

Safety in Mines Research Advisory Committee

Final Project Report

Survey And Assessment Of Techniques Used To Quantify The Potential For Rock Mass Instability

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Executive Summary

The development of techniques to quantify the potential for rockmass instability that may result in rockbursts, has been a priority for researchers in the South African gold mining industry for some decades. These techniques include numeric modelling, seismic monitoring, stress and deformation measuring. Industry, COMRO and SIMRAC funding provided for the development of these techniques. The South African research & development efforts towards early warning of large instabilities (rockbursts) are foremost in the world. The question is whether our application in the field of these methodologies, concepts and techniques is equally advanced.

During this project the research direction changed slightly to cover a wider perspective of seismic risk assessment. It is therefore relevant at the outset 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 defined by the Act and the Tripartite Working Group as *a measure of the likelihood that some specific harm arising from an incident will occur.*

The project did not provide for new technologies to be developed, but concentrated on the evaluation of what is currently available. It surveyed the level of implementation of seismic and rock related risk assessment techniques in South African mines. Although seismicity as such was the main thrust of the audit, the important aspect of the vulnerability of the workplace to seismic driven ground motion and the potential for rockfalls were also covered. The techniques used at fourteen mines and two institutions have been documented.

In accordance with the above hazard and risk definitions, techniques have been recommended for the assessment of the seismic hazard as the maximum event magnitude expected, and the return period for a given magnitude or larger.

Having defined the hazard, the report describes the effect of coupling between the source and the working place, the risk related characteristics of such working place (support standards and effectiveness), as well as the exposure of people.

The tools used in terms of data acquisition and interpretation are described.

Recommendations to improve and standardise seismic risk assessment are given in the conclusion. A procedure for evaluating seismic risk is presented. This procedure is based on a combination of the best aspects of the various risk/hazard assessment techniques as gained from the interviews.

The management of seismic risk is a most important area for subsequent study and is not covered in this report. It is recommended that SIMRAC should consider a review of seismic risk management for future research.

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1 Seismic Risk Assessment – An Overview

1.1 Introduction

The primary output of GAP 608 is to identify the best practice for the application of the numerous methods used to quantify the potential for rock mass instability. The output also includes procedures for rock mass stability evaluation.

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 gold and platinum mines.

A subsequent addition to the originally proposed project was to include a survey of non-seismic risk assessment techniques as used in these mines.

At the outset 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 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 that seismic events which have potential to cause harm. A greater hazard will imply potential to cause greater harm. A study of what the maximum event magnitude in an area might be is typically a seismic hazard determination.

When parameters like probability or likelihood of the occurrence of harm are estimated, it becomes risk assessment.

Seismic hazard is defined often in the literature (for example in SIMRAC GAP 303, and in Chapter 2 of this report), as the 'probability that an event of a certain magnitude or larger will occur within a given time span within a given volume'. For the purposes of this report (other than in Chapter 2) this definition will be associated with seismic risk.

The question that poses itself now is what constitutes seismic risk and what is its impact on safety in mines.

This chapter addresses that question by briefly summarising current knowledge and practice, and provides links (or references) to the rest of the report and also to external references.

Every topic that is discussed also has an indication of the state of knowledge (and of technology if applicable), on that specific topic. A simple arbitrary rating is used. The rating is as follows:

Knowledge / technology level

- 5 excellent
- 4 good
- 3 adequate

- 2 poor
- 1 not existing

The aspects that constitute seismic risk, its impact on mine safety and inputs to risk assessment are:

- Seismic hazard
- Coupling between source and mine excavation
- The characteristics of the mine excavation
- Exposure
- Data acquisition for risk assessment
- Data interpretation for risk assessment

The report concludes with recommendations on how to improve seismic risk assessment and indicates the way towards a standardised approach.

Seismic risk assessment is a continual process in parallel with a strategy to manage the risk. The latter is not discussed in any detail in this report.

1.2 Seismic hazard

1.2.1 The seismic event

Knowledge level 3

For the purpose of this report, the researchers assumed that all seismicity in the South African mining region, is mining induced. This does not take away the importance of major geological structures or even 'locked-in' tectonic stresses.

The mechanism of failure of these events is equally not relevant to this report. However, what should be considered as relevant are the observations of event categories which are of a distinctly different and quantifiable nature. In SIMRAC GAP 303, Mendecki, et al (1997) observed different event populations in an Energy-Moment relation and also observed differences in the cumulative frequency magnitude. They interpreted this difference in the E-M relations in the context of system stiffness.

A further distinction and interpretation was that the smaller events could be associated with the larger E-M ratios and could also be linked with fracturing in the areas of extreme stress concentrations around the excavation. A similar argument is made by Ebrahim-Trollope and Glazer, (1995) who identified further categories of local to medium size events exist in the vicinity and in association with actual mining activity, and other events above magnitude 3.5 that are associated with regional structures and the past history of mining. They categorised events as being face driven, geological driven (local) and regional geologically driven. This may be an oversimplification but the development and application of risk assessment techniques should take cognisance of these distinctly different event categories.

1.2.2 Recognition of seismic hazard.

Knowledge level 3

The recognition of the hazards posed by seismicity can be quantified through seismic observations i.e. experience over time in a particular environment. That is by having had seismic information from a particular environment for a period of time.

The hazard of large events associated with major geological structures can also be inferred by knowledge of the structure and the mine layout. It can be recognised without having had prior seismic information, which then implies that it can be estimated even before mining starts in an area ie. a 'non-monitored' recognition of hazard.

Earlier work is the concept of Excess Shear Stress (ESS) and the work by ISS International (DeepMine task 5.2.1 and SIMRAC GAP 609) towards the numerical simulation of an asperity model on the potential plane of failure

Lasocki describes in Chapter 2 (of this report), the recommended techniques for determining the maximum magnitude (m_{max}) that may occur within a predetermined volume. The determination of m_{max} and mean return period is a classical approach towards seismic hazard quantification.

As mentioned earlier the combination of different seismic event categories in a seismogenic volume may result in a complex magnitude distribution function. Standard methods for determining m_{max} from a model dependent magnitude distribution function may lead to significant errors. Lasocki describes and recommends a non-parametric approach that will provide a much more reliable estimate of m_{max} .

Seismic hazard (as inferred from the Mines Safety Act), is based on parameters such as the maximum magnitude that may occur in an area and the b-value of the classic Gutenberg-Richter relation

$$\log n = a - b \cdot m, \quad (1.2.1)$$

Mendecki and van Aswegen (1997) introduced a concept of seismic hazard magnitude as

$$Hm = 2/3 \log \sum M - 6.1 \quad (1.2.2)$$

Where $\sum M$ is derived from the a- and b- values from equation (1.2.1).

Mendecki and Van Aswegen (1997) and Eneva et al (1998) also attempted to quantify seismic hazard as the volume of ground that would be subjected to the velocities of ground motion exceeding a certain pre-set value. These latter concepts are not discussed further in this report.

The Mean Return Period is a statistical measure of the expected return time for an event of a specific magnitude.

Lasocki (in Section 2.3)and Kijko et al (1999) also describe the estimated expected mean return time for an event of a specified magnitude or larger. They use a classical approach (a cumulative magnitude distribution function defined by the classic Gutenberg-Richter relation), and compares it with a non-parametric approach.

Mendecki and Van Aswegen (1998) motivate a simpler single number in their approach towards quantifying seismic hazard.

1.3 Coupling between the seismic source and mine excavation.

Knowledge level 2 to 3

A component of the seismic energy radiated from the source of the seismic event interacts with mine excavations. This energy can dynamically change conditions with respect to the stability of the immediate rock mass and require the support system to absorb such energy.

A common way of quantifying the potential for damage in an excavation or the performance specifications of the support system is to estimate the peak ground velocity at the excavation boundaries.

A proposed methodology by Spottiswoode is described in Chapter 3, where a model of peak ground velocities in near to far field is presented and applied to a data set from a deep level mine.

Factors determining the peak ground motion as suggested by McGarr (1991) are the source radius, static stress drop, modulus of rigidity, shear wave velocity and the hypocenter distance to the source. The radiation pattern for S-waves is also of importance. For risk assessment a most conservative assumption is used to specify the radiation pattern.

Excellent work by Cichowicz (1997) is also of relevance here. His studies showed that damage is not only controlled by the magnitude and distance between source and mine excavation but also other factors. Very large stress drop events (with smaller source area for similar magnitudes) cause more damage.

Cichowicz observed that small peak ground velocities, down to 0.005 m/s could cause damage.

Mining in a shaft pillar in the Far West Rand, with the shaft in operation, resulted in a practical application of peak ground velocity (Handley, 2000). In this case Matthew Handley undertook a risk assessment approach to estimate the probability of a large seismic event damaging the shaft whilst people were travelling in the cage. His estimate included time of day in terms of exposure and the probability of a seismic event's region of co-seismic inelastic deformation affecting the shaft. He made some conservative assumptions such as assuming a spherical source volume and also that the parameter, Apparent Volume, corresponds directly with the source volume.

This was a practical and meaningful application of a risk assessment philosophy to assess the probability of significant accident in the shaft, resulting from seismicity.

1.4 Characteristics of excavation

The immediate environment around a mine excavation and the support performance defines the response of the excavation to seismic waves.

There is also the situation where the immediate, highly stressed area close to the free surface may be the source of the event. Face bursting may be associated with a small amount of energy, but the excavation is part of the failure, therefore part of the source.

The parameters influencing the contribution made by mine excavation towards seismic risk are:

- Local stress level
- Site effect amplification of ground velocity
- Support
- Local ground condition

1.4.1 Local stress distribution

Knowledge level 3

The effect of the local stress regime may be modelled effectively using ERR. (Spottiswoode, 2000). The correlation between increased ERR and increased seismic risk is not exclusive, but is a valid input in a risk assessment.

Mining rate and time dependant closure and its implications with respect to hazard identification is considered in Chapter 4. Evaluation of the sensitivity of risk assessment to FULCO mining has been undertaken. The emphasis was on the numerical modelling of the effect of blasting everyday on the risk of rockbursting and/or poor ground conditions. Suitable seismic data recorded in areas with FULCO mining in the true sense of the word has not been obtained as yet. Figure 1.4.1 and Figure 1.4.2 shows the effect of mining rate on the steady state closure and on the major principle stress ahead of the face, respectively. Early indications are that the effect of FULCO and mining rate will primarily be seen in the local risk conditions, as in face bursting or possible poor ground condition. Relatively remote or regional seismicity on major structures will not be affected, assuming that the required normalisation for production volume is undertaken.

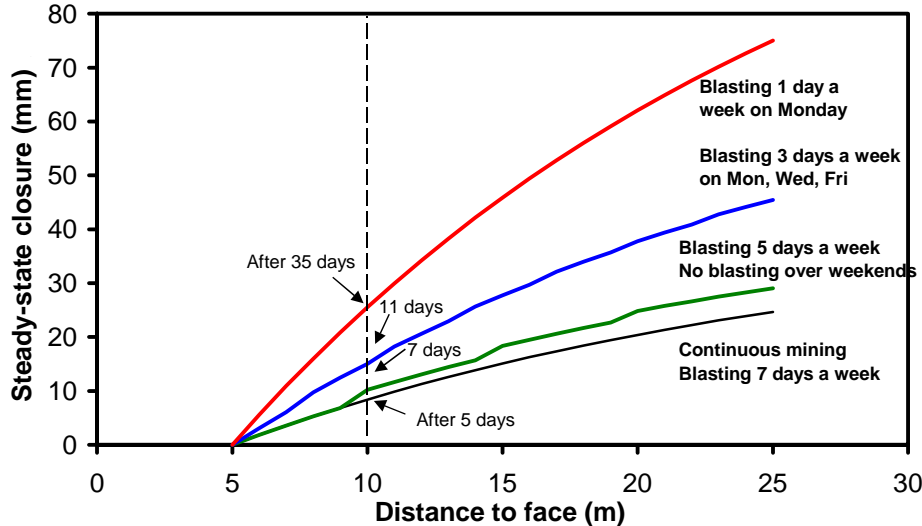


Figure 1.4.1 The effect of mining rate on the total amount of steady-state closure at a point in the stope

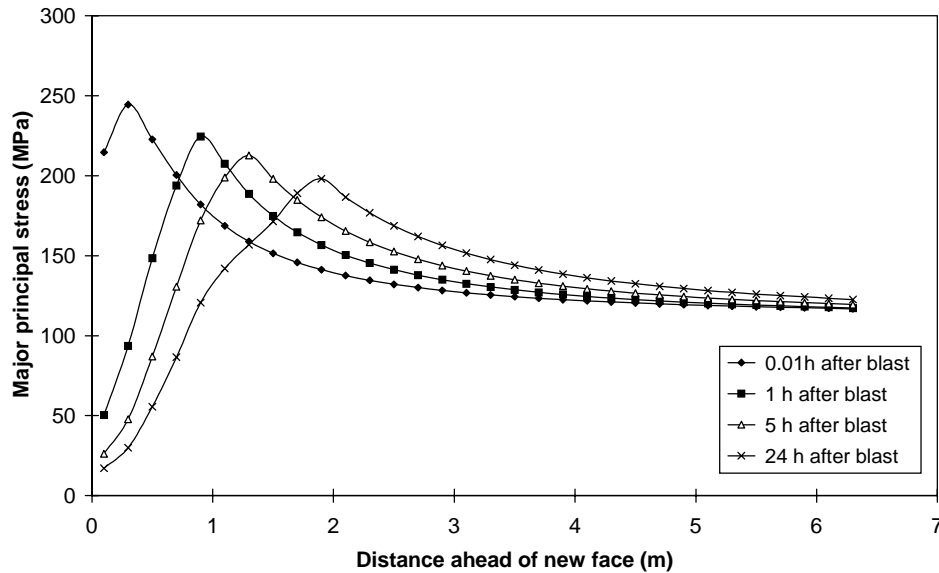


Figure 1.4.2 The effect of time rate on the major principle stress ahead of the face

1.4.2 Site effect amplification

Knowledge level 2

Site effect amplification of the peak ground velocity takes place in the fractured area around the excavation. Hagan et al (1999) looked at a blast simulated rock burst and observed an increased amplification from distant events.

At this stage the lack of understanding of site effect amplification and the unquantifiable degree of site effect amplification, do not allow for site effect to be an input in seismic risk assessment.

1.4.3 Support behaviour and standards of installation

Knowledge level 4

The specification, installation and performance of in-stope and tunnel support have a significant influence on the level of risk to seismicity. Numerous rating systems are in place in South African gold and platinum mines and are detailed in Chapter 7.

1.4.4 Local ground condition

Knowledge level 4

The vulnerability of the excavation to a seismic wave also depends on the local ground condition parameters such as density and orientation of fracturing, and in general the characteristics of the applicable geotechnical area.

1.4.5 Exposure of people

Knowledge level 5

The degree of exposure of people to seismic hazard relates directly to seismic risk. A good example is the earlier mentioned exercise by Handley (2000) who looked at a time density distribution of people in vertical transport. The time distribution of large events was also considered.

The same applies to underground personnel in horizontal transport and at their respective working places.

FULCO and continuous mining is also relevant, to the extent that the time distribution function of people underground and the seismic event time distribution, are significantly different. This may provide a much worsened risk rating.

Time distribution data for both the local seismicity and that of the underground staff at any working place or in transit should be readily available.

The input of these to risk assessment is also described in Chapter 7.

1.5 Data acquisition

1.5.1 Seismic monitoring

Knowledge level 4, Technology level 4

Seismic risk assessment is only possible with the appropriate information to recognise the seismic hazard and to rate the respective contributions of all other parameters influencing the risk. In particular, seismic monitoring up to a certain minimum standard is a prerequisite. These minimum standards include quality acquisition of data, processing and interpretation up to a minimum level.

The definition of this minimum level is not obvious. Mendecki, et al (1999) described the characteristics of seismic monitoring systems. The authors agree with Mendecki's description but wish to point out that the minimum system requirements to achieve a practical and even optimum level of monitoring for risk assessment have not been determined. The authors remain unconvinced that the concept of continuous monitoring will provide a practical input in risk assessment.

Seismic monitoring is more than simply locating events. It was proven in the South African seismic monitoring applications in deep gold mining, that attempts towards the quantification of the response of the rock mass to mining, provided a significant input towards risk assessment.

The relevance of sensor type, sensor density and sensitivity, as well as data quality, are well described by Mendecki, et al (1999)

1.5.2 Mine layout/design

Knowledge level 3

Other inputs to the basic information required for risk assessment are a rating based on mine layout with respect to major geological structures, pillars, remnants and a rating based on ERR (or another measure of the stress distribution).

1.5.3 Panel characteristics

Knowledge level 4

The audit process on panel/stope characteristics including support is described in Chapter 7.

Support includes the appropriateness of the support strategy and the adherence to mine and industry standards.

1.5.4 Experience reference

Knowledge level 2

Ideally every input into in seismic risk assessment should be quantifiable, however a large percentage of the input ratings are obtained subjectively. This implies that an experience base is a prerequisite for any 'meaningful' subjective rating.

An appropriate experience base should exist in a structured, documented and archived format. The researchers in this project found that this is almost non-existent.

The proposed design of a seismic early warning database is given in Charter 6.

A similar argument can be made for an effective accident database and a seismic damage database. An interaction of these datasets should be achieved through implementation in a relational database management system.

1.6 Data interpretation / visualisation

Assuming appropriate and sufficient data and a solid experience base, the final input into seismic risk assessment is an interpretation/ visualisation tool. Some aspects of these tools are discussed.

1.6.1 Seismic early warning

Knowledge level 3 Technology level 4

A substantial research investment is being made in South Africa to understand the physical processes, the development and evaluation of early warning (or prediction) concepts. The most complete references are SIMRAC GAP017, GAP303, Mendecki (1997), and Mendecki (1999).

One of the main objectives of this project was to evaluate the effectiveness of the applied concepts and the tools use for seismic early warning in the industry. To some extent the researchers failed because of a total lack of an experience base (see previous section). Operators at the mines have impressive case studies but for the industry as a whole no definitive statement on issues like success rate can be made.

A review of the state of seismic prediction was done by Spottiswoode and De Beer (1999). No routine practising of short-term prediction on any of the South African mines could be found.

Where used, seismic early warning is primarily based on the following parameters:

- Parameters related to a strain rate (event rate, cumulative Apparent Volume)
- Parameters related to inferred stress (stress drop, Energy Index)
- Parameters relating to the statistical properties of the seismic deformation process.

These parameters are extracted from seismic moment, energy, time and distance between events.

The use of early warning concepts may have rightly or wrongly be discredited in a stand-alone mode, but this does not detract from their importance and validity in seismic risk assessment.

Stewart and Spottiswoode (1993, 1996) developed tools to consider the case of independent parameters that could each provide better than random warning. Finnie (1999) questioned whether any better than random performance can be achieved with prediction, especially when the time window is up to 10 days or more.

1.6.2 Integration of seismic monitoring and modelling.

Knowledge level 3 Technology level 3

Integration of monitoring and modelling is recognised as an area with great potential for enhancing the ability to identify seismic hazard. Substantial progress in integrating monitoring and modelling is reflected in SIMRAC GAP603 and Deepmine task 5.2.1

1.6.3 Interpretation skills

Knowledge level 2

Arguments for a high level of quantification of seismic information are valid, but the practice of seismic early warning is purely heuristic. The operator relies on his prior experience, benchmark case studies and his knowledge of a particular area, to come up with a warning (or level of warning). This leads to the importance of on-site interpretation, bi-direction communication with management and production officials.

Other issues such as the interpretation time window and response time have to be considered. The observed different categories (from face bursting to regional seismicity) will have different optima in terms of interpretation time window and decision-making response time.

This then also implies that the 'sampling rate' for the inputs to a risk assessment rating must correlate with the fastest rate of change of any of the input parameters.

1.7 Risk management

Seismic risk management (as opposed to risk assessment) does not fall into the scope of this project. It is recognised, however, that to complete the management loop from monitoring/data acquisition, hazard identification, and risk assessment, a strategy for the management of seismic risk has to be formulated.

The concept of levels of warning developed by ISSI and in use at Freegold is a step in this direction. Dr. van Aswegen provided an example of the procedure used at Freegold (Appendix to this chapter):

1.8 References

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1.9 Appendix :

An example of the concept of levels of warning as developed by ISS International and used at Freegold. (A description of the terminology is given by Mendecki et al (1999))

Current status of Freegold Seismic Monitoring

Alert System

Alert Criteria

Alert1

- Anomalous spatial seismicity pattern observed in a mining area:
- high stress
- high strain rate
- high gradient in seismic flow

Alert2

- Time history analysis shows one or more of the following:
- a significant increase in rate of ΣV_a
- a significant change in level of $\log(EI)$
- a significant drop in Schmidt number
- occurrence of event(s) mag. > 1.7 (frg local magnitude)

Alert3

- Anomalous temporal seismicity pattern, with one or more of the conditions specified for Alert2 and/or two or more of the following:
- significant increase in rate of ΣV_a
- change in $\log(EI) > .25$
- drop in Schmidt number > 1 order

Alarm:

Conditions as specified for a particular working place, strongly suggesting the occurrence of a significant seismic event within hours.

Suggestions as to action taken by mines under the different levels of warning include:

Alert1: Rock Engineering Department investigates mine layout and geological structure to assess risk and recommend action (e.g. adapt face shapes, slow down mining).

Alert2: Support systems in stopes are checked a.s.a.p. Minimize work force, especially in traditionally hazardous places. Apply maximum safety precautions.

Alert3: Keep shift out of indicated areas until further notice.

Alarm: Evacuate areas indicated and stay out until further notice.

Procedure

The responsible Rock Engineer is contacted and informed of an alert. Alerts are also e-mailed, and a complete log of all alerts issued is kept automatically.

Alerts are reviewed as follows:

- Alert1 - weekly
- Alert2 - daily
- Alert3 - three times per day
- Alarm - hourly

After an alert has been reviewed, the alert can either be confirmed, or it can be cleared when significant recovery has taken place. On clearing an alert, the seismicity of the area is being evaluated for the alert period, and notes are made regarding the largest events, number of large events, etc.

Daily Monitoring

Contour plots

Contour plots are generated of $\log(EI)$, which will indicate areas of high stress, and Inverse Deborah Number, which gives an indication of possible seismic softening of certain areas. From these plots the hazard of an area can be evaluated. In general, if an area of higher than normal hazard is identified from the contour plots, it can be considered to issue an Alert 1.

Anomalous seismic behaviour

The whole area is monitored to identify anomalous seismic behaviour. A check is done for changes of seismicity patterns (e.g. clustering, or unexpected large events). Such areas are usually brought under the attention of rock engineering personnel by phone. An Alert 1 may then be issued.

Time history analysis before night shift

Time histories of all the polygons are done. ΣV_a is plotted against $\log(EI)$. A significant increase in ΣV_a is an indication of an increase in the rate of deformation, while a drop in $\log(EI)$ indicates an area where energy is accumulating. Possible unstable areas can therefore be

identified, and an Alert 2, 3 or an alarm can be considered. The alert is discussed with rock engineering personnel by phone.

Checklists

Checklists are completed daily, and forwarded to rock engineering staff. This compiles the analysis as discussed above.

Hourly Monitoring

Any area that is being mined extensively or that is of particular interest, can be monitored on an hourly basis. In this case a time history (of ΣV_a and $\log(EI)$) will be produced hourly (in hardcopy), for continuous attention.

Weekly Monitoring

Contour plots for a full week are supplied to rock engineering personnel at the end of a week. The daily checklists are also forwarded, for the record of the rock-engineering department; it also serves as a summary of the seismicity of particular areas. Seismic behaviour can therefore be compared over time, and different areas can also be compared.

24-hour service

Personnel are on duty 24 hours a day. Any large events are reported immediately to rock engineering staff. They are also available for any queries regarding events. Events are also processed continuously as they are recorded.

2 Analysis of Currently Used Probabilistic Techniques and Recommended Approach

S. Lasocki

2.1 Summary

Seismic hazard analysis methods in mines are reviewed for the purpose of selecting the best technique that can be used in South African mines. To achieve this goal, the most often used hazard analysis procedure, which is based on the classical frequency-magnitude Gutenberg-Richter relation, as well as alternative procedures are investigated.

Since the maximum regional seismic event magnitude m_{\max} is of paramount importance in seismic hazard analysis, this chapter provides a generic formula for the evaluation of this important parameter. The formula is capable of generating solutions in different forms, depending on the assumptions of the model and/or the available information about past seismicity. It includes the cases (i) when seismic event magnitudes are distributed according to the truncated frequency-magnitude Gutenberg-Richter relation, (ii) when the empirical magnitude distribution deviates moderately from the Gutenberg-Richter model, and (iii) when no specific model of the magnitude distribution is assumed.

Both synthetic, Monte-Carlo simulated seismic event catalogues, and actual data from two deep gold mines in South Africa, are used to demonstrate the discussed hazard analysis techniques.

Our studies show that the **non-parametric** technique, which is independent of the assumed model of the distribution of magnitude, provides an appropriate and **recommended** tool for seismic hazard assessment in mines where the magnitude distribution can be very complex.

2.2 Introduction

Probabilistic seismic hazard analysis is a standard tool used in mines to determine probabilities of occurrence of seismic events that could have an impact on production and/or could be dangerous to underground staff. Such types of analyses had their origin in techniques used in tectonic earthquake seismology (e.g. Cornell, 1968; Mcquire, 1993). However, demands for an increased accuracy in inference have required changes in the standard approaches so as to account for the specific features of mine-induced seismicity. This includes non-stationarity of the seismic event generation processes. Temporal variation of factors controlling seismicity results, in turn, in temporal variation of seismic hazard in the course of mining. Adaptation of stationary hazard analysis to time-variability in mines has led to techniques that allow the assessment of seismic hazard, locally in time and space, in order to monitor present and to foresee future states of rock mass fracturing (e.g., Lasocki, 1993a,b; Stewart and Spottiswoode, 1993; Gibowicz and Kijko, 1994; Trifu et al., 1997).

In this chapter seismic hazard analysis procedures are reviewed for the purpose of selecting the best technique which can be used in South African mines. Based on both simulated and

actual data from deep gold mines in South Africa, it has been shown that the most often used procedure for seismic hazard analysis, which is based on the classical frequency-magnitude Gutenberg-Richter relation, is not optimal and should be replaced by a procedure which is non-parametric and independent of the assumed model of the distribution of magnitude. Since in the seismic hazard analysis the maximum possible seismic event magnitude m_{\max} is of paramount importance, several techniques are presented that can be used for the evaluation of this important parameter.

2.3 Seismic hazard in mines. Definition.

At present there is no generally accepted definition of seismic hazard in mines. In this chapter seismic hazard in mines is defined as: **the probability that the specified value of seismic event magnitude M_p will be exceeded in the next t time units**, and is given by

$$Pr(M_p, t) = 1 - [F(M_p)]^{\lambda t}. \quad (2.3.1)$$

Also, it has become common practice to express seismic hazard in terms of the mean return period. The **mean return period** of events of magnitude M_p and larger is defined as

$$Rp(M_p) = \{\lambda [1 - F(M_p)]\}^{-1} \quad (2.3.2)$$

In both above equations, $F(m)$ denotes the cumulative distribution function (CDF) of seismic event magnitude and λ is the **mean activity rate**. By definition, $\lambda = n/T$, where n is the number of events that occurred having magnitudes greater than or equal to a known level of completeness m_{\min} , and T is the span of the catalogue.

From the above definition of seismic hazard it is apparent that the assessment thereof mainly involves the determination of the CDF of event magnitude. Once this function has been determined, the calculation of seismic hazard is straightforward according to equation (2.3.1).

2.4 Assessment of seismic hazard in mines – a classical approach.

It is clear from equations (2.3.1 and (2.3.2) that the selection of the proper functional form of $F(m)$ the cumulative distribution function of seismic event magnitudes and knowledge of its parameters plays a crucial role in estimating seismic hazard. The most often used formula for the CDF of seismic event magnitudes has its source in the classical Gutenberg-Richter relation

$$\log n = a - b \cdot m \quad (2.4.1)$$

where $n(m)$ is the number of events not less than magnitude m and a and b are parameters. A high b value indicates that a small fraction of the total number of seismic events in the region of interest have high magnitudes, whereas a low b value implies a large fraction of high magnitude events. Establishing the b value for a region can be important for assessing seismic hazard since damage or rock mass weakening are often associated with high levels of ground motion that occur as a result of high magnitude seismic events.

The assumption that the magnitudes of seismic events follow the frequency-magnitude Gutenberg-Richter relation(2.4.1), is equivalently expressed by the CDF, $F(m)$, of the form (Aki, 1965; Utsu, 1965)

$$F(m) = 1 - \exp[-\beta(m - m_{\min})] \quad (2.4.2)$$

where magnitude m is considered as a continuous variable that may assume any value above the threshold value m_{\min} , and $\beta = b \ln(10)$. Aki (1965) showed that the maximum likelihood estimator of the parameter β has the simple form

$$\hat{\beta} = \frac{1}{\bar{m} - m_{\min}} \quad (2.4.3)$$

where \bar{m} is the sample mean magnitude equal to $\sum_{i=1}^N m_i / n$ and n is number of main events that occurred with a magnitude greater than or equal to the level of completeness m_{\min} . Since its first derivation in 1965, the Aki-Utsu formula (2.4.3) has been successfully used in a great number of studies in which entirely different patterns of seismicity were investigated. **This approach has, nevertheless, several significant shortcomings since the assumption is made that magnitudes are unbounded from the top.** The maximum likelihood estimate of β for continuous magnitudes between m_{\min} and an upper limit m_{\max} was derived for the first time by Page (1968). It is easy to show that if an upper limit of magnitude is taken into account, the CDF $F(m)$ takes the form

$$F(m) = \begin{cases} 0, & \text{for } m < m_{\min} \\ \frac{1 - \exp[-\beta(m - m_{\min})]}{1 - \exp[-\beta(m_{\max} - m_{\min})]} & \text{for } m_{\min} \leq m \leq m_{\max} \\ 1, & \text{for } m > m_{\max} \end{cases} \quad (2.4.4)$$

and the maximum likelihood estimator of the β parameter can be obtained from the solution of the equation

$$\frac{1}{\beta} = \bar{m} - m_{\min} + \frac{(m_{\max} - m_{\min}) \exp[-\beta(m_{\max} - m_{\min})]}{1 - \exp[-\beta(m_{\max} - m_{\min})]} \quad (2.4.5)$$

where m_{\max} is the unknown upper limit of magnitude. The parameter m_{\max} is characteristic for specific conditions of rock mass fracturing and has to be estimated.

