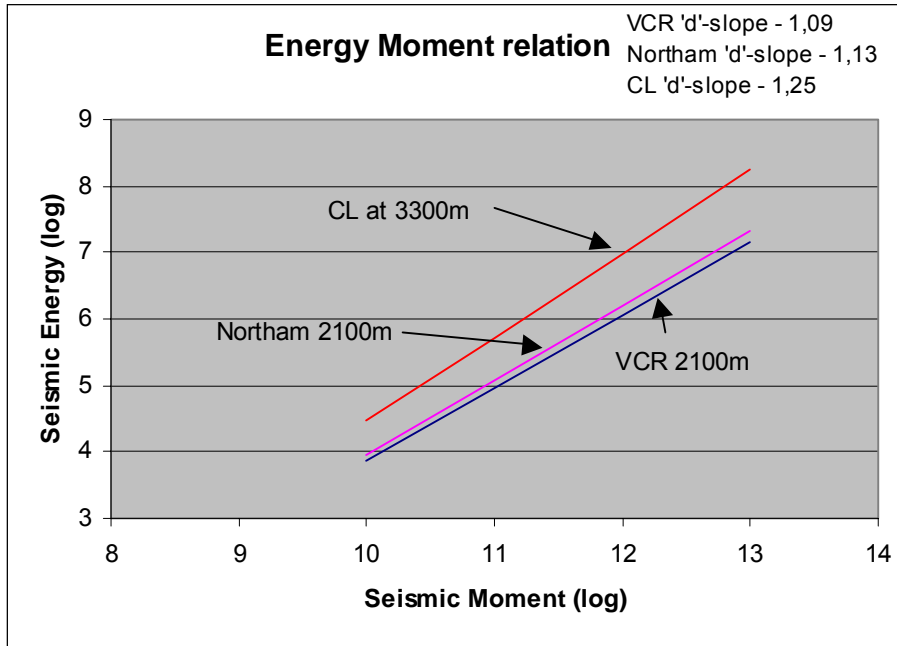


Figure 2.1 A comparison of the Energy Moment relation between Northam at 2100 m, the VCR at a similar depth and the Carbon Leader at 3300 m

2.1.1 A deep Bushveld mine compared with the Far West Rand with reference to peak ground motion

Static stress drop is shown by McGarr (1991) as being directly related to the peak ground motion observed. He also attempted to 'update' this relation to link peak ground motion to Apparent Stress. (McGarr, 2001)

Data from a Far West Rand mine is divided into 100 m depth sub ranges with a total range from 1700 m to 3600 m. The average stress drop is determined for each depth sub range.

Stress drop, $\Delta\sigma = c_2 M f_o^3$, where $c_2 = 1.8 \times 10^{-10}$, in hard rock. Stress drop estimates the stress release at the source. It is model dependant and because of the f_o^3 (corner frequency to the power 3) component, it is most sensitive to proper processing. However, Apparent stress $\sigma_A = GE/M$ is less model dependent and scales the value of Stress drop quite well. The results obtained in this report were confirmed by substituting Stress drop with Apparent Stress.

Figure 2.2 shows the relationship between stress drop and depth as observed in a Far West Rand mine and at Northam. The exponential increase of Average stress drop for increased depth for the Far West Rand is similar to that observed by Andersen and Daehnke (1999) in terms of peak particle velocity. At Northam, however, the effect of depth shows an inverse of what would have been expected.

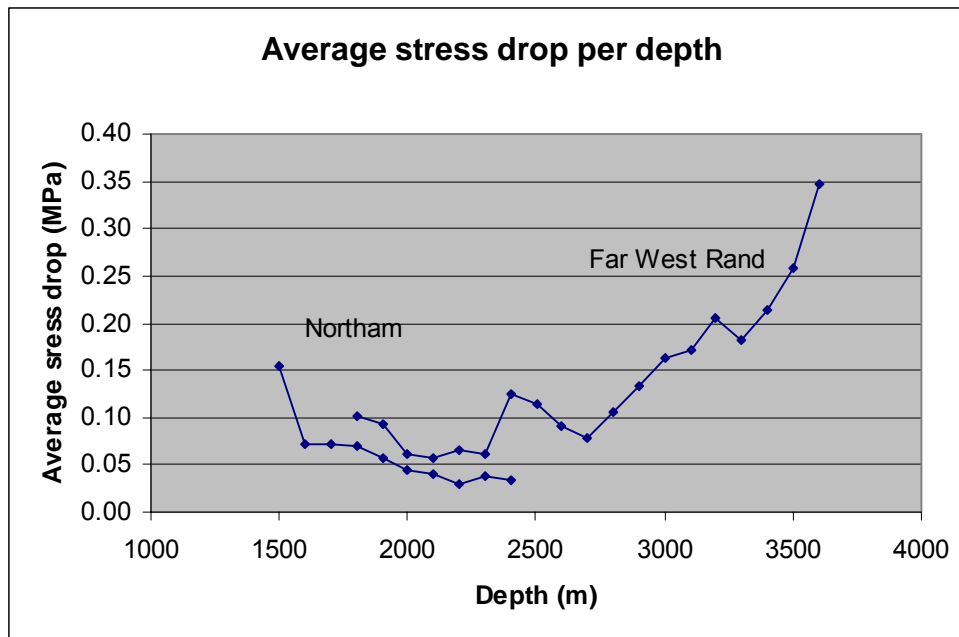


Figure 2.2 The relation between the average stress drop and depth as observed at the Far West Rand and at Northam.

The depth range that overlaps (from 1700 m to 2400 m) shows levels that are in the order of 20%-40% less than those observed at the Far West Rand.

A reason for the discrepancy in the depth dependence observed for the average stress drop may lay in the different mechanisms observed. The shallower mining in the Bushveld is characterised by events mostly resulting from pillars/remnant failure, whereas the deeper mining in the Bushveld (>1500 m) is characterised by seismicity ahead of the active mining faces.

It can be expected, however, that the deeper mining will result in an increase in the average stress drop.

2.1.2 Calculation of peak ground motion

The design criterion for support is currently based on its ability to stop the kinetic energy associated with a certain thickness of hangingwall (H and density ρ) moving downwards at a take-off velocity (v). The hangingwall must then be stopped within a distance (h). The energy required to absorb the potential and kinetic energy, in J/m^2 , is then:

$$E = \rho.H.(g.h + \frac{1}{2}.v^2) \quad (2.1.1)$$

“h” is generally taken as 0,2 m.

When $v^2 > 2.g.h$, more kinetic than potential energy must be absorbed. This occurs when $v > 2$ m/s for $h = 0,2$ m. As the kinetic energy is proportional to v^2 and its effect of support design against rockbursts increases strongly for $v > 2$ m/s, it is critical to be able to estimate the probability of encountering peak ground velocities in excess of 2 m/s. As we have very few actual measurements in this range, we need to

consider models that extrapolate observations made using mine-wide seismic systems.

In this section, we present a model of peak ground velocities in the near to far field and apply it to data from a deep level mine. A picture of the historical occurrence of ground motions is currently the end result of this analysis. Some suggestions are made for providing a comprehensive picture of likely risk.

2.1.2.1 Assumptions

We assume that:

- all seismic events occur on Brune-type circular slip zones in plan around each event location.
- the ground motions are well described by McGarr (1991), but with one alteration motivated here.
- the rock mass is elastic and homogeneous. Site effects and amplification at the skin of the stope are neglected.

Future seismicity is likely to be similar to historical seismicity. This can obviously be qualified by considering likely changes as new mining layouts encounter new geological features.

Models of seismic sources generally consider strong ground motion either in the near field or in the far field. In the near field, the peak velocity is

$$v_N = V_S \Delta\sigma / G \quad (2.1.2)$$

where v_N = near-field ground motion,

V_S = shear-wave velocity,

$\Delta\sigma$ = static stress drop,

and G = modulus of rigidity

Similarly, in the far field, (McGarr, 1991), we have the following:

$$Rv_F = f_{\theta\phi} V_S \Delta\sigma r_0 / G \quad (2.1.3)$$

where $f_{\theta\phi}$ = radiation pattern for S waves

and r_0 = source radius.

