

Compendium of Research Project Summaries on Rockbursts - 1994 to 2016

MHSC
Mine Health and Safety Council

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MHSC

Mine Health and Safety Council

CEO's Foreword

The Mine Health and Safety Council (MHSC) is a national public entity established in terms of the Mine Health and Safety Act, No 29 of 1996, as amended. The entity comprises of a tripartite board represented by State, Employer, and Organised Labour members under chairmanship of the Chief Inspector of Mines. The MHSC is mandated to advise the Minister of Mineral Resources on occupational health and safety (OHS) issues in the mining industry relating to the development and implementation of the MHSC annual OHS research programme. The MHSC also conducts reviewing and development of mining OHS legislation and dissemination (knowledge and technology transfer) of MHSC research outcomes to improve OHS conditions in the South African mining industry (SAMI).

During the 2014 OHS summit, certain milestones were collectively developed by the tripartite stakeholders in order to accelerate the attainment of Zero Harm in the mining sector. These milestones were agreed upon by the SAMI principals in November 2014. These milestones provide a roadmap which will assist the sector in eliminating fatalities, injuries, occupational lung diseases and noise induced hearing loss (NIHL) as well as the reduction and prevention of tuberculosis (TB), HIV and AIDS infections.

For safety, it was agreed that fatalities shall be eliminated by the year 2020 that serious injuries shall be reduced by at least 20% per year from December 2016 onwards and lost time injuries shall be reduced by 20% per year from January 2017 onwards.

Falls of ground still remains the leading cause of mine fatalities apart from other causes such as machinery. Falls of ground accidents caused by seismic events (rockbursts) have been recently occuring at an unacceptable rate despite having a historical declining rate. When they occur, the results are normally catastrophic and result in multiple fatalities. The MHSC has completed numerous projects over the years to assist mines in understanding seismicity in order to enable an effective risk assessment and management during mine design and operational phases of mining. Synopses of work undertaken in this area are highlighted in this publication.

Thabo Dube
Chief Executive Officer
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December 2017

1. Preliminary study of the effect of fault properties and mining geometry on the stiffness of the loading system in fault slip seismic events as a basis for identifying situations prone to seismic activity

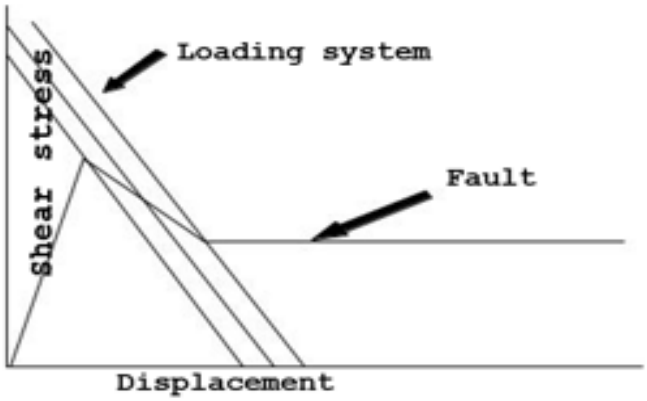
By Dr Esterhuizen E., University of Pretoria, Ref: GAP003, 1994

SUMMARY:

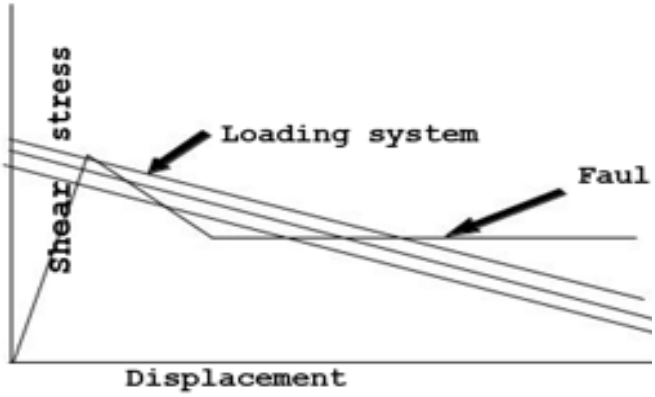
The mechanism of most large seismic events in mines is known to be shear displacement (slip) along geological faults. Seismicity occurs if the slip takes place violently and kinetic energy is released into the surrounding rock in the form of seismic shock waves. Under certain conditions non-violent slip may occur without releasing any kinetic energy. One of the factors which determines the amount of kinetic energy released during fault slip is the shear stiffness of the surrounding strata. The surrounding strata are seen as a loading system which drives a fault to failure. If the shear stiffness of the loading system is high, faults are less likely to slip in a violent manner.

The objective of this investigation was to determine how the relative geometry of a fault and a tabular excavation affects the stiffness of the loading system so that situations which are prone to seismic activity can be identified. Figure 1 shows how stiff loading systems can lead to stable slip.

A simple two dimensional numerical model was used to simulate a fault and a tabular excavation. Only quasi-static conditions were considered.



a) Unstable slip



b) Unstable slip

Figure 1: Sketch showing the effect of soft and stiff loading systems on stability of slip.

CONCLUSION:

The study has shown that the length of fault subject to slip has the most significant effect on loading system stiffness. Any changes to the system which will result in a reduced length of potential slip will therefore also reduce the likelihood of uncontrolled failure. Factors like the ratio of horizontal to vertical stresses, fault strength, fault dip, excavation span and support in the excavation will all affect the system stiffness by affecting the length of potential fault slip. In general it can be said that any action taken to reduce the length of fault slip will also reduce the potential for violent slip.

The potential for uncontrolled slip is decreased by decreasing the excavation span. However, a significant improvement is only achieved by reducing the excavation span to less than the critical half span. The depth below surface has only a minor effect on the loading system stiffness. This means that the potential for violent slip does not change with depth.

Backfill may improve the stiffness of the loading system considerably if there is sufficient closure to load the fill. For example, at a depth of 1000 m with mining spans of 200 m the backfill will have a minor effect, but at 3000 m depth with similar spans there will be a significant effect on the system stiffness. Backfill appears to improve the stiffness of the loading system only along that portion of the fault which dips towards the mining excavation.

2. Seismology for Rockbursts Prevention, Control and Prediction

By Dr Mendecki A.J., ISS International Ltd, Ref: GAP 017, 1996

SUMMARY:

Routine seismic monitoring in mines was introduced over 30 years ago with two main objectives in mind:

- immediate location of larger seismic events to guide rescue operations;
- detection of potential large rock mass instabilities.

The first objective was achieved fairly quickly but with the subsequent development of mine communication systems, its strategic importance has diminished. The very limited success with prediction can, at least partially, be attributed to three factors:

- seismic monitoring systems based on analogue technology that provided noisy and, frequently, poorly calibrated data of limited dynamic range;
- the non-quantitative description of a seismic event by at best its local magnitude; and
- the resultant non-quantitative analysis of seismicity, frequently through parameters of some statistical distributions, with a somewhat loose but imaginative physical interpretation.

The overall objective of this project was to develop methodologies for seismic monitoring for rockburst prevention, control and prediction utilizing new technology seismic monitoring equipment and the recent progress in the theory and methods of quantitative seismology.

A new methodology for the quantification of rock mass response to mining in space and time has been developed and applied. New seismic parameters, i.e. energy index, apparent volume, viscosity relaxation time, seismic Deborah number, diffusion and seismic Schmidt number, have been defined to quantify the complex nature of seismic flow of rock. The phenomenological model of the nucleation of rock mass instabilities has been developed and tested in hundreds of cases in both forward and back analysis mode. In this model seismic rock mass behaviour before instability is characterised by accelerating deformation (e.g. an increase in cumulative apparent volume, ∑VA) and by softening (e.g. decrease in average energy index EI)- see Figures 1 and 2 as an example.

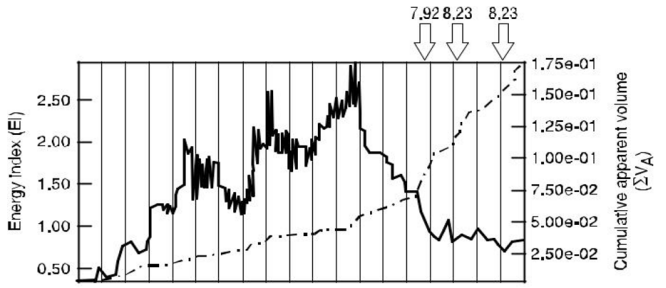


Figure 1: The history of SVA and moving median EI - for the seismicity associated with the WH 6# pillar mining, June 1992 to December 1994. Note that all seismic events larger than log E>7.5 occurred after the drop in EI and during the increased rate in coseismic inelastic deformation.

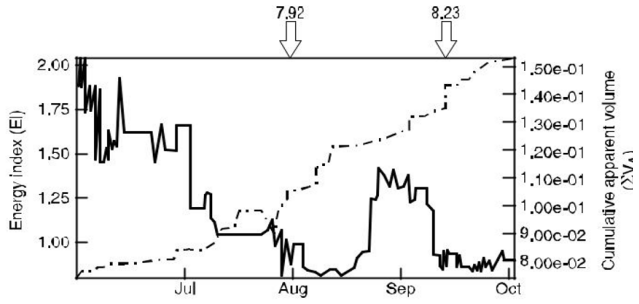


Figure 2: The first two strong events shown in Figure 1, are well predicted by accelerating seismic deformation ∑VA and drop in median energy index EI.

Extremely encouraging is the observation that there seems to be a degree of scale invariance in seismic rock mass behaviour before large and small events.

Basic guidelines for seismic data analysis and interpretation for the purpose of early warning of large instabilities have been defined. The procedure to quantify the potential for instability in terms of “time to failure” has been defined and is currently being tested. Using non-linear dynamics (chaos) theory the predictability of seismic rock mass behaviour has been demonstrated and its limits calculated. In all cases considered, these limits are always manageable and range from a few hours to about three weeks.

A success rate of 35 per cent to 60 per cent has been achieved in the forward prediction of larger rock mass instabilities at a few sites where the new methodologies are used in operational mode.

The success rate is best where best coverage assures a high sensitivity and data throughput. Reaction to the warnings includes improvement of support, the slowing down or temporary suspension of production, and orderly evacuation (e.g. five hours before a magnitude M=4.0 rockburst, a specific part of a mine was evacuated).

It has been shown that the success rate in predicting instabilities depends on the sensitivity and throughput of the seismic monitoring system. There is a need for a research programme on the application of non-linear dynamics to the prediction of rock mass instabilities and for technologies capable of recording, processing and integrating thousands of seismic events per day to improve the success rate of rockburst prediction.

The results of GAP 017 formed the basis of the book, “Seismic Monitoring in Mines”, authored by ISS International and published by Chapman and Hall of London in 1997.

CONCLUSION:

The new standards of seismic data acquisition, seismological processing and interpretation have been defined and technology transferred to the mining industry. It is currently applied on 20 SA gold mines on a routine basis. Fifteen mine seismologists have already been specifically trained for this purpose and are employed in the SA gold mining industry, while several more are in training.

3. An Investigation into the Accessibility of the Seismically Active Fault Plane in which to Perform Controlled Fault Slip

By Goldbach O.D., CSIR: Mining Technology, Ref: GAP 341, 1996

SUMMARY:

Major seismic events (M>3) are often directly associated with slip on geological discontinuities (Figure 1). Rockbursts resulting from slip on such discontinuities account for many fatalities on South African gold mines (Figure 2). The CSIR's Division of Mining Technology has been actively investigating various techniques to reduce the incidence of such rockbursts, including the use of high pressure gas injection to induce controlled fault slip.

Continued research into the feasibility of pro-actively de-stressing seismically hazardous geological faults through the injection of propellant initiated gas is being questioned by the SIMRAC committee in terms of the practical implementation of such a technique in the field. Specifically, concerns exist about the accessibility of geological discontinuities targeted for de-stressing.

This report presents examples of possible field scenarios in which the proposed technique may be practically implemented in scattered, longwall and sequential grid mining environments.

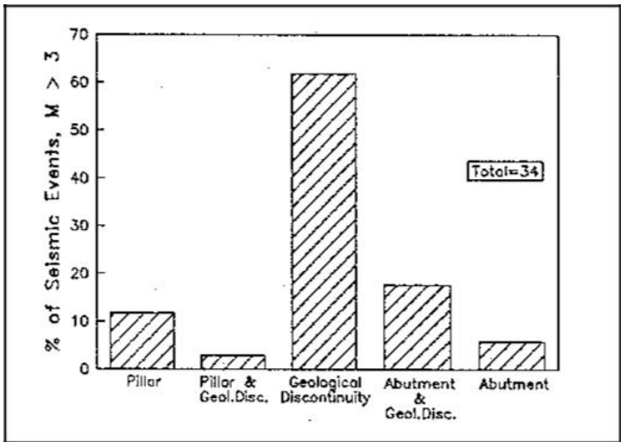


Figure 1: Large seismic events (M>3) for a two year period and where they occurred.