The exact evaluation of β from equation (2.4.5) requires knowledge of m_{\min} and m_{\max} , and can be obtained only by recursive solutions. Nevertheless, a simple approximation of β is possible. With accuracy to the second term of the Taylor expansion of (2.4.5), the $\hat{\beta}$ value becomes (Gibowicz and Kijko, 1994)

$$\hat{\beta} = \hat{\beta}_0(1 - \kappa_{\max}) \quad (2.4.6)$$

where

$$\kappa_{\max} = \hat{\beta}_0 \frac{(m_{\max} - m_{\min}) \exp[-\hat{\beta}_0(m_{\max} - m_{\min})]}{1 - \exp[-\hat{\beta}_0(m_{\max} - m_{\min})]} \quad (2.4.7)$$

and $\hat{\beta}_0$ is the Aki-Utsu estimator (2.4.3).

However, there is a significant number of cases reported where the observed earthquake frequency patterns differ significantly from the Gutenberg-Richter relationship (e.g. Pacheco et al., 1992; Singh et al., 1983; Taylor et al., 1990; Trifu and Radulian, 1991; Umino and Sacks, 1993; Wesnousky, 1994). The evidence for a multi-componental, or in general, non-linear structure of the empirical log-frequency-magnitude distributions is much more substantial in mine-induced seismicity than in natural seismicity (e.g. Dessokey, 1984; Kijko et al., 1987; Johnston and Einstein, 1990; Young et al., 1992; Trifu et al., 1993; Finnie, 1994; Gibowicz and Kijko, 1994; Feustel, 1997; Lasocki and Weglarczyk, 1998; Kijko et al., 1998). Often, these empirical distributions are far more complex than might be expected from the common models used in earthquake seismology, in which some complications such as truncation, (Page, 1968), the randomness of the distribution parameters (Campbell, 1982), and the presence of characteristic earthquakes (Schwartz and Coppersmith, 1984), have been addressed.

In the following section an alternative, and recommended, procedure of seismic hazard assessment in mines will be presented. In this procedure, in place of a specific parametric model of magnitude distribution $F(m)$, a **non-parametric** kernel density estimator of unknown $F(m)$ is used.

2.5 Assessment of seismic hazard in mines - alternative approach.

The procedures derived in the previous sections are parametric and are applicable when the empirical frequency-magnitude graph for the seismic series exhibits apparent linearity, starting from a certain m_{\min} value. However, many studies of seismicity show that, in some cases, (i) the empirical distributions of earthquake magnitudes are of bi- or multi-modal character, (ii) the frequency-magnitude relation has a strong non-linear component or (iii) the presence of "characteristic" events (Schwartz and Coppersmith, 1984) is evident. There are, by way of illustration, some well-documented cases of such deviations and they include the natural seismicity in Alaska (Devison and Scholz, 1984), Italy (Molchan et al., 1997), Mexico (Singh et al., 1983), Japan (Wesnousky et al., 1983), and the United States (Main and Burton, 1984b; Weimer and Wyss, 1997), as well as mine-induced seismicity in the former Czechoslovakia, in Poland and in South Africa (Finnie, 1994; Gibowicz and Kijko, 1994).

In order to use the seismic hazard generic formula (2.3.1) in such cases, the analytical, parametric models of the frequency-magnitude distributions should be replaced by their non-parametric counterparts.

The non-parametric estimation of a probability density function (PDF) is an approach that deals with the direct summation of the kernel functions using sample data. Given the sample

data $m_i, i = 1, \dots, n$ and the kernel function $K(\bullet)$, the kernel estimator $\hat{f}(m)$ of an actual, and unknown PDF $f(m)$ is

$$\hat{f}(m) = \frac{1}{nh} \sum_{i=1}^n K\left(\frac{m - m_i}{h}\right) \quad (2.5.1)$$

where h is a positive smoothing factor (Rosenblatt, 1956; Parzen, 1962). The kernel function $K(\bullet)$ is a PDF symmetric about zero and the specific choice of it is not so important for the performance of the method; many unimodal distribution functions ensure similar efficiencies. In this work the Gaussian kernel function,

$$K(\xi) = \frac{1}{\sqrt{2\pi}} \exp(-\xi^2/2) \quad (2.5.2)$$

was used. However, the choice of the smoothing factor h is crucial because it affects the trade-off between random and systematic errors. Several procedures exist for the estimation of the value of this parameter, none of them being distinctly better for all varieties of real data (Silverman, 1986). For purposes of this report the least-squares cross-validation (Rudemo, 1982; Hall, 1983; Bowman, 1984; Bowman et al., 1984; Stone, 1984) was used. The details of the procedure are given in the Appendix to this chapter. Fortunately, in the application of the non-parametric estimation procedure, the integration of the CDF is not strongly affected by the accuracy of h . The tests show that the final estimates of hazard obtained when the optimal value of h is used do not differ much from those achieved in the case of a reasonable guess of h .

Following the functional form of our kernel (2.5.2) and the fact that the data come from a finite interval $[m_{\min}, m_{\max}]$, the respective estimators of PDF and CDF of seismic event magnitude are (Kijko et al., 1999).

$$\hat{f}(m) = \begin{cases} 0, & \text{for } m < m_{\min} \\ \frac{(h\sqrt{2\pi})^{-1} \sum_{i=1}^n \exp\left[-0.5\left(\frac{m - m_i}{h}\right)^2\right]}{\sum_{i=1}^n \left[\Phi\left(\frac{m_{\max}^{obs} - m_i}{h}\right) - \Phi\left(\frac{m_{\min} - m_i}{h}\right) \right]}, & \text{for } m_{\min} \leq m \leq m_{\max}^{obs} \\ 0, & \text{for } m > m_{\max}^{obs} \end{cases} \quad (2.5.3)$$

and

$$\hat{F}(m) = \begin{cases} 0, & \text{for } m < m_{\min} \\ \frac{\sum_{i=1}^n \left[\Phi\left(\frac{m - m_i}{h}\right) - \Phi\left(\frac{m_{\min} - m_i}{h}\right) \right]}{\sum_{i=1}^n \left[\Phi\left(\frac{m_{\max}^{obs} - m_i}{h}\right) - \Phi\left(\frac{m_{\min} - m_i}{h}\right) \right]}, & \text{for } m_{\min} \leq m \leq m_{\max}^{obs} \\ 1, & \text{for } m > m_{\max}^{obs} \end{cases} \quad (2.5.4)$$

where $\Phi(\xi)$ denotes the standard Gaussian cumulative distribution function.

Despite its flexibility, such a model-free technique as above has only occasionally been used in seismology. One of the first uses was in the estimation of the conditional failure rates from successive recurrence times of micro-earthquakes (Rice, 1975). The non-parametric CDF of seismic event occurrence time was also employed by Sólnes et al. (1994). Another application involved the estimation of spatial distribution of seismic sources (Woo, 1996; Bommer et al., 1997; Jackson and Kagan, 1999, and the references therein) and the non-parametric estimation of epicenter migration in seismic series induced by mining (Lasocki et al., 1997; Lasocki and Idziak, 1998). Some possibilities of this technique for assessing time variations of magnitude distribution in mines were also presented in Lasocki and Weglarczyk, (1998).

In this work the application of the above non-parametric Gaussian (N-P-G) technique for the estimation of the CDF of seismic event magnitudes and their application in the generic formula of seismic hazard (2.3.1) is demonstrated.

Since the maximum regional seismic event magnitude m_{\max} is of paramount importance in both the classical and alternative seismic hazard assessment techniques, the following section presents several techniques that can be used for evaluation of this important parameter.

2.6 Assessment of m_{\max}

To avoid confusion about the terminology, it is to be agreed that m_{\max} , being the magnitude of the largest possible seismic event, is defined as the **upper limit of magnitude for a given region**. Also, synonymous to the largest possible event magnitude, is the **maximum regional magnitude**, which is the **largest event that can be expected to occur in a specified region**.

The value of maximum magnitude so defined is the same as that used by many earthquake engineers (EERI Committee, 1984) and complies with the meaning of this parameter as used by e.g. Hamilton (1967), Page (1968), Cosentino et al. (1977), the Working Group on California Earthquake Probabilities (WGCEP, 1995), Stein and Hanks (1998), and Field et al. (1999). This terminology assumes a sharp cut-off magnitude at a maximum magnitude m_{\max} , such that, by definition, no earthquakes are to be expected with magnitude exceeding m_{\max} . Cognizance should be taken of the fact that an alternative, “soft” cut-off maximum earthquake magnitude is also in use (Main and Burton, 1984a; Kagan, 1991). The later formalism is based on the assumption that seismic moments follow the Gamma distribution. One of the distribution parameters is also called the maximum seismic moment and the corresponding value of earthquake magnitude is called the “soft” maximum magnitude. Beyond the value of this maximum magnitude, the distribution decays much faster than the classical Gutenberg-Richter relation. However, this means that a “soft” cut-off is envisaged since earthquakes with magnitudes larger than such a maximum magnitude are not excluded. This model with the “soft” maximum magnitude has been used by its authors (Kagan, 1994; 1997; Main 1996; Main et al., 1999) and their followers (e.g. Sornette and Sornette, 1999). It must be noted that in this work only a model having a sharp cut-off of maximum magnitude is considered.

Although a knowledge of the value of the maximum regional magnitude m_{\max} is required in many engineering applications, it is striking how little has been done in developing appropriate techniques for an estimation of this parameter. At present there is no generally accepted method for estimating the value of m_{\max} . The current methods for the evaluation fall into two main categories: deterministic and probabilistic.

The deterministic procedure most often applied is based on the empirical relationships between the magnitude and various tectonic and fault parameters. There are several research efforts devoted to the investigation of such relationships. The relationships are different for different seismic areas and different types of faults (Wells and Coppersmith, 1994; Anderson et al., 1996, and the references therein). Another class of deterministic procedures for maximum regional magnitude was developed in the late sixties, and is based on the extrapolation of the classical, log-linear, frequency-magnitude Gutenberg-Richter relation. Among earthquake engineers, the best known is probably the extrapolation procedure as applied recently e.g. by Frohlich (1998), and the “probabilistic” extrapolation procedure applied by Nuttli (1981), in which the frequency-magnitude curve is truncated at the specified value of annual probability of exceedance (e.g. 0.001). As an alternative to the two techniques above, researchers often try to relate the value of m_{\max} to the strain rate or the rate of seismic-moment release (Papastamatiou, 1980; Anderson and Luco, 1983; WGCEP, 1995; Stein and Hanks, 1998 and Field et. al., 1999). Such an approach has also been applied in evaluating the maximum possible magnitude of seismic events induced by mining (e.g. McGarr, 1984). In most cases, unfortunately, the uncertainty of the value of the parameter m_{\max} determined by means of any deterministic procedure is large, often reaching a value of the order of one unit on the Richter scale.

The value of m_{\max} can also be estimated purely on the basis of the seismological history of the area, viz. by using seismic event catalogues and an appropriate statistical estimation procedure. The statistical techniques falling into this category are used in a significant class of problems dealing with extreme values of random variables. The statistical theory of extreme values was known and well developed by the forties, and was applied in seismology as early as 1945 (e.g. Nordquist, 1945). The appropriate statistical tools required for the estimation of the end-point of distribution functions were developed later (e.g. Robson and Whitlock, 1964; Woodroffe, 1972, 1974; Weiss and Wolfowitz, 1973; Hall, 1982) and used in estimating maximum regional magnitude from the eighties only (Dargahi-Noubary, 1983; Kijko and Sellevoll, 1989, 1992; Pisarenko, 1991; Pisarenko et al., 1996).

Also, a very interesting, alternative procedure for the estimation of m_{\max} was described in a recent paper by Ward (1997). Ward’s **computer simulations** of the largest earthquake are very impressive and convincing. Nevertheless, one must realize that all the quantitative assessments given by Ward (1997) are based on the particular model assumed for the rupture process, on the postulated parameters of the strength of the faults and on the configuration of the faults. It is therefore natural to ask, “Is it possible to develop an alternative approach, which has the potential to be free from any subjective assumptions and which is only driven by seismic data?”

The purpose of this section is to provide such a procedure for the evaluation of m_{\max} . The procedure is generic (and therefore very flexible), and is capable of generating solutions in different forms, depending on the assumptions and/or on the information available about past seismicity. The procedure can be applied in the extreme case when no information about the nature of the earthquake magnitude distribution is available, i.e. the procedure is capable of generating a formula for m_{\max} , which is independent of the particular frequency-magnitude distribution assumed. The procedure can also be used when the earthquake catalogue is incomplete, i.e. when only a few of the largest magnitudes are available.

2.6.1 Maximum regional magnitude m_{\max} - Generic formula.

Suppose that in the area of concern, within a specified time interval T , all n of the main earthquakes that occurred with a magnitude greater than or equal to m_{\min} are recorded. Let us assume that the value of the magnitude m_{\min} is known and is denoted as the threshold of completeness. We assume further that the magnitudes are independent, identically distributed, random values with cumulative distribution function $F(m)$. The parameter m_{\max} is

the upper limit of the range of magnitudes and thus termed the unknown maximum regional magnitude, which is to be estimated. Let us assume that all n recorded magnitudes are ordered in ascending order, i.e. $m_1 \leq m_2 \leq \dots \leq m_n$. One observe that m_n is the largest observed magnitude (denoted also as m_{\max}^{obs}) and has a CDF

$$G(m) = \begin{cases} 0, & \text{for } m < m_{\min} \\ [F(m)]^n & \text{for } m_{\min} \leq m \leq m_{\max} \\ 1, & \text{for } m > m_{\max} \end{cases} \quad (2.6.1)$$

After integrating by parts, the expected value of m_n , $E(M_n)$ is

$$E(M_n) = \int_{m_{\min}}^{m_{\max}} m dG(m) = m_{\max} - \int_{m_{\min}}^{m_{\max}} G(m) dm \quad (2.6.2)$$

Hence

$$m_{\max} = E(M_n) + \int_{m_{\min}}^{m_{\max}} [F(m)]^n dm \quad (2.6.3)$$

This expression, after replacement of the expected value of the largest observed magnitude, $E(M_n)$, by the largest magnitude already observed, m_{\max}^{obs} , suggests an estimator of m_{\max} of the form

$$\hat{m}_{\max} = m_{\max}^{obs} + \int_{m_{\min}}^{m_{\max}^{obs}} [F(m)]^n dm. \quad (2.6.4)$$

Cooke (1979) was probably the first to obtain the above estimator of the upper bound of a random variable. If applied to the assessment of the maximum regional magnitude, equation (2.6.4) says that the maximum regional magnitude m_{\max} is equal to the largest magnitude

already observed, m_{\max}^{obs} , increased by an amount $\Delta = \int_{m_{\min}}^{m_{\max}^{obs}} [F(m)]^n dm$. Estimator (2.6.4) is, by

its nature, very general and has several interesting properties. For example, it is valid for each CDF, $F(m)$, and does not require the fulfillment of any additional conditions. It may also be used when the exact number of earthquakes, n , is not known. In this case, the number of earthquakes can be replaced by λT . Such a replacement is equivalent to the assumption that the earthquakes occurring in unit time, conform to a Poisson distribution with parameter λ , with T the span of the seismic catalogue. It is also important to note that, since the value of the integral Δ is never negative, formula (2.6.4) provides a value of \hat{m}_{\max} which is never less than the largest magnitude already observed. Of course, the drawback of the formula is that it requires integration. For some of the magnitude distribution functions the analytical expression for the integral does not exist or, if it does, requires awkward calculations. The drawback mentioned above need not, however, be a real hindrance, since numerical integration with today's high-speed PC's is both very fast and accurate. Estimator (2.6.4) will be called the **generic formula** for the estimation of m_{\max} .

In the following section we will demonstrate how the generic formula (2.6.4) can be used in the assessment of the maximum regional magnitude m_{\max} in the different circumstances that a mine seismologist might face in real life. The cases considered include the following:

the seismic events magnitudes are distributed according to the doubly truncated Gutenberg-Richter relation,

the empirical magnitude distribution deviates moderately from the Gutenberg-Richter model,

no specific model of the magnitude distribution is assumed.

2.6.2 Case I: Application of the generic formula to the Gutenberg-Richter magnitude distribution.

In this section we will demonstrate how to apply the generic formula (2.6.4) to one of the most often used frequency-magnitude relationships, the one known as the Gutenberg-Richter magnitude distribution.

For the classical frequency-magnitude Gutenberg-Richter relation, the respective CDF of magnitudes is described by equation (2.4.4). Following the generic formula (2.6.4), the estimator of m_{\max} requires the calculation of the integral

$$\Delta = \int_{m_{\min}}^{m_{\max}^{obs}} \left[\frac{1 - \exp[-\beta(m - m_{\min})]}{1 - \exp[-\beta(m_{\max}^{obs} - m_{\min})]} \right]^n dm \quad (2.6.5)$$

an integral which does not have a simple solution. It can be shown that an approximate, straightforward estimator of m_{\max} can be obtained through an application of Cramér's approximation. According to Cramér (1961), for large n , the value of $[F(m)]^n$ is approximately equal to $\exp\{-n[1 - F(m)]\}$. Simple calculations show that after replacement of $[F(m)]^n$ by its Cramér approximate value, integral (2.6.5) takes the form

$$\Delta = \frac{E_1(n_2) - E_1(n_1)}{\beta \exp(-n_2)} + m_{\min} \exp(-n) \quad (2.6.6)$$

where $n_1 = n / \{1 - \exp[-\beta(m_{\max}^{obs} - m_{\min})]\}$, $n_2 = n_1 \exp[-\beta(m_{\max}^{obs} - m_{\min})]$, and $E_1(\cdot)$ denotes an exponential integral function. The function $E_1(\cdot)$ is defined as

$E_1(z) = \int_z^\infty \exp(-\zeta) / \zeta d\zeta$, and can be conveniently approximated as

$$E_1(z) = \frac{z^2 + a_1 z + a_2}{z(z^2 + b_1 z + b_2)} \exp(-z),$$

where $a_1 = 2.334733$, $a_2 = 0.250621$, $b_1 = 3.330657$, and $b_2 = 1.681534$ (Abramowitz and Stegun, 1970). Hence, following the generic formula (2.6.4), for the Gutenberg-Richter frequency-magnitude distribution, the estimator of m_{\max} is

$$\hat{m}_{\max} = m_{\max}^{obs} + \frac{E_1(n_2) - E_1(n_1)}{\beta \exp(-n_2)} + m_{\min} \exp(-n) \quad (2.6.7)$$

Above estimator of m_{\max} , is in good agreement with our intuitive expectations: for given values of β and m_{\min} , the larger n is, or the longer the period of observation T , the less the estimated maximum regional magnitude \hat{m}_{\max} deviates from the largest observed magnitude m_{\max}^{obs} .

The estimator (2.6.7) of the maximum regional magnitude m_{\max} was introduced in Kijko and Sellevoll (1989). Equation (2.6.7) has subsequently been used for estimation of the maximum regional earthquake magnitude in several seismically active areas such as e.g. China (Yurui and Tianzhong, 1997), Canada, (Weichert and Kijko, 1989); France (Andre, 1999); Iran (Motazedian, et al., 1997); India (Shanker, 1998); Romania, (Marza, et al., 1991); Greece (Papadopoulos and Kijko, 1991); Algeria (Hamdache, 1998, Hamdache, et al., 1998); Italy (Slejko and Kijko, 1991); Spain (Garcia-Fernandez, et al., 1989), Turkey (Aptekin and Oncel, 1992; Aptekin et al., 1992) and the West Indies (Aspinall et al., 1994). Equation (2.6.4) will be termed the Kijko-Sellevoll estimator of m_{\max} , or, in short, K-S.

It should be noted again that the above estimator of m_{\max} can be used when the number of seismic events, n , is not known. In such a case, the number of seismic events should be replaced by λT and this replacement is equivalent to the assumption that the event occurrence conforms to a Poisson distribution with parameter λ and time span of the seismic catalog T . Calculation of the variance of the estimated maximum magnitude, $\text{Var}(\hat{m}_{\max})$ is the same as for Cases II and III, and is shown in Section 2.6.4.

2.6.3 Case II: Application of the generic formula to the Gutenberg-Richter magnitude distribution in case of uncertainty in the b -value.

A significant shortcoming of the K-S formula for m_{\max} estimation comes from the implicit assumptions that (i) seismic activity remains constant in time, (ii) the proper functional form of the magnitude distribution is specified, and (iii) the parameters of the assumed distribution functions are known without error. As many studies of seismic activity in mines suggest, however, the seismic process can be composed of temporal trends, cycles, short-term oscillations and pure random fluctuations. A list of some well-documented cases of temporal variation of seismic activity in areas from all over the world is given in Kijko and Graham (1998).

When the variation of seismic activity is a random process, the Bayesian formalism, in which the model parameters are treated as random variables, provides the most efficient tool to account for the uncertainties considered above (e.g. DeGroot, 1970). In this section, a Bayesian-based equation for the assessment of the maximum regional magnitude will be derived in which the uncertainty of the Gutenberg-Richter parameter b is taken into account.

Following the assumption that the variation of the β -value in the Gutenberg-Richter-based CDF (2.4.4) may be represented by a Gamma distribution with parameters p and q , the Bayesian (also known as compound or mixed) CDF of magnitudes takes the form (Campbell, 1982):

$$F(m) = \begin{cases} 0, & \text{for } m < m_{\min} \\ C_{\beta} \left[1 - \left(\frac{p}{p + m - m_{\min}} \right)^q \right] & \text{for } m_{\min} \leq m \leq m_{\max} \\ 1, & \text{for } m > m_{\max} \end{cases} \quad (2.6.8)$$

where C_{β} is a normalizing coefficient. It is not difficult to show that p and q can be expressed in terms of the mean and variance of the β -value, where $p = \bar{\beta} / (\sigma_{\beta})^2$ and $q = (\bar{\beta} / \sigma_{\beta})^2$.

The symbol $\bar{\beta}$ denotes the known, mean value of the parameter β , σ_{β} is the known standard deviation of β and describes its uncertainty, and C_{β} is equal to $\{1 - [p / (p + m_{\max} - m_{\min})]^q\}^{-1}$. Equation (2.6.8) is also known (Campbell, 1982) as the Bayesian Exponential-Gamma CDF of earthquake magnitude.

It is important to note that the way of handling the uncertainty of parameter β as above is by no means unique. For example, for the same purpose, Mortgat and Shah (1979) used a combination of the Bernoulli and the Beta distributions. Dong et al. (1984), as well as Stavrakakis and Tselentis (1987), used a combination of uniform and multinomial distributions. Excellent summaries of alternative ways handling all kinds of uncertainties that are present in the parameters, in the model and in the data, are found in papers by Bender and Perkins (1993) and Rhoades et al. (1994).

Knowledge of the Bayesian, Gutenberg-Richter distribution (2.6.8), makes it possible to construct the Bayesian version of the estimator of m_{\max} . Following the generic formula (2.6.4), the estimation of m_{\max} requires calculation of the integral

$$\Delta = (C_{\beta})^n \int_{m_{\min}}^{m_{\max}^{obs}} \left[1 - \left(\frac{p}{p + m - m_{\min}} \right)^q \right]^n dm, \quad (2.6.9)$$

which after application of Cramér's approximation, can be expressed as

$$\Delta = \frac{\delta^{1/q+2} \exp[nr^q / (1 - r^q)]}{\beta} \left[\Gamma(-1/q, \delta r^q) - \Gamma(-1/q, \delta) \right], \quad (2.6.10)$$

where, $r = p / (p + m_{\max}^{obs} - m_{\min})$, $c_1 = \exp[-n(1 - C_{\beta})]$, $\delta = nC_{\beta}$, and $\Gamma(\cdot; \cdot)$ is the Incomplete Gamma Function. Thus, the estimator of m_{\max} , when the **uncertainty of the Gutenberg-Richter parameter b** is taken into account, becomes

$$\hat{m}_{\max} = m_{\max}^{obs} + \frac{\delta^{1/q+2} \exp[nr^q / (1 - r^q)]}{\beta} \left[\Gamma(-1/q, \delta r^q) - \Gamma(-1/q, \delta) \right]. \quad (2.6.11)$$

Equation(2.6.11) will be denoted as the Kijko-Sellevoll-Bayes estimators of m_{\max} , or, in short, K-S-B. An extensive comparison of performances of K-S and K-S-B estimators is given in Kijko and Graham (1998).

2.6.4 Case III: Estimation of m_{\max} when no specific model of the magnitude distribution is assumed.

Making use of the N-P-G estimation of the CDF as given by equation (13), the approximate value of the integral for Δ is

$$\Delta \cong \int_{m_{\min}}^{m_{\max}^{obs}} [\hat{F}(m)]^n dm = \int_{m_{\min}}^{m_{\max}^{obs}} \left[\frac{\sum_{i=1}^n \left[\Phi\left(\frac{m-m_i}{h}\right) - \Phi\left(\frac{m_{\min}-m_i}{h}\right) \right]}{\sum_{i=1}^n \left[\Phi\left(\frac{m_{\max}^{obs}-m_i}{h}\right) - \Phi\left(\frac{m_{\min}-m_i}{h}\right) \right]} \right]^n dm \quad (2.6.12)$$

so that \hat{m}_{\max} based on the **non-parametric** Gaussian estimation of probability density functions takes the form

$$\hat{m}_{\max} = m_{\max}^{obs} + \int_{m_{\min}}^{m_{\max}^{obs}} \left[\frac{\sum_{i=1}^n \left[\Phi\left(\frac{m-m_i}{h}\right) - \Phi\left(\frac{m_{\min}-m_i}{h}\right) \right]}{\sum_{i=1}^n \left[\Phi\left(\frac{m_{\max}^{obs}-m_i}{h}\right) - \Phi\left(\frac{m_{\min}-m_i}{h}\right) \right]} \right]^n dm \quad (2.6.13)$$

Estimator (2.6.13) is very useful. Its strongest point is that it does not require specification of the functional form of the magnitude distribution $F(m)$. By its nature, therefore, it is capable of dealing with cases of complex empirical distributions, e.g. distributions in extreme violation of log-linearity, and/or which are multimodal, and/or which incorporate "characteristic" earthquakes. The drawback of estimator (2.6.13) is that, formally, it requires knowledge of all events with magnitude above the specified level of completeness m_{\min} . In practice, though, this can be reduced acceptably to a knowledge of a few (say 10) of the largest events. Such a reduction is possible because the contribution of the weak events to the estimated value of m_{\max} decreases very rapidly. Another drawback of the formula (2.6.13) is that it requires numerical integration. However, it need not be a real hindrance, since numerical integration with today's high-speed PC's is both very fast and accurate.

One should also mention that it is possible to derive another model-free technique for the estimation of m_{\max} , which is not based on the formalism of the non-parametric kernel estimation procedure. Such a procedure can be developed by means of order statistics, where the CDF of the magnitude distribution is model-free and is based only on the recorded seismic series.

Different approaches can be used in the estimation of the standard deviations of above estimators of m_{\max} . It is clear that, for catalogues that are long enough, the main contribution to the uncertainty in the estimation of parameter \hat{m}_{\max} comes from the uncertainty of the largest observed magnitude m_{\max}^{obs} . This uncertainty has two components: aleatory and epistemic (Toro et al., 1997).

Elementary computations show (Kijko and Graham, 1998) that the approximate variance of the aleatory uncertainty is of the order of the value of Δ^2 . Assuming that the standard error in

the determination of the maximum observed magnitude m_{\max}^{obs} is known and equal to σ_M , the epistemic contribution to the variance of \hat{m}_{\max} is equal to σ_M^2 . Therefore, the approximate, total variance of any of the above estimators [i.e. (2.6.7), (2.6.11) and (2.6.13)] is given by

$$Var(\hat{m}_{\max}) = \sigma_M^2 + \Delta^2, \quad (2.6.14)$$

where the corrections Δ are described by equation (2.6.6), and (2.6.10) respectively.

2.7 Some tests and examples of application.

2.7.1 Assessment of m_{\max} .

2.7.1.1 Tests of procedures based on Monte Carlo simulated catalogues.

Assessment of the performance of the non-parametric based procedure for m_{\max} estimation was carried out using Monte Carlo generated data of a given population distribution. Three models of magnitude distribution $F(m)$ were considered. The first was the pure, truncated Gutenberg-Richter model (2.4.4), the second represented a mixture of two pure, truncated, Gutenberg-Richter distributions and the third consisted of a mixture of the Gutenberg-Richter and normal distributions. The parameters of the three models tested are given in Table 2.7.1.

Model	PDF	Parameters
I	$f(\beta, m_{\min}, m_{\max})$	$b = 0.8$ (or $\beta = 1.84$), $m_{\min} = 1.0, m_{\max} = 5.2$.
II	$0.7f(\beta^{(1)}, m_{\min}^{(1)}, m_{\max}^{(1)}) + 0.3f(\beta^{(2)}, m_{\min}^{(2)}, m_{\max}^{(2)})$	$b^{(1)} = 1.0$ (or $\beta^{(1)} = 2.30$), $m_{\min}^{(1)} = 1.0, m_{\max}^{(1)} = 4.0$, $b^{(2)} = 0.7$ (or $\beta^{(2)} = 1.61$), $m_{\min}^{(2)} = 3.0, m_{\max}^{(2)} = 5.2$.
III	$0.9f(\beta, m_{\min}, m_{\max}) + 0.1N(\mu, \sigma)$	$B = 0.8$ (or $\beta = 1.84$), $M_{\min} = 1.0, m_{\max} = 5.0$, $\mu = 4.5, \sigma = 0.3$.

Table 2.7.1 Studied magnitude distribution models. $f(\bullet)$ denotes the PDF of the truncated, frequency-magnitude Gutenberg-Richter distribution (2.4.4), $N(\bullet)$ is PDF of the normal distribution.

Then, for every model, the two estimated magnitudes \hat{m}_{\max}^{K-S} and \hat{m}_{\max}^{N-P-G} computed according to procedures K-S and N-P-G were compared with the true value of $m_{\max} = 5.2$. This operation was repeated 1000 times in order to discern a general pattern with respect to the relative performance of estimators.

The mean values of \hat{m}_{\max} , estimated for model I [i.e. the pure truncated Gutenberg-Richter CDF (2.4.4)], and model II (i.e. mixture of two truncated Gutenberg-Richter distributions) are shown in Figure 1. The results for model III (mixture of the Gutenberg-Richter and normal distributions) are not provided because this model does not have a definite upper bound of magnitude, that could be used as a reference. For both models of magnitude distribution considered, the values of \hat{m}_{\max}^{N-P-G} result in a slight underestimation. The bias decreases with an increase in sample size. In the case of model I, (Figure 2.7.1), when the assumed distribution function for parametric estimation is exactly the same as the distribution underlying the data, the non-parametric estimate is only slightly worse than the parametric estimate. Yet the discrepancy is negligible – for a number of events equal to 200 or more, this discrepancy is less than 0.05 in magnitude. For model II (Figure 2.7.2), the non-parametric estimates, \hat{m}_{\max}^{N-P-G} , are significantly closer to the "true" value of $m_{\max} = 5.2$ than the corresponding solutions provided by the model based estimate \hat{m}_{\max}^{K-S} .