McGarr used the median value of $f_{\theta\phi} = 0,57$. Using the most conservative value, namely $f_{\theta\phi} = 1,0$, we have $v_N = v_F$ at $R = r_0$. Equations (2.1.2) and (2.1.3) then collapse into a single equation:

$$v = (V_S \Delta\sigma / G) \quad \text{for } R \leq r_0 \quad (2.1.4)$$

$$= (V_s \Delta \sigma / G) * (r_0 / R) \quad \text{for } R \geq r_0$$

By considering the circular source in the X-Y plane, $r_0 = \sqrt{(x_0^2 + y_0^2)}$, we can define the hypocentral distance "R" in terms of elliptical functions around this source as:

$$R' = (\sqrt{((r - r_0)^2 + z^2)} + \sqrt{((r + r_0)^2 + z^2)}) / 2 \quad (2.1.5)$$

and the peak velocity can then be expressed as a single equation :

$$v = (V_s \Delta \sigma / G) * (r_0 / R') \quad (2.1.6)$$

2.1.2.2 Implementation

Figure 2.3 show an implementation of peak ground motion as a risk parameter. A grid with a resolution of 25 m was placed over each of the three areas selected for comparison. (Figure 2.3 specifically shows the VCR area).

Each grid is considered independently. At each grid point all the seismicity in the area is evaluated as to its respective peak ground motion at that point. An arbitrary threshold (say 0,1 m/s) is set and the number of times that the grid point experiences a ground motion of more than the threshold is logged. It is clear that the highest values are experienced at the current face position. It is assumed that each grid point is in fact an excavation and a multiplication factor of 3 is used to allow for site amplification.

Peak ground motion and the number of times that an excavation experiences significant ground motion can be linked to physical damage in the excavation (Andersen and Daehnke, 1999).

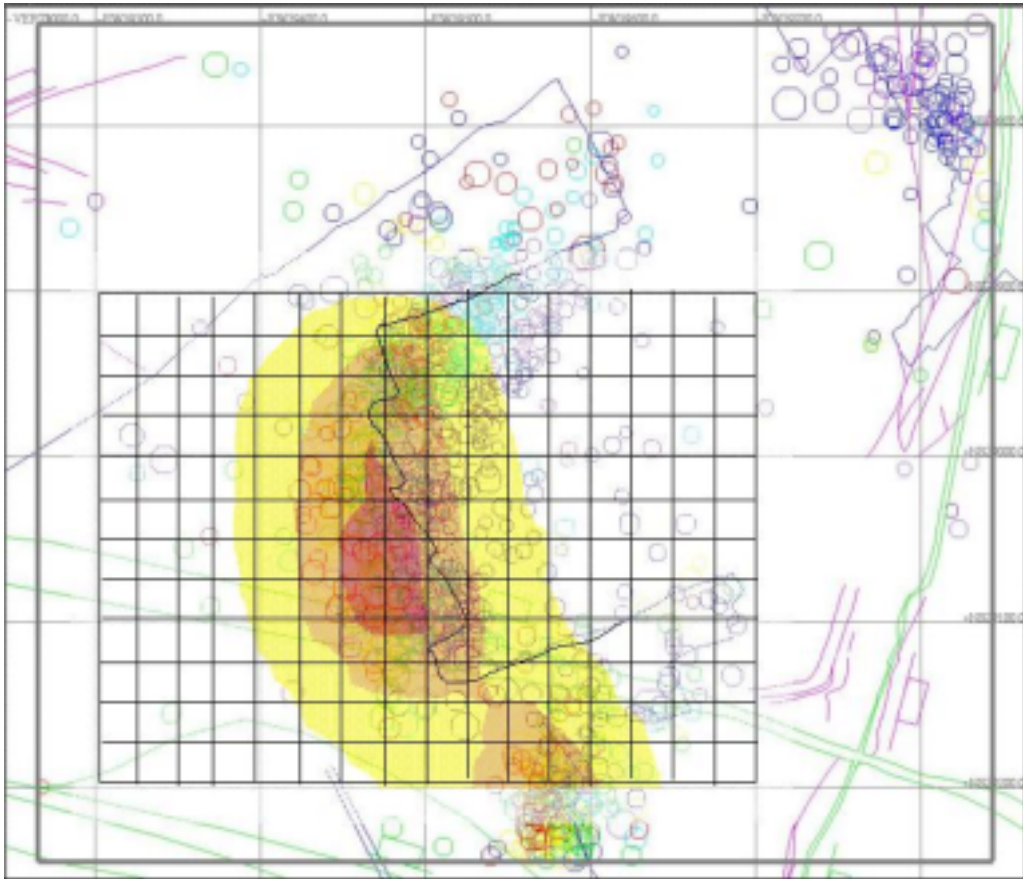


Figure 2.3 An area on the VCR used to demonstrate the application of a grid system to determine the number of times that the peak ground motion exceeds a preset threshold value.

The comparisons between the normalised data sets from the three respective mining areas are shown in Figure 2.4 and Figure 2.5. These figures and the interpretation of such are probably the most significant outcome of this project.

The implication is that, with broad assumptions, the risk posed by ground motion resulting from seismicity at Northam, is approximately 6 times less than comparable mining on the VCR and more than 12 times less than mining at depth on the Carbon Leader reef in the Far West Rand.

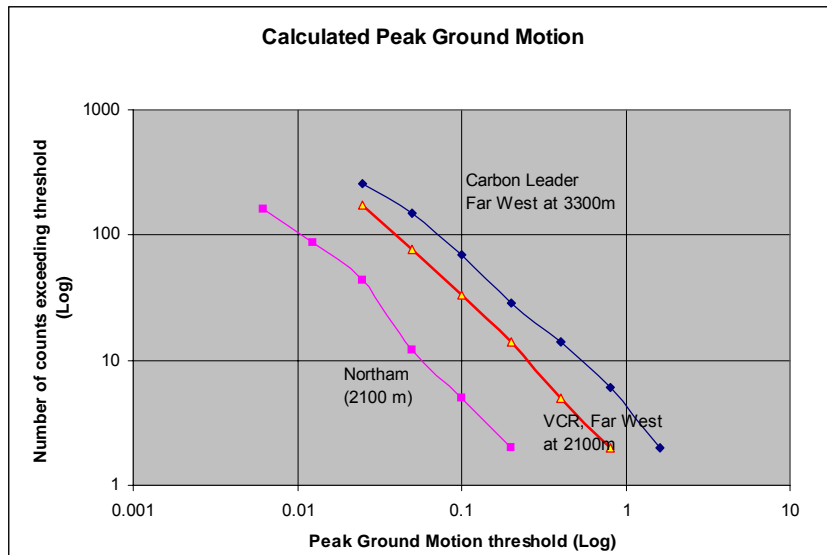


Figure 2.4 A comparison of the observed number of times that a peak ground motion threshold is exceeded.

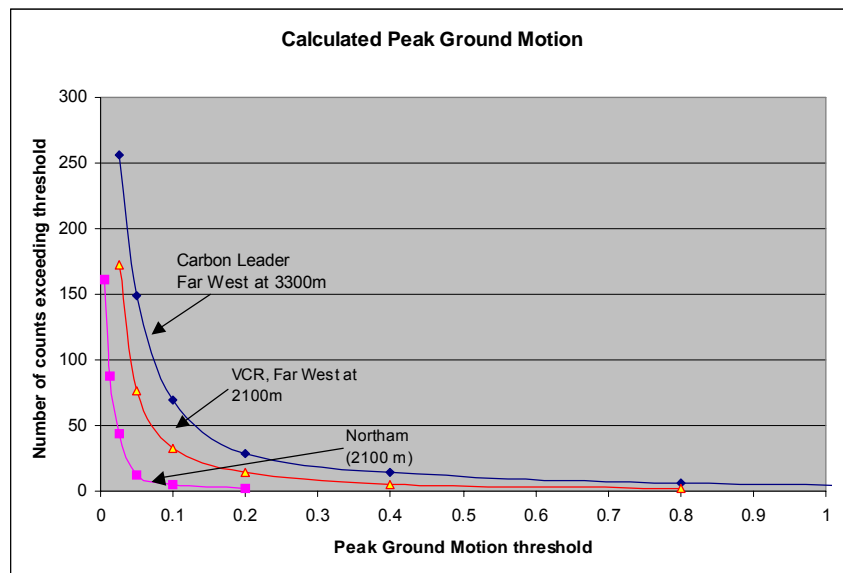


Figure 2.5 As in Figure 2.4 but plotted on linear axes.

These observations were achieved from a data set recorded over a period of six months at each of the three sites. It may be argued that this is not a sufficient sample to make a generalised statement on the seismicity in the Bushveld as compared with similar operations in the gold mines. However, the authors believe that the observations and interpretations are significant and correlate well with the Bushveld experience.

The verification of this observation and the reasons for such a difference needs to be undertaken. A significant difference in most mines in the Bushveld is the higher horizontal stresses (or k -ratio). The analysis done in this project could not quantify nor explain a possible link between higher horizontal stress and the observed lower levels of ground motion.