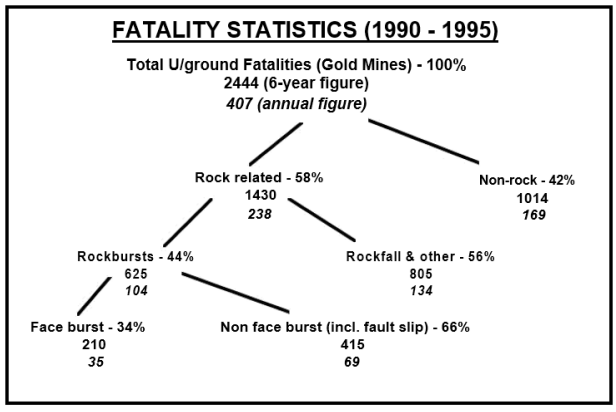


Figure 2: Underground fatality statistics on South African gold mines.

CONCLUSION:

The most likely potential applications for the gas injection method exist in the scattered mining environment. Fault accessibility in a longwall mining environment is more restricted and may require planning of access ways to get close to the targeted structures. In a sequential grid mining layout the advance development of haulages and cross-cuts, together with ledging up to two years prior to the commencement of stoping, provides various possible access points to geological structures requiring de-stressing.

4. Improvement of worker safety through the investigation of the site response to rockbursts

By Durrheim R.J., Milev A., Spottiswoode S.M, Vakalisa B., CSIR: Mining Technology, Ref: GAP201, 1998

SUMMARY:

Rockbursts are a major hazard in South African deep-level gold mines. The severity of damage due to a seismic event often varies greatly over small distances. The goal of this project is to discover the reasons for this variation in the severity of damage. It is believed that a detailed understanding of the damage mechanisms, and the application of this knowledge to the design and support of excavations, will lead to a reduction in the hazard posed by rockbursts.

Ground motion monitors were installed in tunnels and stopes in several deep gold mines. Peak velocity measurements made on the skin of an excavation were found to be four to ten times greater than those within the solid rock at similar distances from the source. Techniques to analyse the measurements were developed. It was found that the amplitude and phase of ground motion at points about 1m apart may vary considerably for small amplitude events, indicating strain across intervening fractures.

This phenomenon could have a pronounced influence on the stability of the hangingwall during larger, potentially damaging events. The damped harmonic oscillator model, widely used in earthquake engineering, was found to be useful for analysing the ground motion. Four hypotheses to explain the amplification of ground motion were investigated: resonance, energy trapping in a channel, energy release due to slab buckling, and energy release due to softening.

Detailed investigations of twenty-eight rockbursts were conducted. A team of specialists inspected damage to the excavations and support elements, mapped mining-induced fractures and other geological features, and interviewed witnesses. Seismograms were analysed to determine the source parameters, and the seismic history was reviewed in each case. Numerical modelling was used to evaluate the mining layouts. Finally, recommendations were formulated and presented to the mines concerned.

It was found that the severity of damage is strongly influenced by local rock conditions and support systems. certain shortcomings in the application of current technology, such as numerical modelling for mine design, were also identified. It was also found that published guidelines, e.g. for pillar design, have limitations which are not explicitly stated. The performance of new technologies such as preconditioning and pre-stressed yielding elongates was assessed under rockburst conditions and it was found that effective implementation depends to a large extend on training and discipline, and there were certain deficiencies in some of these support types.

In the short term, the most important means of reducing the severity of rockburst damage is to ensure that: support is always up to standard, stope support is as close to the face as possible, frequent inspections are conducted by personnel able to identify changes in the rock mass condition and with the authority to recommend and implement appropriate changes to layout and support systems, and sound layouts are adhered to regardless of the demands of production. Further gains may be achieved, in the medium term, through the improvement of areal coverage in gullies and stopes. In the long term, the effectiveness of support systems may be improved by applying earthquake engineering principles which take factors such as the duration of shaking, repeated seismic loading and the structural response into account.

CONCLUSION:

Several techniques were developed to analyse the ground motion measurements. Ground motion measurements made on the skin of the excavation indicate a four to ten fold magnification of the peak velocity when compared to measurements within the solid rock at similar distances from the source.

It was found that the amplitude and phase of ground motion at points about 1m apart may vary considerably for small amplitude events, indicating strain across intervening fractures. This phenomenon could have a pronounced influence on the stability of the hangingwall during larger, potentially damaging events.

5. Deep mine layout design criteria

By Vieira F., Selfe D., Dede T., Maccelari M., York G., SpottiSWoode S., Esterhuizen G, Milev G., Johnson S., Webber S., Arnold D., Handley M., Berlenbach A., Silva A., Ruskovich A., Vlietstra D., CSIR: Mining Technology, Ref: GAP223, 1998

SUMMARY:

This project has developed improved criteria, guidelines and the use of numerical models to design mine layouts at depth. Bracket pillars, strike-stabilizing pillars, concrete pillars, backfill and caving were all studied, using a range of numerical techniques. Data frorn seismicity, GPR, seismic tomography, wire extensometers, stress measurements, fracture mapping, closure-ride stations and precise levelling were used and interpreted. New Mine design criteria were introduced.

Design charts for bracket pillar design were developed to provide rock mechanics engineers with an initial estimate of required bracket pillar width based on mining and geological factors that are easily measured. The rock engineer must also decide on a tolerable event magnitude that could take place on any geological feature. This is less conservative than requiring zero ESS, as was previously suggested. As the design charts are based on assumptions that cannot cover every case, they are meant for initial, comparative, assessment.

Rock engineering and geological data were compiled for numerous dykes and faults in a relational data-base to assist in determining relationships between their characteristic properties and in situ behaviour. A new method has been developed to improve the relative locations of seisrnic events with similar mechanisms.

These methods can be used to identify different source mechanisms within diffuse “clouds” of seismicity. The ERR concept has been extended to include the energy released while the on-reef stress is limited to a pre-defined value. A computer program, MINF, reads MINSIM mining pattern files and implements this concept on shaft-sized mining layouts. Criteria for regional support should be presented in terms of probabilistic concepts.

This approach falls out of the scope of the current project and has not been pursued. A conceptualised design process for stabilising pillar layouts has been proposed.

CONCLUSION:

Dynamic modelling suggests that a minimum angle of 350 should be maintained between a longwall face and a seismically active discontinuity (Figure 1).

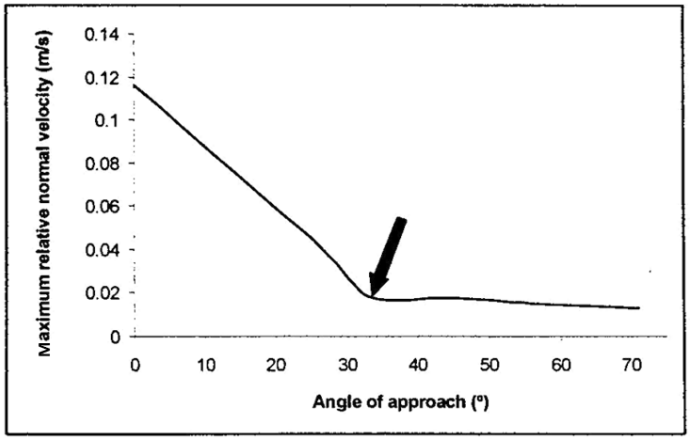


Figure 1: Maximum relative normal velocity in the stope for various approaching angles to a discontinuity.

Many stabilizing pillars yield seismically when stress Is imposed on them at Average Pillar Stresses (APS) perhaps as low as 250 MPa The pillar size or spacing did not significantly influence the back-area seismicity Further deformation is reduced when substantial load is taken in the back area. Large lead-lag gaps between longwalls separated by strike pillars should be avoided. It is recommended that no more that one-year face advance interval should exist between two consecutive longwalls.

The presence of geological discontinuities in the pillars has a small effect on the total seismicity, whereas pillars with irregular pillar generate much more seismicity than do those with regular shapes.

6. Seismology for rockburst prediction

By de Beer W., ISS International Ltd, Ref: GAP017, 2000

SUMMARY:

Fatalities and injuries caused directly or indirectly by a large seismic event are damaging to employee morale, shareholder confidence and public perception of the mining industry. They also diminish progress made in other areas of health and safety. As primary output, a prediction algorithm was developed which automatically quantifies power law accelerated seismic release, learns from history and predicts characteristic events with a 53% to 65% success rate.

Three problems were focussed on:

- quantifying precursory patterns of functions defined using seismic data,
- quantifying limits of predictability and evidence of criticality,
- an automated predictor based on kinematics of failure (time to failure accelerating seismic release curve).

The research perspective is physics, viewing the rock mass as a complex system approaching and retreating from criticality: larger instabilities normalise stress regimes for a time.

Furthermore, a pattern recognition algorithm for characterising the seismic response of rock to mining was developed. The algorithm quantifies previous intuitive, qualitative analyses.

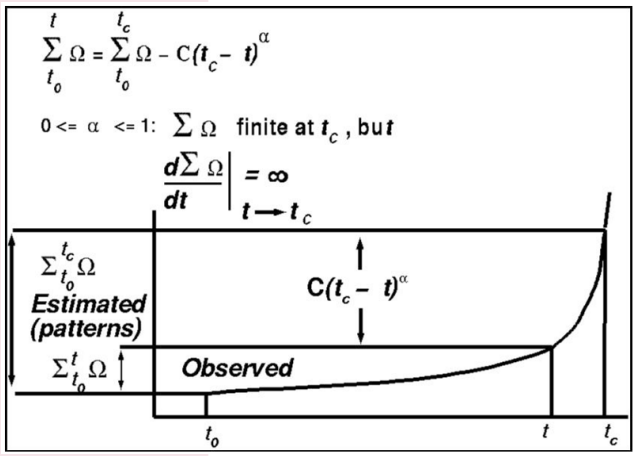


Figure 1: Graphical representation of critical accelerating behaviour by a variable (power law behaviour)

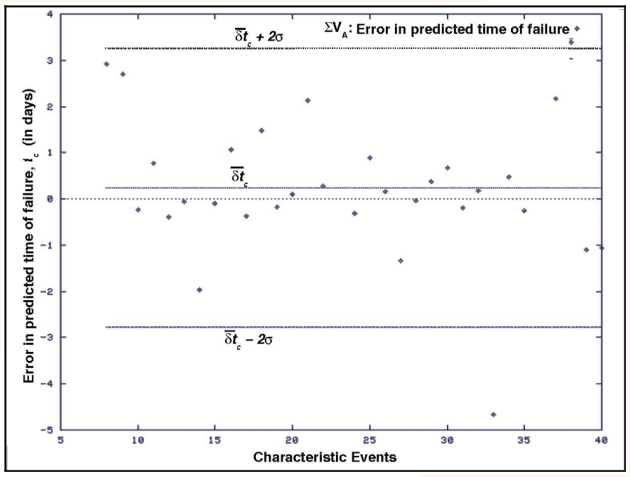


Figure 2: Average error made in the prediction of the time to failure in a Free State pillar extraction by using the SOOTHSAY algorithm and using time series of cumulative apparent volume.

CONCLUSION:

Simple, quantitative examples serve to illustrate that seismicity is not a random phenomenon and therefore not somehow inherently unpredictable.

Temporal patterns in seismicity can be quantified. Abundant, quantitative evidence for criticality and deterministic, reasonable limits of predictability exists. The prediction algorithm can achieve uncertainties in the predicted time of occurrence of a large instability of less than a week.

The prediction strategies followed are very far from random, and present a significant improvement, by a factor of about 4 over a pessimist random forecasting strategy.

7. Support technologies to cater for rockbursts and falls of ground in the immediate face area

By Daehnke A., CSIR: Division of Mining Technology, Ref: GAP 606, 2000

SUMMARY:

The aim of this project is to investigate temporary and face area support systems (with special reference to the use of tendon support). Improved face area support design methodologies are proposed, and a probability analysis is conducted to investigate excavation stability and the requirements of a temporary support system.

It is found that tendons are currently only used under quasi-static conditions. Typically tendons are used in shallow mines and to depths of 1 600 m below surface in situations where the UCS of the hangingwall rock exceeds 170 MPa. Tendons have been successfully used in stoping widths as low as 0,9 m. The use of tendons is primarily based on the presence of at least one pronounced hangingwall parting at a reasonable distance (0,2 to 3 m) from the reef – hangingwall contact (see Figure 1).

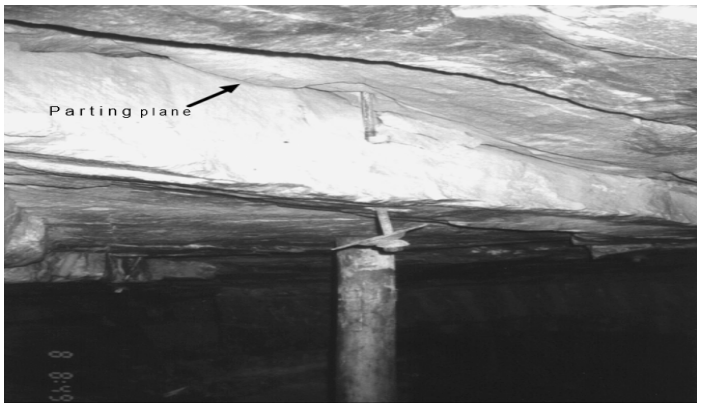


Figure 1: Hangingwall layering suitable for rockbolting (Kinross Mine). Note bearing plate loosened during blast

A detailed investigation into strata conditions, which are most suitable for particular face area support systems, was conducted. The use of tendons is generally recommended for strata conditions entailing a strong hangingwall, minimum hangingwall fracturing, but with problematic roof parallel discontinuities.