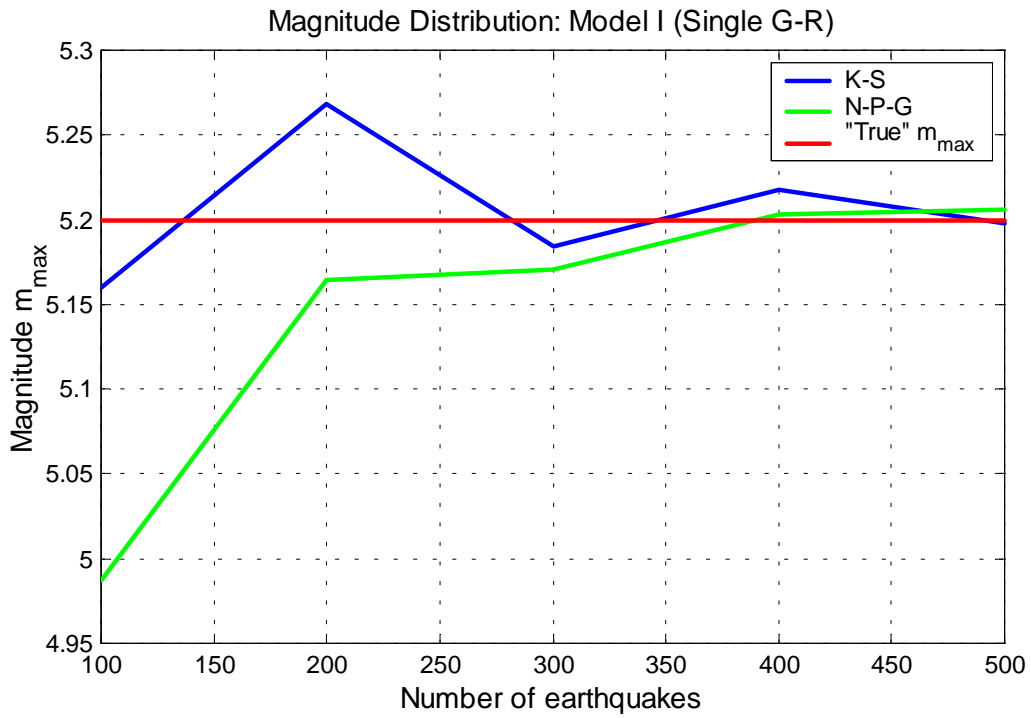


Figure 2.7.1. Mean values of parametric K-S and non-parametric N-P-G estimates of m_{max} for model I.

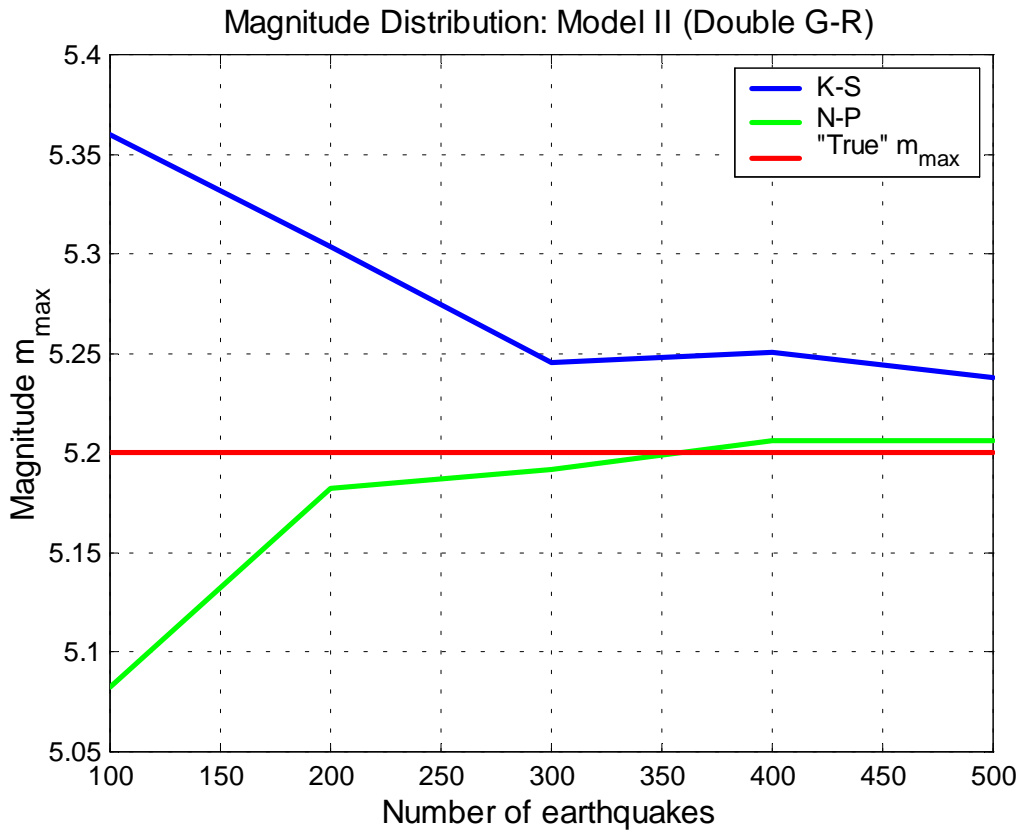


Figure 2.7.2. Mean values of parametric K-S and non-parametric N-P-G estimates of m_{max} for model II.

The above test performed on simulated data showed that the non-parametric estimates of maximum regional magnitude m_{\max} provide correct results with acceptable errors regardless of whether the actual magnitude distribution follows the Gutenberg-Richter relation or not. When the assumed model of magnitude distribution is exactly the same as the distribution characterizing the population the data come from, then non-parametric estimates of m_{\max} are only insignificantly worse than the estimates provided by the traditional parametric approach. On the other hand, when the guess for the magnitude distribution model is wrong, the parametric approach can result in an unacceptably large underestimation or overestimation of the value of m_{\max} .

2.7.1.2 m_{\max} Determination for a selected area of Klerksdorp gold mining district.

The seismic catalogue used consisted of 274 events of moment magnitude above 2.5 recorded in a selected area of the Klerksdorp gold mining district from January 1993 to October 1998. The selection of the analysed data was performed on the basis of epicenter distribution – events that appeared to belong to the same seismicity cluster, were selected. For the sake of the accuracy of the statistical analysis only well-populated clusters were used. This choice was exercised in order not to find or enhance non linear features of magnitude distribution due to an inadequate number of events. From this point of view any appearance of non-linearity in the magnitude distribution was incidental, and so the data analysed can be regarded as typical seismic recordings from mines. Empirical and non-parametric cumulative distributions of seismic event magnitude for the selected data are shown in Figure 2.7.2.

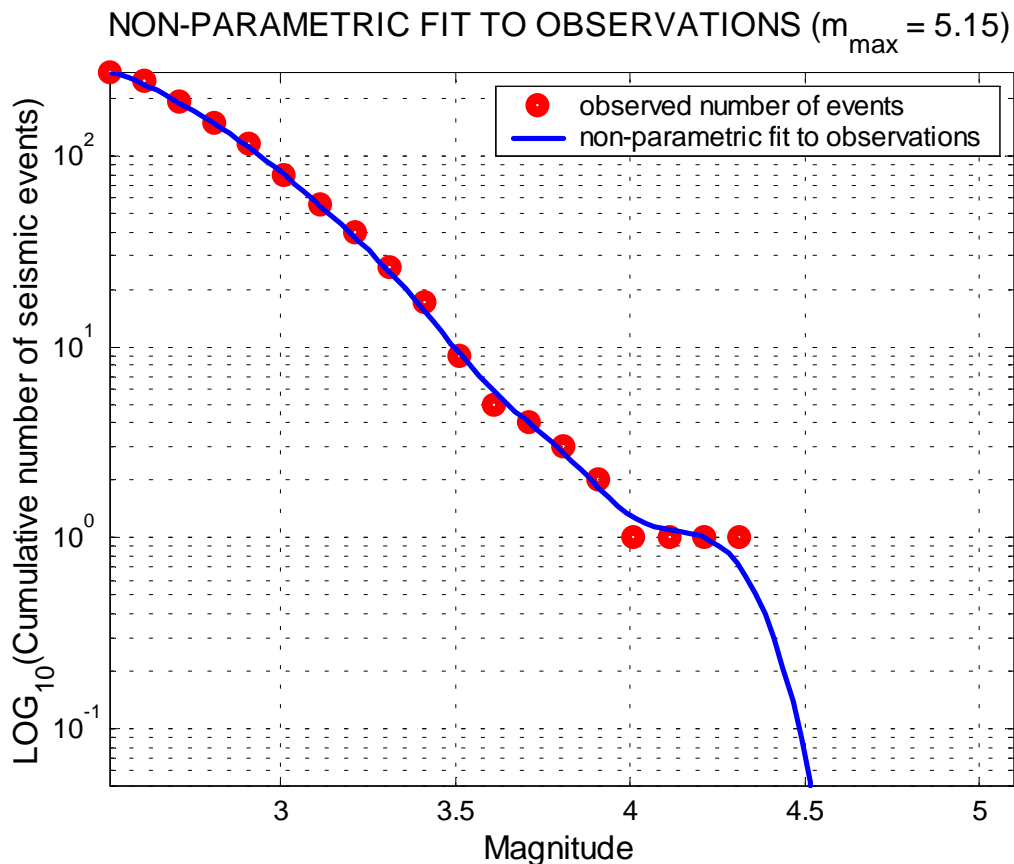


Figure 2.7.3. Observed cumulative number of seismic events and its non-parametric fit for the data from a selected area of the Klerksdorp Gold Mining District.

Application of the three procedures to find estimates of the maximum regional magnitude m_{\max} is shown in Table 2.7.2.

PROCEDURE	$\hat{m}_{\max} \pm SD$
K-S	4.60 \pm 0.27
K-S-B	4.61 \pm 0.27
N-P-G	5.15 \pm 0.81

Table 2.7.2. The values of m_{\max} , with their standard deviations, as obtained by the three procedures discussed in this work for selected area of the Klerksdorp Gold Mining District.

One should not be surprised that K-S and K-S-B estimators of m_{\max} differ so significantly from the N-P-G counterpart. Obviously, the differences follow from the fact that the empirical distribution of magnitude does not follow the standard, log-linear Gutenberg-Richter relation. The first two procedures (K-S and K-S-B) are based on the Gutenberg-Richter relation, while the N-P procedure is model free and therefore is capable of accounting for the presence of characteristic earthquakes, which are clearly seen in the analysed data set (Figure 2.7.3).

Since the last procedure is, by its nature, non-parametric and do not require specification of the functional form of the magnitude distribution, its estimate of the maximum magnitude m_{\max} is considered to be more reliable than the model-based estimators K-S and K-S-B.

2.7.2 Assessment of seismic hazard.

2.7.2.1 Tests of Procedures based on Monte Carlo simulations.

This section presents results of the application of the parametric and non-parametric procedures for the estimation of the cumulative distribution functions of the three models shown in Table 2.7.1. The only difference between the parameters listed in Table 2.7.1 and those used in these tests is that the value of maximum magnitude $m_{\max}^{(2)}$, (model II), was increased from 5.2 to 6.0

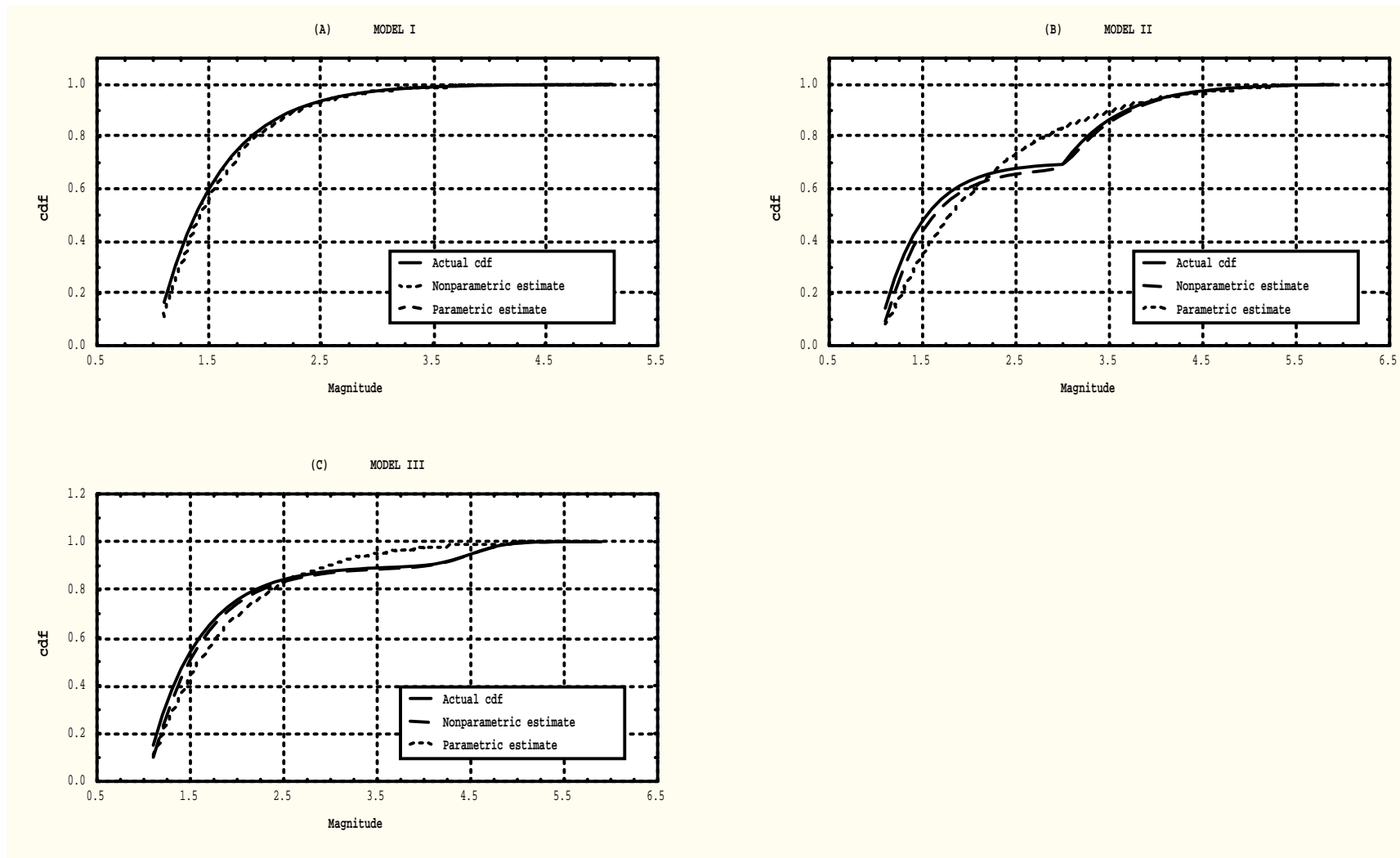


Figure 2.7.4 Differences between assumed CDF and mean values of non parametric and parametric CDF estimates for 3 considered models. The results are obtained as an average from 500 catalogues, each with 200 events.

Figure 2.7.4a presents an estimation of the CDF of seismic event magnitude for cumulative distribution function, model I. The mean values of non-parametric and parametric estimates, obtained from 500 catalogues of 200 events each, are compared with the theoretical CDF of the model. The non-parametric approach underestimates the actual CDF values in the low magnitude range only slightly. As one could expect, in the case of this model, the parametric fit exactly reproduces the actual CDF. Results of CDF estimation for non-log-linear models II and III are shown in Figure 2.7.4b and Figure 2.7.4c. **In both cases the non-parametric estimator reproduces the model CDFs quite well, certainly better than the parametric estimator.** In all three cases considered the catalogue size, varying from 200 to 2000 seismic events, did not have a noticeable effect on the accuracy of the estimate of the CDF.

The differences between the adopted CDFs and their average estimates are presented in Figure 2.7.5. From the figure it is evident that the selection of the wrong model of distribution of seismic event magnitudes, which is often the case for seismic data from mining, leads to considerable errors. The magnitude range for which the parametric estimates are the most erroneous depends upon the shape of the actual magnitude distribution and cannot be predicted, and so cannot be accounted for. Conversely, the errors of non-parametric estimates are not only small but also similar regardless of the complexity of the actual magnitude distribution. The slight underestimation at the lower cut-off of magnitude is probably due to the so-called 'edge effect' of the kernel estimation (Silverman, 1986) and can be reduced by a more subtle estimation technique. The error of the non-parametric estimate is practically zero in the high magnitude range, which is the range that is most important in the assessment of seismic hazard.

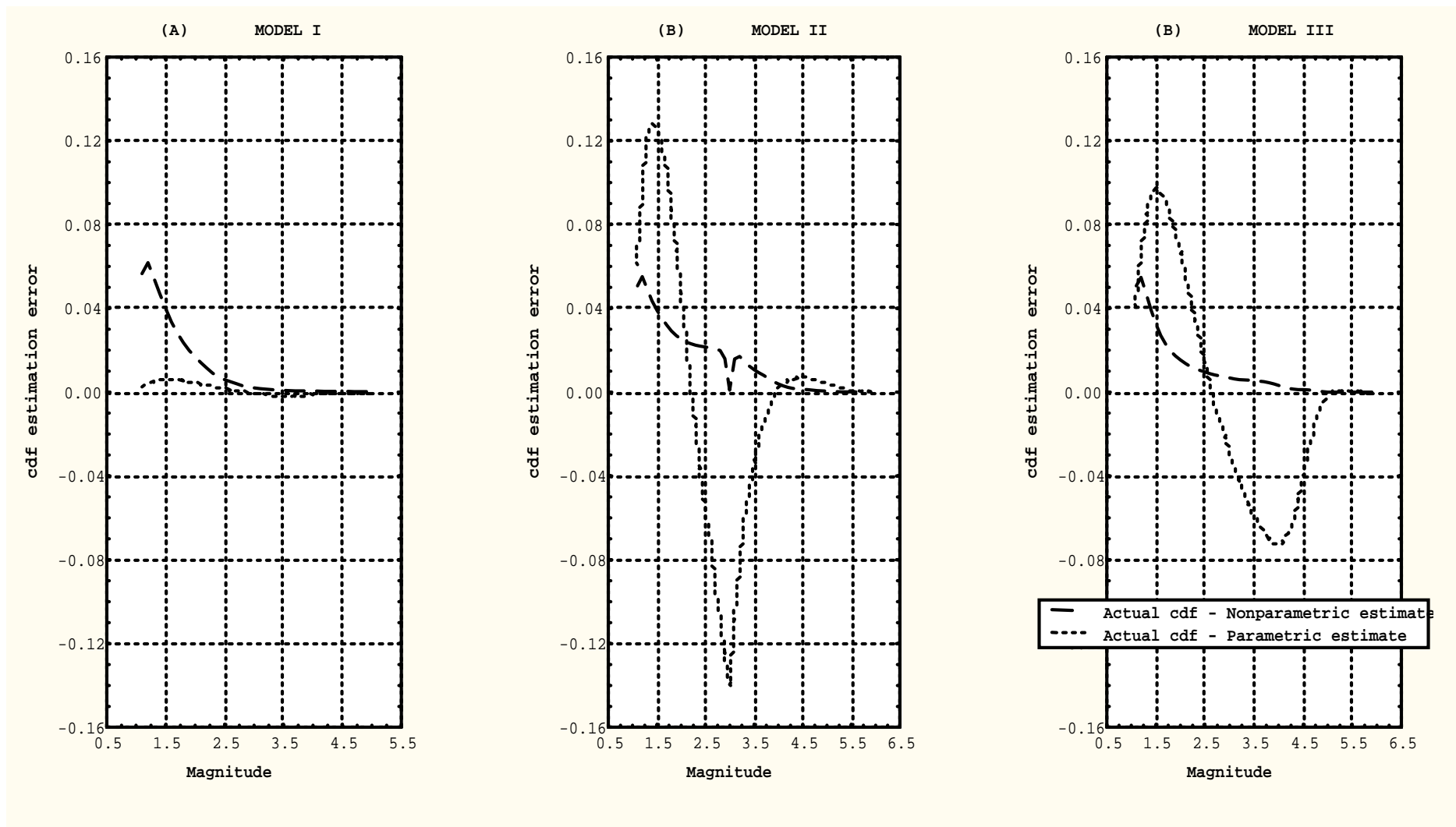


Figure 2.7.5 Differences between assumed CDF and mean values of non parametric and parametric CDF estimates for 3 considered models. The results are obtained as an average from 500 catalogues, each with 200 events.

Since the error of the parametric CDF estimate depends upon the shape of the distribution, its influence on hazard estimates is not determined uniquely. In order to illustrate possible differences between non-parametric and parametric hazard estimates we calculated the value of seismic hazard for magnitude 4.5 in the case of model III. As can be seen from Figure Figure 2.7.5 c, the error of the parametric CDF estimate for $m = 4.5$ was not the largest error produced by this estimate, though it was considerable. We assumed an average of 20 events per time unit ($\lambda = 20$) and the seismic hazard (equation (2.3.1)) was calculated for every catalog of 200 seismic events up to 24 time units. Figure Figure 2.7.6 presents a comparison of the average hazard estimates with the actual hazard. **The non-parametric estimation is quite satisfactory while the parametric approach underestimates the hazard by up to 50%. In practical applications a result like this indicates a meaningless and misleading estimation of hazard.**

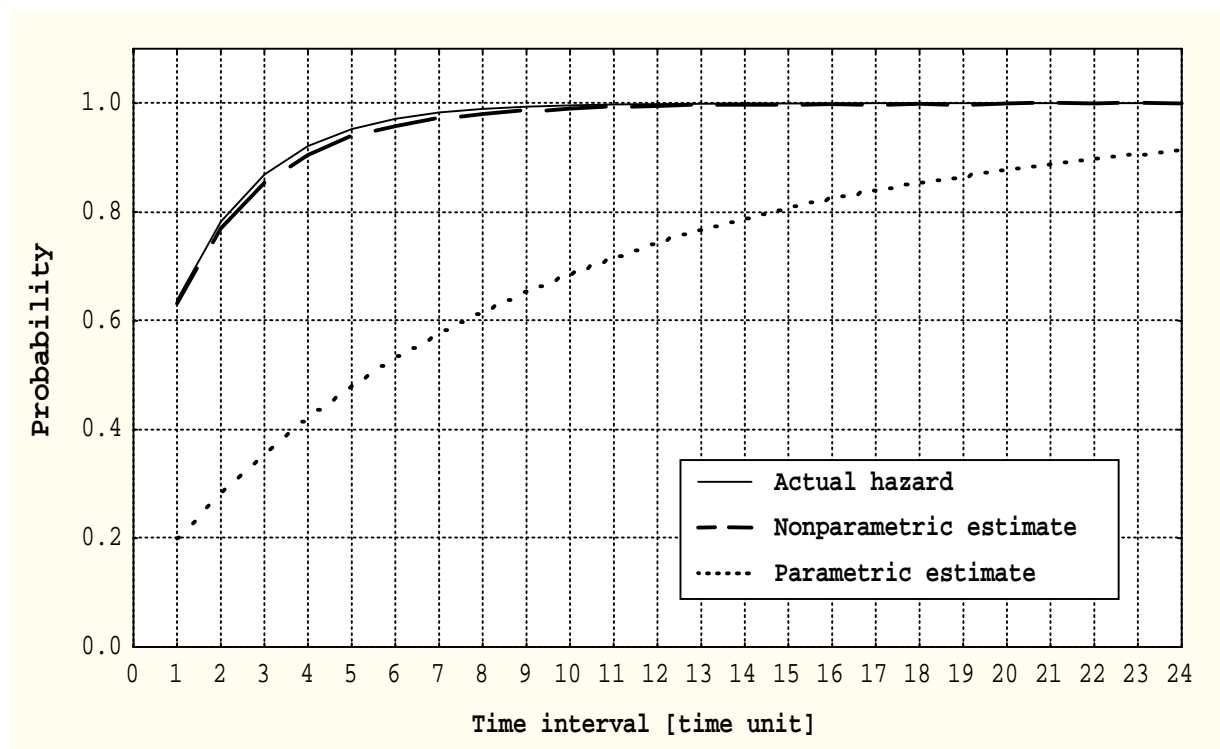


Figure 2.7.6 Actual seismic hazard of $M_p = 4.5$ for model III and mean values of its non parametric and parametric estimates. The results are obtained as an average from 500 catalogues, each with 200 events.

2.7.2.2 Example of application in real mining environment: seismic hazard assessment for a selected area of the Western Deep Levels Mine.

The data set contains seismic events recorded in an area of the Western Deep Levels (WDL) mine from September 1990 to November 1996. The data used consisted of 1715 events of moment magnitude above 0.5.

Only a part of the catalogue was analysed. The selection of the analysed data was performed on the basis of epicenter distribution – events that appeared to belong to the same seismicity cluster, were selected. For the sake of the accuracy of the statistical analysis well-populated clusters were used. The choice was then not meant to find or enhance non linear features of magnitude distribution due to an inadequate number of events. From this point of view any appearance of non-linearity in the magnitude distribution was incidental, and so the data analysed can be regarded as representing typical seismic recordings from mines.

Histograms of magnitude for the data subset is shown in Fig. 6.

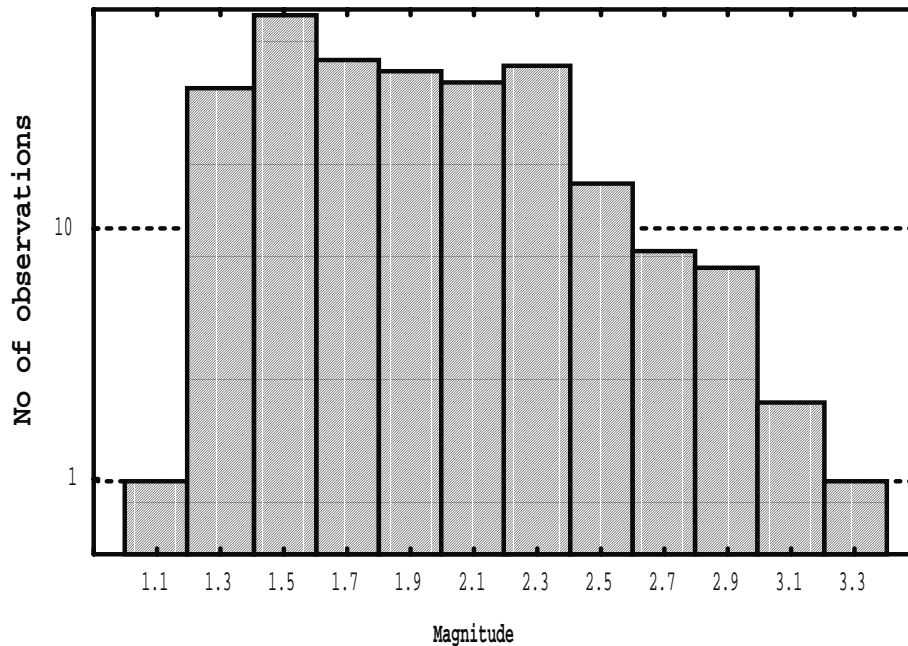


Figure 2.7.7 Histogram of seismic events magnitudes from a selected area of WDL mine.

The data seems to be complete above a local magnitude value of $m_{\min} = 1.4$. Altogether 277 events from the recording period of 51 months exceeded this threshold and were included into the analysis. The average activity rate, i.e. the estimate of the parameter λ was 5.43 events/month.

The analysis included the estimation of the upper limit of the magnitude range m_{\max} , the cumulative distribution function of magnitude, the mean return period and the seismic hazard for selected values of magnitude. The analysis was done both by means of the non-parametric technique and by means of the parametric method with the truncated Gutenberg-Richter based distribution (2.4.4), as a model for magnitude distribution.

The non-parametric estimate of m_{\max} (eq. (2.6.13) is 3.54 and the parametric estimate K-S (eq. (2.6.3) is 3.44. The non-parametric and parametric estimates of the CDF are compared in Figure 2.7.1 a. The differences in estimation of the CDFs result in differences in assessment of the mean return period (Figure 2.7.8b) and the seismic hazard (Figure 2.7.8 c and d). For magnitudes larger than 2.4 the non-parametric mean return period is larger - in the worst case more than 4 times the corresponding parametric estimate. The hazard indicated by the hazard curves obtained from the non-parametric estimator is generally lower than that of the respective curves provided by the parametric approach. The discrepancy reaches 20%.