2.1.3 Exposure

Exposure relates to the exposure of the underground worker to the daily occurrence of seismicity. Again an area on the VCR at 2100 m and Northam at the same depth is used for this comparison. The daily distribution of seismicity of these areas is shown in Figure 2.6 and Figure 2.7 respectively.

The VCR experienced 19% of all the events of $\text{Mag} > 0$ during an 18-hour period from 21h00 to 15h00. Northam experienced 41% of all the events of $\text{Mag} > 0$ during a 17-hour period from 21h00 to 14h00. In general terms it can be stated that the underground worker working in the deeper sections of the Bushveld (specifically in the deeper sections of Northam) is experiencing 100% more events of magnitude > 0 during his underground shift than his counterpart working on the VCR at a similar depth.

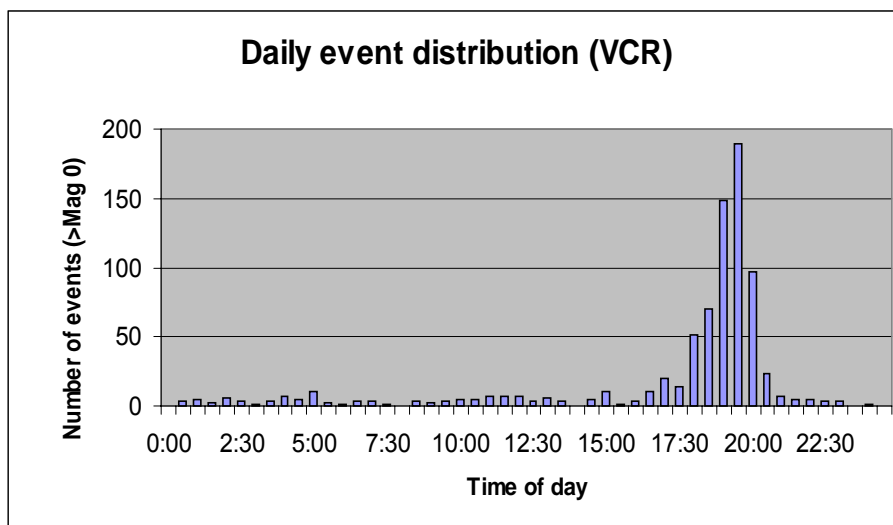


Figure 2.6 The seismic event distribution of all events larger than $\text{Mag} 0$ on the VCR Far West Rand at 2100 m. (30 min resolution)

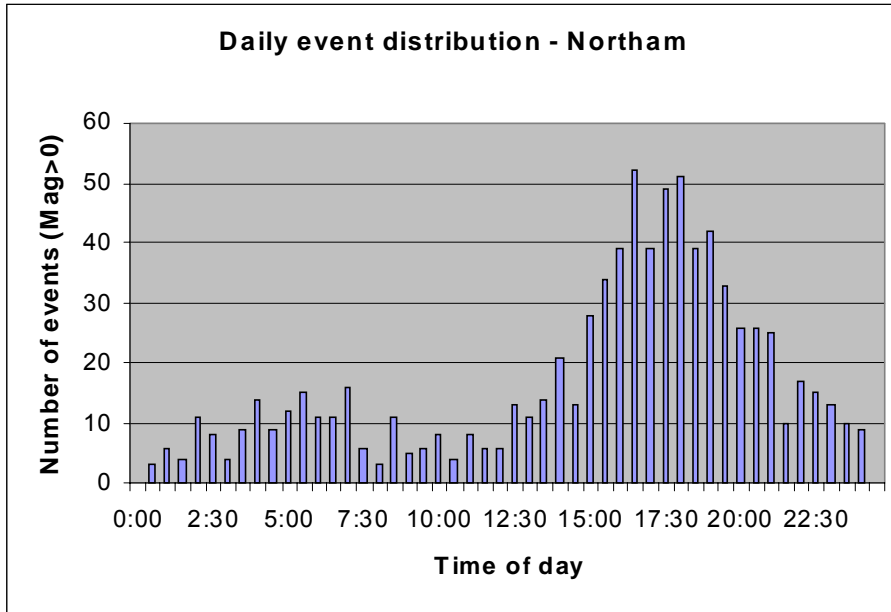


Figure 2.7 The seismic event distribution of all events larger than Mag 0 recorded at Northam at a depth of 2100 m. (30 min resolution)

Van Aswegen and Mendecki (1999) associated a 'softer system' or a lower d -slope with larger m_{max} , and also with a concentration of the time-of-day distribution around blasting times.

We experienced an apparent contradiction to this association. The area selected at Northam exhibited a relative low ' d '- slope, but also a large dispersion of the time-of-day distribution.

3 Stope support for rockburst conditions on Bushveld platinum mines.

A seismic event that caused a rockburst, occurred on one of the Bushveld Complex platinum mines in 2000 resulting in an injury and a fatality. After the event, detailed site investigations were undertaken and an eight geophone seismic system was installed around the stope that had been damaged by the rockburst. Examination of the site showed a considerable amount of convergence in parts of the stope, RSS grout packs showed damage associated with convergence, as did some of the timber elongates. In days subsequent to the rockburst, the installed geophones recorded possible aftershocks from the seismic event and the continued sequence of seismicity. They appeared to plot along a planar structure that was identified as a system of joints and a possible fault. There was evidence therefore, of a slip type event on a geological structure, the type of seismicity that is typical of the deep gold mines.

The determination of the stope support design parameters below has been influenced by the observations made at the site of this rockburst. There is also an influence from the rockburst experience gained on the gold mines and the opinions of persons attending a workshop that was held, at CSIR Division of Mining Technology, after the rockburst described above. The objective of the workshop was to determine how the mine's stope support system should change following the rockburst. The workshop was held on the 26/06/01 and was commissioned by Amplats. The attendees of the workshop were:

Mr A Forbes
Prof M Handley
Mr R Johnson
Mr A Jager
Mr VZ Brink
Mr W Benade
Dr M Roberts
Mr K Noble
Mr G Priest
Mr J Potgieter
Mr P Murton

The intellectual property as captured in the minutes of the meeting is therefore also owned by Amplats and it is with their permission that some of the outcome is repeated in this chapter.

3.1 Stope support design parameters.

3.1.1 Dynamic convergence.

The amount of dynamic convergence allowed should a rockburst occur has been set at 0,2 m for the deep gold mines. Roberts (1999) argued as follows, " *it is also assumed that the hangingwall of the stope displaces downward for an amount of 0,2 m during the rockburst. The amount that the hangingwall is allowed to displace will clearly influence the value of the energy absorption criterion, increasing the criterion if $h > 0,2$ and decreasing the criterion if $h < 0,2$. Many of the Witwatersrand reefs are narrow and the value of 0,2 m used can be justified on practical grounds. Consider a 0,9 m high Carbon Leader stope prior to dynamic closure of 0,2 m. Once this closure has occurred the stope is then 0,7 m high which is sufficient to allow miners, with difficulty, to manoeuvre and get out of the workings. If however a dynamic hangingwall displacement of 0,3 m (as Wagner⁽⁵⁾ assumed) or 0,4 m was allowed it would greatly increase the difficulty of movement that miners would experience in evacuating the workings, indeed the possibility of entrapment exists.*"

Further, inspection of the site of the rockburst at the Bushveld platinum mine, described above, revealed that geodynamic convergence of up to 200 mm had occurred.

It is therefore proposed that a dynamic displacement of 0,2 m should be used for the stope support design.

3.2 Velocity of dynamic convergence.

The velocity of dynamic convergence used for the design of stope support in gold mines is 3 m/s. Roberts (1999) has justified this on the basis of limited technical data.

During the workshop of the 26/06/01 consensus was reached that, for stope support design purposes, a velocity of dynamic convergence of 1 m/s should be used for platinum mines

Work carried out within this project has, to some extent, justified this parameter.

Appendix A described some Ground Motion monitoring (CSIR's Blackbox) The recording was carried out for about 12 weeks and the maximum peak ground velocity recorded was 60 mm/s. This was recorded at 3W panel (11 August 2001, 18H57). This is already a significant value and justifies the use of yielding support in this area.

The calculated peak ground motion results as shown in Figure 2.4 on page 20 also confirms the use of 1 m/s in areas of relative high seismic risk.

3.3 Ejection thickness.

The setting of an energy absorption criterion is based on the principle of back analysis of hangingwall ejection thicknesses involved in fatal rockburst accidents. The energy absorption criterion, E_{ac} , is reef specific and is determined using quantified ejection thicknesses for the reef from a fatal accident database. The ejection thickness for the Ventersdorp Contact Reef on the gold mines is shown as an example in Figure 3.1.