In other strata conditions appropriate columnar support types with adequate areal coverage are recommended. In conditions of weak and fractured hangingwalls, high levels of areal coverage are required. A major output of the investigation is in the form of tables giving recommended support types for various rock classes in shallow and intermediate/deep mining environments.

Underground investigations, as well as analytical and numerical models, resulted in an improved understanding of tendon interaction with a discontinuous hangingwall in quasi-static and dynamic conditions. An engineering approach for the design of stope face support systems is proposed and facilitates the convenient evaluation of support resistance, energy absorption and spacing requirements of tendons, props and packs in the stope face area.

Consultations with production personnel led to insights into operational constraints of temporary and face area support systems. Specific constraints investigated include labour availability, time, availability of support units, transport and storage, stope width, dip of reef, effect of mine geometry, and position of marked shot holes. Various solutions to overcome the operational constraints are proposed. The SDA II software (support design tool) was used to investigate periods in the production cycle when face area support systems are least able to meet their performance requirements. It was found that the shift, which normally enters the panel after the blast, is most vulnerable. To improve worker safety at the face, it is essential to reduce the unsupported hangingwall span before the workers enter the panel after the blast. In certain circumstances tendons are effective in meeting this requirement. Spray-on membrane support could also potentially provide this support in highly fractured conditions but the effectiveness and practicability of this type of support has yet to be evaluated in the aggressive face area environment. Practical constraints (appear to) militate against the use of coal mining type shield support in the face area of deep, narrow stoping width gold mine stopes mined by blasting.

CONCLUSION:

A probabilistic study was conducted to quantify the risks of injury, depending on the type of support, condition of the rock, mechanisms of deformation of the rock and support, support installation constraints

and personnel exposure. A methodology was formulated to determine the risks of injury associated with various support types recommended as optimal for representative classes of strata conditions. Detailed guidelines for applying the methodology and recommendations on the verification, calibration and expansion of the methodology are given in conclusion.

8. The design and development of an effective support system for tabular stopes in gold and platinum mines

By A. Daehnke et al., CSIR: Division of Mining Technology, Ref: GAP 708, 2000

SUMMARY:

Rock mass instabilities represent the single largest cause of injuries and fatalities suffered by the workforce in South African gold and platinum mines. The majority of rock-related fatalities (± 55 per cent) occur in the immediate vicinity of the stope face.

The objective of Phase I of SIMRAC Project GAP 708 is to develop a support system concept that addresses the deficiencies in current stope support systems, and that will significantly reduce fatalities in the stope face area in the short to medium term.

In order to achieve the objective, the deficiencies of current stope support systems were identified and include

i. poor area cover,
ii. poor installation practices, and
iii. poor face area support during cleaning (almost 50 per cent of all stope fatalities involve people whose activity at time of incident is related to cleaning or making safe).

Any successful alternative stope support system would have to comply with certain rock engineering and operational requirements related to gold and platinum mining. These were identified and are discussed in the final project report.

The following eleven alternative concepts were developed by means of reviews, workshops and brainstorming sessions, and are described in the final project report:

- High Pressure Stope
- Pneumatic Support
- Powered Shields
- Remote Miner
- Safety Cell
- Modified Spiling System
- Walking Beam Wishbone Support
- Twin Beam Support System
- Rockbolt Reinforcement
- Longhole Drilling
- Remotely Advanced Headboards

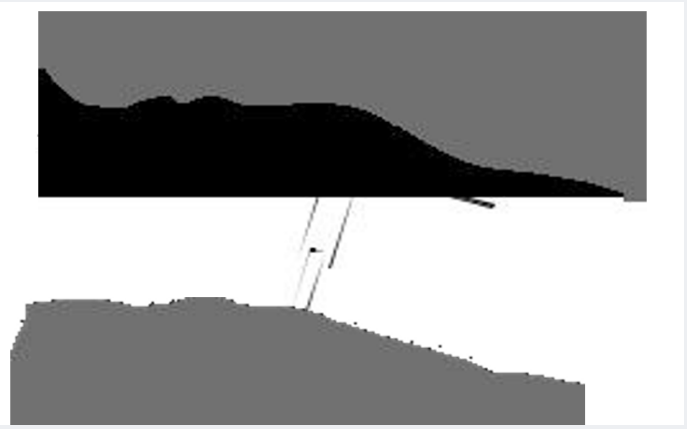


Figure 1: Remotely advanced headboard system.

These concepts were evaluated in terms of rock engineering and operational requirements, and a rating system (using current stope support systems as a benchmark) was devised to determine the relative merits of the different systems. The rating system is based on the likely improvement in safety due to the introduction of the system (reduction in fatalities and injuries in the face area), the practicality of the system (its ability to function in the difficult, varying conditions typically encountered in gold and platinum mines) and the research and development requirements (to get the system to a point where it can be widely implemented in South African gold and platinum mines).Detailed costing and planning of Phase II of SIMRAC Project GAP 708 was also carried out for the three recommended systems.

9. A Study of Rockburst Source Mechanism

By Ortlepp W.D., Steffen, Robertson and Kirsten, Ref: GAP 524, 2000

SUMMARY:

The research project comprising GAP 524 involved detailed examination of a group of 3 rockburst ruptures or burst-fractures discovered in a VCR stope panel on a peninsular remnant on Mponeng Mine at a depth of 2550m below surface.

A careful visual study of the exposures was made, samples were taken and minerological and geochemical analyses conducted on the fractured rock and the finely comminuted fault gouge.

CONCLUSION:

In every important detail the 87-50 6E ruptures were effectively identical to the type example studied in 1974 on ERPM Ltd. This confirms that the shear rupture type of origin is an important sub-class of rockburst source mechanism. This type of rockburst is probably more common than previously believed and does not require any unfavourable geological structure or 'bad ground' as a necessary pre-condition for its occurrence.

The relative direction of shear movement can easily be determined even without a stratigraphical marker and the sense of the movement is normal, dip-slip.

There are strong indications that the rupture has its origin at some considerable distance from the reef plane. The fracture front is driven through pristine rock at high velocity, towards the stope abutment.

Although there are intense frictional effects and probably quite high temperatures on the sheared surface; no chemical or mineralogical changes occur. In particular no coesite was found in the finely comminuted rock flour. Considerable similarity exists between the mechanics of the rupture process and that involved in crustal earthquakes.

As a result of this study a better foundation of understanding of rockburst source mechanism now exists on which to build future conjecture regarding important practical matters such as the magnitude of ground motion parameters like free-surface particle velocity and amplitude.



10. The meaningful use of peak particle velocity at excavation surface for the optimisation of the rockburst support criteria for tunnels and stopes

By Cichowicz A., ISS International Ltd, Ref: GAP 709B, 2001

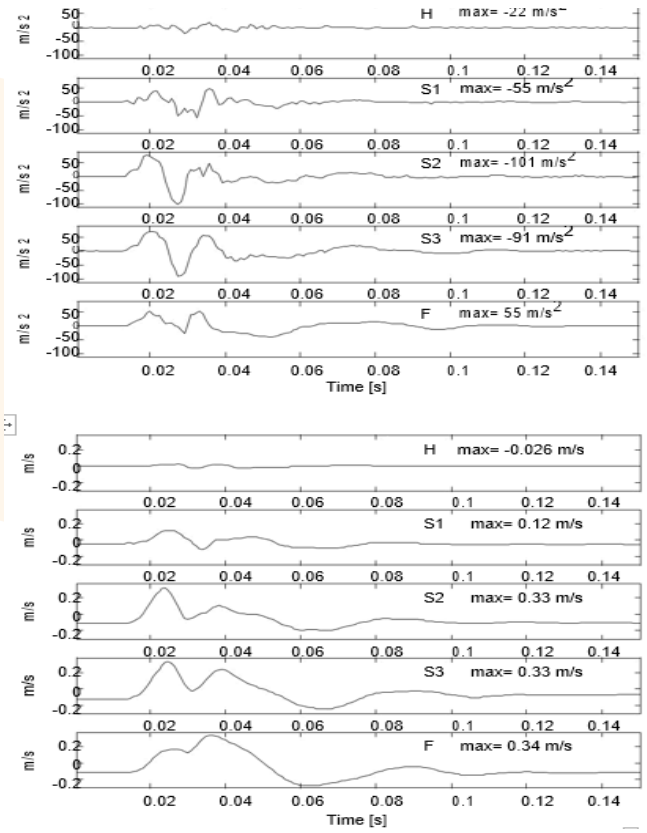
SUMMARY:

A set of data, collected from Mponeng and East Driefontein Mines, provides a new insight into the response of support units under seismic load. The most important accomplishment of monitoring is the measurement of peak ground acceleration caused by strong ground motion in the hangingwall, footwall and support units. The strongest monitored peak ground accelerations are in the range of 70-118m/s^2 which, after integration, relate to peak ground velocity in the range of 0.34-0.46 m/s. These parameters of strong ground motion are caused by seismic events of magnitude 1.2-2.4 observed from distances 80-140m. The strongest ground motion was monitored by horizontal accelerometers and not, as could be expected, by vertical ones.

A semi-empirical model of peak ground velocity is proposed to predict the ground motion of future seismic events. The ground motion of moderate seismic events with magnitude 0.7 and 1.2 is amplified in the stope, however ground motion generated by a large seismic event with magnitude 2.4 is not amplified in the stope.

Data shows that support response to strong ground motion has radically different characteristics compared to support response to weak ground motion. The weak ground motion reveals a total lack of coherence between ground motions in the hangingwall and footwall.

The near-field strong ground motion data provides a base for proposing a new model of support response due to seismic loading. The cyclic response of the support (brick composite pack) to cyclic seismic loading was unexpected (see figure).



Seismic event with magnitude 1.2 located below the stope, 69m from strong ground motion sensors, Mponeng Mine, November 25, 2000, 11:25:22. Hangingwall (H), support (S1,S2,S3),and footwall (F).

11. Relationship between ERR, system stiffness parameters and seismic energy release for different geotechnical areas

By Lachenicht R., ISS International Ltd, Ref: GAP 612A, 2001

SUMMARY:

The primary objective of the research was to develop and test modelled system stiffness methodologies and to undertake comparative analyses with seismic energy release for different geotechnical areas. From this objective, the results of the research can be used as a standard reference during the design of mining areas with rockburst potential.

A parallel project, 'The Relationship between ERR and Seismic Energy Release for different geotechnical areas (GAP612c) investigated ERR as a layout design parameter. As such, the applicability of ERR as a mine layout design parameter does not form part of this project.

This report summarises the results of extensive research into the delineation of appropriate mine layout design criteria incorporating stiffness as a design criterion. The methods documented are specifically designed to assess the seismic potential of the general rockmass fabric in terms of a mine layout design. The proposed methods are simple, robust and easily applied to mine design:

- The methods allow for the quantification of the long-term seismic potential of a mine layout design: facilitating for long term layout optimisation.
- The methods allow for the quantification and spatial delineation of the short-term seismic hazard of a mine layout: facilitating for controlled mining.

The detailed scope of work undertaken was:

- Undertake a literature survey to investigate energy and stiffness as design criteria
- Define the role and importance of stiffness in design methods for seismic hazard assessment
- Develop and test practical design methodologies (incorporating stiffness) for the assessment of the seismic potential of the general rockmass surrounding mine layouts.
- Compare the methods in terms of the applicability to the assessment of the seismic potential of mine layouts within deep level tabular orebodies
- Undertake detailed tests of the optimised methods against the observed seismic hazard associated with mine layouts in different geotechnical environments
- Refine the proposed design methodologies into detailed procedures which can be applied by Rock Engineering practitioners and consultants during mine planning
- Investigate the benefits of combining integration with the proposed design methods
- Develop guidelines to optimise a mine layout design against seismic hazard for South African conditions

The recommended methods can be summarised as follows:

Seismic Hazard Assessment Method (long term layout assessment):
The objective of the design method is to assess the seismic potential of a long-term mine layout design incorporating both a mine layout and a planned extraction sequence. This facilitates both mine layout and extraction sequence design optimisation. The design method is applicable to the South African context. In addition, as a result of the model parameter accurately representing the observed seismic hazard in combination with the strong empirical basis of the method, the resultant seismic hazard assessment of long term planned mining layouts can be directly utilised for risk assessment studies.

Stability Assessment Method (short term layout assessment):
SAM constitutes a design method which can be used in the short term to delineate potentially hazardous 'asperity regions', and in real-time assess the stability of these areas through a combination of modelling, seismic observations and integration – this forms the basis for controlled mining. This method must be applied and tested through a controlled mining set-up, i.e. the model must be updated on a regular basis (e.g. weekly) in conjunction with continual seismic monitoring and stability chart updates of potentially hazardous 'asperity regions'.

CONCLUSION:

In conclusion, the proposed design method is applicable to the South African context and facilitates controlled mining. The project report includes details and comparisons between the proposed design methods, comparisons of the applied design methods to quantitative seismic parameters and guidelines for the assessment of seismic hazard.