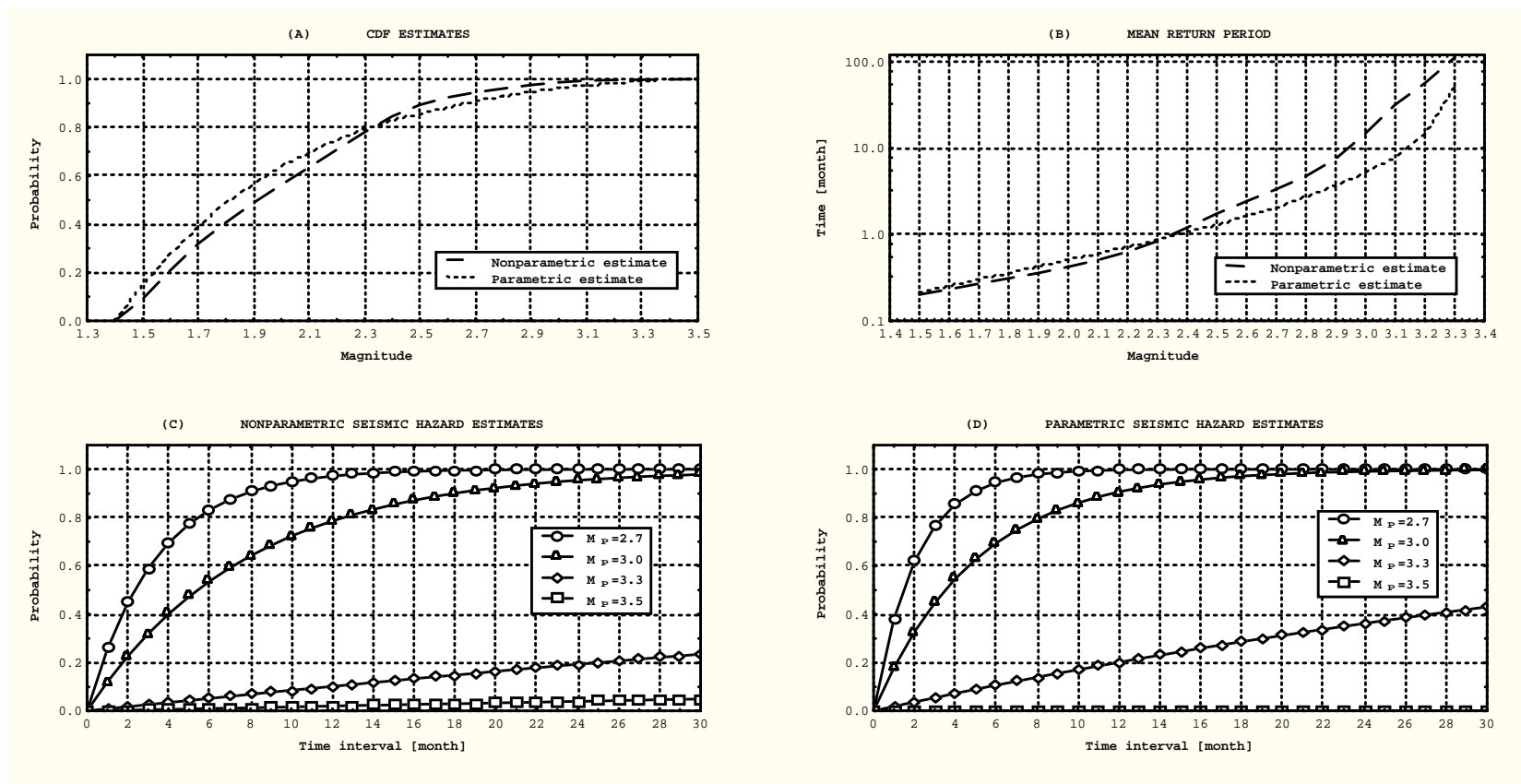


Figure 2.7.8 Non parametric and parametric estimates of CDF of seismic event magnitude (a), mean return period (b); seismic hazard (c) and (d) for the data from WDL gold mine. Dashed lines – non parametric estimates. Dotted lines – parametric estimates.

2.8 Discussion, conclusions. Recommendations.

The non-parametric approach to a seismic hazard assessment is independent of the underlying magnitude distribution and is therefore an appropriate tool for seismic hazard assessment in mines when the magnitude distribution is complex.

The studies performed on simulated data showed that the non-parametric estimates of magnitude distribution are correct regardless of whether the actual magnitude distribution follows the Gutenberg-Richter relation or not. The technique turned out to be effective for sample sizes starting from approximately 200 events.

When the assumed model of magnitude distribution is exactly the same as the distribution characterizing the population the data come from, then non-parametric estimates are only insignificantly worse than the estimates provided by the traditional parametric approach. On the other hand, **when the guess for the magnitude distribution model is wrong, the parametric approach can result in an unacceptably large underestimation or overestimation of the seismic hazard and related parameters.**

In the discussed examples it was intended to simulate the typical scenario of seismic hazard estimation in mines. However, although the real conditions in mines are often considerably different from the scenario considered, it is not expected that the conclusions regarding the superiority of the non-parametric approach over the parametric, Gutenberg-Richter based relation will change significantly. It is can expected that use of one of the re-sampling procedures (e.g. Hall, 1992; Shao and Tu 1995) will limit the bias implied by the individual sample analysis.

In the examples given, which are based on real data from the mines, the results of the non-parametric analysis differ significantly from the results provided by the parametric approach. these studies of Monte Carlo generated data show that for well-populated samples, such differences can only result from a wrong assumption of magnitude distribution model. In this context it is concluded that the non-parametric hazard estimation is more precise while the parametric approach significantly overestimates the actual hazard. Owing to the complexity of the factors controlling the generation of seismicity in mines, the distribution of magnitude is variable, often non-log-linear and multimodal. From the point of view of magnitude distribution the actual mining data studied were chosen at random. In practice there are more complex magnitude distributions and the assumption of a simple Gutenberg-Richter based distribution or any other parametric model of mining induced events is frequently inadequate. **Hence, the non-parametric, seismic hazard estimation procedure is particularly useful and strongly recommended for use in seismic hazard assessment in mines.**

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APPENDIX

LEAST-SQUARE CROSS-VALIDATION FOR GAUSSIAN KERNEL FUNCTION

Presented here is the procedure of least-squares cross-validation as presented by Silverman (1986) and the approach which we used in selecting the smoothing factor h , for the Gaussian kernel function (Equation (2.5.2)).

Given a sample of n elements of the random variable X , x_1, \dots, x_n , the kernel estimator $\hat{f}(x)$ of the actual PDF $f(x)$ is given by (10) and (11). The integrated square error is

$$\int_{-\infty}^{\infty} [\hat{f}(\xi) - f(\xi)]^2 d\xi = \int_{-\infty}^{\infty} [\hat{f}(\xi)]^2 d\xi - 2 \int_{-\infty}^{\infty} [\hat{f}(\xi) f(\xi)] d\xi + \int_{-\infty}^{\infty} [f(\xi)]^2 d\xi \quad (A1)$$

The last term of equation (A1) does not depend on h . It has been shown (Silverman, 1986) that the optimal choice of h that minimizes first two terms of (A1) requires minimizing of the score function

$$M_0(h) = \int_{-\infty}^{\infty} [\hat{f}(\xi)]^2 d\xi - 2n^{-1} \sum_{i=1}^n \hat{f}_{-i}(x_i) \quad (A2)$$

where

$$\hat{f}_{-i}(x) = \frac{1}{(n-1)h} \sum_{j \neq i} K\left(\frac{x - x_j}{h}\right) \quad (A3)$$

Further on the score $M_0(h)$ is

$$M_0(h) = \frac{1}{n^2 h} \sum_{i=1}^n \sum_{j=1}^n K^{(2)}\left(\frac{x_i - x_j}{h}\right) - \frac{2}{n(n-1)h} \sum_{i=1}^n \sum_{j=1}^n K\left(\frac{x_i - x_j}{h}\right) + \frac{2}{(n-1)h} K(0) \quad (A4)$$

where

$$K^{(2)}(x) = \int_{-\infty}^{\infty} K(\xi) K(x - \xi) d\xi \quad (A5)$$

It is then assumed that the minimizer of $M_0(h)$ is close to the minimizer of the expected value of $M_0(h)$, $E[M_0(h)]$, hence close to the minimizer of the mean integrated square error

$$E \left[\int_{-\infty}^{\infty} [\hat{f}(\xi) - f(\xi)]^2 d\xi \right]. \quad (A6)$$

Since for reasonably populated samples $(n - 1) \cong n$ minimization of the simplified score

$$M_1(h) = \frac{1}{n^2 h} \sum_{i=1}^n \sum_{j=1}^n \left\{ K^{(2)}\left(\frac{x_i - x_j}{h}\right) - 2K\left(\frac{x_i - x_j}{h}\right) \right\} + \frac{2}{nh} K(0) \quad (A7)$$

can usually replace minimization of $M_0(h)$.

Silverman (1986) suggested a use of the Fourier transform for finding the minimum of $M_1(h)$ in the general case of any kernel function. However, in case of the Gaussian kernel, the direct minimization of $M_1(h)$ turns out to be quite efficient and fast. For this particular kernel $K^{(2)}(x)$ is the pdf of Gaussian distribution with expected value 0 and variance 2 and $M_1(h)$ becomes

$$M_1(h) = \frac{1}{n^2 h \sqrt{\pi}} \sum_{i,j} \left\{ 0.5 \exp\left[-\frac{(x_i - x_j)^2}{4h^2}\right] - \sqrt{2} \exp\left[-\frac{(x_i - x_j)^2}{2h^2}\right] \right\} + \frac{\sqrt{2}}{nh \sqrt{\pi}} \quad (A8)$$

The smoothing parameter h that minimizes (A8) results from solution of the equation

$$\sum_{i,j} \left\{ 2^{-0.5} \left[\frac{(x_i - x_j)^2}{2h^2} - 1 \right] \exp\left[-\frac{(x_i - x_j)^2}{4h^2}\right] - 2 \left[\frac{(x_i - x_j)^2}{h^2} - 1 \right] \exp\left[-\frac{(x_i - x_j)^2}{2h^2}\right] \right\} - 2n = 0 \quad (A9)$$

3 Risk Assessment of Possible Rockburst Damage, Based on Peak Ground Velocity

3.1 Introduction

The risk of rockburst damage in certain deep level mining is ever-present. Rockburst-resistant support is installed in areas that are considered to be “seismically active”. The design criterion for support is currently based on its ability to stop the kinetic energy associated with a certain thickness of hanging-wall (H and density ρ) moving downwards at a take-off velocity (v). The hanging wall must then be stopped within a distance (h). The energy required to absorb the potential and kinetic energy, in Joules / m², is then:

$$E = \rho.H.(g.h + \frac{1}{2}.v^2) \quad (3.1.1)$$

“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. 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 observation 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.

3.2 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

$$v_N = V_S \Delta \sigma / G \quad (3.2.1)$$

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, equation 38), we have the following:

$$R v_F = f_{\theta\phi} V_S \Delta \sigma r_0 / G \quad (3.2.2)$$

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 (1) and (2) then collapse into a single equation:

$$\begin{aligned} v &= (V_S \Delta \sigma / G) && \text{for } R \leq r_0 && (3.2.3) \\ &= (V_S \Delta \sigma / G) * (r_0 / R) && \text{for } R \geq r_0 \end{aligned}$$

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 :

$$R' = (\sqrt{((r-r_0)^2 + z^2)} + \sqrt{((r+r_0)^2 + z^2)}) / 2 \quad (3.2.4)$$

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

$$v = (V_S \Delta \sigma / G) * (r_0 / R') \quad (3.2.5)$$

This is shown graphically in Figure 3.2.1.

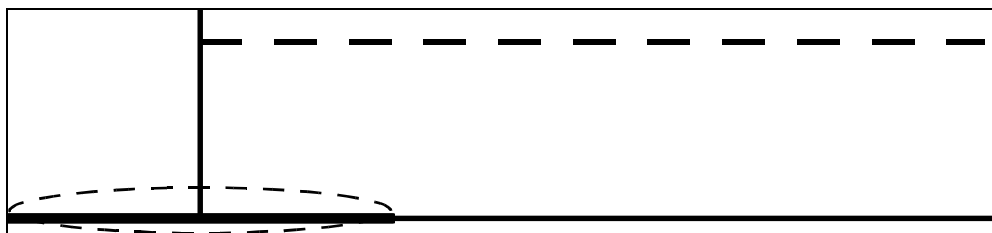


Figure 3.2.1 Equation (3.2.5) shown graphically

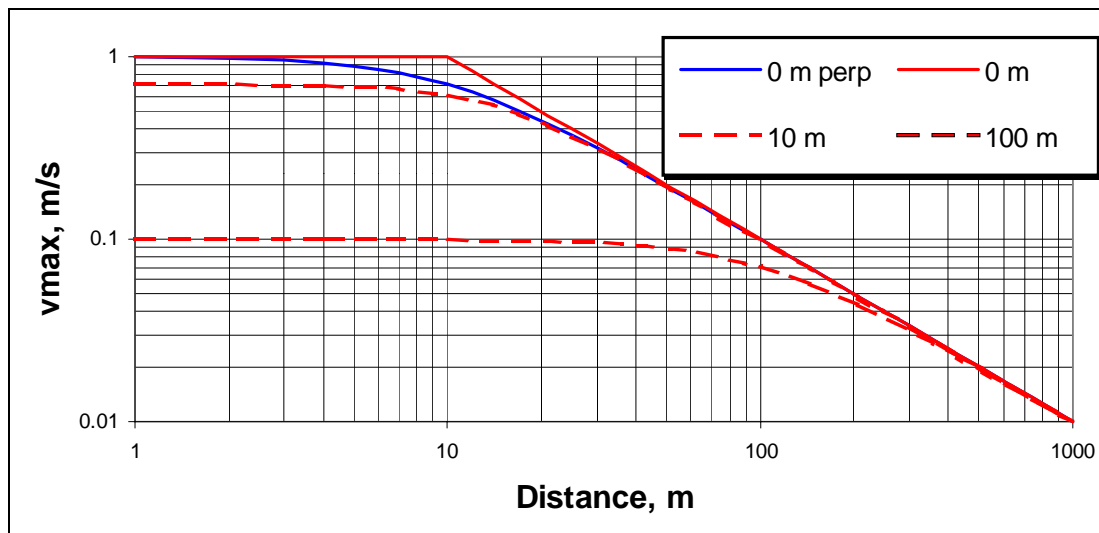


Figure 3.2.2 Dependence of peak velocity as a function of distance from the centre of a circular with radius 10m and constant stress drop seismic source, as proposed by Brune (1970).

Figure 3.2.2 From top to bottom, as sketched in Figure 3.2.1

Measured in the plane, from the centre of the source outwards.

Measured at right angles to the plane, from the centre outwards

Parallel to (1), 10 m from the source

Parallel to (1), 100 m from the source

In this section, “x” and “y” in equation (3.2.5) are on the plane of the reef and $z = 0$.

3.3 Analysis

Equations (3.2.4) and (3.2.5) have been applied by in Deepmine project 4.1.2 (Andersen and Daehnke, 1999).

We use the, the display features described in Spottiswoode (1997) are used to apply equation (3.2.5) to data from a portion of a deep level mine. Ddata for three months are displayed.

These figures show:

- 1.) Face outlines, obtained from the MINSIM digitizer program.
- 2.) Seismic locations, from the mine network
- 3.) Contours of the number of times each point was exposed to strong ground motion of v m/s, using equation (3.2.5). The location in X and Y, the static stress drop and source radius were used to generate this picture.

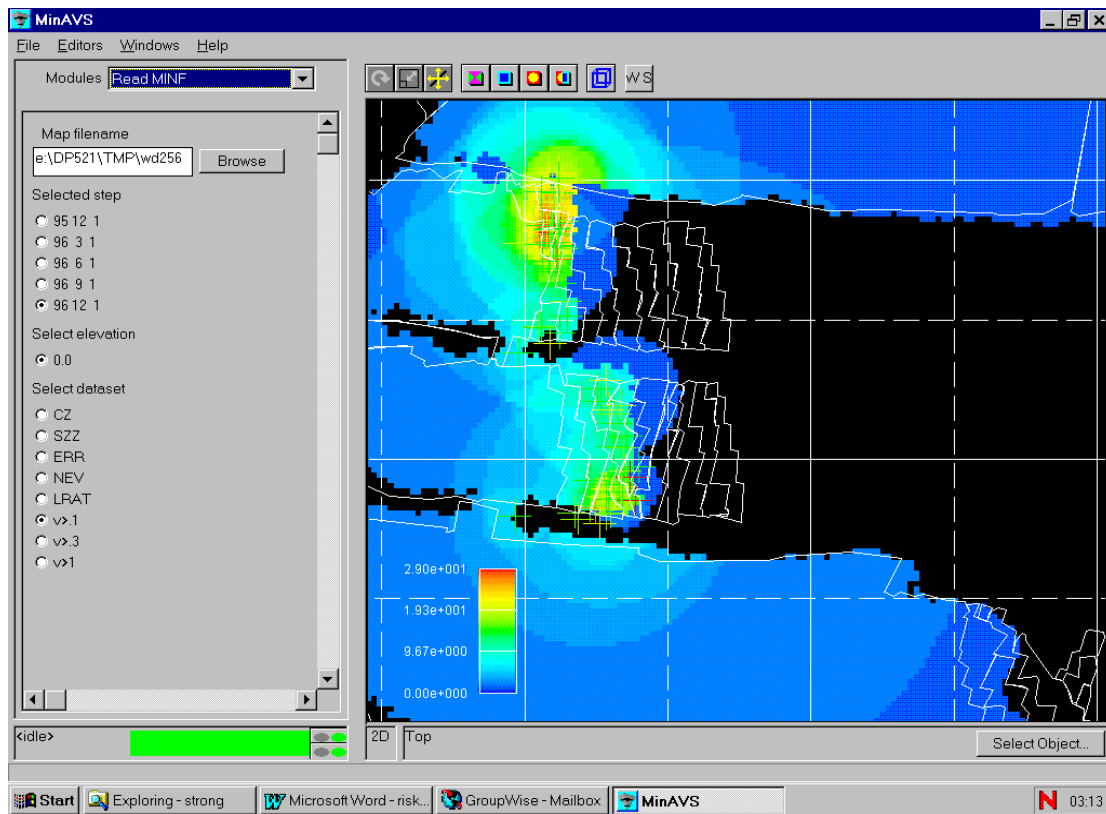


Figure 3.3.1 Contours of the number of times that each on-reef point was subjected to a ground velocity in excess of 0.1 m/s, assuming solid rock. This would be equivalent to 0.3 m/s if, say, the amplification in stopes is 3.0. The maximum number of “hits” was 29, in the upper longwall

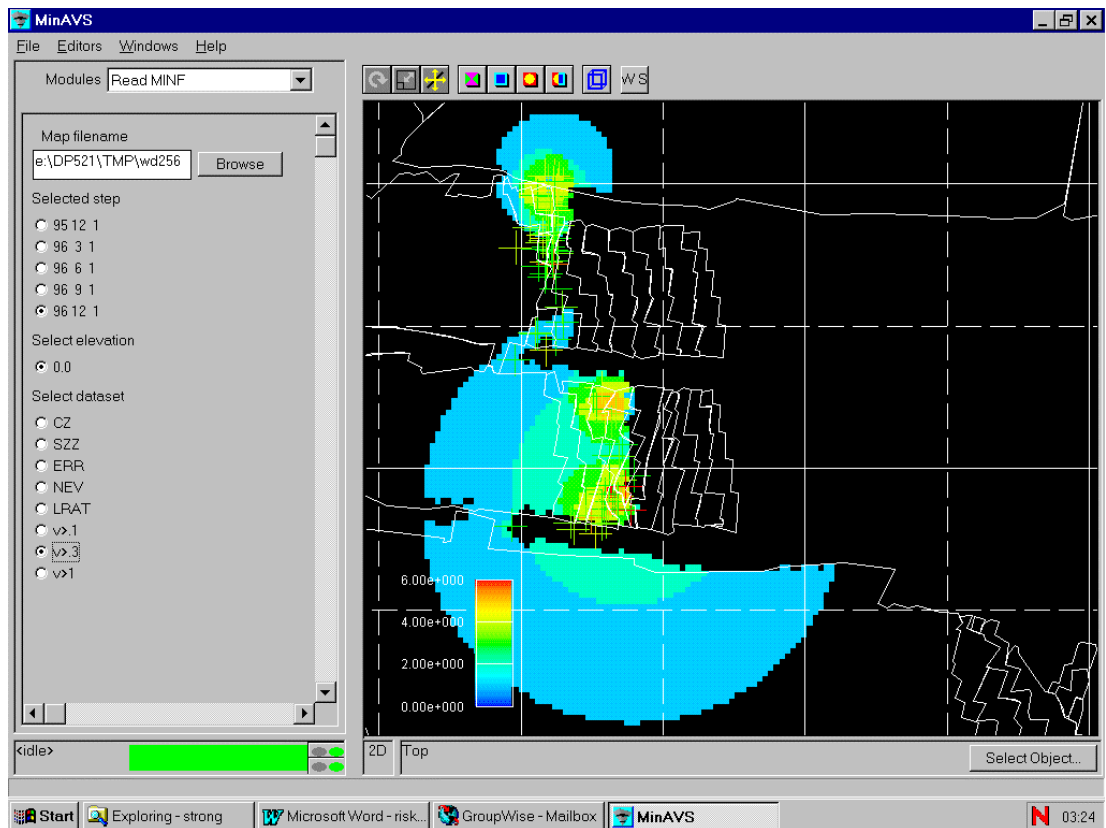


Figure 3.3.2 The same as figure 2 for ground velocity in excess of 0.3 m/s.. The maximum number of “hits” was 6, in the lower longwall.

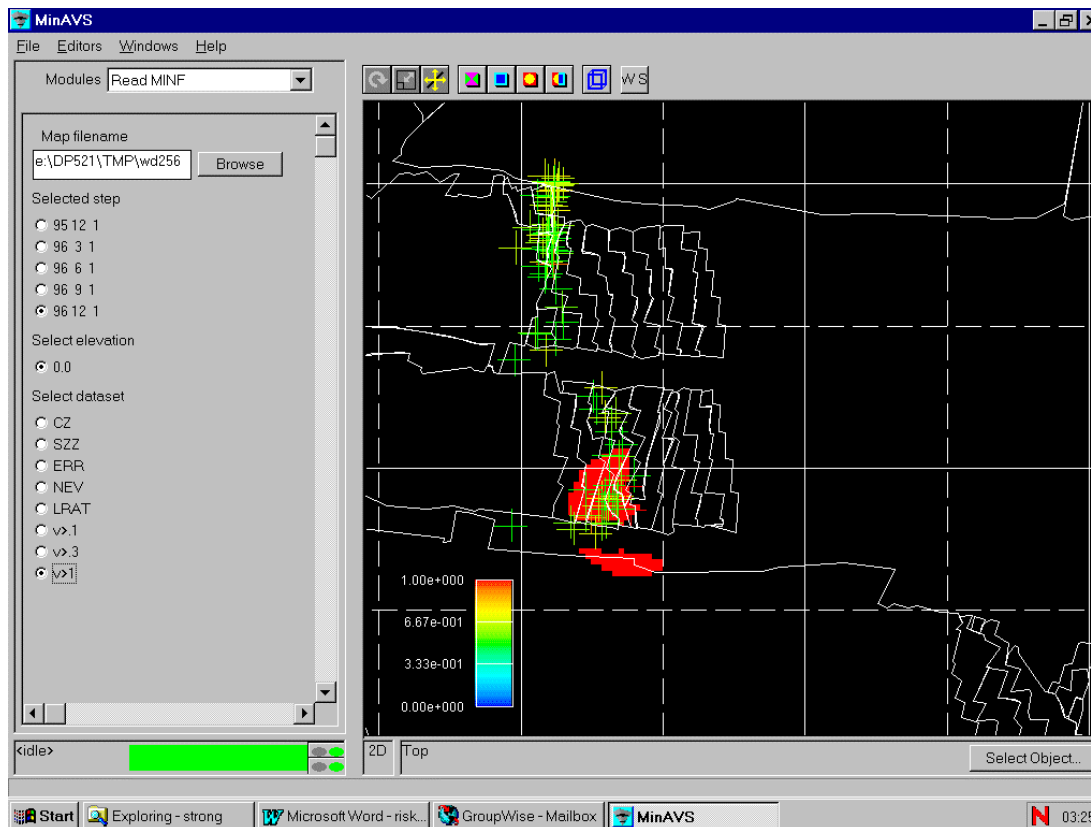


Figure 3.3.3 The same as Figure 3.3.1 and Figure 3.3.2 for ground velocity in excess of 1.0 m/s. One event generated this expected ground velocity over $\frac{1}{2}$ of the lower longwall.

3.4 Proposed methodology

To use this methodology on a mine for decisions on support design requires a number of steps:

1. The data should be available in suitable format.

1.1 Mine seismic data with locations and source parameters (moments & radii) for areas concerned and for an extended time period.

1.2 Digitized outlines for comparison. If these can be used for generating MINSIM input files over the same time periods, then we have information that can be used to test mine design criteria and to provide input to various research projects, including SIMRAC 612 ERR.

2. Use of a suitable graphical viewer. The figures shown here were prepared using MINAVS, a purpose-built package. The MINSIM 3D display software is now available and is well suited for this application.

3. Direct field evidence to calibrate this model. This should preferably be done using the following observations:

3.1 Quantification of rockburst damage, such as maps of panel-shifts lost.

3.2 Direct measurements of ground velocities by, for example, the CSIR Ground Motion Monitor.

In addition, the strong ground motion model described above is only approximate and could be improved in a number of ways that would require further research. Some important topics that should be addressed are:

1. Reducing the spurious effect of location errors. A conservative approach is to move all locations to the vicinity of likely sources, such as nearby faces, pillars, geological features, or abutments. This issue was addressed in part by Spottiswoode (1997). It is planned to complete this work during the year 2000 under the SIMRAC GAP722 project.
2. Reducing the spurious effect of source extent. This can be addressed, in part, through work in conjunction with that needed to address the location problem just mentioned.
3. Considering whether v_{\max} as radiated from the seismic source is, in fact, the best estimator of damage. For example, a shear slip a few metres ahead of the face can cause face crushing and large ground motions. The role of co-seismic horizontal stresses is also poorly understood. These are extremely difficult issues, but must be addressed if we are to design support systems for rockburst conditions. Some of these issues will be addressed in the new SIMRAC project GAP 709.

3.5 References

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4 Implications of the Time-Dependent Behaviour Of Deep Level Stopes in Hard Rock

4.1 Abstract

Although hard rock is not usually associated with large creep deformation, significant time-dependent behaviour is observed in the deep tabular excavations of the South African gold mines. This behaviour is the result of the rheology of the fracture zone surrounding these excavations and the time-dependent extension of this zone following a mining increment. It appears that continuous closure measurements are a useful diagnostic measure of the rock response when enlarging the excavation. The effect of rate of mining on stope closure is also investigated in the paper. A novel technique to estimate changes in average closure rate for different mining rates is developed by examining steady-state closure behaviour. Calibration of the developed model indicated that the mining rate has a noticeable effect on the closure rate. This should be accounted for when designing appropriate stope support.

4.2 Introduction

Analysis of the rock mass at depth requires some knowledge about the response of the rock to enlargement of excavations. Limited techniques are available to characterise the in situ behaviour of the rock. Seismic data has been used extensively in an attempt to characterise the rock in the deep gold mines in South Africa (e.g. Legge & Spottiswoode 1987, Mendecki 1997). Another potential source of valuable data is the closure behaviour of excavations. Continuous time-dependent closure data is a good reflection of the localised rock response and can provide important information which may be more difficult or even impossible to obtain with other methods. Rate of stope closure is also an important input parameter in the support design analysis (SDA) used in the South African industry (Roberts 1999). Although stope closure was frequently monitored in the South African gold mining industry since the early 1930's (Altson 1933), and probably before that, there are several issues that still need to be resolved. For the purpose of support design, the effect of mining rate on stope closure should be investigated.

4.3 Field Observations Of Time-Dependent Closure And Implications For Hazard Identification

The stopes of the South African gold mining industry are tabular in nature implying a negligible height (typically 1.2 m) compared to the lateral extent of the orebody (see Malan 1999). The high stresses ahead of the working faces result in an extensive fracture zone surrounding these excavations. Recent studies (Malan 1998; Malan 1999) indicate that the rockmass in these deep tabular excavations undergoes significant time-dependent deformation. This behaviour is important as it causes a gradual non-violent reduction of stress concentrations and hence diminishes the danger of rockbursts. It is therefore vital to determine the conditions associated

with the transition from stable deformation to rockbursts to identify potentially hazardous conditions.

To investigate the time-dependent behaviour of the rock mass, continuous measurements of stope closure were recorded in different geotechnical areas (Malan & Napier 1999). It should be noted that these continuous closure measurements contain much information that is lost with the more conventional daily or longer period measurements. The continuous closure behaviour typically consists of an instantaneous response at blasting time, followed by a primary phase of decelerating closure lasting approximately five hours, and a steady-state closure phase. This pattern is repeated after the next blast (Figure 4.3.1). This time-dependent behaviour is the result of the rheological behaviour of the fracture zone that surrounds these excavations. After a mining increment, the fracture zone extends in a time-dependent fashion ahead of the working faces. The majority of new fractures appear to form within approximately the first five hours after the blast; thereafter the number of new fractures decreases until the next blast. In areas with prominent bedding planes, creep on these discontinuities may also play a significant role in determining the closure response.

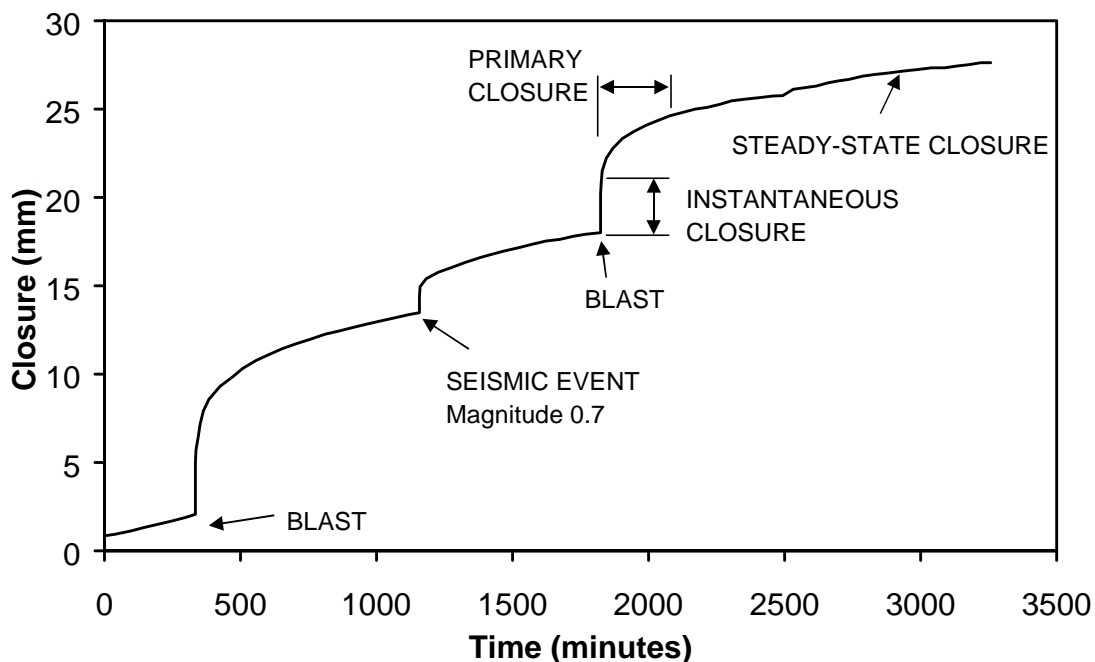


Figure 4.3.1. Typical continuous closure measurements in a tabular excavation of the Ventersdorp Contact Reef (hard lava hangingwall). The closure instrument was 9 m from the face.