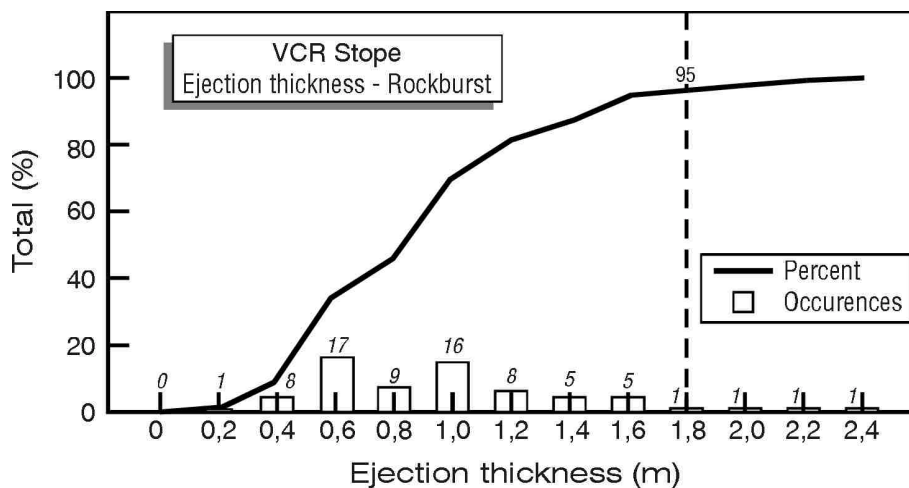


Figure 3.1 Cumulative percentage of ejected block thickness for Ventersdorp Contact Reef.

Figure 3.1 shows a histogram of ejected block thickness from rockbursts as a cumulative percentage of increasing block thickness measured at the sites of fatal accidents on the Ventersdorp Contact Reef. The ejection thickness representing 95% of the cumulative percentage is used for the purpose of developing an energy absorption criterion.

On the Bushveld platinum mines there is no history or data to determine what 95% *ejection thickness* represents, however, what is known is the 95% cumulative *fallout thickness* for individual mines.

The Ventersdorp Contact Reef has a blocky lava hangingwall and in some ways it is similar to the hangingwall of the platinum reefs of the Bushveld. Figure 3.2 shows the 95% cumulative fallout thickness determined at sites of fatal accidents caused by rockfalls on the Ventersdorp Contact Reef.

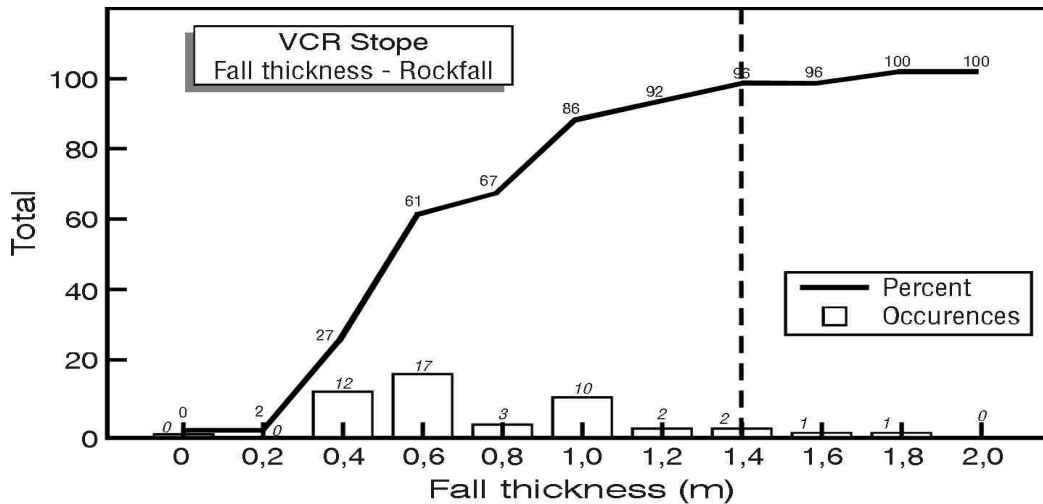


Figure 3.2 Cumulative fallout thicknesses for the Ventersdorp Contact reef.

The ratio of 95% cumulative fallout thickness to 95% cumulative ejection thickness for the Ventersdorp Contact reef is 1.4 m: 1.8 m. As the 95% fallout thicknesses for the Bushveld platinum reefs are known, it is proposed to use this ratio to determine a value that could represent the 95% cumulative ejection thickness for the Bushveld platinum reefs.

For example, if the 95% fallout thickness for the Bushveld platinum reefs is 1,0 m then the value that could represent the 95% cumulative ejection thickness for the Bushveld platinum reefs will be 1.29 m, say 1.3 m.

3.4 The determination of an energy absorption criterion for the platinum reefs of the Bushveld Complex.

Based on the data above:

$$\text{Energy absorption criterion (J/m}^2\text{) or } E_{ac} = 1/2 mv^2 + mgh$$

where m = mass of hangingwall per m^2 (height determined by 95% ejection thickness)

$$v = 1,0 \text{ m/s}$$

$$h = 0,2 \text{ m}$$

$$g = 9,81 \text{ m/s}^2$$

$$E_{(ac)} = (\frac{1}{2} \times 3900 \times 1^2) + (3900 \times 9.8 \times 0,2) \text{ J/m}^2$$

$$E_{(ac)} = 1950 + 7644 \text{ J/m}^2$$

$$E_{(ac)} = 9.6 \text{ kJ/m}^2$$

It is proposed therefore that the design of rockburst resistant stope support systems would require that they satisfy the energy absorption criterion of 9.6 kJ/m².

Figure 3.3 shows how the energy absorption criterion varies for a range of ejection thicknesses using the parameters described above.

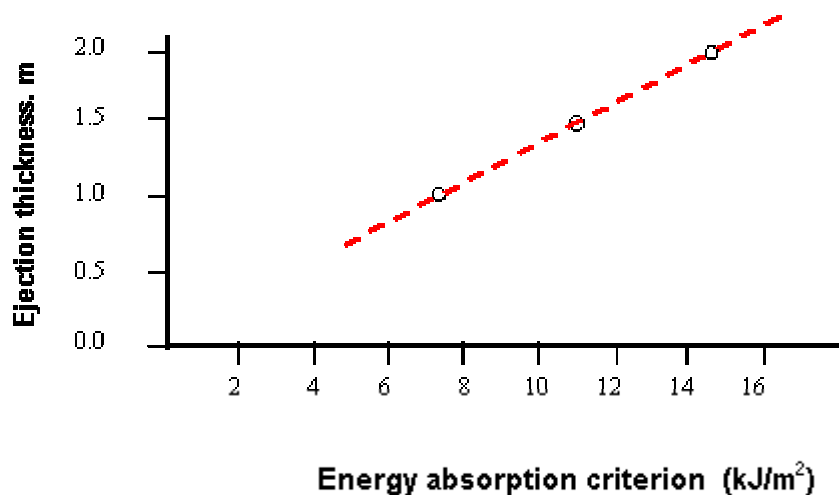


Figure 3.3 The variation of the energy absorption criterion for a range of ejection thicknesses.

3.5 The type of support system that would satisfy the energy absorption criterion.

This section describes the type of stope support system that could meet the energy absorption criterion of 9.6 kJ/m². It is not prescriptive, many support systems at a variety of support unit spacing have the ability to meet the energy absorption criterion. Below is an example of one such support system.

3.5.1 The permanent support.

Consider a timber elongate that can yield 200 mm at a yield force of 200 kN, these units could be spaced 1.8 m on strike and 1.5 m on dip. A support system made up of such support units would have a support resistance of 74 kN/m² and an energy absorption capacity of 14.8 kJ/m² before convergence acts on the support system. Assuming a convergence rate of 5 mm a day and a two day cycle and the front line of elongates is installed not further than 3,0 m from the stope face, the support design methodology, (Roberts, 1999) can evaluate this support system, as shown in Figure 3.4. From Figure 3.4, the energy absorption capability of the yielding timber support system can be determined as a function of the distance behind the stope face, as in Figure 3.5. From Figure 3.5, the distance behind the stope face where the support system no longer meets the energy absorption criterion is determined as 10,5 m at **A**. This can define the sweeping line and is barricaded off.