12. The meaningful use of peak particle velocities at excavation surfaces for the optimisation of the rockburst criteria for tunnels and stopes

By Milev A.M., Spottiswoode S.M., Noble B.R., Linzer L.M., van Zyl M., Daehnke A., Acheampong E., CSIR Miningtek, Ref: GAP 709, 2002

SUMMARY:

The aim of this project is optimisation of rockburst support systems by the systematic evaluation and quantification of the peak particle velocity recorded underground. To achieve this goal, a number of research tasks were carried out.

A cost-effective instrument especially designed for recording strong ground motion was developed and manufactured. The instrument, a Peak Velocity Detector (PVD) was used for extensive underground seismic measurements of the Peak Particle Velocities (PPVs) in Carbon Leader Reef and Ventersdorp Contact Reef, stopes and tunnels. A total of 41 sites were monitored. The results are summarised below.

- TauTona gold mine: 15 139 seismic events were recorded during 2 437 site days. The maximum PPV was 3 m/s.
- Kloof gold mine: 6 066 seismic events were recorded during 659 site days. The maximum PPV was 3 m/s.
- Mponeng gold mine: 1183 seismic events were recorded during 403 site days. The maximum PPV was 2.3 m/s.

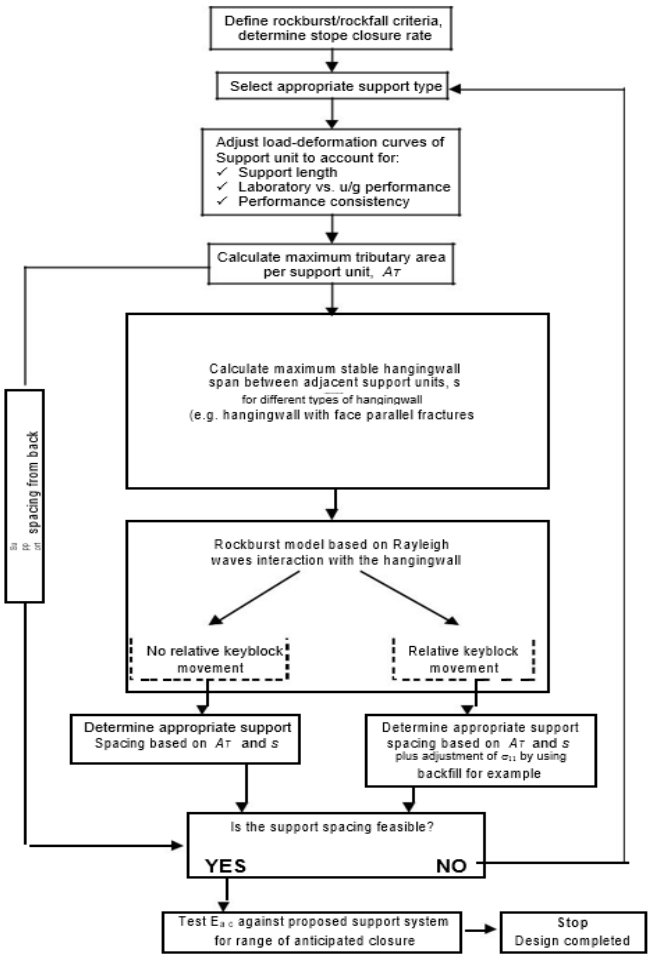
Back-analyses of mine network data to derive the site response on the skin of the excavations were conducted. The site response obtained at the surface of excavations was found to attenuate with the hypocentral distance.

The relationship between the site response and additional seismological parameters, such as hypocentral distance, source radius, wavelength (Lambda) and maximum velocity are discussed in this report. In addition, the relationship between source radius and hypocentral distance, and the relationship between the source radius and magnitude were outlined in an attempt to characterise the comprehensive rock mass behaviour under dynamic loading.

A statistical procedure using the maximum likelihood curve fits, was applied to estimate the probability of occurrence of a particular PPV. The curve fit for the PPVs recorded at TauTona was very good, while, the curve fits to PPVs recorded at Kloof and Mponeng were less satisfactory, particularly towards the high end of the velocity range.

The maximum expected PPVs estimated using the maximum likelihood approach (with a minimum and a maximum) were 4 295 mm/s at TauTona; 1 799 mm/s at Kloof; and 1 862 mm/s at Mponeng.

The velocity criterion of 3 m/s was found to be an adequate value to meet the requirements of support systems during a rockburst. In general, the present support systems and technologies do not need to be modified significantly to cater for rockfall and rockburst conditions.



Principles of ISSDM

A new physical model explaining the near-field and far-field distribution of the strong ground motion was proposed. A rockburst model, which takes the discontinuous nature of the hangingwall into account, was developed. The model focuses on the stress changes caused by the passage of Rayleigh waves along a stope.

An Improved Stope Support Design Methodology (ISSDM) was developed for rockburst conditions combining zones of support influence with hangingwall stability controlled by buckling, rotating and shearing keyblocks and the physical processes during the interaction between the hangingwall and propagating Rayleigh waves. The flowchart illustrating the principles of the ISSDM is shown on the following page.

13. Review of past research areas – seismology and mine layout design

By Milev A.M., Spottiswoode S.M., Noble B.R., Linzer L.M., van Zyl M., Daehnke A., Acheampong E., MF Handley (Institution), Ref: GAP816B, 2002

SUMMARY:

The Safety in Mines Research Advisory Committee (SIMRAC) has managed research on behalf of the mining industry since 1993, and is now becoming concerned that the industry may not have implemented many of the results. SIMRAC selected six projects, valued at R52.89-million, to be evaluated under Project GAP 816b for their level of implementation in industry. The six projects with abbreviated titles are:

1. SIMRAC Project GAP 017: Seismology for Rockbursts: Control, Prevention and Prediction;
2. SIMRAC Project GAP 034: Deep Mine Layout Design Criteria;
3. SIMRAC Project GAP 223: Deep Mine Layout Design Criteria (continued);
4. SIMRAC Project GAP 303: Mine Layout, Geological Features and Geological Hazard;
5. SIMRAC Project GAP 336: Preconditioning;
6. SIMRAC Project GAP 409: Seismology for Rockburst Prediction.

The objectives of GAP 816b as defined by SIMRAC are:

1. To compile a succinct and clear summary of the research outputs of the six project reports listed above;
2. To list those research findings that can be implemented together with guidelines on how to implement them;
3. To analyse the research done in the six projects and identifying gaps in the research;
4. To provide general guidelines for future research.

The work is tackled by studying the six project reports, summarising them, noting their outputs, and evaluating them for their implementability in industry. Then gaps in the research are identified and future research guidelines are given. There is a danger that this work may have a bias, originating from the personal perceptions and prejudices of the project leader. In order to avoid such a possibility, a survey of the level of implementation of the research in industry was carried out. An outside collaborator in industry then evaluated the work.

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

By Brink A.V.Z., Roberts M.K.C., Spottiswoode S.M., CSIR Miningtek, Ref: GAP 821, 2002

SUMMARY:

The objective of this project was to assess the seismic risk in the Bushveld Complex platinum mines and to provide guidelines for regional and local support in areas where the seismic risk is considered to be high. In order to achieve this objective, seismic data from some recently installed seismic networks were analysed. A significantly increased risk is observed with mining at depth (down to 2000 m below surface), and with larger mined-out areas. Potholes, a feature unique to the Bushveld Complex, do not show as increasing the potential for larger seismicity.

The deteriorated ground condition experienced when approaching potholes, however, enhances the likelihood of 'shakedown' damage and therefore increases the probability of an undesirable consequence (an increased risk of experiencing seismic related damage/injuries). Comparable mining areas were selected in the Bushveld and on the Far West Rand. Based on the analyses of the seismicity, and in particular the peak ground motion that an excavation is subjected to, the seismic risk associated with mining in the deepest areas of the Bushveld is about 6 times less than a similar area on the VCR.

An industry workshop on local support requirements in areas of higher seismic risk resulted in the specification of support requirements. A maximum design parameter for yielding support in terms of the ground motion velocity is 1 m/s. Subsequent seismic monitoring provided the confirmation of this specification. A stope support design methodology and parameters are given. A single example of the need for better regional support was demonstrated. An abutment pillar greatly reduced the convergence rate and seismicity in an area of mining down-dip from an extensively mined-out area.

The following conclusions were reached in this project:

1. The vast majority of the mining areas in the Bushveld do not experience seismicity that poses a significant risk to the safety of underground workers.
2. Seismicity is assumed by a significant portion of the industry to only occur in or around highly stressed pillars/remnants. The project experienced this to be true in the majority of areas that can be described as shallow to medium depth mining.
3. Accepting the above observation, seismic risk in the Bushveld is due to being close to the source (for example in the case of strain bursting) and also the apparent inability of the support units to yield while maintaining a stable hangingwall.
4. No seismicity was found that could uniquely be attributed to the existence of potholes, except where potholes were left as small remnants.
5. The researchers found clear evidence of dynamic failure on a geological structure. The deeper operations are experiencing normal (as in gold mining) seismicity ahead of the working face.
6. The amount of seismic energy released in the selected area in the Bushveld mines at comparable mining depths is in the same order (within an order of magnitude) as the selected and similar area on the Far West Rand.
7. There are strong indications that the daily blasting has a less concentrating effect in terms of temporal distribution of seismicity. A larger percentage of the seismicity occurs during shift times. In general terms it can be stated that the underground worker in the deeper sections of the Bushveld is experiencing 100% more events of Mag > 0 during normal shift time than his counterpart working on the VCR at a similar depth. Again this is an observation that requires a larger data set for validation. It furthermore requires an understanding of why the time dependant behaviour of the Bushveld exhibits such a different response to the daily blasting.
8. The risk posed by ground motion resulting from seismicity at Northam, is approximately 17% of the risk posed by ground motion at comparable mining on the VCR and 8% of the ground motion risk at

depth on the Carbon Leader reef in the Far West Rand. This aspect is the most significant observation in this report and requires to be confirmed by comparing more areas and the inclusion of data on seismic damage in these areas

9. The project found substantially different behaviour in the interpreted seismological behaviour between the familiar gold (Wits) environment and the Bushveld. In particular, in terms of time of day distribution, peak ground motion, influence of depth. The project did not succeed in quantifying the possible contributing factors, such as k-ratio or higher horizon stress. This aspect requires addition research and monitoring.

10. The design parameters for the support system provides for a relatively easy achievable energy absorption requirement (9,6 kJ/m^2).



15. Criteria for preconditioning at varying stoping widths in different geotechnical areas

By Toper A.Z., van Rensburg A.L.J., Milev A.M., Grodner M.W., Noble B.R., CSIR: Mining Technology, Ref: GAP 811, 2003

SUMMARY:

Following the completion of the research projects GAP030 and GAP336 certain mines have tried to implement the technique in faceburst -prone high stoping width areas but unfortunately these attempts were claimed to be unsatisfactory. The issue of why the current design of the technique could not meet the faceburst control requirements for these areas or whether the preconditioning technique was incorrectly implemented in those areas was unknown.

A few experimental sites were established in high stoping width areas and the research team ensured that preconditioning was implemented regularly and properly in these sites. After a certain period of regular and continuous preconditioning at these sites, it was observed that preconditioning provided a safer working environment in terms of protecting the mining faces from potential facebursts.

However, overhanging face shapes could not be eliminated when the preconditioning holes were drilled in the middle of the stope height (i.e. equal or greater than 1 m from the reef / hangingwall contact). When preconditioning holes were positioned at about 60 cm below the reef / hangingwall contact, overhanging face shapes were eliminated and improved hangingwall conditions were observed (Figure 1). The production personnel reported that the face advance per blast was increased after the introduction of preconditioning.

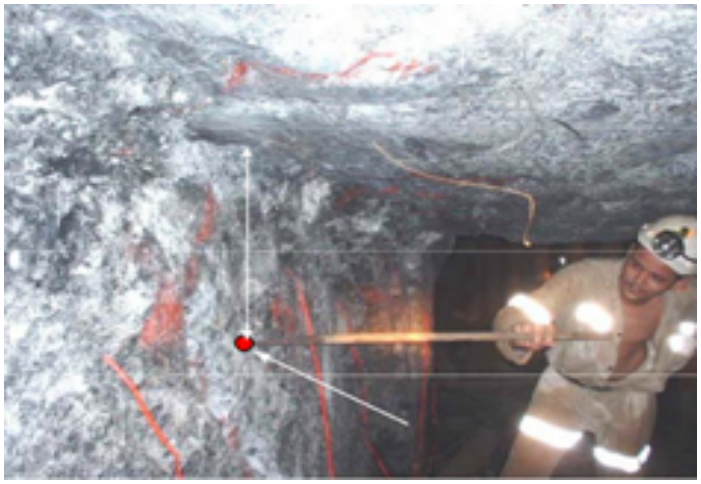


Figure 1: Improved hangingwall conditions where the preconditioning hole was drilled at about 60 cm below the reef / hangingwall contact

The temporal and spatial distribution of micro-seismic events following face-perpendicular preconditioning was studied to evaluate the effect of preconditioning on face induced seismicity. Two underground stopes, 109/48 and 109/51 at Mponeng mine were monitored during the course of this project.