Time-dependent closure data collected in stopes of the Ventersdorp Contact Reef (hard lava hangingwall) and Vaal Reef is shown in Figure 4.3.2. It was found that the closure behaviour of the Ventersdorp Contact Reef (hard lava) typically includes large instantaneous responses after blasting and low steady-state closure rates. These areas also appear to have an increase in the risk of face bursting. In comparison, in some areas of the Vaal Reef and the Ventersdorp Contact Reef (soft lava), the instantaneous closure response after blasting is small, followed by a large steady-state closure rate. In these areas, there appears to be a low risk of face bursting, but the rapid unravelling of the hangingwall might lead to enhanced fall of ground problems.

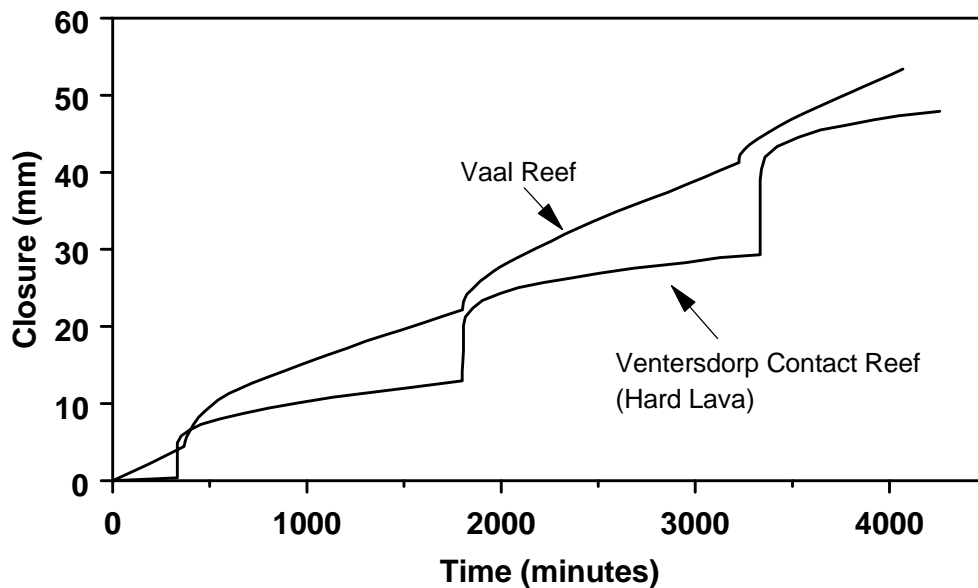


Figure 4.3.2 Comparison of typical continuous closure profiles of the Ventersdorp Contact Reef (hard lava) and the Vaal Reef. The data for each area contains the effect of three blasts. The behaviour of the Ventersdorp Contact Reef (soft lava) not shown here is similar to the Vaal Reef with high rates of steady-state closure and small instantaneous responses at blasting time.

It appears that the instantaneous closure response at blasting time gives an indication of the magnitude of stress in the face area. The larger the stress, the bigger the instantaneous closure response following the mining increment. This hypothesis was successfully tested by simulating two stopes with different rock conditions using a continuum viscoplastic approach (Malan 1999). It should also be noted also that a large steady-state closure rate may be undesirable as it indicates a high mobility of the fracture zone that can lead to an increase in the risk of falls of ground.

It is suggested from these measurements that continuous closure data is useful in identifying different geotechnical areas and in the possible identification of hazardous conditions. Further work should focus on quantifying the fall of ground and face bursting risks in different geotechnical areas (possibly using accident statistics) and on correlating this with the continuous closure behaviour. The effect of parameters such as stope span, measurement position, type of support and face advance per blast on the closure behaviour should also be investigated before a robust hazard measure can be developed.

4.4 Effect Of Mining Rate On Closure And Implications For Support Design

As stope closure contains a time-dependent component (see Figure 4.3.1 and Figure 4.3.2), the closure measured as a function of face advance will be dependent on mining rate. It is therefore important to record the rate of mining when collecting long period closure measurements. If

stope support is designed using measured closure values, significant changes in mining rate in future may result in the original support design not meeting the desired criteria anymore. This section illustrates how continuous closure measurements can be used to estimate the changes in total closure for different mining rates.

In order to estimate the effect of mining rate, the steady-state closure will be examined in more detail. Ideally an analytical solution is needed where the cumulative increase in closure at a specified point in a stope can be computed for different mining rates. Malan (1998) derived a Burgers viscoelastic solution for an isolated stope under plain strain conditions. Although this model can simulate the time-dependent closure, viscoelastic theory predicts an increase in the rate of steady-state closure as the distance to face increases. This behaviour is only true for some geotechnical areas in the mining industry and is therefore not ideally suited to investigate the mining rate problem. The problems associated with the use of viscoelasticity are a result of the inability of the theory to simulate the fracturing around the stopes. Numerical models based on continuum (Malan 1999) and discontinuum (Napier & Malan 1997) viscoplasticity overcame these problems and allowed for the direct simulation of fracture zone rheology and the resulting time-dependent closure. Simulating mature stopes and calibrating the many parameters in these models are however problematic and therefore a simple empirical approach will be investigated in this study.

Studies showed that the rate of steady-state closure appears to be constant in the short term but it gradually decreases when there is no blasting or seismic activity. This is illustrated in Figure 4.4.1. This particular data set was obtained in a Ventersdorp Contact Reef (hard lava) panel in the Carletonville area. The measurements were obtained over a long weekend when there was no mining activity for several days.

The steady-state closure is best approximated by a function of the form

$$\Delta S_{SS} = a(1 - e^{-bt}) \quad (4.4.1)$$

where a and b are parameters and t is time. The steady-state closure for Station No. 2 in Figure 4.4.1 after the seismic event was plotted in Figure 4.4.2 together with the model given in equation (2.3.1). The parameters used to obtain this fit were $a = 3.85 \text{ mm}$ and $b = 0.015 \text{ h}^{-1}$. Note that these calibrated values are only applicable to this particular stope. From equation (2.3.1), the rate of steady-state closure is given by

$$\frac{dS_{SS}}{dt} = ce^{-bt} \quad (4.4.2)$$

where

$$c = ab \quad (4.4.3)$$

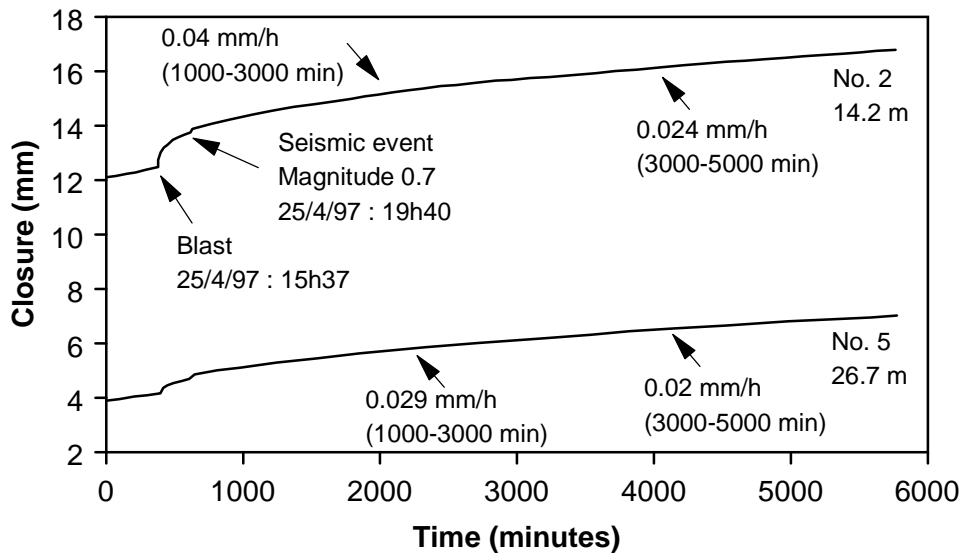


Figure 4.4.1 Closure measured in a Ventersdorp Contact Reef (hard lava) panel when there was no mining activity for a period of four days. The time periods in brackets indicate the intervals used to calculate the steady-state closure rates. Two closure instruments at different distances to the stope face were used to collect the data (after Malan 1998).

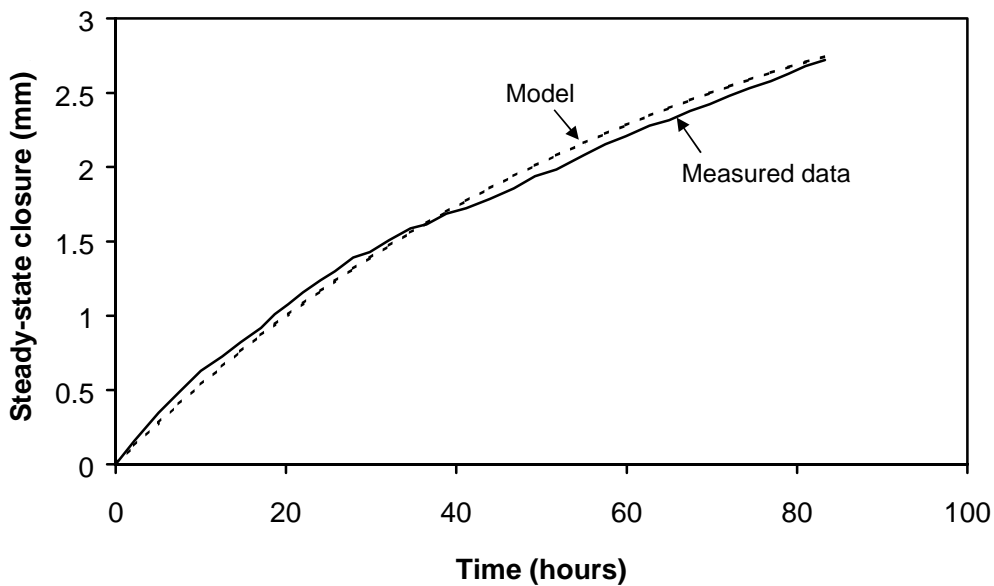


Figure 4.4.2 Measured and simulated values of steady-state closure for the Ventersdorp Contact Reef (hard lava) panel.

From equation (4.4.2) the rate of steady-state closure at $t = 0$ is given by c . For convenience, equation (2.3.1) will be written as

$$\Delta S_{SS} = \frac{c}{b} (1 - e^{-bt}) \quad (4.4.4)$$

Continuous closure measurements (Malan 1999) indicated that the rate of steady-state closure is also a function of measurement position in the panel. This is indicated in Figure 4.4.3 where

the rate of steady-state closure appears to decrease as the distance to face increases. The parameter c in equation (4.4.4) is therefore a function of the distance to face. As the rate of steady-state closure is also a function of the length of face advance on a particular day and the position in the panel along strike, there is some scatter present in the data as illustrated in Figure 4.4.3. From studies described in Malan (1998) where three closure meters were installed at increasing distance to the face, it is however clear that, after any particular blast, the rate of steady-state closure decreases into the back area. The parameter c will therefore be approximated by the following function

$$c = \alpha e^{-\beta d} \tag{4.4.5}$$

where d is the distance to face. From the fit of equation (4.4.5) in Figure 4.4.3, calibrated values for α and β are 0.1195 mm/h and 0.0454 m^{-1} , respectively. Inserting equation (4.4.5) in (4.4.4) gives

$$\Delta S_{SS} = \frac{\alpha e^{-\beta d}}{b} (1 - e^{-bt}) \tag{4.4.6}$$

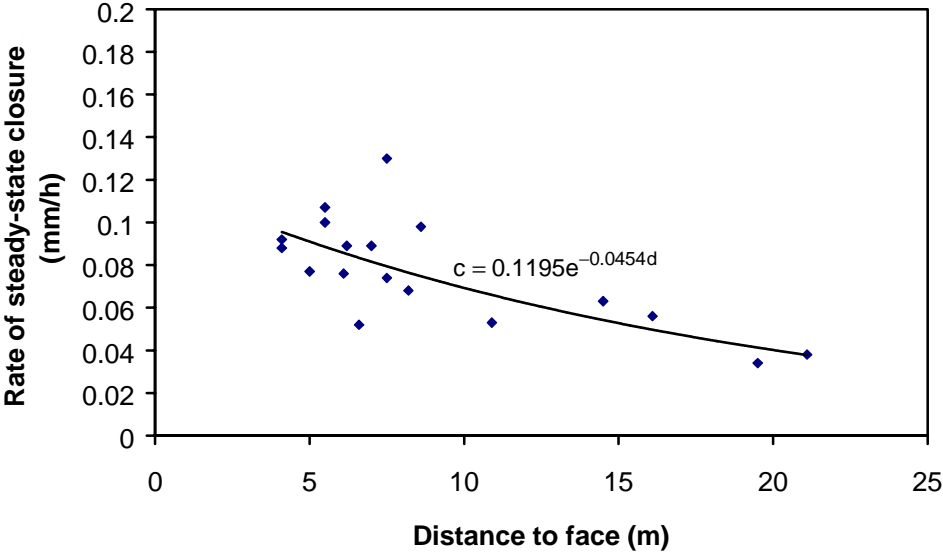


Figure 4.4.3 Effect of distance to stope face on the rate of steady-state closure. Although there is some scatter present in the data with a resulting poor fit to the given function, it will be used as a useful approximation of the trend. The period of time used to calculate the rate of closure was taken from six hours after the blast (to avoid the effect of the primary phase) to 24 hours after the blast (or until the next blast occurs, whichever comes first).

As the decrease in rate of steady-state closure illustrated in Figure 4.4.2 is repeated after every blast, equation (4.4.6) should be further modified to allow for the incremental enlargement of the

stope. If a closure meter is installed at a fixed position in the stope and a number of increments are mined, the total amount of steady-state closure measured at that position will be given by

$$S_{SS}^T = \sum_{k=1}^n \frac{\alpha e^{-\beta(k\Delta\ell+f)}}{b} \left[1 - e^{-b(\tau_k - \tau_{k-1})} \right] \quad (4.4.7)$$

where n is the number of mining increments and τ_k is the time when the k^{th} increment is mined. The distance to face is given by

$$d = k\Delta\ell + f \quad (4.4.8)$$

where $\Delta\ell$ is the size of each mining increment and f is the original distance to face.

Equation (4.4.7) was used to simulate the effect of different mining rates (for a total face advance of 20 m) on the steady-state closure at a measuring point 5 m behind the original face. The size of each mining increment was assumed to be 1 m. The calibrated values for α , β and b obtained from Figure 4.4.2 and Figure 4.4.3 were used. The results are illustrated in Figure 4.4.4. It is assumed that the parameters α and β are not functions of the mining rate.

The effect of different mining rates is clearly visible in Figure 4.4.4. It should be emphasised that the closure plotted in Figure 4.4.4 is only the steady-state closure and does not include the instantaneous or primary closure components. It is assumed that the cumulative contributions of the instantaneous and primary closure phases are not affected significantly by mining rate (to be further investigated in future). If long period closure measurements are available for a particular panel at a known mining rate, equation (4.4.7) can then be used to estimate how a wide range of mining rates will affect the rate of closure. Limited continuous closure measurements in the panel will however be required to calibrate the parameters α , β and b .

From Figure 4.4.4, note that at a distance of 10 m from the face (for support originally installed 5 m from the face), the cumulative steady-state closure is 8.4 mm for continuous mining operations. This gives a steady-state closure rate of 1.7 mm/m. If there is however only one blast a week, the steady-state closure at the same distance to face will be 25.5 mm. The corresponding rate of closure is 5.1 mm/m which is an increase of 3.4 mm/m. Imagine then that the stope is mined using continuous operations (blasting every day). Long period closure measurements during this period (for a closure station installed 5 m from the face) might indicate that the rate of total stope closure for a face advance of 5 m is some value, say x mm/m. If it is decided to decrease the mining rate to just one blast a week, the new rate of closure that can be expected is $(x+3.4)$ mm/m. The current support design should then be tested using this new rate to establish if any changes are required.

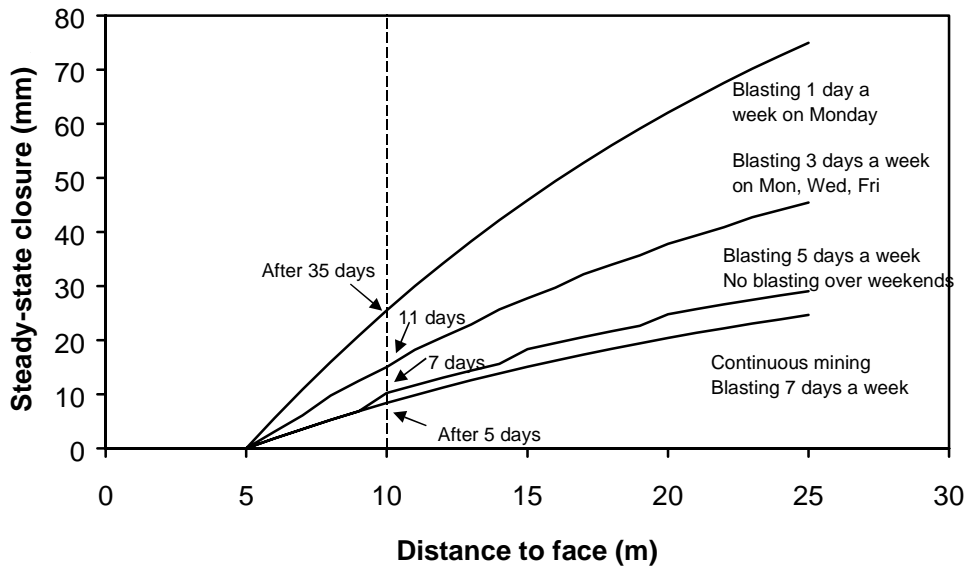


Figure 4.4.4. The effect of mining rate on the total amount of steady-state closure at a particular point in the stope. This is for a VCR (hard lava) panel.

4.5 Conclusions

Although excavations in hard rock are not usually perceived to undergo significant time-dependent deformation, data from the deep gold mines in South Africa illustrates time-dependent closure rates as high as 0.6 mm/h in certain areas. This behaviour is the result of the rheological behaviour of the fracture zone that surrounds these excavations. Continuous closure measurements appear to be a useful diagnostic measure of the rock mass response in different geotechnical areas. It seems that the instantaneous closure response after blasting is a reflection of the face stress before that blast. For tabular excavations in the Ventersdorp Contact Reef (hard lava), the instantaneous closure response is very prominent, but the steady-state closure rate is low in comparison with other areas. These areas also appear to be prone to face bursting. For excavations in certain areas of the Vaal Reef and the Ventersdorp Contact Reef (soft lava), the steady-state closure rate can be as high as 15 mm/day. For these excavations, the instantaneous closure response at blasting time is small and the risk of face bursting appears to be low. Further work is, however, necessary to quantify the possible correlation between risk of face bursting and the closure response of the stopes.

The effect of rate of mining on stope closure was also investigated. A novel technique to estimate expected changes in closure rate for different mining rates was developed by examining the steady-state closure behaviour. Calibration of the analytical model indicated that the mining rate has a noticeable effect on the closure rate. This should be accounted for when designing appropriate stope support.

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5 Seismic Time Sequences

5.1 Foreshocks and aftershocks

Seismic risk is typically managed on three time scales:

Long-term or strategic

Major mine layout decisions such as commitment to the use of strike or dip pillars are made very few times during the life of a mine.

Medium-term

Planning on the scale of months or longer is done in response to unexpected geological conditions.

Short-term

Alerts based on certain seismicity patterns form the basis of short-term seismic risk management. The success of alerts is based on identifying times and places with increased probability of larger, potentially damaging, events.

In this section, we investigate aftershocks and foreshocks of mining-induced induced events at three mines:

Blyvooruitzicht (BVZ) (SIMRAC GAP 020, Toper et al, 1998)

East Rand Proprietary Mines (ERPM) (Milev et al, 1995) and

WDLE (Western Deep Levels East or Tau Tona shaft).

In addition, the data from BVZ included precondition blasts:

Aftershock sequences typically follow Omori's law:

$$R(t) = a(t + c)^{-p} \quad (5.1.1)$$

where

$R(t)$ = the rate of seismicity,

t = time after the main shock,

a = a constant.

c = a small time offset and

p = a constant ≈ 1.0 .

Main shock-aftershock sequences have not been commonly studied for mine seismic events. A noteworthy exception is the aftershock sequence studied by Scott Phillips et al (1997).

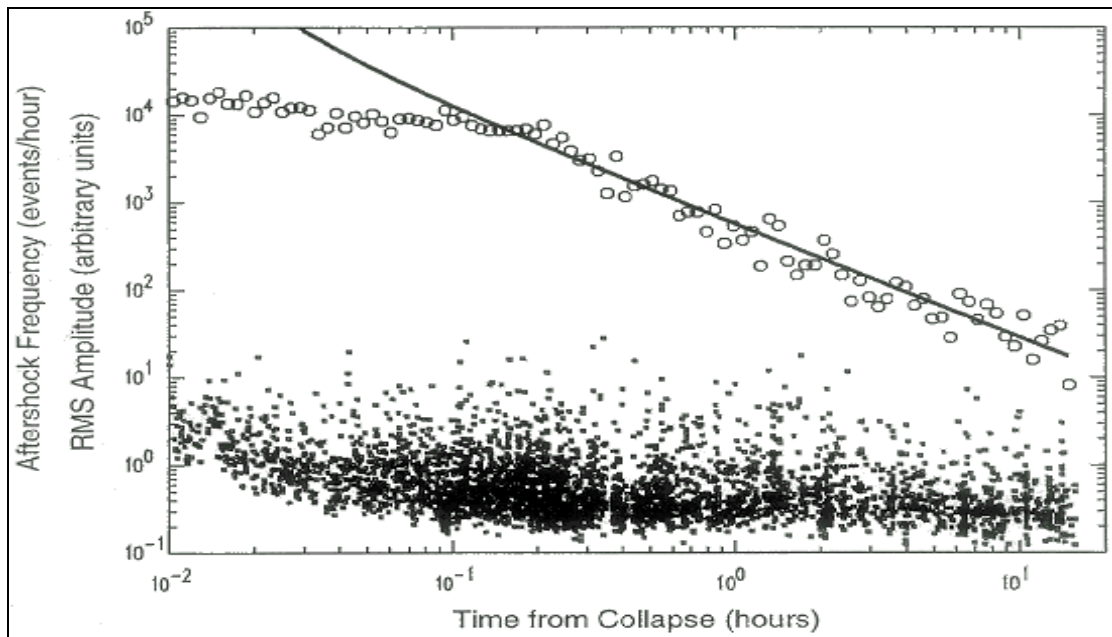


Figure 5.1 Fall-off rate of seismicity following a mass blast (Scott Phillips et al., 1997)

Here we introduce a stacking procedure in which many seismic events within defined times of day and size ranges are considered to be main shocks. The time distributions of all seismic events before and after these main shocks are studied in terms of equation (5.1.1) above using a computer code written for the purpose. Tests performed on a synthetic seismic data set consisting of events randomly distributed in time did not show any time decay and, as was expected, $R(t)$ was not a function of time t for values of time less than the average rate of seismicity.

In Figure 5.2, we show the seismicity rate for events with $M > -1.7$ following 65 preconditioning blasts (Toper et al, 1998). The $\log(R(t))$ vs $\log(t)$ plot is used as a test of the power-law behaviour implicit in equation (5.1.1). Time bins 5 seconds wide, followed by exponentially increasing bin widths were used. We can see that the seismicity decayed as t^{-p} until a background seismicity rate of 8 events per day was reached after about one day.

The decay of seismicity after the precondition blast as seen in Figure 5.2 was very well developed because the mining at this site was isolated from other mining and the face advance rate was very slow. The increased rate of seismicity following the blasts also indicated that the blast resulted in large changes in the rock mass, as planned.

Several other data sets were tested and showed a rapid fall-off, but with a less clear transition from the Omori-type decay to a background level (e.g. Figure 5.2). The sharp fall-off at small times is missing in the WDLE data set. This was offset by a higher rate of seismicity about one minute after the main shocks (see Figure 5.5 below).

As it was difficult in all data sets to separate aftershocks from the background seismicity, it was therefore necessary to adjust equation (1.1.1) as follows:

$$R(t) = R_{\infty} + a(t + c)t^{-p} \quad (5.1.2)$$

where R_{∞} represents the background rate of seismicity.

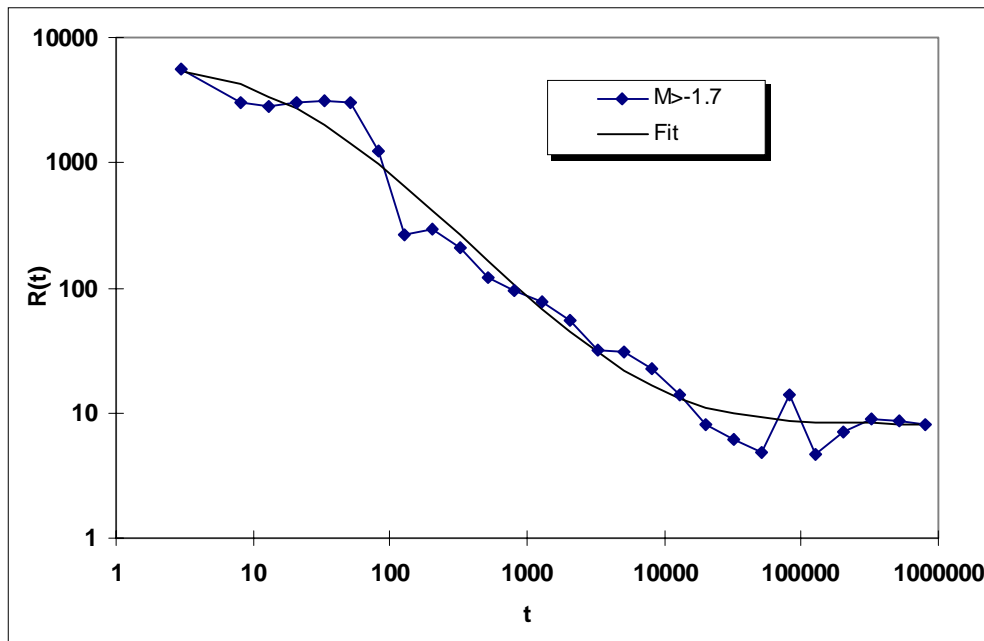


Figure 5.2 Seismicity rate for $R(t)$ in events per day per precondition blast as a function of time t in seconds following 68 precondition blasts. The event rate $R(t)$ is in units of events per day, normalised by the number of main shocks. The smooth line is the fit according to Equation(5.1.2)

Data were then inverted for R_{∞} , a , c and p while minimising the logarithm of the absolute value of the error. Figure 5.2 shows the application of Equation (5.1.2) to events with $M > -1.7$ following 65 precondition blasts. Values of R_{∞} , a , c and p are listed in Table 1.

We then tested the behaviour of larger seismic events outside the blasting windows. Main shocks occurring between 20:00 and 12:00 each day were chosen. In Figure 5.3 we show the decay rates **before and** after seismic events with $M > -1.7$ and $M > 1.0$. Data for fore- and after-shocks were superimposed in Figure 1.3. The only difference between them is caused by the exclusion of blast events when selecting main shocks. Without this exclusion, these graphs would coincide more closely.

At short times, the aftershocks of $M > 1.0$ events were generally several times more numerous than aftershocks of all the events. On the other hand, the $M > 1.0$ events exhibited very few foreshocks, with the fit to $R(t)$ barely rising above the noise level. The irregular curve for the foreshocks of events with $M > 1.0$ was due to the lower number of events (50) and the smaller number of foreshocks. From these data, it seems that we have potentially a better chance of anticipating that small events ($M < 1.0$) are imminent compared to larger events ($M > 1.0$).

The background level following and preceding the larger events was lower than that for the smaller events. One possible explanation is that the larger events occur on structures less intimately associated with the stoping than is the case for the smaller events. It seems that, in general, the smaller events are spatially distinct from the larger events. The most important seismic feature at this site is the up-dip and down-dip edge of the stabilising pillar that was mined (Tooper, 1998). Larger events tended to follow the pillar, whereas smaller events were associated with the advancing faces.

How many precursory events are involved? In Table 5.1.1 we list the cumulate number of seismic events in the four hours prior to, or following, the main shock, with the background rate of seismicity (R_{∞} in Equation (5.1.2) above) removed. This supports the contention that smaller events are more “predictable” than larger events.

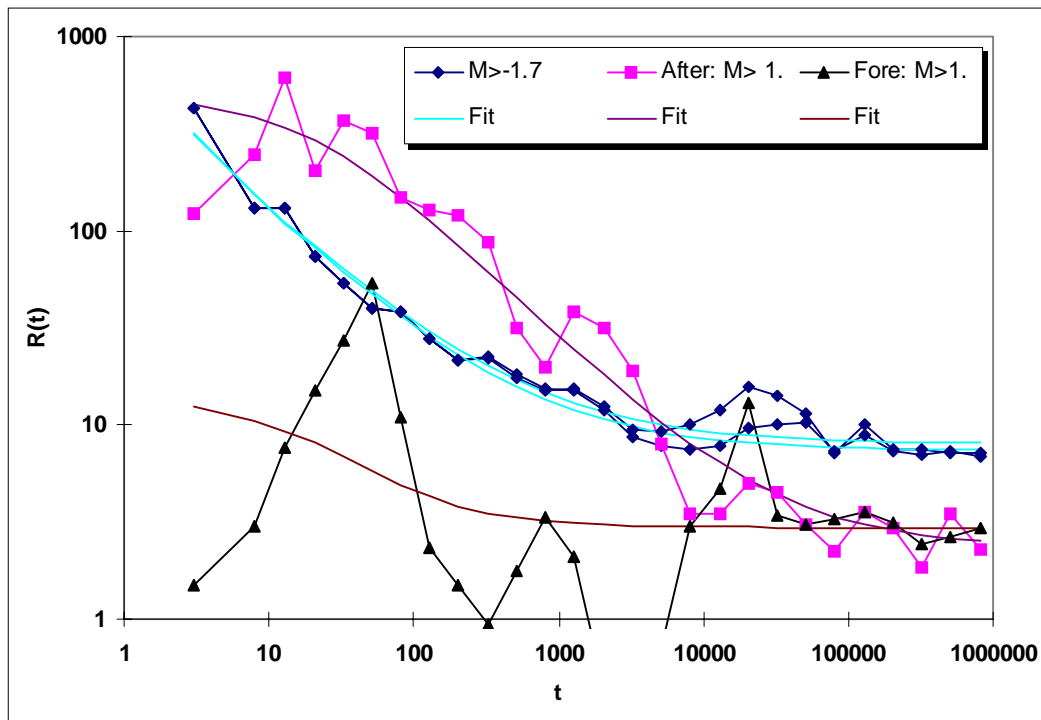


Figure 5.3 Seismicity rate at BVZ in events per day preceding and following events with $M > -1.7$ and $M > 0$. Symbols as for Figure 5.2.