3.5.2 The face support.

From Figure 3.4, it can be seen that there is no energy absorption capability in the stope face area from 0 m to 3,0 m from the face. Clearly the energy absorption criterion is required to be met here. One way of achieving this is the use of yielding mechanical props. The energy absorption criterion requirement could be met by installing two rows of these props at a spacing of 1.5 m on dip and 1.2 m on strike, 1.2 m from the face. The proviso is that the props are fitted with a yield mechanism capable of yielding 200 mm at a force of 120 kN. This will give this temporary support system an energy absorption capability of 13.3 kJ/m².

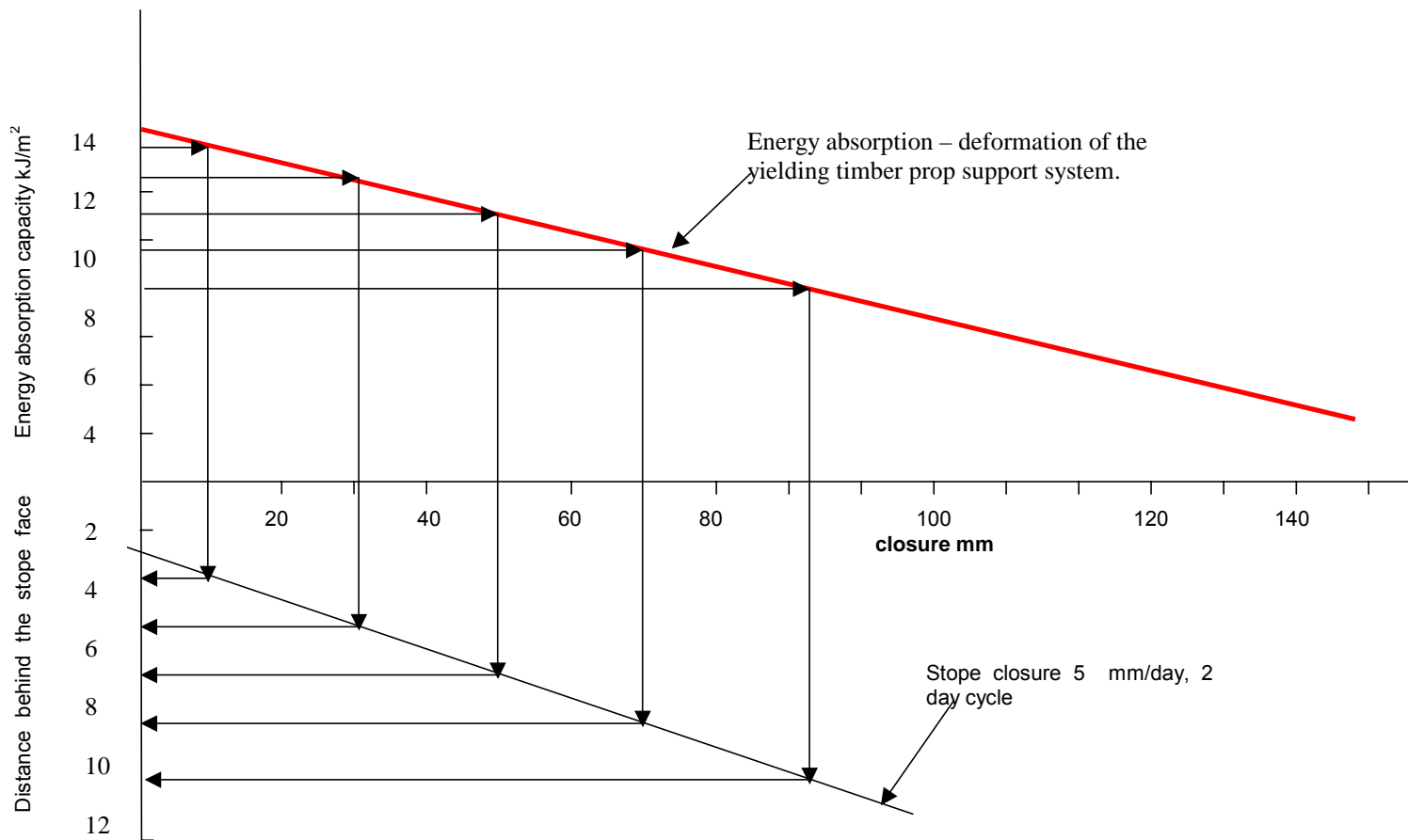


Figure 3.4 The energy absorption capability of the yielding timber support system plotted as a function of the stope convergence and distance behind the stope face.

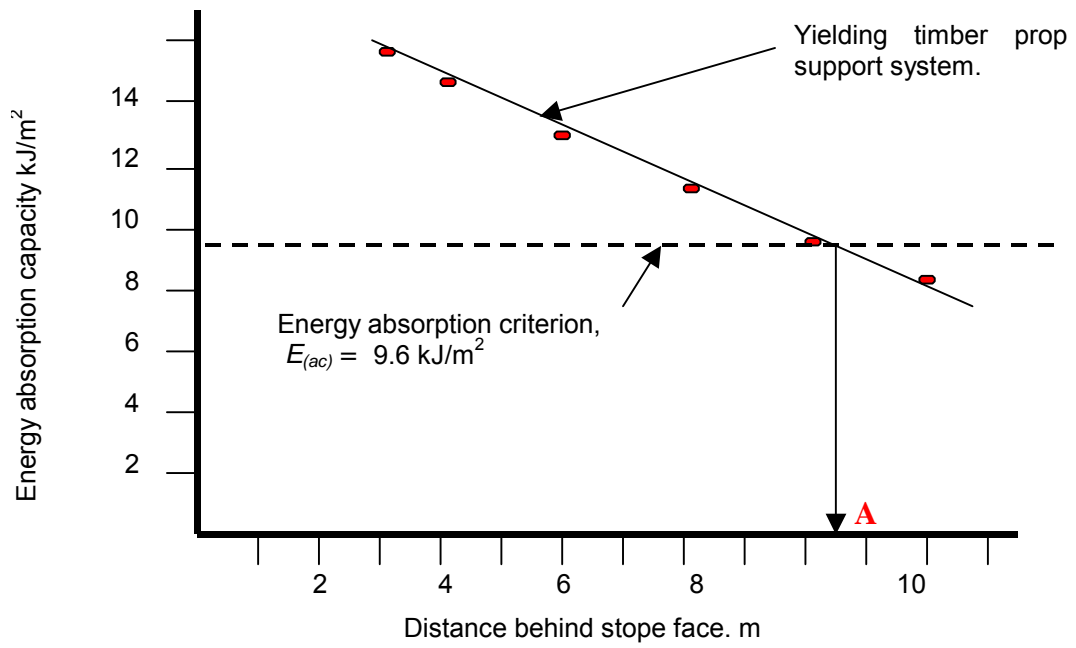


Figure 3.5 The determination of the distance behind the slope face where the energy absorption criterion is not satisfied.

4 Conclusions and Recommendations

The following conclusions were reached in this project:

1. The vast majority of the mining areas in the Bushveld do not experience seismicity that poses a significant risk to the safety of underground workers.
2. Seismicity is assumed by a significant portion of the industry to only occur in or around highly stressed pillars/remnants. The project experienced this to be true in the majority of areas that can be described as shallow to medium depth mining.
3. Accepting the above observation, seismic risk in the Bushveld is due to being close to the source (for example in the case of strain bursting) and also the apparent inability of the support units to yield while maintaining a stable hangingwall.
4. No seismicity was found that could uniquely be attributed to the existence of potholes, except where potholes were left as small remnants.
5. The researchers found clear evidence of dynamic failure on a geological structure. The deeper operations are experiencing normal (as in gold mining) seismicity ahead of the working face.
6. The amount of seismic energy released in the selected area in the Bushveld mines at comparable mining depths is in the same order (within an order of magnitude) as the selected and similar area on the Far West Rand.
7. There are strong indications that the daily blasting has a less concentrating effect in terms of temporal distribution of seismicity. A larger percentage of the seismicity occurs during shift times. In general terms it can be stated that the underground worker in the deeper sections of the Bushveld is experiencing 100% more events of $\text{Mag} > 0$ during normal shift time than his counterpart working on the VCR at a similar depth.

Again this is an observation that requires a larger data set for validation. It furthermore requires an understanding of why the time dependant behaviour of the Bushveld exhibits such a different response to the daily blasting.

8. The risk posed by ground motion resulting from seismicity at Northam, is approximately 17% of the risk posed by ground motion at comparable mining on the VCR and 8% of the ground motion risk at depth on the Carbon Leader reef in the Far West Rand.