The temporal distribution of the number of events after preconditioning blasting was found to be similar for both monitoring sites, 109/48 and 109/51, Mponeng mine. The only difference was the longer decay time at site 109/51 indicating that the mining layout may influence the post preconditioning seismicity.

The spatial migration of the seismic events after the preconditioning blasts was obtained for both sites. Most of the seismic events moved up to 15 m ahead of the face. This occurred within the first 10 minutes after the blast, which can be used as an indicator of the time for relaxation or transfer of stresses after the preconditioning blasts. This time was found to be faster than the relaxation or transfer of the stresses after the production blasts reported in previous studies.

Geological and geotechnical mapping studies of the high stoping width preconditioning sites has shown that preconditioning was effectively used at higher stoping widths. In addition, this study has also shown that preconditioning improves rockmass conditions when a potentially dangerous feature such as a dyke is encountered during stoping operations. These improvements include an increase in the number of face-parallel fractures (which act as a barrier to absorb the seismic energy of a rockburst and thus reduce the potential of face bursting) and an increase in the shearing along the stope hangingwall contact, resulting in fewer fractures penetrating into the hangingwall. This reduces the potential for fall of grounds and reduces dilution.

The major problems with the implementation of preconditioning were identified. Much has been learned from this recent study of preconditioning in

the mining industry in terms of issues that need to be addressed in order to ensure successful future implementation of the technique. Based on the results obtained from this study the guidelines for the best implementation of preconditioning were updated and a structured implementation process was proposed.

A booklet and a CD-based interactive training module for demonstrating the correct implementation of the preconditioning technique were produced to help the training departments of mines to continue the education and training process.



16. Experimental validation of a mine-wide continuous closure monitoring system as a decision making tool for gold mines

By Malan D.F., CSIR: Mining Technology, Ref: GAP 852, 2003

SUMMARY:

Previous fundamental research projects have indicated the value of continuous closure measurements for improved support design and hazard assessment and for quantifying the effect of seismicity on stope closure. This project examined the feasibility of a mine-wide continuous closure monitoring system. A RMT remote reading telltale system was modified to operate as a closure system and installed at Mponeng Mine. Unfortunately, this system did not succeed in collecting data from a large number of closure stations. Nevertheless, some valuable lessons were learnt. The two most serious problems experienced were communication problems from underground to surface and maintaining continuity of the cabling in the stope.

Three types of closure meters were designed and evaluated during the project. These designs operated satisfactorily. The cable network used in the stope presented a major challenge in terms of maintenance. Mining activity and falls of ground resulted in frequent cable damage. In conclusion, cable connections in the stope for a mine-wide closure system is not seen as a viable option. Radio communication is probably the best method to link closure meters to a data logger located elsewhere in the stope. Further problems were experienced with the communication system to surface. The RMT system required a copper link to surface. This was not available at Mponeng and attempts to use the fibre optic network were not successful. In future, mine-wide closure systems should be designed with the necessary flexibility to link-in with the existing mine communications systems, whether it is fibre optic, copper or leaky feeder.

Following the problems experienced with the RMT system, SIMRAC requested Miningtek to use the remaining funds to collect data from one site only using standalone closure meters. The site chosen was the 109/51 area at Mponeng Mine. Four panels were instrumented with CSIR closure meters. During the three month period of monitoring, only two significant events occurred. These were seismic events on 21 November 2002 and 17 January 2003 causing significant falls of ground in some of the panels. A very significant finding for mine-wide closure monitoring was that there is a very good correlation between the amount of seismic closure in the panels and where the damage occurs. In both cases the falls of ground occurred in the panels with the highest amount of seismic closure, even though the seismic events located closer to other panels that remained undamaged.

A useful parameter calculated from the closure data is the closure ratio (CR), which is the ratio of the instantaneous blasting closure to total closure following a blast. Note that this parameter is only defined for the closure following a blast and not a seismic event. Calculation of this closure ratio for closure data collected in earlier projects showed that it is a very good measure to identify different ground conditions and possible hazards. Closure ratio values of greater than 0.4 are typically associated with strain bursting conditions, while low values (typically < 0.1) are associated with significant risks of falls of ground. The average closure ratio calculated for the experimental site at Mponeng Mine is 0.5 (Figure 1). No face bursting occurred during the period of monitoring so any possible changes in closure ratio preceding these bursts could not be investigated.

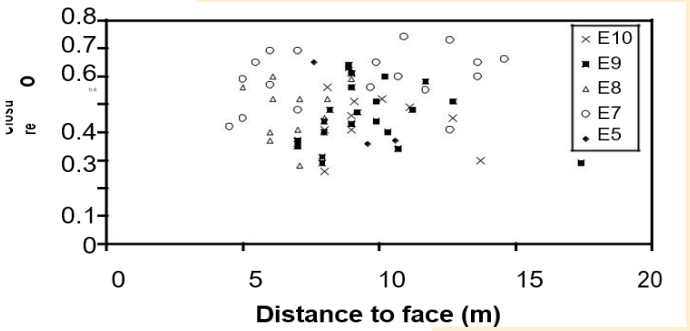


Figure 1: Closure ratio as a function of distance to face for the panels at Mponeng Mine in the 109/51 area.

Some numerical modelling using DIGS was conducted to verify the usefulness of the CR parameter. It was found that these initial results support the utility of the closure ratio in discriminating the response of different geotechnical environments. However, the present numerical simulations do not support more definite trends of the closure ratio as a function of time or as a predictor of incipient face stability.

“20 Years of Positive Contribution to Zero Harm for Mine Workers.

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17. Proactive approaches to rock mass stability and control

By Dr Lynch R.A., ISS International Limited, Ref: SIM 02 03 02, 2005

SUMMARY:

Primary Outputs

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

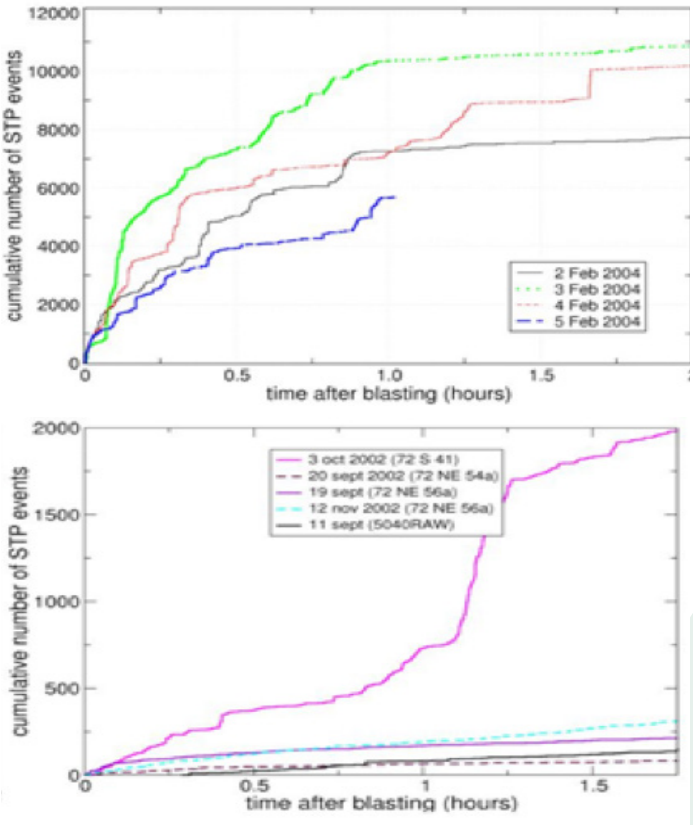


Figure: Short -term seismic responses to blasting for two mines with different levels of seismic activity. The responses on the right come from the mine with lower levels of seismicity. Note the different vertical scales.

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

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

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

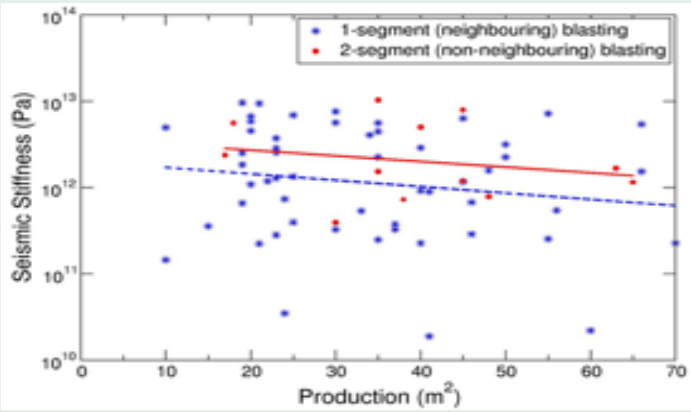


Figure: Blasting distributed panels tends to degrade the seismic stiffness less (and therefore leads to less seismic hazard) than blasting neighbouring panels at this pillar at Mponeng mine, even though the same amount of ore is extracted.

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

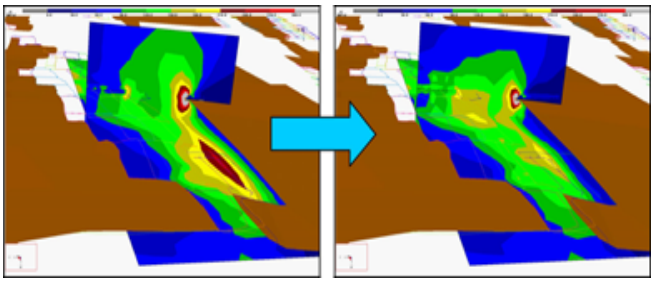


Figure: Integration of numerical models with recorded seismic data has the potential to offer improved seismic hazard assessments in the short - and medium-term, although the technique is only at the implementation stage now.



18. Improved seismic locations and location techniques

By Cichowicz A., Spottiswoode S.M., Linzer L., Drent D., Heyns P.S., and Handley M.F.,
University of Pretoria, Ref: SIM 02 03 04, 2005

SUMMARY:

Seismic locations are the most important information derived from mine-wide monitoring systems. The typical error of location varies from 30 – 100m, mostly depending on the density of geophones in the volume of interest. This location accuracy is sufficient to pinpoint which mining excavations may be damaged by the seismic event (hence providing guidance for a search and rescue effort), and to draw reasonably credible event location – damage distribution correlations. This permits a broad picture of relative seismic hazard mine-wide, but has now reached the limit of usefulness in further improving mine safety. Furthermore, there has been no real improvement in seismic locations in the last 10 years.

There are two ways to improve seismic data:
1. Increase seismic sensor density;
2. Use improved seismic location algorithms. The first option relies on the notion that the more seismic traces, the better the location. It also means that the sensitivity of the system increases, resulting in a greater processing workload for seismic staff. There is an associated increase in the seismic monitoring system operating and maintenance costs. The latter option provides a means of "getting the best out of existing systems" without any increases in workload or operating costs. It also provides a path towards fully automatic seismic data processing, relieving seismic staff of manual processing and quality control, leaving more time for interpretation and hazard management.

Absolute location methods use some or all the traces from a single event in order to provide a location for that event. Both earthquake and mine seismologists use absolute location methods and manual processing as standard. In contrast, relative location techniques use traces from two or more neighbouring seismic events to obtain locations relative to each other. There are many spectacular results from relative location techniques scattered throughout the earthquake seismology literature, while mining seismologists, aside from a few experiments in waveform similarity, are yet to implement these techniques. A literature survey revealed that the double difference location algorithm is more effective than waveform similarity techniques in the mining environment, because the latter is computationally intensive, hence slow, when large numbers of events must be processed. The summary objectives of the research were to:

1. Investigate weaknesses and strengths of the waveform similarity technique;
2. Identify and quantify sources of noise in seismograms;
3. Determine options for identifying seismic signals in noisy seismograms;
4. Develop relative location software for inclusion in current seismic location algorithms;
5. Review and refine automatic means of P- and S-onset identification;
6. Test improved seismic location techniques;
7. Estimate the error of seismic parameter determination.

CONCLUSION:

The prime objective is to reduce location error by an order of magnitude, since researchers in earthquake seismology had claimed 10-fold error reductions using the double difference method, and a 5-fold reduction in error using the waveform similarity method. This project has achieved a 2- to 5-fold reduction in location errors with minimum operator input and no extra cost. Source parameter estimation errors are reduced proportionately using the standard seismogram-source parameter relationships in seismology. Errors should be further reduced when geophone misalignments and site effects have been quantified and corrected. Methods to achieve this were developed during the course of this work.

Future work involves implementing the results of this work, compensating for geophone misalignments, automating the P- and S-wave pick inconsistency algorithm, and coupling this with an automatic relative location algorithm to produce good seismic locations with a reliability of 99.99%. More work is necessary to develop an automatic means of determining the quality and reliability of relative location results.

19. Calibration of Integration-Ready Numerical Models to Real Seismic Data

By Ilchev A.S., Mendecki A.J., ISS International, Ref: SIM 020307, 2005

SUMMARY:

The project is a continuation of GAP603: Fundamental Aspects of the Integration of Seismic Monitoring with Numerical Modelling.

It was decided that the parallel study of two integrated models as different as MAP3Di and IDRM, the Integrated Damage Rheology Model, will highlight the common advantages and potential problems facing any present or future implementations of the modelling with real seismic data.