The value of this precursory pattern for small events is, however, of doubtful value as it is results from symmetry. If all events are considered to be main shocks and stacked at time zero, then the time sequence of foreshocks and after shocks is identical. We have, then, a situation in which aftershock sequences also appear as foreshock sequences.

Table 5.1.1 List of parameters for fore- and after-shock sequences analysed here. “Dist” is the search radius on plan for identifying fore- or after-shocks. “Additional events” is the excess over the background in the fitted curve up to 4 hours.

Mine	Number of main shocks	Dist	M_{MIN}	M_{MAIN}	After or fore	R_{∞}	p	Additional events
BVZ	113	50	-1.7	Precon	after	8.1	1.1	6.54
BVZ	47	50	-1.7	1.0	fore	3.0	0.66	0.00
BVZ	47	50	-1.7	1.0	after	2.3	0.74	1.98
SEV	82	300	0.0	2.0	fore	0.22	1.04	0.17
SEV	82	300	0.0	2.0	after	0.16	1.10	0.19
WDLE	326	250	0.0	2.0	fore	4.4	0.54	0.63
WDLE	326	250	0.0	2.0	after	4.2	0.76	1.03

In Figure 5.4, we show an analysis of data from ERPM mine (Milev et al, 1995). These graphs are not as well separated as those for data from BVZ. Aftershocks are only slightly more numerous than foreshocks.

In Figure 5.5 we show foreshock and aftershock time sequences for WDLE. As mentioned previously, there is anomalous behaviour at times less than about one minute due, presumably, to the association methodology of the ISS system involved. Nonetheless, the general behaviour is similar to that of the data from BVZ.

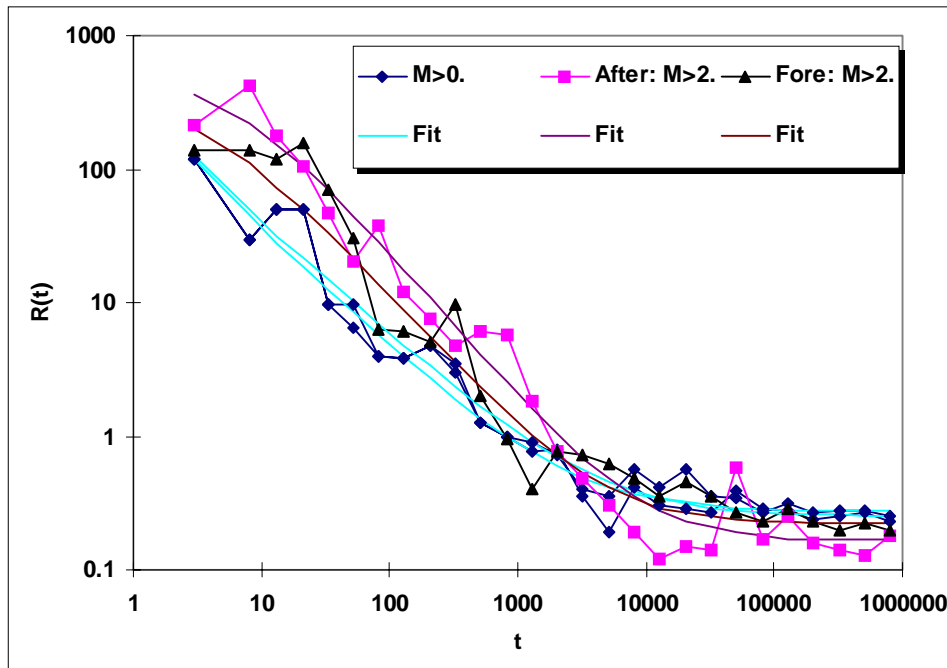


Figure 5.4 Seismicity rate at ERPM in events per day preceding and following events with $M>0.$ and $M>2.0$. Symbols as for Figure 5.2

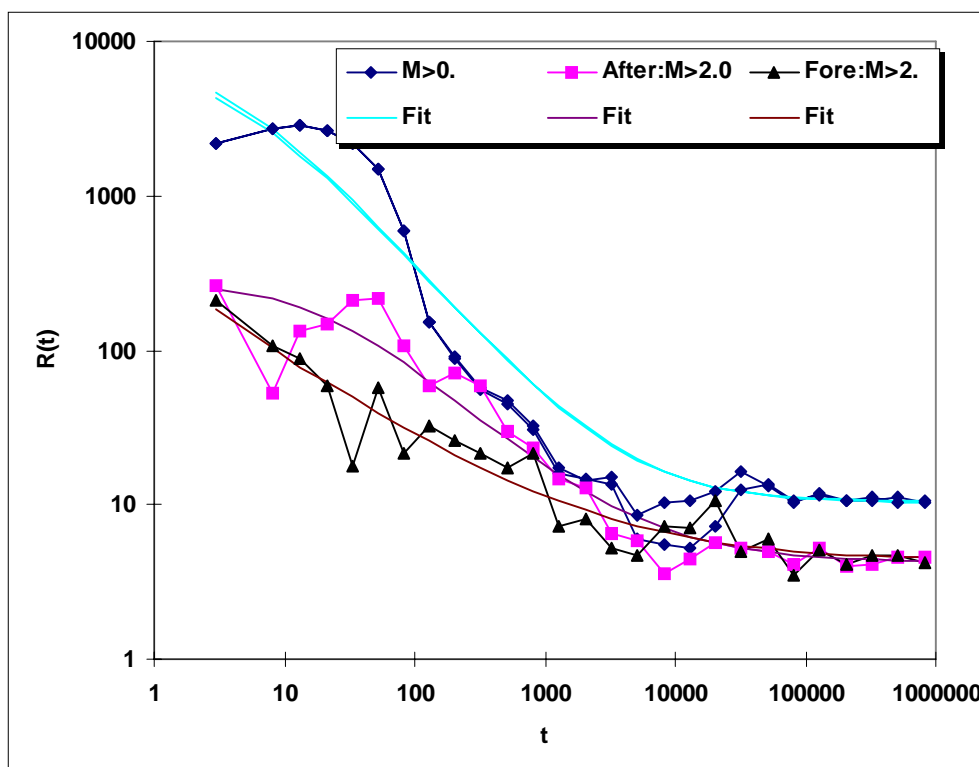


Figure 5.5 Seismicity rate at WDLE in events per day preceding and following events with $M>0.0$ and $M>2.0$. Symbols as for Figure 5.2

The large increase in seismicity, typically 1000-fold in the figures shown here, occurs over only a few seconds to minutes. Table 5.1.1 shows a number of derived parameters, including the “additional events” over and above the background of R_{∞} and extending for four hours before or after the main shock. Only in the case of the precondition blasts did an additional number of more than one event, on average, take place, either before or after the main shocks.

5.2 Conclusions

Aftershock time sequences follow Omori’s law. Foreshocks also show the same patterns, particularly for small events. These results were obtained through stacking of the time sequence of many events outside of the blasting time. On average, less than one additional event took place in the four hours preceding main shocks. Smaller events were better “forecast” than larger events

There are too few events preceding larger events to be used in consistently successful predictions. Perhaps the concentration of seismicity with time could be better described as swarm behaviour than in terms of foreshocks and aftershocks. Seismicity increases due to stress transfer from previous events, but also decreases as the rate of creep, or viscous deformation, dies down.

Although aftershocks are not very common, they occur at a rate well in excess of the background rate for a few hours. This fact should be considered in seismic risk management and people should be withdrawn over a region around larger events. The recommended times and areas should be decided locally, based on agreed concept of “acceptable” risk and using the methodology described above.

5.3 References

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6 Classification of High/Low Seismic Risk Mining Scenarios

6.1 Introduction

In order to quantify high/low risk mining scenarios and the success or failure of seismic early warning, access to detail case study information is required. The researchers found that such information does not exist.

A database in ACCESS was designed for the purpose of this project and with the specific aim to answer question such as the above.

6.2 Available data

Numerous individual case studies on prediction and early warning attempts are reported in the literature. These case studies were conducted in the Welkom, Klerksdorp and Carletonville mining regions. The overwhelming majority was documented with hindsight knowledge.

Appendix A (to this chapter) provides probably the best available set of early warning results that is available. With additional information an evaluation of how much better the warning were than a random issue of warnings. Similar data was not available from other mining regions.

Glazer reports in Chapter 8, that the following was recorded at his #5 Shaft experiment:

- 27% of events took place within 24 hours after issuing the warning.
- 51 % of events took place within 3 days after issuing the warning.
- 49% of events took place between 4 and 23 days after issuing the warning (with one after 41 days).

On the same data set Ebrahim- Trollope et al (1999) reported the following early warning success:

The results of stability analysis and seismic warnings within the 5B north area are as follows:

- Of the 200 events (magnitude>2.0) that occurred where the network was sufficiently sensitive, 160 (80%) had a seismic warning between 0 – 10 days prior to the event;
- a further 11 (6%) occurred between 11 to 40 days after a seismic warning;
- 29 events (14%) occurred without an instability being identified or only manifesting itself very shortly (minutes – a few hours) before failure;
- 9 seismic warnings were not followed by a relatively large event; and
- 29 seismic warnings resulted in events with magnitude between 1.7 and 2.0.

The above were found to be the only quantification of the early warning results in the industry

6.3 Proposed database structure

An ACCESS database was compiled for capturing the appropriate information associated with each individual early warning. This should only be a component of a larger risk database including seismic data, stope information, geotechnical data and worker exposure.

An input sheet for the early warning information is given in Appendix B. The input parameters were specified as follows:

Event Parameters

- Event ID
- Day of the week
- Seismic Moment
- Seismic Energy
- Magnitude

Polygon Information

- Approx. volume
- Number of events > Mag. 2 in previous 6 months
- Gutenberg-Richter b-value
- Gutenberg-Richter a=value
- Mining method
- Preconditioned?
- Backfill?
- Average face advance rate
- FULCO?
- Average ERR?
- Geotechnical area
- Time since warning

Early Warning

- Seismic system
- Event rate
- Apparent volume
- Energy Index
- Apparent Index
- Stability concept
- Schmidt number
- Deborah number
- Other

Damage and Mechanism

- Reported u/g damage?
- Production lost
- Event mechanism

Management decision

- Was early warning communicated?
- Was warning considered in the production decision-making?
- Actions taken

6.4 Conclusion

The researchers found that a coherent data set of seismic early warning case studies did not exist. Similarly, it was not possible to correlate seismic data with any safety or damage information. A relational database including the above will be a prerequisite for risk assessment.

6.5 References

Ebrahim- Trollope, R., Glazer, S.N., Stankiewicz, T. Kijko, (1999) A. Seismological Analysis for a Scattered Mining Environment- Methods and their Limitations, SARES 99 2nd Southern African Rock Engineering Symposium

1.6 Appendix

RESULTS OF THE 5B EXPERIMENT USING

THE INSTABILITY CONCEPT (DECEMBER 1994 - MAY 1996)

•	POLYGON	WARNING	DATE ISSUED	DATE OF EVENT	TIME SPAN (Days)	M _L
1	Pillar 1a	X	28/12/94	30/12/94	2	3.4
2	Pillar 34			05/01/95		2.7
3	5 Bn	X	14/01/95			
4	Trb 2			18/01/95		2.9
5	Pillar 11	X	01/02/95	02/02/95	1	2.8
6	Pillar 34	X	02/03/95	02/03/95	> 1	3.3
7	56 a	X	03/03/95	06/03/95	3	2.7
8	55 - 54	X	27/02/95	22/03/95	23	2.7
9	Trb 2	X	09/03/95	25/03/95	16	2.7
10	Pillar 14	X	24/03/95	25/03/95	1	1.7
11	Pillar 34	X	05/04/95 08/04/95	10/04/95	5 2	2.7
12	56	X	09/03/95 06/04/95	20/04/95	41 14	2.4
13	Trb 2	X	10/05/95			
14	56 b	X	26/05/95	06/06/95	11	2.5 2.2
15	56 a			08/06/95		2.7
16	58	X	29/05/95	09/06/95	11	2.2
17	55, 56, 34, 56 a	X	28/06/95			
18	54			16/07/95		2.7
19	34			03/08/95		3.3
20	34			03/08/95		3.5
21	54 b, 55, 56 a	X	07/08/95			
22	58	X	07/08/95	11/08/95	4	2.6
23	54 b, 56 Pillar 14	X	30/08/95 20/09/95	22/09/95	23 2	2.1
24	54 b, 56 Pillar 14	X	22/09/95	25/09/95	3	2.6

25	54	X	20/09/95	29/09/95	9	2.3
26	56 a	X	20/09/95	29/09/95	9	2.6
27	55 s	X	04/10/95	04/10/95	2 hrs	2.1
28	55 - 56	X	12/10/95	12/10/95	6 hrs	2.7
29	54	X	16/10/95	16/10/95	2 hrs	2.2
30	54	X	20/10/95	27/10/95	7	2.2
31	54			07/11/95		2.5
32	55 s			08/11/95		2.5
33	55 - 56	X	06/11/95	10/11/95	4	2.3
34	54	X	22/11/95	24/11/95	2	1.8
35	54	X	22/11/95	30/11/95	8	2.1
36	55 s			01/12/95		2.5
37	54	X	02/12/95	07/12/95	5	1.7
38	54, 55 w, 55 e, 55 s	X	14/12/95	14/12/95	3 hrs	1.7
39	55 w	X	14/12/95	15/12/95	1	2.8
40	56 a	X	06/12/95	15/12/95	9	2.1
41	5 b	X	18/12/95 20/12/95	21/12/95	3 1	2.8
42	56 a	X	20/12/95	23/12/95	3	2.7
43	55 s, 54	X	28/12/95	30/12/95	2	2.1
44	54	X	28/12/95	03/01/96	6	2.5
45	55 s	X	28/12/95	11/01/96	14	2.9
46	55 s	X	12/01/96	12/01/96	10 hrs	2.6
47	54	X	08/01/96			
48	55 - 56			13/01/96		2.2
49	56 a	X	02/01/96	16/01/96	14	2.3
50	56 a	X	22/01/96	29/01/96	7	2.8
51	55 - 56	X	22/01/96	02/02/96	10	1.8
52	56 a	X	01/02/96	07/02/96	6	2.6
53	55 - 56	X	22/01/96	11/02/96	19	2.8
54	56 a	X	09/02/96	11/02/96	2	1.9
55	56 a	X	13/02/96	15/02/96	2	1.8
56	54	X	12/02/96	15/02/96	3	2.3

57	5 b	X	13/02/96 22/02/96	22/02/96 23/02/96	9 1	2.0 2.7
58	55 - 56, 54	X	23/02/96	24/02/96	1	3.4
59	55 w	X	23/02/96	24/02/96	1	2.0 2.6
60	55 e	X	23/02/96	25/02/96	2	3.0
61	5 b	X	09/03/96	09/03/96	4 hrs	2.5
62	5 b	X	20/03/96	25/03/96	5	2.3
63	56 a	X	20/03/96	03/04/96	14	2.2
64	54	X	29/03/96	04/04/96	6	2.8
65	56 a	X	16/04/96	17/04/96	1	1.9
66	55 - 56	X	04/04/96 16/04/96	17/04/96	13 1	1.8
67	55 - 56	X	16/04/96	24/04/96	8	2.2
68	54 South	X	29/04/96	01/05/96	2	3.0
69	55 - 56	X	29/04/96	02/05/96	3	2.7
70	56 a	X	29/04/96 05/05/96	08/05/96	9 3	2.2
71	54 n	X	10/05/96	10/05/96	2 hrs	2.5
72	56 a	X	10/05/96 14/05/96	15/05/96	5 1	2.3
73	56 a	X	15/05/96	18/05/96	3	2.3
74	55 - 56	X	21/05/96	27/05/96	6	2.2
75	55 - 56	X	27/05/96	28/05/96	1	2.3
76	56 a	X	21/05/96	30/05/96	9	2.7
77	5 b	X	21/05/96	30/05/96	9	2.6

7 Rock Related Risk Assessment Techniques on the Gold and Platinum Mines

7.1 Introduction

Enabling output No 1 calls for the 'Evaluation of seismic and rockburst risk assessment techniques'. This was interpreted as seismic risk assessment only. Later in the course of the project it was realised that, to fully appreciate the extent of activities in this regard, out in the field, it would be necessary to evaluate all rock engineering risk assessment techniques. This was therefore subsequently included in the work plan. No additional funding was requested.

An attempt was made to cover all major groups and regions both for gold and platinum. A technique employed in Western Australia has also been included.

The findings have been systematically tabled in Appendix A. All the specific mines mentioned were visited and appropriate rock engineering staff was interviewed. It must be stressed that the techniques described are those currently in use on these mines.

7.2 Methodology

The areas and mines covered were:

- Western Areas/Placer Dome South Shaft
- Kloof Gold Mine
- Anglo Gold West Wits Operations
- Tau Tona Mine
- Elandsrand Gold Mine
- Vaal River Operations and African Rainbow Minerals
- Great Nologwa Mine
- Tau Lekoa Mine
- Hartebeestfontein Gold Mine
- Matjhabeng Mine
- Rustenberg Platinum Mines – Rustenberg Section
- Rustenberg Platinum Mines – Amandelbult Section
- Rustenberg Platinum Mines – Union Section
- Impala Platinum Limited
- Dept. of Minerals and Energy – Western Australia
- CSIR Miningtek – Rock Risk Expert System

The techniques used are basically similar in that they cover the assessment of both regional and local rock engineering parameters in order to reach a risk assessment level. The complexity, however, varies considerably. Each technique is customised to the particular mine, geotechnical conditions and purpose of the assessment. Weightings are generally used to emphasise the risk level of certain parameters.

In most cases the assessment is based largely upon data readily available on surface such as that measurable on a mine plan, seismic data, results of numerical modelling etc. The inclusion of significant, up to date, results of underground inspections, is rare. Where this is the case, i.e. where resources exist to collect such data, the results of the assessment and efficiency of the corrective actions appears to be greatly improved.

The management of the assessed risk takes a number of forms on the various mines and includes

- discussions and decisions (at planning meetings) regarding required actions to manage high risk levels;
- drawing up of summary sheets (normally monthly) to facilitate discussion at planning meetings;
- input of assessments on to a database mainly for the determination of trends;
- use of summary sheets as part of the production planning process;
- reporting of risk ratings to senior management;
- inspections by senior personnel and rock engineers of very high risk panels;
- declaration of special precautionary areas;
- presentation of monthly risk assessment results to the Mine Overseer of the section prior to planning meetings;
- a zero tolerance approach for high risk levels, i.e. panels are stopped;
- a monitoring programme to check on compliance and efficiency of the recommended actions;
- use of trained observers underground who are empowered to make on the spot decisions regarding corrective actions or whether it is safe to continue mining; and
- active involvement of trained production personnel in risk assessment to the extent that they are empowered to recommend corrective action.

The first nine points are all important to expedite and facilitate the risk management process. The last three points are rarely present but strongly recommended to improve the efficiency of both the assessment and the management of rock related risk.

7.3 Conclusions

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:

- Subjectivity.

It is extremely difficult to ensure even reasonable consistency in the assessment of risk levels. This is despite the detailed procedures and careful weighting of parameters etc. that are in place in some cases.

- The general lack of assessments done by personnel external to the mine.

It is strongly recommended that external audits be performed regularly to ensure that risky practices condoned (in some cases unintentionally) by the mine be minimised or eliminated.

- Lack of sufficient rock engineering resources particularly trained and certificated personnel.

In most cases the part of the assessment that is possible, using readily available data on surface, is reasonably well covered. Other essential data from underground in the form of up to date assessment of rock conditions, support standard compliance and effectiveness etc. is not adequately covered, however. On two of the mines visited this problem has been largely overcome by the use of trained observers and production personnel.

- One of the most important factors, namely that of rockburst risk, has proved 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.
- 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.

Results and advantages as seen by users out in the field at the moment are:

- 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 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.
- On one mine the system is believed to have contributed significantly by reducing the number of large seismic events. Assessment also reliably forecasted most working areas with increased seismic levels. The system was, however, less successful in predicting rockburst accident levels.

-

7.4 Future Work

The need for future work to improve the systems in place on the various mines was recognised in all cases. This includes:

- the training of observers, safety and/or production personnel to assist with risk assessment and management;
- the introduction and implementation of follow-up procedures;
- initiating regular external audits;
- improving the techniques by making them less qualitative and more quantitative;
- inputting all information on to a database to allow trend analyses;
- introducing direct, up to date, underground data such as rock mass ratings, support compliance and efficiency, into the equation;
- using instrumentation to detect unfavourable structures in the hangingwall and ahead of the mine face;
- finding ways to reduce subjectivity when assessing risk.

7.5 References

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ROCK-RELATED RISK MANAGEMENT SYSTEMS ON VARIOUS GOLD AND PLATINUM MINES - APPENDIX A (Page 1)

	Vaal River Operations and African Rainbow Minerals	Matjhabeng Mine	Western Areas/Placer Dome South Shaft	Great Nologwa Mine	Tau Lekoa Mine
Name of system	The Panel Rating System	Stope and Development Rock Rating System	Stope Appraisal Sheet and Panel Audit System	Hazard Register	Shallow Mining Panel Risk Rating System
Contact and/or ref.	George Brinch, Dunn and Laas (1999)	G Myburgh, Interactive software on MS Excel	Sandor Petho, Gerritsen and Saloojee (1998)	Johan Oelofse, Koeloe Joubert, George Brinch	Johan Oelofse, Judeel and Laas (1999)
Aim and background	Identify rock-related hazards and associated risks and then to provide guidelines for actions required to reduce the risk. This would be designed to satisfy the requirements of the DME guidelines for codes of practice (MRAC Task Group, 1999) and would also formalise the declaration of special areas. The latter had always been a problem on these mines. First started in 1994 and has evolved since then.	The Stope and Development Rock Rating System currently in use consists of a simplified questionnaire and allows a particular stope or development end to be rated A,B or C depending upon the assessed risk. Corrective actions and support measures to fit the rating are then recommended.	The rock-related risk management system on Western Areas Ltd (known as the REMUS system) consists of a "Stope Appraisal Sheet" functioning as regional hazard identification system and a Panel Audit System functioning as a local (panel specific) hazard identification system. In both cases simplified, one-page sheets have been designed for ease of use. The system was initiated as a result of recommendations given to the "Guidelines for the compilation of a Rock Engineering Code of Practice", issued by the DME.	The "Hazard Register" is an AngloGold corporate initiative currently being introduced via the Safety Departments on AngloGold mines. The aim is to identify hazardous risks underground by section. The system includes mechanical, electrical and environmental hazard groups in addition to strata control.	Tau Lekoa mine (mining VCR at between 900 and 1 600 m) uses crush pillars as part of their in-panel support system. The relationship between fall-out-height and span between these pillars is used in their design. The "Shallow Mining Panel Risk Rating System" was developed as a monitoring and risk management tool to determine the degree of compliance to pillar standards.
Applicable to:-	Geologically complex, intermediate to deep tabular mines.	Intermediate depth, scattered mining environment	Intermediate depth	Intermediate to deep tabular mines	Shallow mines incorporating crush pillars as part of the support system.
Risk assessment parameters	Local factors <ul style="list-style-type: none"> • Inter-panel mining sequences (lagging) • Stress concentrations due to leads/lags • Siding lag or gully lead (flat fracturing problems) • Approach to geological structures (distance and angle of panel face to structures) • Presence of adequate second accesses Regional factors (weighted) <ul style="list-style-type: none"> • Geological complexity (number and type of structures) • Mining span (takes into account effect of closure) • Rockburst risk (position of panel relative to seismically active structures) • Face stress regime (by numerical modelling) 	Local features (Stope) <ul style="list-style-type: none"> • Stress fractures, brows, bed separation • Blast damage • Distance to holing • Face shape and lag • Support to face distance • Stopping width and appropriate support • Presence of faults, dykes, cross bedding • Stope access & escape ways (condition) • ASG and gully (span, lead & brows) • Escape Gully (presence and condition) Local features (Development) <ul style="list-style-type: none"> • Sidewall fracturing (extent & support) • Faults, dykes (presence and support std) • Bedding planes (presence & thickness) • Temporary support (presence and std) 	Regional factors (Stope Appraisal Sheet): <ul style="list-style-type: none"> • Leads/lags • Face shape • Special area or not (?) • Presence of major geological features • Stopping widths • Backfill to face distance Local factors (Panel Audit System) <ul style="list-style-type: none"> • Support type and standard of installation • Ground conditions <p>The abovementioned are divided into sections relating to the face area and to the gully area.</p>	Strata control parameters: <ul style="list-style-type: none"> • Presence, standard and condition of support of brows; • Adequacy of barring; • Presence, standard and condition of support in vicinity of faults; • Marking, drilling and charging of blast holes; • Presence and positioning of barricades; • Presence, standard, condition and type of support in special areas; • Mining configuration in special areas; • Backfill standard in special areas; • Seismic alert (or not) for the area; • Presence, standard and condition of temporary and permanent support. Non-strata control parameters: <ul style="list-style-type: none"> • Mechanical • Electrical • Environmental. 	The following parameters are assessed from the 1:200 plans: <ul style="list-style-type: none"> • Face length (the maximum stable dip span between strike crush pillars); • Mining spans (the minimum span on strike between pillars); • Crush pillar cutting (compliance to a width to height ratio of between 1.8 and 2.0 m); • Geology (complexity); • Weak side safety pillars (the positioning of pillars relative to the "weak" or "strong" side of geological feature); • Panel lag on geological structure (extent that a panel lags onto a geological feature); • Local high stress conditions (causes scaling on unconfined crush pillars and slip on adversely orientated geological features).
Risk management	Five action levels have been identified, each associated with a specific panel rating and certain conditions. Rock Engineers scrutinise the recently surveyed panel plans monthly and rate the panels. Summary sheets are drawn up and discussed with the relevant personnel. The relevant actions, calling for applicable support resistance/energy absorption requirements, are immediately effective. On Great Nologwa Mine use is made of what is known as SHIC (Safety Hazard Identification Index). Conditions are predicted and recommendations made accordingly. These are prominently displayed on the 1:200 plans.	Appropriate production personnel are informed at regular meetings. Recommendations are followed.	The Stope Appraisal Sheet aspect has become part of the production planning process and is undertaken on a monthly basis. The Panel Audit System forms part of a quarterly underground evaluation of all panels. In view of the vast quantity of data, it was decided to set up a database to facilitate analysis and access by appropriate production personnel. Ratings on the Panel Audit System are combined to give overall rating of poor, moderate or good. The system allows the user to search for data relating to a specific panel. The risk assessment may then be viewed or printed.	Safety officers underground assign an A, B or C rating to each parameter. This is entered into a database on surface after being automatically weighted according to a pre-determined list. An accumulated rating is then calculated and categorized into a low, medium, high and very high-risk index. The software was written by GMSI, i.e. external to AngloGold. Follow-up procedures are carried out and depend upon the risk index category. The follow-up normally includes visits to the sections by senior production and safety personnel. Recommendations are then made for corrective action. The level and urgency of the visit depends upon the calculated level of risk.	The abovementioned parameters are rated between 0 and 3. Each is allocated a weighting ranging between 3 and 6 depending on the severity. Ratings are multiplied by the allocated weighting factor and then added together to determine a total. This total is then related to a final report rating between 1 and 5. The panel risk rating results are discussed during monthly pre-planning sessions. The rating results are also reported in writing to the General Manager of Tau Lekoa Mine and down to Mine Overseer level. The report consists of the following: <ul style="list-style-type: none"> • The risk ratings are obtained for each working panel as well as the recommended actions for each of the seven risk rating parameters; • The three worst panels of each Mine Overseer section are highlighted for intensive care to reduce the risk; • A summary is given in table form showing the percentage of panels, per mine overseer section, not to standard regarding, face length, mining span, mid-panel pillar width to height ratio and strike pillar widths. The previous month's results are also included in the table to determine whether there are improvements or not; • A summary is made in table form showing the percentage of panels per risk rating 5,4 and 3, for each mine overseer section. The previous month's results are also included in the table to determine whether there are improvements or not; • Trend graphs are included. Members of the Rock Engineering Department visit panels with a high-risk rating. They decide whether actions can be taken to reduce the risk. If the risk cannot be reduced it is recommended that these panels be stopped.