This aspect is the most significant observation in this report and requires to be confirmed by comparing more areas and the inclusion of data on seismic damage in these areas

9. The project found substantially different behaviour in the interpreted seismological behaviour between the familiar gold (Wits) environment and the Bushveld. In particular, in terms of time of day distribution, peak ground motion, influence of depth. The project did not succeed in quantifying the possible contributing factors, such as k-ratio or higher horizon stress. This aspect requires addition research and monitoring.
10. The design parameters for the support system provides for a relatively easy achievable energy absorption requirement (9,6 kJ/m²).

5 References

Akermann, K.A. (1999). A support design methodology for shallow scattered mining environments. SAIMM Colloquium on Successful Support Systems for safe and Efficient Mining, Mintek, Randburg, May 1999.

Andersen, L. and Daehnke, A. (1999) Final project report for Deepmine 4.1.2.

Brink, A.v.Z., Hagan, T.O., Spottiswoode, S.M., and Malan, D.F. (2000) Survey and assessment of the techniques used to quantify the potential for rock mass instability SIMRAC Report GAP 608, Department of Mineral and Energy Affairs, South Africa.

Brink, A.v.Z., Hagan, T.O., Roberts, M.K.C. and Milev, A. (2000) Preliminary assessment of seismic hazard and risk in the Bushveld Complex platinum mines. SIMRAC Report GAP 711, Department of Mineral and Energy Affairs, South Africa.

Brink, A.v.Z., Spottiswoode, S.M. and Hagan, T.O. (2002) Software tool for managing seismic hazard in seismically active mines. SIMRAC Report GAP 714, Department of Mineral and Energy Affairs, South Africa.

Durrheim, R.J., Spottiswoode, S.M., Jager, A.J., van der Merwe A.J. Seismic Monitoring of Pillar System performance in Bushveld Complex Platinum Mines. SARES97, Edited by RG Gurtunca and TO Hagan. ISRM Regional Symposium, Johannesburg. 1997.

Haile, A.T., and Jager, A.J. (1995) Rock mass condition, behaviour and seismicity in mines of the Bushveld igneous complex, SIMRAC Report GAP 027, Department of Mineral and Energy Affairs, South Africa.

McGarr, A. (1991) Observations constraining near-source ground motion estimated from locally recorded seismograms, *J. Geophys. Res.*, pp 16495-16508.

McGarr, A. (2001) Control of strong ground motion of mining-induced earthquakes by the strength of seismogenic rock mass, *Rockburst and Seismicity in mines – RaSiM5, South African Institute of mining and Metallurgy*

Menoni, S., Petrini, V. and Zonno, G., 1999, Seismic Risk evaluation through integrated use of Geographical Information Systems and Artificial Intelligence techniques (Project SERGISAI - contract n. ENV4 - CT96 - 0279), *Proceedings of the review meeting on "Seismic Risk in the European Union", Brussels, Belgium, November 27-28, 1997*, edited by Maria Yeroyanni, EUR 180033 EN, Volume II, pages 117-149.

MRAC Task Group. (1999). Guidelines for the Compilation of a Mandatory Code of Practice to Combat Rockfall and Rockburst Accidents in Metalliferous Mines and Mines other than Coal. Published at website <http://www.dme.gov.za/mhs/gold.htm>. Pretoria, 1999 (35 pages).

MOSHAB, (1999). Surface Rock Support for Underground Mines – Code of Practice. Mines Occupational Safety and Health Advisory Board. Western Australia. February 1999. 16 pages.

Nicolau, N (1999). Stope Panel Rating System. Association of Mine Managers of South Africa. Papers and Discussions. Paper presented in August 1999. To be published. Johannesburg South Africa.

Roberts, M. K. C. 'The Design of Stope Support in South African Gold and Platinum Mines.' Ph.D. Thesis, 1999, University of the Witwatersrand, South Africa.

Spottiswoode, S.M. (1997) Energy Release rate with limits to on-reef stress' Proc. 1st Southern African Rock Engineering Symposium, Ed. RG Gurtunca and TO Hagan, SANGORM, pp 252-258.

Van Aswegen, G. (2001) Analysis of the processes which lead to potentially damaging seismic events associated with geological structures and highly stresses areas in extensively mined areas and old mines, SIMRAC Report GAP 605, Department of Mineral and Energy Affairs, South Africa.

Van Aswegen, G, Mendecki, A.J. (1999) Mine layout, geological features and geological hazard. SIMRAC Report GAP 303, Department of Mineral and Energy Affairs, South Africa.

6 Appendix A – Results from a 12 week GMM monitoring exercise at Northam

The purpose of including this appendix is that it describes a project where the Ground Motion Monitor independently monitored seismicity in the area at Northam that was extensively used in this project. The main objective was to determine reasons for the excessive convergence that was observed in this area. The recommended abutment pillar was also the only seismic observation of the effect of changes in the regional support design.

The Ground Motion Monitor also allowed for the only available data of ground motion recorded on the free surface in the excavation.

A secondary objective was to calibrate the seismic system at Northam.

This is not the full report and is abbreviated in order to only record the results that are appropriate to this report.

6.1 Introduction

During July 2001, CSIR Miningtek was invited to visit an area at Northam mine where excessive convergence was observed, (12 Level 5W). A consensus was that the convergence was due to high abutment stress. However, it was suggested that a Ground Motion Monitor, 'Blackbox', be installed. The objectives of the local seismic monitoring were to extend the coverage of the mine wide network; to determine whether the convergence is associated with significant seismicity (an increased seismic risk); and to use the output of the local system to calibrate the velocities of the mine-wide network.

6.2 Results

The 'Blackbox' was installed on 8 August 2001. A total of 328 events were recorded up to 30 October 2001. A five-week period of no mining (due to a strike) is included in this total period. Towards the end of September, the system got progressively worse due to cable damage.

Figure 6.1 shows a plan view of the processed events. The events tend to concentrate around the 3W panel. On the eastern side, a larger scatter is observed. This could be due to a location scatter, but also because of the presence of a large pothole. The 5W panel was stopped to form an abutment pillar. It is probably too early to observe the loading of this pillar and little seismicity occurred in this area.

A section view showing the vertical scatter is shown in Figure 6.2. Again, this scatter is larger than expected with some events up to 100 m from reef. The reason for the

relative inaccuracy in the vertical direction is due to the very planar nature of the sensor configuration and most ray paths being through the fractured zone around the mining excavation, resulting in lower wave propagation velocities.

Figure 6.3 and Figure 6.4 show the seismic distribution with respect to the day of the week and the time of day. The significance of these distributions is the effect that larger events (assuming a similar distribution) may have on the exposure of underground personnel to the seismic hazard. A typical hard rock deep level gold mine is experiencing a larger concentration of seismicity around blasting time.