The main result from the reported research is the demonstration that the calibration of integrated models to observed seismicity is possible. This simple statement of fact can change the status of the integration concept from that of an interesting idea to the level of a practical tool.

There is virtually no previous experience in the use of integrated models. In the case of MAP3Di, the prescribed calibration procedure has not been tested and for IDRM similar procedures were not even conceptualized before this project. This gap was filled at the start of the research. The main difference between the calibration strategies of MAP3Di and IDRM is in the choice of calibration criterion: while for MAP3Di this is a comparison between quantities which were inferred from the observed and modelled data, in the case of IDRM to calibrate means to force the model to reproduce one or another pattern in the observed data.

IDRM is still in the development stage. The work on its calibration supplied valuable feedback which was evaluated and used to improve the model. The most significant development in this direction was the addition of visco-elastic behaviour to the previously implemented non-linear elastic constitutive equations. Thus IDRM became a visco-elastic non-linear damage rheology model with enhanced capabilities of:

- Distinguishing between seismic and aseismic release of irreversible deformation
- Emulating the Kaiser effect under cyclic loading-unloading

The case studies conducted in the course of the reported project were set for Tau Tona gold mine, area 336 and for the Mponeng gold mine, level W109. The sites were chosen because of the high level of local seismicity and the quality of the available seismic data. The integration of MAP3Di with the seismic events observed during the interval of time between two mining advances was performed by the voxels method. The computed seismic displacement in the voxels was converted to stress increment and added to the stress field computed for the last mining step. This constitutes integration of real seismic data with the numerical model and it could lead to significant changes in the elastic stress field as shown in Figures 1 and 2.

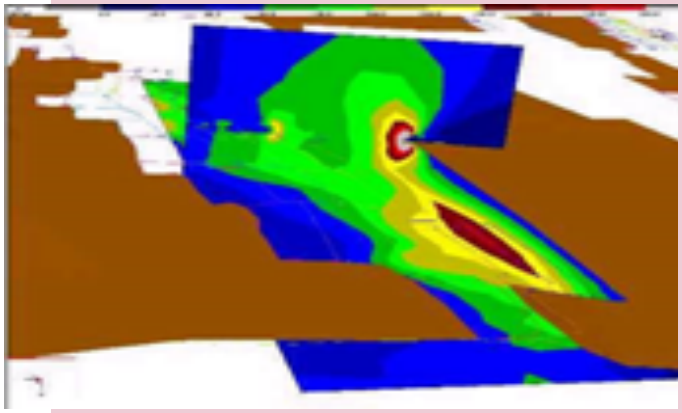


Figure 1: Major principal stress before the integration

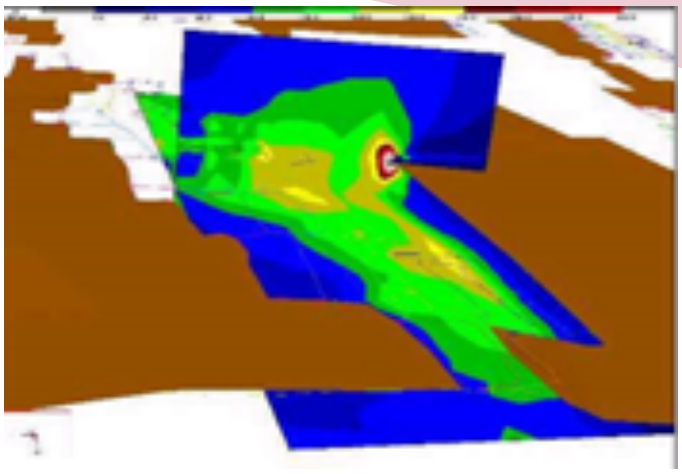


Figure 2: Major principal stress after the integration

The calibration of the integrated MAP3Di is done by comparing the strain energy in selected test volume as evaluated by the modelled elastic stress with the work done by the seismic displacements on the test volume.

The IDRM approach to the problem of calibration to observed seismicity is different from the one used by MAP3Di. The most valuable feature of IDRM is its capability to model the build-up and localization of sub-critical da ma ge whic h c a n lead either to a loss o f stability or to the co nc ur re nt process of healing. The fact that these processes can be modelled with very high resolution in the time domain opens unique possibilities for calibration: not just to an observed quantity or a set of quantities as it is the case in MAP3Di, but to a pattern.

The following procedures can be employed for the calibration of IDRM:

- Calibration to total potency of the events observed inside the IDRM domain within a given interval of time.
- Calibration to observed time distribution of potency release.
- Calibration to observed spatial distribution of potency release.
- Calibration to observed spatio-temporal distribution of potency release.

The IDRM potency release shows the typical features seen in local seismic data, namely:

- Delayed response
- Accelerated potency release after the blast
- Gradual slowin g-down of the seismicity after the blasting-driven increase.
- Accumulation of the excitation in the rock mass with the advance of the mining inside the IDRM domain.

The capability of IDRM to model the time dependence of the potency release in response to production mining is shown in Fig.3

The practical side of the integration is in the possibility to formulate new and, hopefully, more reliable criteria for structural stability. MAP3Di and IDRM use different approaches to this question.

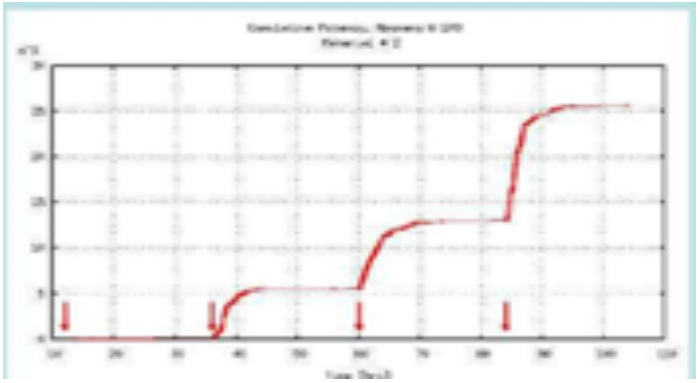


Figure3: Modelled cumulative potency v. time

The MAP3Di viewpoint;

The stability of structures can be assessed by calculating the stiffness of the structure from the integrated model. Another quantity to monitor is the energy stored in the structure of interest. The corresponding procedures are prescribed in the MAP3Di as structure stability assessment. The result of the structure stability assessment is presented in the form of a graph and the plotted quantity is called stability parameter. The choice of stability parameter is not one and the same for all cases.

For instance, if a general assessment of the stability of rock mass is sought for, the general stress level represented by the principal stress can serve as a stability parameter. The principal stress can be averaged over the structure and plotted against time. Another possibility is to plot energy versus stiffness. In that case higher energy is interpreted as higher seismic hazard. Likewise lower stiffness also means higher seismic hazard.*The IDRM viewpoint*
IDRM allows access to detailed information about the state of the modelled rock mass and the processes in it. This information can be analyzed for any signatures of an approaching instability.

One possible criterion for the approach of instability is the localization of higher level of damage in one or more parts of the modelled domain. The elastic modulae are damage-dependent and the effect of high damage level is softening of the material. Then these softer sub-domains will act as loading system for the adjacent, stiffer parts of the IDRM domain. This may lead to violent modes of failure.

Another possibility is to relate any spatio-temporal clustering of failed elements to the stability of a larger volume of rock. The size, location, orientation and shape of modelled seismic clusters reflect the weakening of the material due to damage evolution

and loading history. The formation, localization and spread of weakened zones can be interpreted in terms of hazard levels for the occurrence of structural instability.

Finally, here are some observations made in the course of the research on the project:

- MAP3Di and IDRM are very different numerical models. In fact, it is difficult to imagine any other two models which are as diametrically opposite as the above. Yet both MAP3Di and IDRM successfully implement the integration concept. This shows that the idea of modelling with real data does not impose any serious restrictions on type of models.

- The differences between MAP3Di and IDRM are reflected in the different calibration methods. This indicates that there may never be a universal calibration methodology which could be applied to any integrated model and in every case. The variety of calibration options should be seen as an advantage rather than as a handicap.

- MAP3Di, the first and still the only commercially available package for integrated modelling, is at the end of its development curve in terms of the integration. It is impossible to push a boundary integral method model of static stress any further without rewriting it from the beginning and adding some dynamics to it. IDRM, on the other hand, is at the beginning of its development and, without major rewriting, it can be extended to include: temperature effects, fluid flow and effective stress from pore pressure, dynamical loading and other phenomena of practical importance.

- One serious problem for IDRM is the numerical instability resulting from large deformations of the Lagrangian grid. It seems that the correct way to address this problem is by implementing a meshless algorithm preserving the fundamental equations of the model. This idea is being seriously considered in ISS International.

20. Minimising the Rockburst Risk

By Durrheim R.J., Essrich F.E., Ortlepp W.D., CSIR Division of Mining Technology, Ref: SIM 05 03 02, 2006

SUMMARY:

Phase 2 of project SIM050302 (Minimising the rockburst risk) has four primary outputs, of which three are covered in the report.

Output 1: is a detailed proposal for a 5-year rockburst research project. The duration of the project is sufficient to recruit and develop new expertise. The research priorities were identified using scenario planning and technology road mapping techniques. The proposal was tabled at the Rock Engineering Technical Advisory Committee (RETAC) meeting on 10 May 2006, and approved subject to minor amendments. These changes were made, and the final proposal submitted to the Mine Health and Safety Council (MHSC) programme manager on 19 May 2006. On 6 July 2006 the MHSC approved inclusion of the project in the Annual Research Programme for 2006/7.

Output 2: is a plan to coordinate and stimulate the transfer and implementation of rockburst knowledge and technology by improving the quality of training. It was found that training providers are generally competent and well informed with respect to recent research findings related to seismicity and rockbursting. There are sufficient training providers to meet current demand. The quality of training methods and materials are deemed to range from satisfactory to excellent. MHSC publications, especially the Handbook on rock engineering practice for tabular hardrock mines (Jager and Ryder, 1999, SIMRAC) was widely acclaimed as an effective means of

transferring knowledge and research findings to practitioners. However, there is a definite need to improve the general level of knowledge, competence and compliance in the mining industry. The standard of training and throughput of trainees is only likely to increase if it is driven by mining companies and/or regulation. Many of the constraints, such as low educational levels and difficulties in achieving changes in behaviour, are systemic and can only be overcome through the combined efforts of training institutions and authorities, the mining inspectorate, organised labour, employers, and researchers.

It is recommended that MHSC take the following actions: publish a poster summarising rock-related research relevant to hardrock mines; publish a handbook dealing specifically with mining-induced seismology; engage industry champions, university staff and other training providers in research activities. While there is scope for research into ways to improve the effectiveness of knowledge and technology transfer and implementation, the issue is generic and research should be undertaken accordingly, not as part of the rockburst project.

Output 3: seeks to formulate standard operating procedures and best practice in seismic hazard assessment and rockburst risk management, will be dealt with in a separate report.

Output 4 is a laboratory investigation into the formation of polyhedral quartz particles during dynamic brittle shear fracture. A high-pressure cell capable of quasi-statically applying loads approaching 1000 kN and pressures of the order of 2 GPa to a 25 mm diameter glass sphere was designed, built and proven. A method of rapidly unloading the apparatus was developed,' but the' rate of unloading was not sufficient to permit the elastically rebounding sphere to self-shatter. A simplified method of sealing a film of mercury around a smaller sphere has been devised that will permit effective loading to higher stress levels, and make the estimation of stress and strain simpler and more robust. Most importantly, the new method will eliminate problems of friction and large inertial mass inhibiting the rebound recovery rate.

21. New criteria for rockmass stability and control using integration of seismicity and numerical modelling

By Hildyard M.W., Napier J.A.L., Spottiswoode S.M., Sellers E., Linzer L.M., Kataka M.O., CSIR Division of Mining Technology, Ref: SIM 02 03 01, 2006

SUMMARY:

The main objective of the research project SIM 020301 was to investigate improved criteria for the evaluation of deep-level tabular mine layout design using integration of seismicity and numerical modelling. The final report addresses the basic formulation of rockmass stability assessment criteria, current practical strategies for integration of seismicity with numerical modelling, and future static and dynamic approaches to numerical modelling of mine seismicity, together with possible optimisation procedures for performing integration exercises.

Issues relating to the assessment of rockmass stability are examined in detail. An initial assessment is made of suggested parameters that can be evaluated from records of seismic activity in designated mining areas, together with the definition of measures of effective rockmass stiffness. It is argued that an explicit computation of ongoing changes in released energy, taking off-reef deformations into account, provides a direct method of assessing progressive seismic activity evolution as mining proceeds. The computation of the Generalised Energy Release (GER) can be accomplished by extending the generic displacement discontinuity method that has been used routinely in the mining industry. Examples are used to motivate the utility of the GER concept in layout stability evaluation and to indicate how this may also be used to monitor stability transitions in sub-critical crack growth that may precede rapid deformations.