<p>Shortcomings and Future work</p>	<p>Face stress regime should, but as yet cannot, take into account the tectonic setting of the panel. The rockburst risk is difficult to quantify and is probably the most significant of all factors. The probability of rockbursting based upon percentage of damaging events, damaging magnitudes and distance between damaging events is being considered. (the outcome of this SIMRAC project i.e. GAP608 and that of GAP530 that looked at site response should help in this regard.)</p> <p>Following up of recommendations by all parties concerned is of utmost importance.</p> <p>Further work is required to develop a less qualitative and more quantitative method for rating different parameters. Also the weighting factors need to be reviewed by statistical back analysis of the agents contributing to rock-related accidents in stopes.</p>	<p>Much of the current mining at Matjhabeng Mine consists of the removal of remnant pillars. The Stope and Development Rating system described above is not applicable. A form outlining special instructions for remnants and special areas exists and is used but in the opinion of the Section Head, Rock Engineering is often incorrectly applied and in most cases inadequate, be it correctly applied or not.</p> <p>CSIR Miningtek was requested to assist in the development of an effective risk assessment technique for remnant pillars. (It was decided to do this as part of SIMRAC project GAP608 without any direct charge to Matjhabeng Mine). (See comments below).</p>	<p>A method needs to be determined to combine the two systems, i.e. the Stope Appraisal Sheet and the Panel Audit System. There is also much subjectivity associated with the systems and a statistical approach for the determination of final risk is required to alleviate this problem.</p> <p>The acceptance of erroneous data into the database also needs to be addressed by means of a more rigorous code to reject such data.</p> <p>The scope of information derived from the database also needs to be broadened allowing depiction of statistical trends, for instance, thereby facilitating overall management of risk.</p> <p>Another area of future research would be the integration of data from the mines seismic system into the estimation of hazardous ground conditions. As the extent of mining in the area increases, seismicity is expected to play an increasingly more influential role in risk determination.</p> <p>Unfortunately the abovementioned enhancements cannot be undertaken until a suitably qualified and experienced person is employed.</p> <p>A further development would be the incorporation of an outcomes based training module for underground personnel to mitigate recurring unfavourable trends identified by the risk assessment systems.</p>	<p>The "Hazard Register" is not used in conjunction with the Panel Rating System on Great Nologwa Mine. The former deals with ratings on a section and not a panel level. In the view of the Rock Engineering Section Head, this is a serious shortcoming.</p> <p>Subjectively, according to those consulted, is a serious problem and is currently being addressed on Great Nologwa mine.</p>	<p>Geological complexity will, in future, be augmented by underground rock mass ratings. Consideration is also being given to using GPR and seismic surveys to delineate dome structures with other previously unknown geological features in the hangingwall.</p>
<p>Results</p>	<p>Many of the past problems associated with the declarations of special areas have been cleared up. The system is accepted both by management and production personnel and forms an integral part of the monthly planning meetings. Improved communication between rock engineers and production personnel.</p> <p>The scrutinising of stope plans monthly allows the rock engineer to be pro-active and can be used to focus on certain areas and panels. The system does not replace underground trips.</p> <p>According to Nicolau (1999) use of the system has resulted in the lowest Lost Time and Reportable rates in the past five years, and the achievement on one million fatality free shifts – twice – at Great Nologwa mine.</p>	<p>The Stope and Development Rating System is useful in normal stoping situations at Matjhabeng Mine. The effect that it has had on accident statistics since it was first introduced has as yet not been fully evaluated. The system is not applicable for risk assessment and management when dealing with the removal of remnant pillars.</p>	<p>The utility of the system as it is currently used is somewhat limited by the following:</p> <ul style="list-style-type: none"> • The Rock Engineer who originally conceived the system and wrote the software has since left the company; software limitations and required enhancements have not been attended to as a result. • Erroneous data is accepted into the database; • Subjectivity needs to be addressed; • Weighing of the various factors needs refinement; • The stope appraisal sheet and panel auditing system needs to be combined; • Facilities to statistically analyze the data to expose trends needs to be developed; • A method of introducing seismic risk assessment needs to be developed; <p>The data input aspect is time consuming and needs to be simplified without reducing the quality of the system.</p>	<p>The system has been in operation for about three months. It is still too early to judge the effectiveness thereof.</p>	
<p>Comments</p>	<p>Cannot be seen as a "black box" providing exact solutions. A degree of engineering judgement and experience is still necessary.</p> <p>Nicolau (1999) says that the system is under constant review and revision and it is hoped to train all stoping employees in the use of the system, It is believed that this will further improve the safety statistics. The risk assessment requirements of the Mine Health and Safety Act are met on an on-going basis. Falls of ground are addressed in a cost effective manner. Although the system is based on fairly complex input and calculations, the outputs of these calculations are translated into meaningful actions by production personnel.</p>	<p>The advantage of an effective risk assessment technique for remnant pillars is (i) improved safety and productivity by means of the application of an appropriate mine layout, mining method and support system and (ii) the determination (prior to any major expenditure on access ways, equipping etc) of whether a remnant is in fact worth mining or not.</p> <p>Essentially what is required is something similar to a 'Panel Rating System' as described by Dunn and Laas (1999) but focussing in on remnant pillars. A first step could be a modification of the aforementioned system by bringing in and emphasising certain aspects that play a major role in determining conditions when mining in the vicinity of the Basal Reef. One such obvious aspect would be the thickness of the quartzite beam that lies between the Basal Reef itself and the overlying shales. The thickness of this beam has both safety and economic implications as it determines whether undercutting the shales is deemed possible or not.</p> <p>CSIR Miningtek (as part of SIMRAC project GAP608) is currently carrying out a survey of risk</p>			

		<p>assessment techniques in use on the gold and platinum mines. In addition to satisfying the required outputs of the SIMRAC project it is intended to recommend a list of likely criteria to be included in a risk assessment system for remnant pillars on Matjhabeng based upon the findings of the survey. Based upon the initial discussions with the Rock Engineering Section Head at the mine this would not likely include as broad headings:</p> <ul style="list-style-type: none"> - Geology (Beam thickness, presence of major geological features, etc.) - Seismic History - Energy Release Rate (gives idea of stress span, etc.) - Reef Geometry (Dip, stoping width in the immediate vicinity.) - Escape ways (Presence and condition) <p><i>As the current survey progresses it will become possible to add to and expand upon the existing listed parameters, outlining possible ways in which they can be used for risk assessment.</i></p>			
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ROCK-RELATED RISK MANAGEMENT SYSTEMS ON VARIOUS GOLD AND PLATINUM MINES - APPENDIX A (Page 2)

	Dept. of Minerals and Energy – Western Australia	CSIR Miningtek – <i>RockRisk</i> Expert System	Amandelbult Section - Rustenburg Platinum Mines	Kloof Gold Mine	Amplats. Rustenburg Platinum mines. Union Section.
Name of system	MOSHAB Geotechnical Risk Assessment	RockRisk (An expert system)	Risk Assessment Programme	Continuous Rock-Related Risk Assessment	Stope Panel Risk Assessment
Contact and/or ref.	MOSHAB (1999)	Webber (1996)	Akermann 1999, Karl Akermann	Deon Geyser	Gavin Potgieter
Aim and background	The purpose of the Geotechnical Risk Assessment (GRA) is to Identify areas of the mine where the level of ground support needs to be upgraded. It is assumed that the necessary primary rock reinforcement is in place to ensure general structural stability of the excavation. GRA forms part of a more general Code of Practice. Installation of surface (areal) rock support in addition to existing or primary reinforcement methods may be necessary to control the risk of injury or death that can result from small rockfalls from between the installed rock reinforcement methods. Surface (areal) rock support is required in <i>all</i> headings greater or equal to 3.5 m in height.	RockRisk is an expert system designed to assess the risk of rockbursts occurring under a given set of circumstances allowing the user to then take proactive measures to minimise the risk. It is possible to assess the risk very quickly and also it serves as an educational tool for inexperienced rock engineers. The development of RockRisk was to a large degree prompted by the reasonable success of two systems namely the "Mining Alert Level" developed by F Naude and " PillRisk " developed by P K van der Heever	Used to identify high risk areas requiring special support specifications and recommendations.	A stope panel risk rating database is maintained by the rock engineering dept as an aid in the determination of special areas and as a monthly hazard assessment.	A rock engineering resource problem sparked the need for the active involvement of production staff in risk assessment. Two risk assessment spread sheets i.e. for the UG2 and the Merensky reef horizons were drawn up and are completed monthly by a trained shift supervisor. Aim is to identify high risk areas requiring special support specifications and recommendations.
Applicable to:-	Western Australian underground mines. The Code and GRA as used here are only applicable to gravity-induced rock falls. Dynamic rock failure mechanisms are not covered.	Intermediate to deep tabular mines with a rockbursting problem.	Shallow, scattered mining environments	Intermediate to deep tabular VCR mines with a rockbursting problem.	Shallow, scattered mining environments particularly on the platinum mines.
Risk assessment parameters	Ground conditions: Geology of the rock mass; Geotechnical information on the planes of weakness; Mechanical properties of the intact rock, planes of weakness and the rock mass; Magnitude and orientation of the three-dimensional, pre-mining rock stress field; Magnitude and orientation of the induced rock stresses caused by the mining process (creation of voids); Potential rock failure mechanisms; Blast damage to the rock mass; Likely scale and nature of the ground movement; Possible effects on the working places and installations; Previous experience and relevant historical data; and Groundwater and exposure to the atmosphere The above factors are considered in relation to any proposed excavation designs and any existing excavations. Determine areas of the mine where rock support is likely to be needed such as: Areas where ground conditions are such that loose rock is likely to develop over time; Areas which generate loose material and constantly require scaling; Areas which are subject to increases or decreases in stress; and Areas, which are subject to significant, blast vibrations. Mining method: Where workforce is exposed to wide spans for example. Exposure risk: Surface (areal) rock support is required where there is regular employee access, for example workshops	Layout: Mining method Type of mining Stoping width Panel layout K-ratio Depth Reef uniformity Reef dip Regional support: Type Effectiveness Local support: Type Effectiveness Support resistance Number of rows Management strategies: Safety record Audits held? Planning strategies Numerical modelling Geology: Structural properties Method of approach Extent of mining Distance from structure Seismic history: Rockburst history Seismic activity Diurnal changes Spatial changes B-value changes Access ways: Gully support Second escape way? Tunnel properties	Main factors considered in the risk assessment are: - Hangingwall pyroxenite beam thickness Rock Mass Rating (RMR) Geological complexity Support spacing Excavation span Mining sequence Pillar size Water	Eight parameters are taken into account and weighted, resulting in a risk rating for each individual panel. This is done monthly. Ratings are limited to the values of 0,1,2,3 equating to low, medium, high and very high risk respectively. The parameters are:- ERR, calculated monthly by means of numerical modelling. Stoping width. High stoping width i.e.>2,2m considered as very high risk. Seismic rating. From seismic database and is the frequency of seismic events multiplied by the severity of the potential hazard. Leads and lags. 1:1000 plans. Mining type. Includes remnant mining. Type of structure. Geological data. Throw of structure. Geological data. RD factor. Subjectively determined the rock engineer. Examples that may lead the rock engineer to judge the panel as high risk would be a talcose hangingwall or dense jointing.	14 weighted parameters are taken into account, they are: - Mining sequence Lead and lag distances Distances from holing Sidings Regional stability Yield pillar geometry Panel spans between pillars Distance to fault or dyke Mining on side of fault or dyke Orientation of face to geological structure Orientation of major joint set to face General hangingwall conditions Extent of mining spans – an area overview Other i.e. anything else affecting risk Risk rating ranks as 1,2,3,4,5 being very good, good, average, poor and very poor respectively.
Risk management	Where surface rock support is required, then the geotechnical risk assessment will provide the following information: To what extent is the surface rock support needed? What excavations need to be treated? How will the surface rock support be integrated with the	The system was tested in hindsight but to my knowledge never used proactively to assess risk. A management system to follow initial assessment was therefore never developed. (The above needs to be validated.)	All relevant geological information collected by the geology and rock engineering departments is transferred to a plan and discussed in detail at planning meeting. High-risk panels are declared special precautionary areas and support recommendations are to reduce the risk. High-risk	The rock engineering department recommends mining procedures or support systems based on the risk ratings. The recommendations are recorded in a Special Areas File.	The risk assessment sheets are filled in monthly for producing panels. If a very poor rating is determined then the rock engineering department is consulted immediately. A database is kept of all sheets. They are discussed in detail at a monthly planning meeting where the appropriateness of standard support recommendations to

	<p>current ground control systems? How will the surface rock support be integrated into the current mining cycles and are any changes to procedures required?</p> <p>A regular inspection and monitoring programme is established and is carried out by competent persons and conducted for all areas identified in the risk reassessment. The frequency of inspections is relative to the risk and takes account of changes in ground and operating conditions.</p> <p>In the event of a change in conditions, such as a surface rock failure in an area scheduled for surface rock support; the priority of the area is reassessed. Where such events occur in areas where no surface rock support was considered necessary, the area is reassessed, taking into consideration the risk factors and the change in conditions.</p> <p>Any schedule and action plan is developed in consultation with relevant employees and safety and health representatives.</p> <p>If the assessment concludes that surface rock support is not required, then the Mine Manager justifies this in a documented geotechnical risk assessment. The risk assessment is included in a Ground Control Management Plan.</p>		<p>panels therefore become medium or low risk panels in future risk assessments because special area instructions have been enforced. Compliance is assumed.</p>		<p>suit particular risk ratings is discussed. The shift overseer is also empowered to make additional recommendations that are checked at the planning meeting.</p>
<p>Shortcomings and Future work</p>		<p>RockRisk is part of a dynamic process and as such, refinements will have to be undertaken and additional features added as the need becomes apparent and as understanding of the risk parameters improves. RockRisk requires calibration for individual mines. Variations in mining methods and mining history have meant that the relative risk varies from mine to mine. In addition, some reef horizons carry greater risks than others. Neither of these factors has been quantified.</p> <p>The relative risk of casualties for each mine and each reef can be assessed using the current database at CSIR Miningtek. This can be incorporated into the programme. (In my opinion a facility needs to be introduced to calibrate the weighing system and address specific problems on each mine by rock engineers on that mine.)</p>	<p>Wedges/keyblocks bounded by serpentine-filled joints are considered potentially hazardous especially if water is present. Present rock engineering department resources are insufficient to ensure that all potentially hazardous wedges/keyblocks are detected. The drive now is to train the workers and supervisors to identify the hazards and therefore be in a position to take effective precautions.</p>	<p>Introduced recently and still too early to say.</p>	<p>Main problem was the lack of sufficient rock engineering staff to cover the mining area adequately. This prompted the training and use of the shift overseer in this respect. Some staff are more motivated than others to perform the assessment and the quality is affected accordingly.</p>
<p>Results</p>	<p>Introduced very recently. Too early to judge</p>	<p>Two major rockbursts were considered to test the RockRisk systems reflection of reality. An acknowledgement expert with detailed knowledge of both events made an assessment of each situation using a personal classification scheme. The expert then used RockRisk to assess the risk and compare the results.</p> <p>In the first case RockRisk differed by 10% from that estimated by the expert. In the second case the actual risk was far higher than that assessed by the expert. The lack of good quality seismic data was that problem. Dr Webber states that the single most important factor in RockRisk is whether the area has a history of damaging rockbursts. In both of the above cases this was unknown.</p> <p>RockRisk was tested in the Carletonville and Klerksdorp areas and initial feedback was extremely positive (Webber 1996). It, however, does not seem to have caught on. Reasons need to be established. One may be insufficient follow-up. Dr Webber has left the country.</p>	<p>Highly significant safety and productivity improvements since 1996 have been largely attributable to an improved support system incorporating the use of pre-stressed elongates. This has been possible at these shallow depths because of the high horizontal stresses at Amandelbult. The introduction of the Risk Assessment Programme together with hazard awareness training has assisted in ensuring that the safety and productivity improvements are maintained. Millionaire Shield achieved twice in two years.</p>	<p>There has been a significant improvement in rock-related accidents. The system has certainly assisted but it is still too early to say to what extent.</p>	<p>As in the the case of the Amandelsbult Section significant improvements in safety and productivity were realised when an improved support system was introduced. The current system has rally on been in full use for six months and although it must have contributed of late to the continued improvements, the extent of the contribution is difficult to assess.</p>
<p>Comments</p>	<p>There are two more documents that may be useful and need to be acquired: DME Guideline. <i>Geotechnical Considerations in underground mines.</i></p>		<p>Each factor is assigned a maximum index of 10 and the <i>probability</i> of a fall of ground is determined by calculating the average index for all factors. Severity is determined from a weighted average. Geology,</p>		<p>Unstable wedge/keyblock and pillar punching are the main rock related hazards. A seismic system is currently being installed to augment data available for risk assessment.</p>

	DME Draft Guideline. <i>Risk Management</i>		beam thickness, span, pillar size and water are weighted. A relationship between probability, severity is then used to categorise stope panels as high medium or low risk.		
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ROCK-RELATED RISK MANAGEMENT SYSTEMS ON VARIOUS GOLD AND PLATINUM MINES - APPENDIX A (Page 3)

	Hartebeestfontein Gold Mine	TauTona Mine	AngloGold West Wits Operations	Rustenburg Platinum Mines – Rustenburg Section	Elandsrand Gold Mine – Anglogold West Wits Ops
Name of system	Rock Mechanics Risk Assessment for Pillar Mining	The Hazard Rating System	Seismic vulnerability assessment	Stope Plans Risk Assessment	Seismic Hazard Assessment
Contact and/or ref.	Koos Bosman	Shaun Murphy, Lourens Scheepers	Matthew Handley	Johan Lombard, Lando Sloane	Jannie de Lange, Essrich (1997)
Aim and background	To assess the risk involved when mining remnant pillars	Identify rock-related hazards and associated risks and then to provide guidelines for actions required to reduce the risk. The system is essentially a customised version of "The Panel Rating System" as applied on many of the Vaal River Operations and African Rainbow Minerals mines. See page 1 of this table.	A practical, fast check on seismic vulnerability. The assessment is done quarterly Twenty questions requiring yes or no answers are covered for each section of a particular mine. This can be considered to be an external audit and is normally done by a senior rock engineer not directly involved with the day-to-day running of the mine.	To identify problem areas and risks in stopes underground and to ensure action is taken to minimise the risk.	To evaluate the seismicity-related risk in production areas on the mine. Based upon the "Mining Alert Level" procedure developed on Western Deep Levels South mine. Naude (1995)
Applicable to:-	Intermediate to deep tabular mining of remnant pillars	Deep tabular mines	Deep, tabular rockburst-prone mines	Relatively shallow, scattered mining environment particularly on the platinum mines.	Deep tabular rockburst-prone mines
Risk assessment parameters	The following parameters are assessed for accessways: - Proximity to geological features Abutments Prevailing stress levels Middling Escapeways Waiting places The following parameters are assessed for stopes: - Rock mass competence and the presence of geological features Seismic history Prevailing stress levels Pillar geometry Other e.g. water from fissures, mining through excavations etc.	Local factors Inter-panel mining sequences (lagging) Stress concentrations due to leads/lags Gully position relative to lead/lag and North siding Approach and distance to geological features (distance and angle of panel face to features) Presence of adequate second accesses Regional factors (weighted) Geological complexity (number of structures) Mining span Rockburst risk (position of panel relative to seismically active structures and/or pillars) Face stress regime (Depth)	The questions revolve around: - Recent modelling Mining according to plan Layout Negotiation of geological features Face shape Regional support – pillars Regional support – backfill Panel support Gully/roadway position and layout Gully/roadway support Development layout Development support Travelling way support Travelling way layout Escape way layout (2 nd entrance) Escapeway support (2 nd entrance) Refuse chamber position Rock mechanics dept coverage Seismic coverage Record and acknowledgement systems The support and conditions in eight randomly chosen panels are checked underground each month. The results are used in the assessment. Six rock engineers from the AngloGold West Wits Operations perform the assessment.	Risk factors taken into account are: - Panel length Pillar width Leads/lags compared to adjacent panels Rock mass ratings Support types (actual vs. planned) Rock engineering dept visits underground Adjustments for faults, dykes, etc. A panel starts with a score of 100. Marks are deducted for non-compliance and the final score is expressed as a percentage. The lower the score the higher the risk.	Seismic Hazard Analysis (SHA) parameters: - Average seismic index (reflects state of stress) Cumulative apparent Volume (reflects the co-seismic inelastic deformation taking place) Energy Release Rate (ERR) Face Configuration Rating (reflects effect of lead/lags, abutments, mine layout and remnants) Geology (Approach to features, presence of slopes between terraces, flat faulting and jointing). Production (reflects the total area mined with respect to a certain cohesive set of producing panels)
Risk management	A special form is available where the risks are noted for each of the abovementioned parameters. A probability of occurrence and possible consequence thereof are judged and a high, medium or low "risk rank" is assigned. Low risk is accepted as tolerable. Controls are recommended where necessary for each of the potential hazards associated with medium and high risk parameters. The potential hazards are listed below: - Accessways Proximity to geological features Fallout on geological features in tunnels Rockburst induced collapse in tunnels Rockburst induced violent ejection in tunnels Abutments Deformation of tunnels due to adverse fracturing resulting from high stress Rockburst induced collapse in tunnels Prevailing stress levels Collapse of blocky hangingwall due to adverse mining induced fractures Collapse of slabbed sidewalls Middling Collapse of small middlings Deformation of tunnels due to water seepage from stope above Escapeways Availability of second escapeway back to access route or to another level or crosscut Exposure to high stress or to stress change	The system came into operation in December 1999 and at this stage is used in combination with a detailed monthly assessment of what is known as the 1:1000 Seismic Plan. The need for the latter will eventually fall away. The results of the monthly risk assessment for each panel are given to the Mine Overseer of the section prior to the monthly planning meeting. Recommendations and corrective actions are agreed upon and minuted. The actions may include a visit by Rock Engineering if conditions warrant it. At this stage no formal follow-up system is in place. A summary sheet is used where problem areas are highlighted. One of the main advantages is that this formalised approach allows one to detect trends. Subjectivity is to some extent minimised as well.	Is essentially an external audit. An entire mine typically consisting of six sections and 120 panels can be covered in a few hours. The 1:1000 "Seismic Plans" are used where details of ERR's, face shape index, geology etc are depicted. Zero tolerance applies and management is expected to act immediately on the findings and recommendations resulting on the audit.	0-25 – Very high risk panel 26-50 – High risk panel 51-75 – Medium risk panel 76-100 – Low risk panel A panel is stopped immediately if the percentage score is less than 25. Mining can recommence after precautions (approved by the Rock Engineering Dept) have been taken. The information is distributed to the Business area Manager and to the appropriate operations managers, mine overseers and shift supervisors.	Rating is carried out for each set of panels being mined together. Scores of 1to5 for each parameter are given based upon the gauged risk. Geology score ranges between 1 and 8. Scores are added and then the SHA rating determined as follows: Score SHA 1-10 1=low 11-14 2=below average 15-20 3=average 21-26 4=above average >26 5=high The SHA is then recorded on plan next to the group of panels to which it applies. ERR and Face Configuration Rating are also shown with the SHA. It is considered useful to do this even though they form part of the SHA Results are communicated to senior management and other appropriate persons at a quarterly meeting where strategies to minimise risk are decided. These are communicated to mine overseer level at monthly meetings. Compliance is checked at the following quarterly meeting.

	<p>Fallout on geological features in escapeway Rockburst induced collapse in escapeway Waiting places Exposure to stress and/or rockbursts</p> <p>Stopes Rock mass competence and presence of geological features Collapse of small wedges in stopes Collapse of brows and undercuttings Fallout on geological features in stopes Seismic history Probability of mag >3.5 within period of mining Structure – known to be hazardous or not History of damaging events or not Prevailing stress levels Anticipated fracture density High closure rates Pillar geometry Mining towards hazardous geological feature More than two final remnants</p>				
Shortcomings and Future work	<p>Somewhat more subjective than many other assessments.</p>	<p>It has been difficult to coordinate the assessments with underground measuring dates and planning meetings.</p> <p>The system took some two months to initiate. Some resistance to change has been experienced.</p> <p>A lack of sufficient resources in the Rock Engineering Department is a problem. There is a danger that the system may be relied upon too much without sufficient underground inspections by these individuals. It is the intention to involve the Safety Department in the near future to at least partially address this problem.</p>	<p>The assessments are fast and practical but may be somewhat superficial as a result of the limited resources.</p>	<p>The assessment is seen as a useful management tool but, given more resources, it could be far more efficiently done with more measurements and visits underground.</p>	<p>SHA deals with the rockburst hazard but not does not adequately address the fall of ground problem. For this a far more detailed assessment of the ground conditions (joint patterns combined with mining-induced fractures) needs to be done. Here resources are a problem.</p> <p>Follow-up on compliance to recommendations is not adequate – again lack of resources.</p> <p>Often high risk ratings on the geology side are given to counteract uncertainty in interpretation of the attitude and hazard presented by features such as dykes, faults and joints.</p> <p>A quarterly external audit in the form of a “Seismic Vulnerability Analysis” is no longer in operation. There is a strong need for this. This included the audit of a number of randomly selected panels each month. The “Seismic Vulnerability Analysis” is also described briefly in this table.</p> <p>There have been major changes in management recently on Elandsrand Gold Mine. The importance of the existence of this rating scheme and other external audits is not yet fully appreciated.</p>
Results	<p>Useful in determining whether a particular remnant is mineable or not.</p>	<p>Too early to judge. In the opinion of the Rock Engineering Department staff the potential for significant safety improvements is there, however. The more formal approach ensures that problem areas are more effectively highlighted and addressed.</p>	<p>There have been significant improvements in rock-related accident statistics recently. It is believed that the system has had a significant part to play.</p>	<p>The assessment is seen as a very useful tool enabling managers and supervisors to focus in on high risk areas and take appropriate actions to minimise the risk.</p> <p>The system has been in operation for approximately six months and results appear to be highly satisfactory at present. This improvement may, at least in part, be attributable to the recent introduction of active elongate support.</p>	<p>As reported by Essrich (1997). Since the introduction of the system in September of 1994, it has contributed by reducing the number of large seismic events on the mine. While the assessment reliably forecasted the majority of working areas with increased seismic levels, it was less successful in predicting rockburst accident areas.</p>
Comments					<p>SHA should be seen as a hazard rating rather than a risk assessment per se. SHA does not take into account the quality and type of support and the presence of backfill for example. Therefore high hazard ratings may not necessarily correlate with severe losses from rockbursts.</p>

ROCK-RELATED RISK MANAGEMENT SYSTEMS ON VARIOUS GOLD AND PLATINUM MINES - APPENDIX A (Page 4)

	Impala Platinum Limited				
Name of system	Rock Engineering risk assessment				
Contact and/or ref.	Les Gardner, Noel Fernandez				
Aim and background	To identify problem areas and risks in stopes underground and to ensure action is taken to minimise the risk				
Applicable to:-	Relatively shallow, scattered mining environment particularly on the platinum mines				
Risk assessment parameters	<p>Risk factors taken into account are: - Panel length Pillar width Leads/lags compared to adjacent panels Rock Mass Ratings (RMR) Support types (actual vs. planned) Compliance to support standards Rock engineering dept visits underground Adjustments for faults, dykes, etc.</p> <p>Impala has, at present, over 30 trained observers who measure support standard compliance and determine RMR's underground.</p>				
Risk management	<p>The form, filled in by the observer at the time of each visit, is checked by the Rock Engineer or Strata Control Officer and goes via the Mine Overseer to the Manager of the section.</p> <p>The detailed support compliance data, RMR's and layout risk factors are combined to determine an overall risk rating for each panel. This is summarised monthly for each section of the mine.</p> <p>The observers are able, at present, to visit 75% of the panels monthly on Impala mine. They are trained to a level where they can stop the operation if deemed necessary and/or make recommendations to minimise any observed risk.</p>				
Shortcomings and Future work	<p>The ideal is to visit each panel at least once per month. 75% of the panels are presently visited each month.</p> <p>Maintaining consistency in the assessment of conditions and risks between more than 30 observers has proved somewhat problematical.</p> <p>The spread of observer resources across the mine needs attention. The work load differs significantly between shafts and sections.</p>				
Results	<p>With the large number of underground measurements and an efficient database it is now possible to detect trends and take more timeous and effective action.</p> <p>The increase in cost of support, due to the fact that it is now complying readily to standard, is far outweighed by the safety advantages that the situation offers.</p>				
Comments					

8 Seismic Warning as Used in Areas Where Seismicity Is Driven By Geological Structures.

S.N.GLAZER

8.1 Abstract

During recent years some uncertainty has arisen as to what constitutes a seismic warning. Is it some kind of prediction, which can then be graded in terms of alert, alarm and scam? In the first part of this chapter try understand these phenomena and hopefully solve some of the misunderstandings. The second part describes in some detail how the seismic warning system was developed (using back-analysis techniques) and then applied in practice. All practical examples are warnings that were issued before the event took place. A statistical analysis of results is given for about 200 such warnings issued from the end of 1994 to the middle of 1999.

This chapter also part deals with some the perceptions resulting from practical experience with the warning system. Initially, the mining personnel, the seismologists, as well as the Rock Engineering personnel, were very enthusiastic about the method. The general feeling was that at last there was a proactive method that can help with improving the safety of the underground workers. With the introduction of the Health and Safety Act, this perception changed. The problems that arose were the legal responsibility and transparency issues.

The fourth part of this report gives the present status and limitations of the seismic warning method. It should be treated as a method that has a developed methodology that makes it a practical tool for managing mine seismicity. It should be used to raise the level of awareness of seismic hazard. It is limited to indicating an area that might experience a seismic event and it is not time-specific and also it cannot be used to indicate the size of the expected event.

Finally, in the conclusions seismic warning is recommended for use in practice. Its applications and limitations should be made known to all interested parties. This should involve not only mining personnel, but also mine inspectors and unions. The effectiveness of the method can be improved by application of such methods as statistics and tomography. This would then require some additional focused research.

8.1.1 Summary

As it stands at present, the warning system has a developed methodology that makes it a reliable and practical tool for managing seismicity.

The warning system cannot be used to prevent seismic events taking place. It must be used to raise the level of awareness of seismic hazard and this can only result in safety improvement.

Seismic warning is limited to indicating an area that might experience a seismic event and it is not time specific and it cannot indicate the size of the future event.

Seismic warning is in fact a method that allows monitoring of seismic hazard for a specific area for a given time.

There are several limitations to the method, most important being that it cannot be used with regional events. In other words the method is limited only to those events that result from actual mining.

It is possible to improve the method. Improvement should include better selection of space and time of the input data. This could be done by application of statistical methods. The other more important improvement is that it should include a method for more detailed location of the future event. This should come out of tomography.

Possibly more information can be gained by back-analysis of big or medium sized events. For this a consistent database is required.

At this stage it is unlikely achieving a great improvement using quantitative interpretation methods. There will be no global solutions but there might be some improvements that will apply only locally, since the instability is very site dependent

Deterministic prediction of mine induced seismic events is not possible for all practical reasons. The main argument is the nature of the input data itself. This input is a result of a measurement so in effect any possible prediction can only be given with some error bonds. For this reason the prediction would be given in probabilistic terms.

8.2 Recommendations

There is still room for improvement in the hardware used in mine seismology (communication speed). The location methods can still be improved, for example by the introduction of relative location techniques, as well as the source calculation methods (near-field). The emphasis should be in the field of practical applications of mine seismology, mainly in the field of combating the rockburst hazard. For this reason, good and reliable interpretation methods of recorded and processed data is at present the most important part of mine seismology.

The stress index, probably when used in its cumulative form, should be applicable to quantitative methods. The derivative of cumulated apparent volume was also used for experimental (test) purposes.

These are relatively easy changes. These can be done by rewriting the interpretation software. The main problem remains how to define the optimal volume of rock mass, to which the

interpretation methods apply. From observations we already know, that the used polygons are of optimal size for predicting events of magnitude between 2.1 and 2.4. This observation confirms that the bigger the event, the bigger should be its preparation zone.

Establishing a relationship between the event size and optimum quantity of data will still not solve the problem. There are even more fundamental questions to be answered. Where and when to place the polygon and start the observations? At present it is a trial and error process, which depends on the individual making this type of analysis. Even so, there are some indicators that are used quite frequently. These are space concentrations of events, and changes in activity rate in both directions, i.e. increasing or decreasing. This indicates that methods, as described for example in Kijko (1985), Lasocki (1993), Kijko and Funk (1994) and Kijko (1996) could be used to solve this problem.