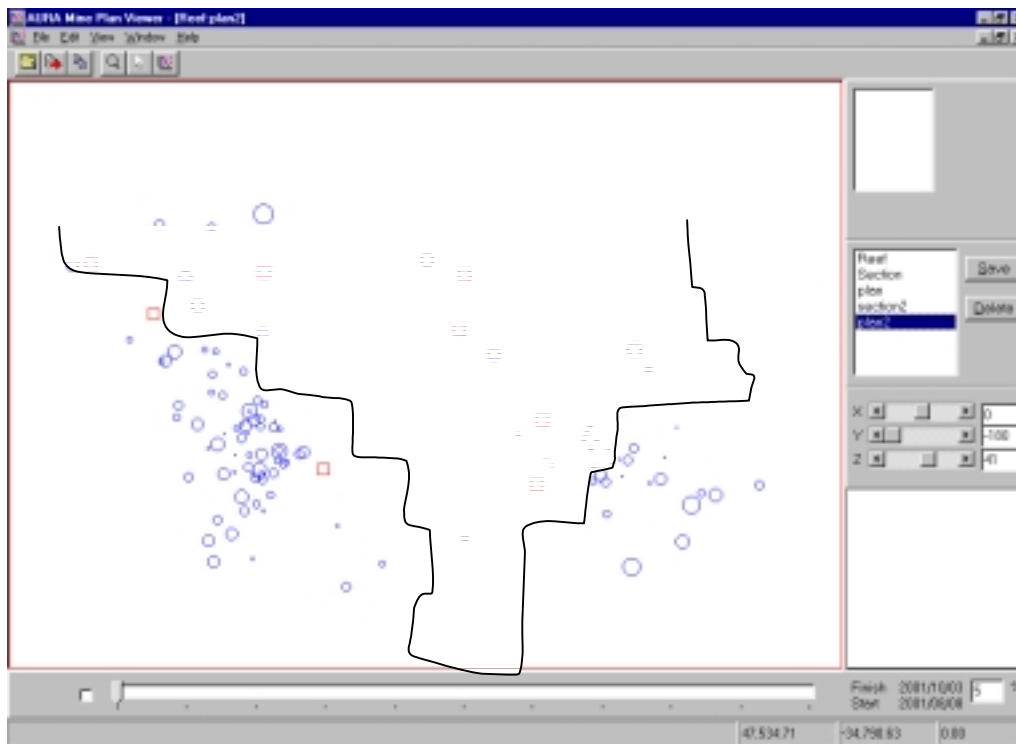


Figure 6.1 Plan view of the seismic event location

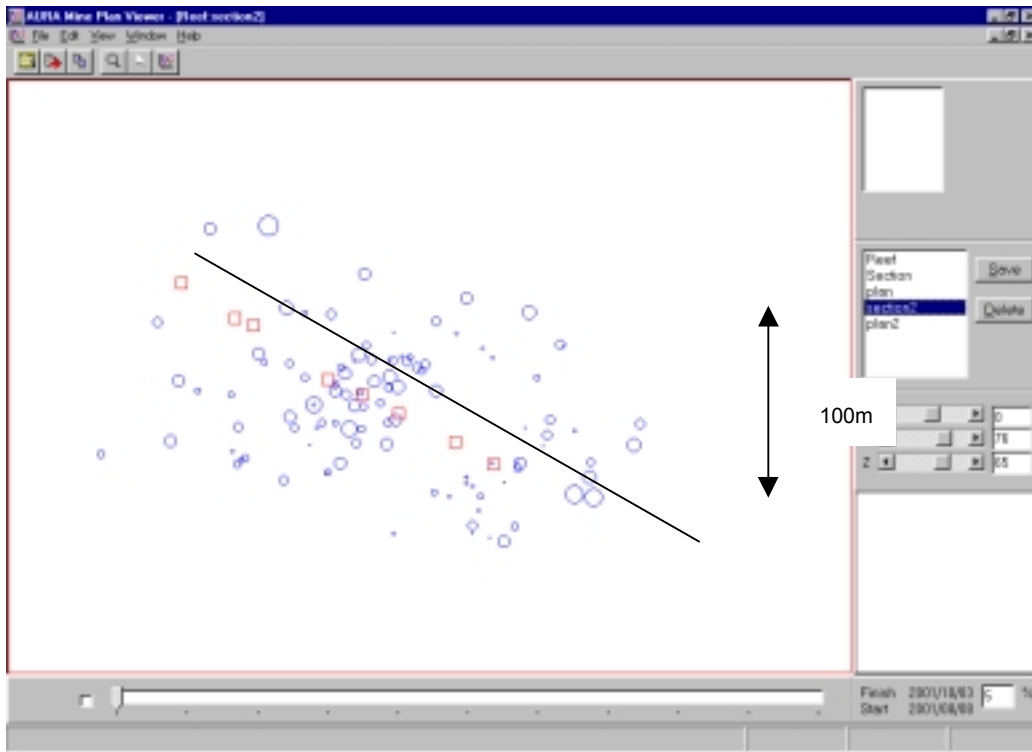


Figure 6.2 Section view of the seismic event location

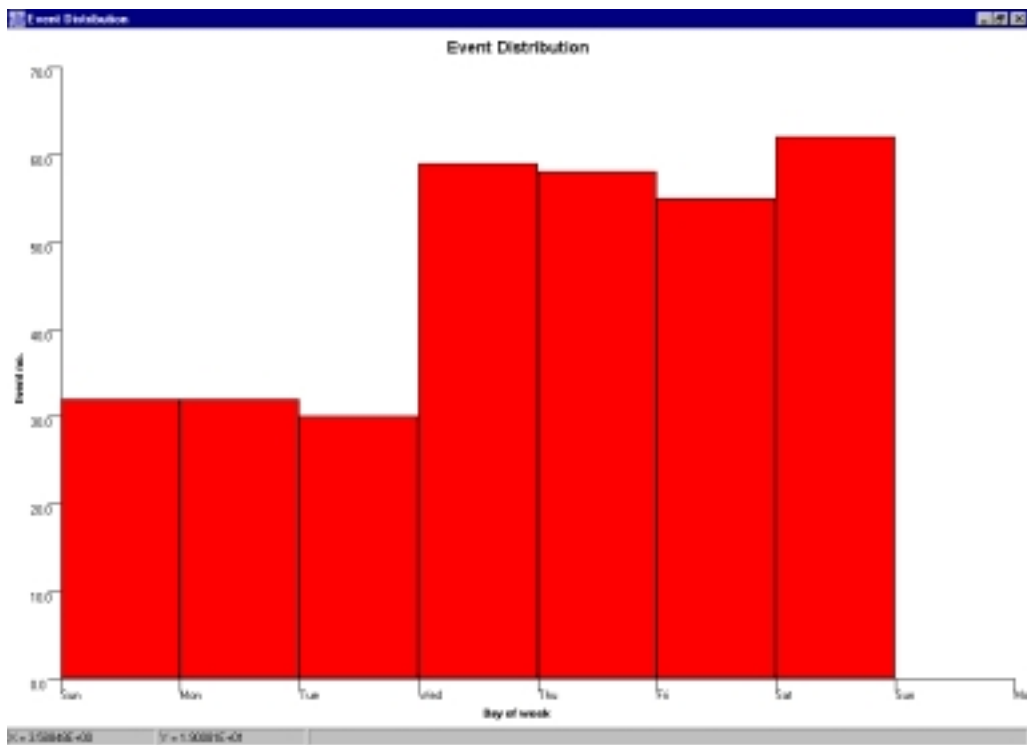


Figure 6.3 Seismic event distribution as a function of the day of the week

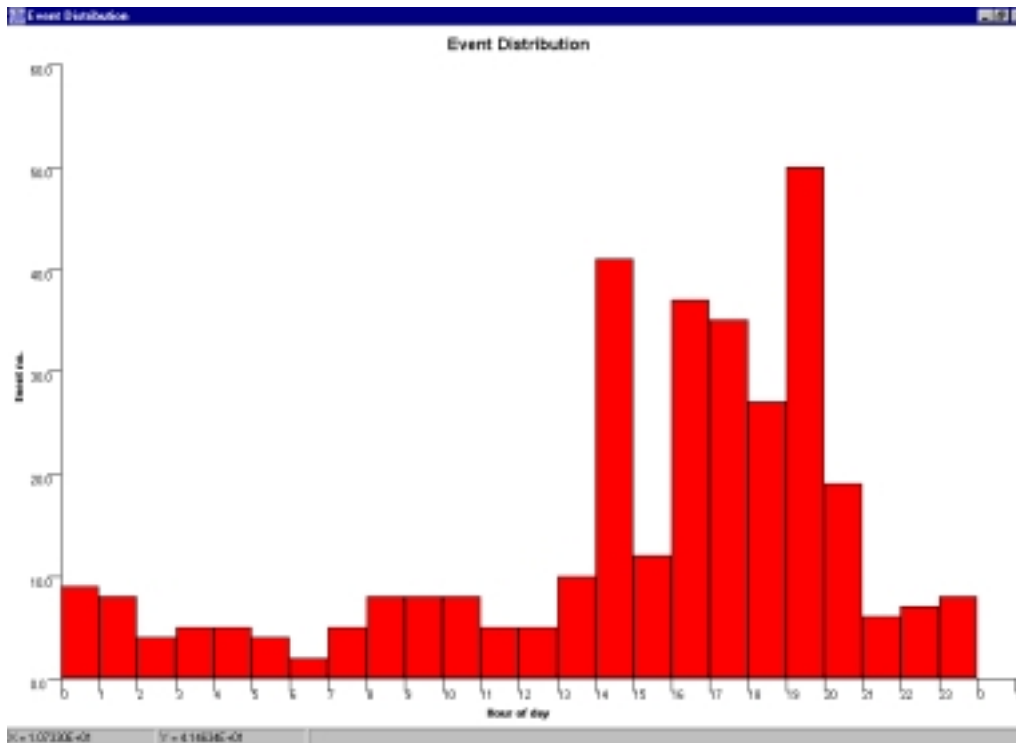


Figure 6.4 Seismic event distribution against time of day

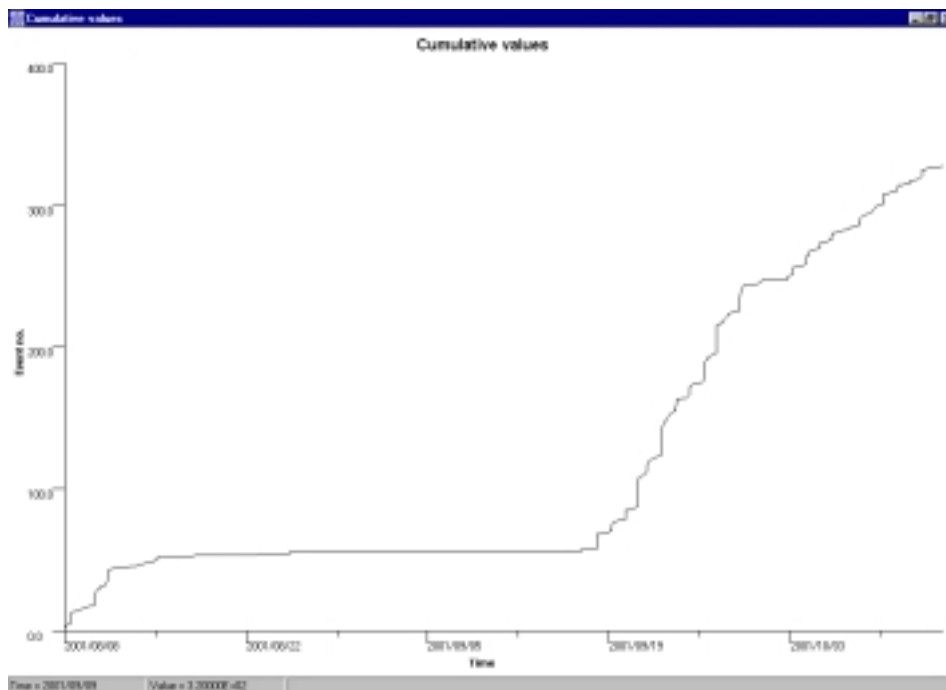


Figure 6.5 The distribution of events versus date/time, showing the 5-week strike period

Figure 6.5 shows the distribution of events versus date/time and the 5-week strike period is clearly recognisable as a seismically quiet period. Figure 6.6 shows the Frequency/Magnitude distribution of seismic events. The non-linear distribution may be attributed to some blast data being part of the total data set.

The peak ground velocity recorded was 60 mm/s. This was recorded at 3W panel (11 August 2001, 18H57). This is already a significant value and justifies the use of yielding support in this area.

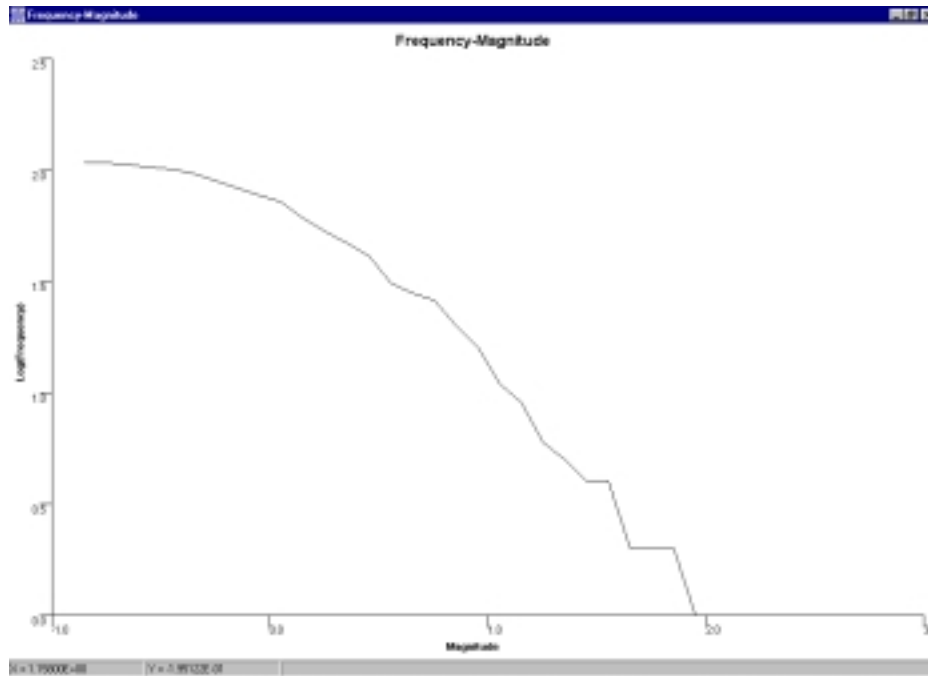


Figure 6.6 Frequency magnitude distribution for events recorded by Ground Motion Monitor

6.3 Calibration

An objective for the 'Blackbox' monitoring was to calibrate the mine wide system. Different options may be used to determine P- and S- velocities.

The following events seem potentially good reference events for calibration of the mine-wide system.

Date/time	X coor	Y coor	Z coor	Loc error	Mag
2001/08/08 23:29	47408,05	-34902,6	-1051,8	15	1,19
2001/08/09 23:07	47411,8	-34859,6	-1019,39	6	1,04
2001/08/10 19:58	47431,61	-34930,3	-1035,35	5	0,99
2001/08/11 18:57	47427,87	-34935,9	-1037,51	18	0,98
2001/08/08 22:59	47329,66	-35092	-968,917	17	0,96
2001/08/10 19:47	47386,18	-35060,2	-986,627	14	0,93

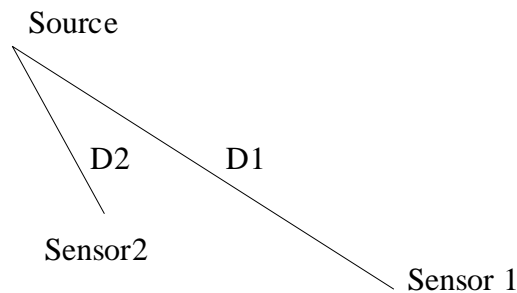
A dense sensor array will invariably result in a greater accuracy in absolute terms and the locations achieved from this may serve the same purpose as a calibration blast. The calibration with "Blackbox" events proved difficult because the mine-wide system only trigger at four stations at the time and these stations again formed a plane. It was obvious that higher P- and S-velocities are experienced at Northam but could not be tested.