A practical method for the integration and reconciliation of observed and simulated seismic activity is outlined. A simplified procedure, based on the MINF computer code, is proposed, where no feedback of observed seismic activity for adapting the model properties is attempted. MINF, and a new integration tool called MINSINT, can be used to estimate the amount of seismicity expected from any sequence of planned mining, based on past seismicity and mining, and the planned mining. The MINF and MINSINT programs have been integrated as a suite to test the concepts of large -scale back-analyses of mining and seismicity, covering mined areas of about one square kilometre or more in extent. MINSINT performs the integration of seismicity and modelling, and estimates the amount and distribution of expected seismicity within planned areas of mining.

Finally, a number of future developments for the integration of seismic activity with numerical modelling and stability assessment are described. The development of a computer code to simulate three-dimensional rock damage and fracture evolution is described. This code provides the capability of nucleating a series of crack growth elements at designated positions in space (seed points). The key assumption is made that rock damage zones can be simulated as an assembly of non-intersecting but mutually interactive circular crack elements. It is found that this approach is able to simulate non-trivial problems, such as the formation of coherent shear band structures and the development of plausible fracture patterns that arise near the edges of lead -lag and crush pillar configurations in tabular layouts. However, it is also found that a number of restrictions encumber the application of the current test code. Typical fracturing around a lead-lag is shown in Figure 1.

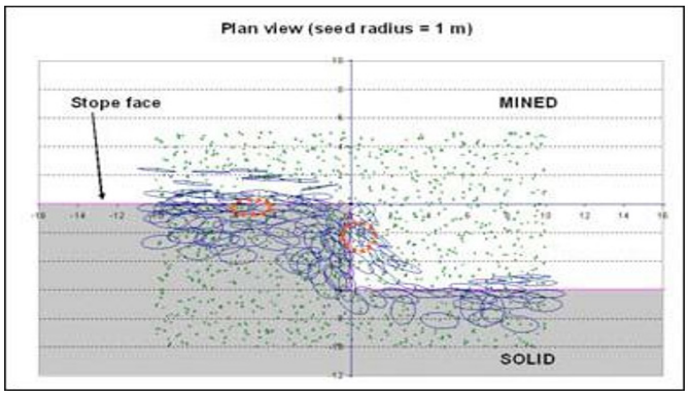


Figure 1: Fracture development around a lead-lag configuration with fixed radius seed points

Integration of numerical modelling with seismicity, as well as with other underground observations, can be considered as an optimisation problem. The first step of the integration process is an optimisation of the model parameters to minimize differences between the models of past mining steps and that of the observed seismic data during that mining. A genetic algorithm approach has been tested for a simplified stope model using successive solutions generated by the MINF program with different values of Young's modulus and strength parameters. The observed data is considered to be the closure at a point in the model and the genetic algorithm was able to find an optimal solution which lies within the input resolution. A "synthetic annealing" approach was also investigated. A sequence of runs is considered, where the intact strength is decreased by some specified interval and the other parameters are varied within certain ranges until there is a match between observed and predicted behaviour.

Various approaches for integrating dynamic numerical modelling with seismic measurements are also presented. The ultimate dynamic integration is for a numerical code to correctly project the dynamic rockmass response throughout a mined region, simply given inverted seismic parameters such as seismic moment, stress drop or moment tensor solutions. It is shown that this approach requires considerable further development due to uncertainties and limitations in both models and seismic data. Dynamic integration can also be used to run models to test and improve seismic inversions. This approach was used to evaluate the effect of mined out areas on traditional seismic inversions. Synthetic seismograms generated from models were inverted using standard seismic techniques. Results are presented where the inverted source magnitude and mechanism differs considerably depending on whether a stope was present in the model. Integration of dynamic modelling with the seismic inversion process could therefore lead to improved inversions and a better knowledge of the source mechanisms. A third form of dynamic integration is in the use of active seismic velocity scans. Active velocity scans would provide useful diagnostics on the degree of fracturing, and dynamic models provide a means of interpreting this fracturing. These velocity scans could also provide an important technique for evaluating and optimising models of fracture zone development. Examples are presented of simulated velocity scans and the effect of fracturing is shown to be detectable in the resulting waveforms.

While much has been achieved, continued work is required in a number of areas to fully realize their benefit. The integration methodology should be

implemented more routinely and encouraged to evolve. The novel methods of 3D fracture growth simulation showed that they can provide important insight into mining problems, such as the fracturing in pillars and lead-lag layouts. Nevertheless, such fracture generation remains a challenging problem for 3D mining models, and its application to a wider range of problems requires further novel development. Dynamic modelling requires more thorough investigation. Its use for more rigorously testing the meaning of seismic inversions is in its infancy, yet already indicates important, possibly far-reaching consequences for understanding the sources of mine seismic events. Dynamic fracture growth has yet to be suitably investigated, yet it remains a fundamental component of the rockburst problem.

22. The determination of loading conditions for crush pillars and the performance of crush pillar under dynamic loading

By Canbulat I., Groundwork Consulting (Pty) Ltd., Ref: SIM040302, 2006

SUMMARY:

Many gold and platinum mines use crush pillars with strike dimensions of 2, 3, 4, 5 or 6 m and 2 or 3 m on dip, separated by 0.5 to 3 m wide ventilation holings. These pillars are normally located on strike and with a 1 m siding on the down dip side of the strike gully. The mechanisms involved in pillar behaviour are complex and generalised formulations may not be applicable as many factors can affect pillar performance including local and regional geology, rock mass characteristics and mining. The rock mass and mining conditions of forty different sites at nine different shafts have been quantified. In addition, detailed geotechnical monitoring has taken

place at eleven sites. Data from closure-ride stations, extensometers, borehole camera surveys, geological mapping, geotechnical mapping and seismic monitoring has been collected. The response of crush pillars to seismic events (that is their dynamic behaviour) showed that firstly, the fracturing of the pillar continues to occur well behind the face and secondly, that crush pillars scarcely respond to larger distal events. The influence of geological discontinuities and mining induced fractures on crush pillar behaviour has also been established. In the different stratigraphic and structural domains studied, there does not appear to be a possibility of the pillar crushing to form a blocky rock-mass analogous to a UCS specimen.

Borehole logging and mapping of pillar holings created an extensive database of fracture characteristics of crush pillars across many different reefs, depths and ages. From this database the Pillar Fracture Index (PFI) was developed to quantify, especially, crush pillar behaviour. It is defined as

PFI = ∑(lengths of hole with fractures > 10cm apart) x100

The PFI provides a single number that encapsulates all the characteristics of the pillar and thus allows the easy comparison of different pillars. It was found that all crush pillars had Pillar Fracture Indices of less than 45. Slight variations in the PFI can be ascribed to variations in the geotechnical conditions and age of specific types of pillars.

23. Lead-lag design criteria and seismicity patterns

By Goldbach O.D., Rangasamy T., Linzer L.M., Kataka M.O., Spottiswoode S.M., du Pisani P.,
CSIR, Ref: SIM 040303, 2006

SUMMARY:

South African tabular gold mining operations typically involve different mining advances in adjacent panels, resulting in panels "leading" or "lagging" each other. It is believed that the lead-lag distance between adjacent panels can affect the distribution and characteristics of fracturing and damage within the mining faces, and may affect roof conditions and hence safety. The normal procedure is to minimise the lengths of lead-lags. Mines have generally adopted layouts where optimum lead-lags between adjacent panels are deemed to be between 5 and 10 m. However, these values are based on experience rather than on detailed measurements or an understanding of the deformation processes. There is thus a need to scientifically quantify the effects of lead-lags on fracturing, stability, support and seismicity to either confirm or modify current design guidelines. The research findings contained in this report aim to provide quantified guidelines on acceptable lead-lag lengths in gold mines.

This report begins with a comprehensive review of the problems associated with lead-lags and resulting strike gully stability. The available published material is summarised in a detailed literature review. Various rock engineering practitioners were interviewed in order to gain their perspectives about the problems surrounding lead-lags. A number of accident reports are summarised as examples of fatal situations where lead-lags were a contributing factor.

Current industry practice around lead-lags is evaluated by reviewing codes of practice (COP) to determine the currently adopted inter-panel lead-lag distances. The stipulated lead-lag lengths range between 4,5 m and 10 m. However, an analysis of in-situ lead-lag lengths shows that the adherence to the recommended standards is seldom above 50%.

Two sets of field measurements were made around lead- lags. Daily closure readings were made in three adjacent panels of a deep-level gold mine site for almost a year in an attempt to relate closure variations to lead-lag length. Detailed fracture mapping around lead-lags was undertaken at 14 sites across eight gold mines representing a variety of reef types, depths and mining methods.

The closure results produced very little useful information about lead-lags, apart from some evidence of an increase in closure rate in the gullies for lead-lags longer than approximately 10 m. The fracture mapping work produced convincing evidence that optimal lead-lags are in the range 4,0 m – 16,5 m, depending on the reef type. However, the guidelines for the selection of lead-lag lengths relate only to quasi-static conditions and any potentially damaging seismic events will impose further restrictions on the choice of optimum lead-lag length.

Seismicity associated with lead-lags was identified from mine seismic catalogues. Moment tensor inversions were used to calculate the source mechanisms of the lead-lag events. Fault plane solutions were used to confirm the selected seismicity as lead-lags events and to determine the focal mechanisms.

At Kloof, most events occurring on the VCR lead-lags were of the oblique-slip type. The largest lead-lag event had a magnitude ML=1,7 and it occurred on a lead-lag that was only 15 m long. The mechanisms of the lead-lag events were mostly implosive and had lower energy/moment ratios than shear or tensile events occurring on the faces or abutments. Seismic event magnitudes were larger for lead-lags in the range 14 m to 23 m. There was evidence of a seismic event occurring on a flat-dipping failure plane that cut obliquely across the corner of a 22 m long lead-lag.

At Tau Tona, none of the selected seismic events occurred on inter-panel lead-lags. However, seven of the 21 fault plane solutions were associated with abutments on the lower Carbon Leader Reef. The events occurring on abutments were predominantly oblique -slip or dip-slip type. The mechanisms of the abutment events were mostly shear events. The event magnitudes were larger for abutments lengths in the range 36 m to 52 m. A possible corner-cutting fracture was noted for one seismic event, which occurred in the corner of a 52 m long pillar abutment.

The effect of varying lead- lag length on the rock mass was simulated in two stages of numerical modelling: elastic modelling using MINSIM and inelastic modelling using Elfen. The results from the MINSIM modelling

showed, as expected, that increases in lead-lag length resulted in large increases in the vertical stresses, ERRs and elastic convergence. It was concluded that short lead-lags should be used to reduce the vertical face stresses and ERR. The intensity of fracturing and the absolute values of the vertical stresses and ERRs cannot be predicted by MINSIM. However, the results seem to indicate extreme vertical stresses and ERRs for lead-lags greater than 10m at 3000m depth.

The discrete fracture generation capability of Elfen was used to simulate damaged and failed regions around lead-lags. The Elfen model showed that damage increases as the lead- lag distance increased. However, the modelled fracture extents were much smaller than the underground observations because the increase in the friction angle (with increasing plastic strain) of the Mohr-Coulomb strain softening model needs to be less severe than has been modelled here. Further work is required to carefully calibrate the material model with both lab test data and underground observations to obtain a reliable indicator of the extent and magnitude of damage. While the results are not definitive, the work undertaken in this project indicates that the currently adopted lead-lag guidelines of 4,5 m to 10 m stipulated in COPs are similar to the findings in this project. In particular, the fracture mapping work provided some compelling results. It is proposed that the following lead-lag distance ranges be adopted in the South African gold mining industry:For reef types not addressed in this project, a match for similarities in geotechnical settings should be checked against the four reef types analysed and the appropriate range chosen. If no match is established, the ranges suggested for the combined reef should be used. Although not extensive, the seismic findings from the Kloof analysis seem to corroborate the guidelines derived from the fracture mapping work.

Reef type	Approx. depth (m)	Proposed optimal inter-panel lead-lag distances for quasi-static conditions from fracture mapping analysis (m)						
		RMR	MRMR	Q-rating	Fracture spacing	Siding-parallel fractures	Wedge analysis	Current practice
VCR	2400 to 3000					5,5–10,5		5–10
CLR	3200					9,5–16,5		10–15
Vaal	2300					4,0–9,0		5–10
	2000 to							
All reefs	2000 to 3200	15	17		<16,5	6,5–12,0		5–15

Figure : Proposed Lead-lag Guidelines

The Elfen model needs to be further calibrated for the study of fracturing around lead-lags before it can be successfully used as a forward modelling tool to determine optimal lead-lag lengths during mine design.

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

By Spottiswoode S.M., Milev A., Linzer L.M., Majiet S., CSIR, Ref: SIM 04 03 01, 2008

SUMMARY:

We analysed seismicity in two deep gold mines in terms of strain energy release during mining. Mining of the Carbon Leader Reef (CLR) and the Ventersdorp Contact Reef (VCR) took place with regularly-spaced dip pillars and backfill for regional support. More than 500 000 m2 of reef were mined over several years and induced more than 10 000 events of Local Magnitude greater than 0.0 per reef. Four approaches were taken to study the behaviour of these pillars, namely: comparison between modelling and observed seismicity; seismic source mechanisms; strong ground motion studies; and ground tilting. The best research results came from the studies of seismicity and numerical modelling and ground tilting.