Some probabilistic procedures in the assessment of seismic hazard were tested at Vaal Reefs during 1994/1995. A summary of these results is given by Nel (1995a), where an analysis of the probabilities of occurrence of seismic events in the polygons used for the instability concept, is done for the time period between January and June 1995. It was concluded that the probabilities are strongly dependent of the threshold value and the number of events in a time window. It was observed that after a time period with smaller events only, or in which there is a decrease in the number of events it becomes very difficult to predict. Results were mixed and changed from polygon to polygon, with the overall success rate being in general very low. At the same time the first attempt was made to use the logistic distribution model (Nel, 1995b). The results obtained from the logistic distribution model were good. The advantage of this method is that it can still be used even if there are not that many events in the polygon. The disadvantage that was found was that the model requires a regular occurrence of events above the prediction limit in order to gain experience. In those cases where there were no such events for some time it would "forget" all gained experience.

There is at least one more field where the probabilistic approach should be implemented to improve the instability concept, and that is the polygon determination. One of the limitations of the instability concept is related to separating areas of interest (Glazer, 1997b). At present they are based on three main factors: seismic space clusters, geology and the position of underground working places. The seismic cluster at present is not defined in an objective way and is limited to a space cluster as seen on the monitor. This is a very crude way of doing it, and there is a need for improvement. The cluster definition should be improved by using appropriate statistical procedures, which should also include the time clustering of events. Such techniques already exist, and their description can be found in Kijko and Funk (1994), Kijko et al (1993), Gibowicz and Kijko (1994) and Kijko and Funk (1996). Instead of using a space cluster, a space-time cluster should be analysed in the instability method. At present the time window is arbitrary, and very dependant on the amount of recorded data. Additionally, the polygon includes only the geology that is close to the underground working places. As a result, seismic events on this or any other intersecting geological feature which contributes towards the instability area can be located outside the polygon, and therefore not taken into account.

Clustering should also help in defining not only the data set that should be used in the instability analysis, but should also define events that are associated with one another. The bimodal distribution of events has already been described in several publications and works, for example, Kijko et al. (1987), Stankiewicz (1989), Glowacka et al. (1992) and Kijko (1996). This bimodal distribution is very evident when analysing the stress index values for different mining areas, namely those where the mining has reached an advanced stage, and those where only development work has taken place. In development areas for the lower moment ranges, there is more energy release than in well mined-out areas. The same phenomena can be observed for events located close to working areas and which are caused by blasting. Those events do not contribute towards instability, at least not when the working face is still at some distance from a geological feature.

It must be remembered that in the case of the Vaal Reefs or the Klerksdorp areas, the damaging events take place mainly on geological features (v.d.Heever, 1982, Potgieter and Roering, 1984) and are of a different type to those at the West Rand, where the damaging events occur often in stoping (Ortlepp, 1984) or are connected with pillar failure (Hagan, 1990).

As with the events close to the working area, there must also be some events at the lower ranges of moment, associated with dykes or faults. A method of differentiating between these two should improve the results of the instability concept. This bimodal distribution could also be one of the reasons why some techniques applied earlier on (with exception of logistic model) were not as successful as was hoped (Nel, 1995a).

On the other hand, events with large moment also do not seem to follow a pattern, as can be observed for medium size events (magnitude between 1,0 – 3.0) based on the moment – apparent stress relationship.

Heunis (1977) made an observation that faults of large displacements (40m+) present special rockburst and rockfall difficulties, because the risk of very large seismic event is considerably amplified by their presence. Little is known about the methods of preventing the occurrence of these very large events. According to Gay et al. (1984) there is a very good correlation between the number of events and the total area mined. From the plots of cumulated energy released by seismic events against centares mined, a very interesting conclusion was made. It is that the occurrence of big events seems to be independent of the mining activity. Johnston (1992), after a comprehensive study of mine induced seismic events from various parts of the world, divided them into two categories. The first category includes events of low to medium size magnitudes, whose event rate is, in general, a function of mining activity. They locate generally within 100m of the mining faces on some geological discontinuity. Events of the second category are events of high magnitude and are difficult to correlate with mining and occur on prestressed faults. These events are then, in some undefined way, triggered by mining activity. Direct proof that such events take place is given in part 2.14.

From the above one can conclude that the mine induced seismic events at Vaal Reefs follows a triple-mode distribution, that than the bimodal one, and that the events can be divided as follows (Ebrahim-Trollope and Glazer, 1997)

- After-blast events, which are due to fracturing in front of the faces.
- Small events up to magnitude 0.5.
- Events connected with local small to medium size faults and dykes. Those events are up to magnitude 3.0, and are connected with actual mining.
- Events above magnitude 3.5 that are associated with regional structures and past history of mining.

Experience indicates that some of the events of magnitude between 2.0 and 3.0 can be predicted through instability analysis 0-10 days before they take place (Glazer, 1997c). Small events form noise as far as the instability analysis is concerned, while the big regional events above magnitude 3.5 seem to be out of the instability method prediction range.

There is one more technique that can be used in order to improve the reliability of the seismic warning method.

Seismic tomographic imaging can be used to map stress changes, faults and density variations in the rock mass. Repetition of tomographic imaging for a given area would provide means for monitoring changing stress conditions in mines (Gibowicz and Kijko 1994, Kijko, 1996). A combination of monitoring results and distress blasting techniques can be used to recognize, locate and control highly stressed rock masses. (McGaughey et al., 1987, Young et al, 1989).

In rockburst investigations comparisons between velocity images and seismicity have shown that the induced tremors are associated with regions of high velocity while low velocity are generally aseismic (Maxwell and Young, 1993).

Temporal investigations of velocity structure in mines have shown that the induced seismicity is related to temporal velocity reductions, when decreased normal stress may unclamp critically stressed fractures, resulting in induced tremors (Maxwell & Young 1997). The low velocity regions map relatively weak zones, incapable of storing sufficient strain energy to be the site of a strong event. In other words the region is deforming through aseismic stable sliding. High velocities correspond to strong stiff regions capable of storing significant strain energy (Maxwell & Young 1994).

Results of a velocity imaging study at the Western Holdings Mine, Welkom indicated that a large tremor located in a region of a very high velocity (Maxwell and Young, 1994).

In Polish mines tomography is used to monitor changing stress conditions and the efficiency of destressing operations (Dubinski et al., 1998, Dubinski, 1996, Baranowski et al., 1996 Goszcz, 1996, Gerlach, 1996).

From the above listed theoretical and practical applications it appears that seismic tomography could become a useful tool for estimating rockburst hazard, by mapping stress distribution and its temporal changes in relatively small rock mass volumes already covered by dense seismic networks.

8.2.1 Conclusions

The seismic warning method should be regarded as already proven in practice for monitoring of seismic risk in time.

It should be used to raise the level of awareness of seismic hazard. This should result in safety improvement.

The users should be aware of the method limitations.

It should be understood that seismic warning has nothing to do with predictions as such. Seismic warning is limited only to indicating an area that might experience a seismic event and is not time specific and it cannot indicate the size of the future event.

Seismic warning includes not only the detection devices, but also implies the judgements, decisions and actions that follow receipt of the sensor information. Warning encompasses communication, analysis of information, decisions and appropriate actions. Use of warning systems indicates that the most expensive portion and weak link in them is not the sensor but the communication and evaluation system.

There is still significant room for improvement of the effectiveness of seismic warning. As far as selection of area (volume), time span and input data is concerned, the application of statistical method should result in improvements.

9 Summary of recommendations

9.1 Recommendations

Each chapter in this report dealt with particular aspects of seismic risk assessment. Some of the recommendations and conclusions made in the chapters are repeated here.

From Chapter 1, Seismic Risk Assessment – An Overview

- Seismic prediction of event magnitude, position of potential source and expected time of occurrence is not practised anywhere.
- Seismic risk assessment includes knowledge of the seismicity; rock mass conditions and support standards pertaining to the mining excavation, the geological environment and stress distribution around the excavation, and the degree of exposure of the worker to the seismic hazard.
- The researchers found that a coherent data set of seismic early warning case studies did not exist. Similarly, it was not possible to correlate seismic data with any safety or damage information. A relational database including the above is a prerequisite for risk assessment.
- Seismic monitoring up to a certain minimum standard is a prerequisite. These minimum standards include acquisition of data, processing and interpretation up to a minimum level.
- A minimum level of training for on-site evaluation of seismic risk is required.

From Chapter 2, Analysis of Currently Used Probabilistic Techniques and Recommended Approach

- The non-parametric, seismic hazard estimation procedure is strongly recommended for use in seismic hazard assessment in mines, that is the determination of the maximum even magnitude and the mean return time of an event larger than a certain magnitude

From Chapter 3, Risk Assessment of Possible Rockburst Damage, Based on Peak Ground Velocity

- The strong ground motion model described is only approximate and could be improved in a number of ways that would require further research. Some important topics that should be addressed are:

- Reducing the spurious effect of location errors. A conservative approach is to move all locations to the vicinity of likely sources, such as nearby faces, pillars, geological features, or abutments.
- Reducing the spurious effect of source extent.
- Considering whether v_{max} as radiated from the seismic source is, in fact, the best estimator of damage. For example, a shear slip a few metres ahead of the face can cause face crushing and large ground motions. The role of co-seismic horizontal stresses is also poorly understood.

From Chapter 4, Implications of the Time-Dependent Behaviour Of Deep Level Stopes in Hard Rock

- Continuous closure measurements appear to be a useful diagnostic measure of the rock mass response in different geotechnical areas. It seems that the instantaneous closure response after blasting is a reflection of the face stress before the blast. Further work is, however, necessary to quantify the possible correlation between the risk of face bursting and the closure response of the stopes.

From Chapter 7, Rock Related Risk Assessment Techniques on the Gold and Platinum Mines

The need for future work to improve the systems in place on the various mines was recognised in all cases. This includes:

- The training of observers, safety and/or production personnel to assist with risk assessment and management;
- The introduction and implementation of follow-up procedures;
- Initiating regular external audits;
- Improving the techniques by making them less qualitative and more quantitative;
- Inputting all information on to a database to allow trend analyses;
- Introducing direct, up to date, underground data such as rock mass ratings, support compliance and efficiency, into the equation;
- Using instrumentation to detect unfavourable structures in the hangingwall and ahead of the mine face;
- Finding ways to reduce subjectivity when assessing risk.

Results and advantages as seen by users out in the field at the moment are:

- 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 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.
- 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.
- On one mine the system is believed to have contributed significantly by reducing the number of large seismic events. Assessment also reliably forecasted most working areas with increased seismic levels. The system was, however, less successful in predicting rockburst accident levels.

From Chapter 8, Seismic Warning as Used in Areas Where Seismicity Is Driven By Geological Structures.

- The contribution of seismic warning to seismic risk assessment has been proven in practice.
- It should be used to raise the level of awareness of seismic hazard. This should result in safety improvement.
- The users should be aware of the limitations of the current techniques used for seismic early warning.
- It should be understood that seismic warning has nothing to do with predictions as such. Seismic warning is limited only to indicating an area that might experience a seismic event and is not time specific and it cannot indicate the size of a future event.
- Seismic warning is more than seismic monitoring and includes not only the detection devices, but also the judgements, decisions and actions that follow receipt of the sensor information. Warning encompasses communication, analysis of information, decisions and appropriate actions.
- There is still significant room for improvement of the effectiveness of the seismic warning method. Other methods may complement that of seismic warning.
- The application of statistical methods should improve the selection of area (volume), time span and input data.

10 Suggested procedure for evaluating seismic risk

10.1 Introduction

The researchers had the opportunity to evaluate the various risk assessment methodologies and tools during their consultation with the industry.

This chapter combines all this input and suggests a best approach for seismic risk assessment in the South African gold and platinum mines. A fundamental problem with many risk assessment procedures is the degree of subjectivity involved. The same accusation may be made to this proposed procedure. However, the researchers are confident that the holistic approach adopted in compiling this rating system, will provide a better assessment of seismic risk. It should be noted that this is presented as a conceptual approach and it would trigger significant debate on some detail. It is not implied that this approach is ready for implementation by the industry.

Table 10.1.1 summarises the proposed procedure for evaluation seismic risk. 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

Each category has effectively the same relative importance or weighting.

An overall seismic risk assessment is achieved by combining individual category ratings. The process of achieving a single rating is discussed later.

Each category is subdivided in various parameters contributing to that category. The parameters are rated individually and averaged to provide a category rating. A risk rating ranges from 1 to 5, where 1 implies a very low risk and 5 an always unacceptable high risk.

The overall risk rating is obtained by multiplying the category ratings. The reason for multiplication rather than addition (as in averaging) is that any single category with a low rating should have a larger 'alleviating' effect. Multiplication of the respective risk ratings allows for the outlier to have a larger effect. The ratings of 5 with a single rating of 1 will provide a lower overall rating through multiplication as opposed to averaging. Examples are as follows:

Individual ratings	Average rating	Multiplication-based rating
5 5 5 1	4	3
4 4 4 1	3	3
3 3 3 1	3	2
4 3 2 1	3	2

}

Rating	Risk Assessment Category	Parameter	Parameter rating
P ₁	Level of Ground Motion	M _{max}	p ₁
		Distance from source	
		Mean Return Time (Frequency)	p ₂
		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	p ₅
		(Site Effect Amplification)	(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

Rating ranges from 1 to 5, 1 being very low risk to 5 being a high risk

Rating per category $P_1 = (p_1 + p_2 + \dots + p_n) / n$

Overall Risk Rating $P_{combined} = P_1 * P_2 * P_3 * P_4$

$P_{risk} = 1$ (if 1# $P_{combined} < 4$)

$= 2$ (if 4# $P_{combined} < 36$)

$= 3$ (if 36# $P_{combined} < 144$)

$= 4$ (if 144# $P_{combined} < 400$)

$= 5$ (if 400# $P_{combined} < 625$)

Table 10.1.1 A proposed risk assessment methodology

To describe the suggested rating system each category will be described individually.

10.2 P₁ - Level of ground motion

The concept here is to assess the risk level relating to the level of ground motion that the excavation (all excavations including stopes, haulages and shafts) may be subjected to.

This category is divided into three parameters:

10.2.1 P₁p₁ – M_{max} and distance from the source

The concept of determining the maximum possible event is described in Chapter 2. This may not be necessarily the only way, nor may sufficient data exist to achieve a proper quantification of M_{max}. The risk assessor should recognise different modes of failure within his volume of interest such as pillar foundation failure, slip on a fault ahead of the mine face, or even face bursting.

It is known that the local magnitude is not the only parameter determining the level of ground motion and an associated parameter with a significant influence, is the static stress drop. However, this is a very difficult parameter to bring into a first order risk rating.

The level of ground motion is as much a function of the distance as the magnitude (or more correct, the stress drop). Again we assume a far-field situation between the excavation and the source of the event.

For combining both event magnitude and distance in a single risk rating, a risk matrix is proposed.

M _{max}	>4	5	5	5	5	4	3
	3-4	5	5	5	4	3	2
	2-3	5	5	4	3	2	1
	1-2	5	4	3	2	1	1
	0-1	4	3	2	1	1	1
		0-20	20-50	50-100	100-200	200-500	>500

Distance to the source (in metres)

Table 10.2.1 A possible risk matrix for combining M_{max} and distance to the source

10.2.2 $P_1P_2 - M_{\text{ean}}$ return time

The mean return time of the event has a significant risk implication. The above matrix (Table 10.2.1) provides the rating related to the magnitude (or possible amplitude) of the ground motion, but not how often such ground motion can be experienced. The following provides a means of linking mean return time (as described in Chapter 2) with a risk rating.

Risk rating for P_1P_2	Description of return period
1	Once within life of mine
2	Longer than 5 years
3	Longer than a year
4	Longer than a month
5	Less than 1 month

Table 10.2.2 A risk rating based on the mean return period

10.2.3 $P_1P_3 - \text{Seismic/ Time distribution}$

The assumption of an equal probability for an incident resulting in large ground motion, during a 24 hour period is unnecessarily conservative. Every mine with a history of seismic monitoring has a known hourly distribution function for the occurrence of seismicity at that mine.

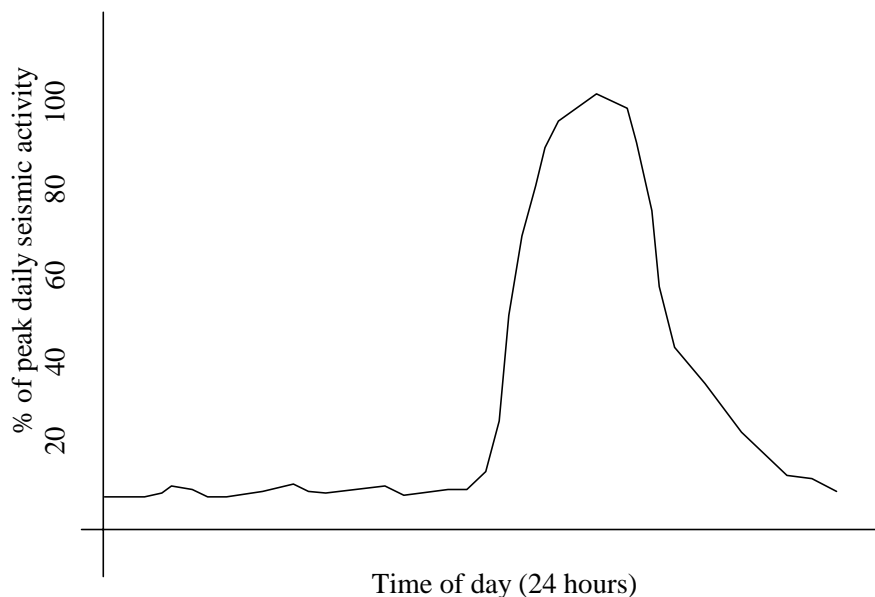


Figure 10.2.1 A schematic presentation of a possible seismic distribution function

It is often claimed that larger events have a more random distribution through the day and also have a unique weekly distribution. Care should be taken to ensure that the event category recognised as having the highest ground motion risk rating should be considered in terms of its magnitude diurnal distribution.

Figure 10.2.1 shows a schematic version of a typical seismic distribution on a mine practising daily blasting in a narrow time slot. It is obvious that other mining methods may have a different distribution that may result in a higher P_1P_3 rating at a particular time.

The method of rating the risk associated with the time distribution of seismicity is based on a magnitude range and source position identified as resulting in the largest ground motion

Risk rating for P_2P_3	Description of relative seismic event rate
1	0 to 20% of max. seismic event rate
2	20 to 40% of max. seismic event rate
3	40 to 60% of max. seismic event rate
4	60 to 80% of max. seismic event rate
5	80 to 100% of max. seismic event rate

Table 10.2.3 Risk rating based on the time distribution of seismic event rate in the area/volume of interest. Seismic event are from the magnitude range and source position identified as resulting in the largest ground motion.

This is an unsatisfactory method of determining a risk rating in the sense that it is a relative measure. An absolute measure, as in an event rate of x , is site dependent and should be developed per individual site.

10.3 P_2 - Vulnerability of the excavation to ground motion

Where the first category, P_1 , relates to a possible level of ground motion and the time distribution of such ground motion, P_2 refers to vulnerability of the excavation to this occurrence of a high level of ground motion.

The P_2 category includes the following parameters

- p_1 Energy Release Rate (ERR)
- p_2 Local geological structure
- p_3 Support
- p_4 Ground condition
- p_5 Escape ways
- p_6 Local site amplification

10.3.1 P_2P_1 – Energy Release Rate (ERR)

The Energy Release Rate is an ambiguous parameter in defining the vulnerability of the excavation to large ground motion. It implies that active mining steps are taken. It also can be argued that it should rather relate to the source and the probability of having a large ground motion rather than the vulnerability of the excavation.

ERR scales with the stress levels ahead of the mining face and reflects local factors such as stress concentrations due to lead/lags and mining span.

The numerically modelled ERR may show large dependencies on some arbitrary selected input parameters. It is therefore suggested that the risk rating, P_2P_1 , be relative to the maximum levels within the mine (or mine section).

Risk rating for P_2P_1	Description of the level of ERR
1	0 to 20% of max. ERR in area
2	20 to 40% of max. ERR in area
3	40 to 60% of max. ERR in area
4	60 to 80% of max. ERR in area
5	80 to 100% of max. ERR in area

Table 10.3.1 A risk rating based on ERR

The comment in 10.3.1 on ratings based on relative values, is equally valid in this section. An alternative approach may be to determine a range of ERR values, for example 0 to 10 MJ (rating 1), 10-20 MJ (rating 2), 20-30 MJ (rating 3), 30-40 MJ (rating 4), and > 40 MJ (rating 5).

10.3.2 P_2P_2 – Geology

Again an argument around cause and effect exists in terms of the influence of geology on the risk rating. The emphasis in this case is on the influence of geology as for the vulnerability of the excavation. Input parameters to consider will be the presence or close vicinity of a dyke/fault contact, the approaching of significant structure (distance and angle to be considered), and the complexity (number and type of structure).

A quantifiable rating is suggested to include the effect of geology on the vulnerability of the excavation.

Risk rating for P_2P_2	Description of local geology
1	No risk associated with local geological structures
2	Below average risk associated with local geological structures
3	Average risk associated with local geological structures
4	Above risk associated with local geological structures
5	Unacceptable risk associated with local geological structures

Table 10.3.2 Risk rating associated with local geological conditions

10.3.3 P_2P_3 – Support

The appropriateness of the support type and adherence to industry and mine standards are reflected in this risk rating and in most cases will be the result of a support audit.

- Haulages, access ways, gullies and stope conditions
- Adherence to appropriate support types, installation standards for the above conditions

- Quality and placement of backfill (if installed)
- Support audit parameters should include aspects around adherence to temporary support standards

A conservative, worst case approach should be adopted in determining the risk associated with support.

Risk rating for P ₂ P ₃	Description of quality of support
1	Significantly better than the required support type and installation standard
2	Adherence in all aspects to the appropriate support standard and installation standard
3	Adherence in most aspects to the appropriate support standard and installation standard (the non-compliance is not significantly increasing the risk)
4	Adherence in most aspects to the appropriate support standard and installation standard. (the non-compliance is significantly increasing the risk)
5	Unacceptable support type or poor non-adherence to support installation standards

Table 10.3.3 Risk rating associated with quality of support

10.3.4 P₂P₄ – Ground condition

This parameter describes the vulnerability of the excavation due to the observed ground conditions. Some inputs to consider are:

- Adequacy of barring
- Blast damage
- Sidewall fracturing
- Bedding planes
- Rock mass competence

A subjective rating for the influence of local ground condition is suggested:

Risk rating for P ₂ P ₄	Description of local ground condition
1	There is no risk associated with the local ground condition
2	The risk associated with the local ground condition is low
3	The risk associated with the local ground condition is acceptable
4	The risk associated with the local ground condition is above average
5	The risk associated with the local ground condition is high

Table 10.3.4 Risk rating based on local ground conditions

10.3.5 P₂p₅– Escape ways

The existence, quality and compliance of escape ways to the industry accepted standards are rated by this parameter. The worst-case escape scenario should be rated.

Risk rating for P ₂ p ₅	Description of standard of escape ways
1	Excellent alternative escape ways exists
2	The escape ways are better than the mine and industry standard
3	The escape ways meet the mine's code of practice
4	The escape ways are not providing save alternative escape routes
5	The escape ways are not usable

Table 10.3.5 Risk rating associated with the standard of the escape ways

10.3.6 P₂p₅– Site effect amplification

As described in Chapter 1, the local site effect amplification can provide a significant amplification of the ground motion. A current SIMRAC project (GAP709) intends to quantify the effect of site amplification.

At this stage we can recognise the importance of such amplification but cannot provide a method that can distinguish the different areas or changes in time in terms of the risk associated with site effect amplification.

It is suggested that site effect amplification is not rated in terms of the vulnerability of the excavation to the ground motion at his stage.

10.4 P₃p₁– Exposure of people

A rating for the exposure of people to the risk of seismic driven ground motion is required. A basic example is the seismic risk to people in a shaft that varies greatly between the start of the day shift and, say, three hours later.

A rating method similar to that used in rating the distribution of seismicity, P₁p₃, is proposed.

Risk rating for P ₃ p ₁	Description of time distribution of people
1	0 to 20% of max. number of people exposed
2	20 to 40% of max. number of people exposed
3	40 to 60% of max. number of people exposed
4	60 to 80% of max. number of people exposed
5	80 to 100% of max. number of people exposed

Table 10.4.1 A proposed risk rating based on the time distribution of people in a particular working area or in transport.

10.5 P₄– Quality of information

Seismic risk assessment is only possible with structured and easily accessible data/information. The data could be provided by mine design, geological mapping, seismic monitoring, seismic interpretation and numerical modelling.

Other relevant factors determining the quality of information is the risk assessment interval and the volume of rock mass considered. Furthermore, effective communication for input for assessing risk as well as the effective communication of the results, is a prerequisite.

10.5.1 P₄P₁– Mine plans/Structure/Layout

The risk assessor requires accurate input as to the

- current position of mining,
- the correlation with the planned position of mining,
- knowledge of the position and orientation of significant geological structure
- the nature of the structure and its contact with the host rock

Risk rating for P ₄ P ₁	Description of quality of Mine plans/Structure/Layout
1	Excellent mine planning processes with accurate knowledge of all significant structures
2	Good mine planning processes with accurate knowledge of all significant structures
3	Adequate mine planning processes with reasonable knowledge of all significant structures
4	Poor mine planning processes <u>or</u> poor knowledge of significant structures
5	Poor mine planning processes <u>and</u> poor knowledge of significant structures

Table 10.5.1 The risk associated with mine plans/structure/layout

10.5.2 P₄P₂– Seismic monitoring

It is impossible to meaningfully assess seismic risk without quantified seismic monitoring. Monitoring also implies the ability to process and interpret seismic data.

The risk rating of seismic monitoring is primarily based on the

- quality of the recorded data, including accuracy of location and the quantification of seismic parameters;
- the sensitivity of the network;
- and the quality of processing and interpretation

Risk rating for P ₄ P ₂	Description of quality of Seismic monitoring
1	Adherence to the monitoring standard as suggested in the "Guide to routine seismic monitoring in mines", ISSI, 1999
2	Quantitative seismic monitoring, processing and interpretation with a sensitivity at least one Magnitude unit less than the smallest event being manifested as a rockburst
3	All seismic events manifested as rockbursts recorded with an accuracy well within the dimensions (20%) of the risk assessed volume of the rock mass
4	The recording is part of a regional seismic system with only the larger events recorded (>Mag 2) and/or with accuracies worse than 200 m
5	No seismic monitoring

Table 10.5.2 The risk rating based on the quality of seismic monitoring

10.5.3 P₄P₃–Seismic early warning

Some debate still exists on the validity of the concept and the practical value of seismic early warning in South African deep gold mines. The value of early warning, even if it only creates awareness, cannot be disputed and is therefore included as a risk parameter.

Risk rating for P ₄ P ₃	Description of seismic early warning
1	The concept and methodology of seismic early warning are accepted and implemented to a level of evacuating people, or preventing entry, from areas with a perceived high probability of experiencing large seismic driven ground motion.
3	Seismic interpretation forms the basis for pro-active management decision in preventing, control and prediction of rockbursts
5	No pro-active management decision in preventing, control and prediction of rockbursts

Table 10.5.3 The risk rating based on the practice of seismic early warning

10.5.4 P₄P₄–Assessment interval and assessment volume

A long time interval between risk assessment, as well as a larger area of mining being assessed, will lead to an averaging of the risk rating. This averaging will result in being of less value as an input towards managing seismic risk. An average good (say 3) rating for the total mining associated with a shaft, may hide some high-risk anomalies in specific sections or panels.

A similar argument can be made for assessing risk at long intervals, such as every six months, it is true that some parameters may change slowly, but others would be more dynamic.

Risk rating for P ₄ P ₄	Description of assessment interval and assessment volume
1	Risk assessment done on individual working areas, for example a single panel and its gully. Time interval prescribed by the fastest changing parameter, for example an increase in seismicity on a structure close to the working place
3	Risk assessment done monthly per panel/working place
5	Risk assessment done at intervals more than quarterly and/or for all mining associated with a shaft

Table 10.5.4 Risk assessment rating based on the assessment interval and assessment volume

10.5.5 P₄P₅–History

The importance and relevance of having an experience reference was described in Section 1.5.4 (page 21). An effective database should exist and allow for easy access to:

- earlier risk assessment exercises,
- risk management decisions and outcomes,
- seismic data,
- seismic damage,
- seismically linked accidents,
- production,
- mine face positions and that of major structures.

A suggested rating is given in Table 10.5.5, which could be significantly improved by rather defining absolute levels.

Risk rating for P ₄ P ₅	Description of quality of experience reference
1	An excellent experience reference
2	A good experience reference
3	A reasonable experience reference
4	An ineffective experience reference
5	No structured experience reference

Table 10.5.5 Risk rating based on the quality and availability of historic data.

10.5.6 Communication

Effective risk assessment relies on effective communication for the input of data on individual parameter ratings and also communicating the results of the assessment process as input for pro-active risk management.

Risk assessment is of little value if it is not part of risk management decision-making

Risk rating for P ₄ P ₆	Description of quality of communication
1	Excellent communication in terms of input to, and output from the risk assessment process
2	Good communication in terms of input to, and output from the risk assessment process
3	Average communication in terms of input to, and output from the risk assessment process
4	Poor communication in terms of input to, and output from the risk assessment process
5	No communication in terms of <i>either</i> input to, <i>or</i> output from the risk assessment process

Table 10.5.6 The risk rating based on the quality of communication

10.6 Conclusions

This chapter has provided a suggested procedure for evaluating seismic risk. The approach was holistic and attempted to address all the factors contributing to seismic risk and also to provide an appropriate weighting as for their respective importance.

It is not supposed to be seen as a 'universal' best practise, but rather a general approach or methodology to be adopted. Different environments may experience the parameters contributing to seismic risk of being of more (or less) relevance.

11 Conclusions

The expectations around seismic prediction in the South African gold mining industry were to some extent shattered. However, the concepts and tools developed to achieve the goal of prediction are appropriate and valid in the more holistic approach towards the assessment and management of seismic risk.

This report reviewed the techniques used to quantify the possibility of rock mass instability, and recommends the use of these techniques in assessing seismic risk. A similar approach towards defining effective management of seismic risk is recommended.

The project did not intend to describe Rock Engineering risk assessment in general. The emphasis was on the risk associated with dynamic ground motion. There is a large overlap between this and non-seismic driven falls of ground. The assessment procedures suggested in Chapter 10 can, with some exclusions, still form the basis for risk assessment in a low seismic risk environment.

A logical extension of the work described in the project is to incorporate the suggested risk assessment procedures in an expert system environment for easy application in the industry.