A subsequent visit allowed for the use of ISS data. By selecting events with clear P- and S- arrivals and with one sensor relatively close to the event, the relative arrivals time can be use to calculate the wave velocities. It is calculated in sensor pairs. The ideal is to have a sensor pair with one sensor much closer to the source. If the pair of sensors is relatively at the same distance to the source, the time difference in p arrivals is small and the velocity calculation therefore inaccurate.

Obtain the exact arrival times of the particular wave front (P or S) and determine the wave velocities from:

$$V_p = (D1-D2)/(t1-t2)$$

where D1 and D2 are the source/sensor distance and t1 and t2 are the respective arrival times.



The inherent inaccuracies are well handled by selecting a number of observations and getting an average velocity.

Twenty individual triggers were processed with the following velocity values:

Observation	P velocity	S velocity	
1	6784	3937	m/s
2	6924	3705	m/s

3	7000	3898	m/s
4	6817	3569	m/s
5	6814	4058	m/s
6	6800	3948	m/s
7	6802	3741	m/s
8	7051	4083	m/s
9	6856	4073	m/s
10	6820	3792	m/s
11	6737	3957	m/s
12	6625	3843	m/s
13	6700	3915	m/s
14	6048	3452	m/s
15	6643	3931	m/s
16	6816	3870	m/s
17	6999	4005	m/s
18	6873	3791	m/s
19	6832	3648	m/s
20	6949	3924	m/s
Average	6794	3857	m/s

The implemented velocities were a P- velocity of 6750 m/s and S- velocity of 3870 m/s. The variation was due to the minimum resolution that the seismic system allowed for.

6.4 Summary

The objectives for the system were achieved. The monitoring stopped a bit prematurely, because of the theft of the sensor cabling. There was little merit in re-installing the cabling.

The original objectives of the local seismic monitoring were to extend the coverage of the mine wide network; to determine whether the convergence is associated with significant seismicity (an increased seismic risk); and to use the output of the local system to calibrate the velocities of the mine-wide network.

It could not be determined whether the relatively high convergence rate was associated with increased seismic risk. A mining decision to leave an abutment pillar in the area of high convergence had an immediate positive effect. Very little seismicity was observed around the pillar, which was just being formed. The general area however is subjected to a safety risk due to seismicity.

A higher rate of seismicity is recorded on the west faces. A peak ground velocity of 60 mm/s was recorded. This is already a significant value and justifies the use of yielding support in this area.

The mine-wide system was calibrated and event locations can be given with much greater confidence. It would be advisable to re-locate the total seismic data set with these new velocities