It was necessary to develop automatic polygon (auto-poly) definitions to clearly separate areas of localised

mining and seismicity. Here follow six conclusions based on work done in this project that are of direct rock engineering value:

1. ERR is, on average, a direct measure of likely seismicity. Seismicity per area mined is proportional to Energy release Rate (ERR) when seismicity is measured through summation of a range of parameters such as number of events with Magnitude (M) > 0, Seismic Moment and Apparent Volume.
2. There was no sign of run-away pillar failure. The seismicity rate per strain energy release appeared, if anything, to decrease at the highest levels of ERR.
3. The effective pillar strengths are at least 300 MPa. The modelling only showed an accelerated rate of strain energy release at high values of ERR if the pillar strengths were assumed to be less than 300 MPa.
4. The benefit of backfill in reducing strain energy release is greater as pillar stresses approach their ultimate strengths. In other words, backfill is most useful for regional support when regional rock support is at its most stressed.
5. Seismicity was controlled far more by mining than by the presence of geological features (faults and dykes). This was probably the result of good mining practice such as using bracket pillars within final pillars and mining through features at a good angle of incidence. Geological structures appeared to increase seismicity measured as Apparent Volume by about 9% on the VCR and 37% on the CLR. There was a greater influence for radiated energy. The greater influence on the CLR mining was at least partly attributable to the faults and dykes interrupting aseismic slip on bedding planes.
6. Mining from both sides of a pillar at the same time does not generate any more seismicity than if only one side is mined at any time. Modelling work suggests that reducing the amount of mining on each side of a pillar is likely to be less hazardous than mining each side in turn. This work places some doubt on the value of the so-called 70 m rule as a rockburst-control measure.

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

1. Stopes and / or a yielding facture zone ahead of the face created a soft loading system that influenced the character of seismic events at high levels of ERR. Apparent stress was found to be lower for events associated with higher ERR mining where the face stresses should be higher. This contradiction is resolved when loading stiffness was calculated and it was found that P/S moment ratios, apparent stress and seismic stiffness all increased with increasing loading system stiffness. These effects were more strongly developed for mining in the Carbon Leader. The amount of seismicity per area mined increases with increasing span but, as the loading system softens, strong ground motions and therefore the damaging potential of each event decreases.
2. A new mechanism is needed to explain time-dependent failure of brittle rock. Several types of observation are incompatible with current rock mechanics theory, such as:

- The absence of measurable changes in tilt even at a level of 0.1 microradians following seismic events that caused co-seismic tilt changes up to 100 microradians as measured 90 m below reef;
- Tilt signals similar to those reported for “slow earthquakes”;
- More seismic events occurring far from active mining than was predicted from modelling; and
- A lower fall-off rate of aftershock density with distance, D, than expected, namely ~D-1.3 compared to the current theory of D-3. (Data analysis part of SIM-05-03-02: Managing Rockburst Risk.)

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

25. Minimising the increasing seismic risk in the platinum sector

By Essrich F., Hanekom J.W.L., Stankiewicz T.B.A., Rangasamy T., SiM Mining Consultants (Pty) Ltd., Ref: SIM050302, 2011

SUMMARY:

The seismic and rockburst data made available by the Council for Geoscience and collaborating mines in the Rustenberg area show fluctuations over recent years, but not necessarily an alarming trend for the near future. Some shafts experienced general increases in events M>1, the magnitude range widely seen as responsible for most rockbursts; other shafts experienced reductions. The event rate normalized by production is 2 to 3 events M>0 and 0.2 to 0.4 events M>1 per 1 000m2 mined, roughly five to ten times lower than in an ultra-deep West Wits gold mine. The most rockburst prone shaft in the study area experiences rockburst frequencies per area mined similar to those in the West Wits mine, but rockburst injury rates are significantly lower.

On the investigated platinum mines, rockbursts are almost exclusively associated with in-stope pillars: Three quarters are on-reef pillar failures, a further 20% are pillar foundation failures. Geological discontinuities (minor faults and dykes, joint sets, potholes and domes) are found to be associated with roughly one fifth of investigated rockbursts. Taking all seismic events into account, not only damaging events, 70-90% have a burst-type failure mechanism with some variation depending on the shaft. The magnitude range of most damaging events lies roughly between 0.5 and 3, but can stretch to below -1.5 as is the case with gold mines.

The typical Western BVC rockburst results in on-reef damage either close to the face or in the back area. On at least one of the shafts damage to tunnels, crosscuts and raises is more frequent than to the face area. Rock ejection and shake-down are the most frequent damage mechanisms, while bulging, buckling, lifting and dynamic closure all play minor roles. Depending on the shaft, 5 to 20% of reported rockbursts result in injuries compared to 75% on an ultra-deep West Wits gold mine.

In general, identified as leading indicators for rockbursts were mixed mining directions in a panel set, an extraction ratio >80% within a 200x200m area of a pillar, the formation of abutments and poor face shape control, and the placing of tunnels close to mining abutments.

The capacity to quantify seismic hazard and rockburst risk on the collaborating mines offers opportunities for improvements. Even though the quality standards of seismic data are generally high, routine and ad-hoc data analysis and reporting are not subject to rigorous performance assessments. Daily polygon assessments perform better in predicting safe periods (green ratings) than unsafe periods (orange or red).



26. Integration of the South African National Seismograph Network and Database with Mining Networks as per the recommendation in chapter 1 of the Presidential Mine Health and Safety Audit 2008

By Grobbelaar M.R.G., Council for Geoscience, Ref: SIM 100201, 2013

SUMMARY:

The work conducted within this project aims to address the recommendation in the Presidential Mine Health and Safety Audit 2008.

During this project, the Klerksdorp-Orkney-Stillfontein-Hartbeesfontein (KOSH) gold mining region was identified as the pilot study area. Through this pilot phase, the Council for Geoscience (CGS) has bridged the gap in the monitoring of mine seismicity within the KOSH region between the underground mining networks and the South African National Seismograph Network (SANSN) using the newly installed surface stations.

These stations send data in real time to the central data centre in Pretoria where it is automatically processed and the events located. The information can then be sent to the relevant authorities. The automatic locations remain preliminary until they have been checked by a seismological analyst and then they are released as the final locations.

These automatic locations are performed by analytical software which is compatible with the new equipment and which can alert relevant personnel when there is an event.

In order to supplement the rich supply of data from the 25 new stations (Figure 1 and 2 below), this project aims to also integrate the wealth of information concerning mining-related seismic events which resides with each of the mines. This information will be of great value for research into rockbursts, especially if it is integrated with data from each network and the SANSN.

By working intimately with the database, and through consultation with the mines, the DMR, and the MHSC, and during a recent workshop with other stakeholders, the CGS has gained a better knowledge of what type of research is still outstanding and the areas where research is urgently required in order to address the safety in the mines.



Figure 1: Map of the 25 seismological stations installed in and around the KOSH region.

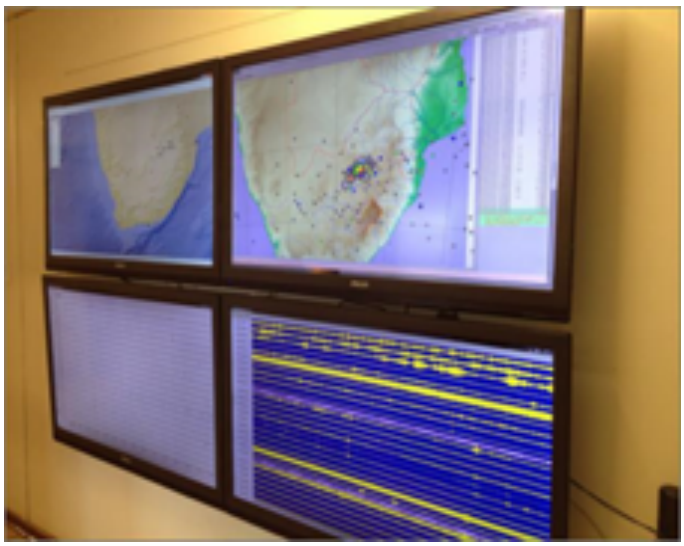


Figure 2: The display used to monitor the SOH of the stations and the data flow to the CGS.

27. Fluid-Induced Seismicity in the Central Basin Area: Ground Motion Prediction and the Development of an Early Warning System for Risk Reduction

By Cichowicz A., van Aswegen G., Birch D., Diop S., Kijko A., Labuschagne L., Manzunzu B., Midzi V., Smit A., Zulu B., Council for Geoscience, Ref: SIM 110201, 2013

SUMMARY:

Three and a half years after pumping out water in the ERPM mine was stopped, the evolution of seismicity has indicated that the Central Rand Basin area is not yet in equilibrium. Because the seismicity is located directly beneath the densely populated area of Johannesburg, a full understanding of its mechanism is required. Seismic data clearly shows that the level of seismicity in the rock mass influenced by water ingress tends to be much larger than in the surrounding rock mass.

3D modelling was used to identify areas of instability within the abandoned mines. This led to a description of the spatial distribution of seismic activity with a possible explanation and an estimation of maximum shear displacement along a geological feature. There was good agreement between the spatial distribution of seismicity and the positions of potentially unstable features, which suggests two possibilities: (1) Seismicity is being controlled by stresses induced by the shape and orientation of the mined-out reef on intersecting geology rather than tectonic stresses and (2) The shallow crust can be described by a homogenous material and uniform stress model.

Probabilistic seismic hazard analysis was conducted for the area. A hazard map was produced for spectral acceleration at T = 0.1s and T = 1.0s, both at return periods of 475 years and 979 years. Maximum expected peak ground acceleration values are 0.16 g and 0.2 g for the return periods of 475 years and 979 years respectively. The acceleration values are almost the same at OR Tambo and Johannesburg, whilst values at Soweto are much less. The expected damages for 12 of the most characteristic building classes found in the Johannesburg CBD are included. The assessment of the maximum regional seismic event magnitude is also provided.

One of the objectives was to determine the subsoil and groundwater conditions at the site in order to delineate the most favourable locality for a first level microzonation map in support of the strong ground motion amplification prediction and the development of an early warning system for risk reduction. The results have indicated that the most favourable topographic situation for thick soil deposits in the study area occupy the Johannesburg graben valley.

A catalogue of almost 1000 earthquakes was created and the spectral parameters of the events were estimated. A new relation between ML and Mw was obtained. An increase in static stress drop with increasing seismic moment is observed. The attenuation of the P- and S-waves in the Central Rand Basin is estimated using the coda normalization method. The frequency dependence of parameters for the P- and S-waves is calculated for a set geometrical spreading constant. The dependence between seismic moment and radiated energy in different studies shows a similar pattern to the one observed here. However, there are some significant differences; static stress drop for the Central Rand Basin area is one order of magnitude larger than in the case of observed seismicity induced by a water reservoir in China. The amplitudes of ground motion are controlled by the amount of static stress drop during a seismic event; therefore, areas where large stress drop is observed are areas where strong ground motion is expected. The spatial variation of seismicity in the area was analysed. The number of events increased significantly within the last year of monitoring. The distribution of seismicity shows a reduction in the level of seismicity in the western part of the Central Rand Basin during 2012. The strongest events in 2012 were concentrated in the eastern part the Central Rand Basin. However, the earthquakes with the largest stress drops are observed in the eastern and central parts of the Central Rand Basin.

The variation of cumulative seismic moment and radiated seismic energy as functions of time indicates that a constant deformation rate model is not suitable for the Central Rand Basin area. However, evolution with time of the cumulative parameters and estimated strain rate could be used as indicators of future seismic activity.

28. Integration of the South African National Seismograph Network and Database with Mining Networks as per the recommendation in chapter 1 of the Presidential Mine Health and Safety Audit 2008.Phase 2

By Grobbelaar M., Cichowicz A., Council for Geoscience, Ref: SIM 14-02-04, 2015

SUMMARY:

This project stemmed from the recommendations in Chapter 1 of the Presidential Mine Health and Safety Audit where it was recommended that the mining data and the national data be integrated.

AGA network corner frequency is up to 200Hz, while the KOSH maximum corner frequency is around 50Hz. Both differences are caused by instrument limitations. The AGA network distorts the corner frequency below 10 Hz and the KOSH network distorts the corner frequency above 50 Hz.

The objectives of this study were to:

I. Maintain the existing extended seismograph network within the Klerksdorp-Orkney-Stilfontein-Hartbeesfontein (KOSH) region, and
II. Calibrate/compare the data from the network in KOSH to the data from the mines within the region, and
III. Present the results to the stakeholders for inputs.

Methodology

The process used to undertake the project were:

- Regular maintenance and state of health checks for the network and procuring of spares for rapid deployment.

Results of the study

The data from the KOSH network was compared to that from AngloGold Ashanti (AGA). As can be observed from Figure 1 the relationship between the Mw magnitudes from the different networks is very close to one, therefore in the practical application both the Mw magnitudes can be used interchangeably.

The scalar seismic moment obtained by the KOSH network has a tendency to be larger than that obtained from the AGA network, but it is to influence the estimation of the Mw magnitude significantly.

There are significant differences between the corner frequencies obtained by the two networks. The KOSH network estimates the corner frequencies within the range varying from 6 to almost 30Hz, while the AGA network has an almost fixed value of 10 Hz.

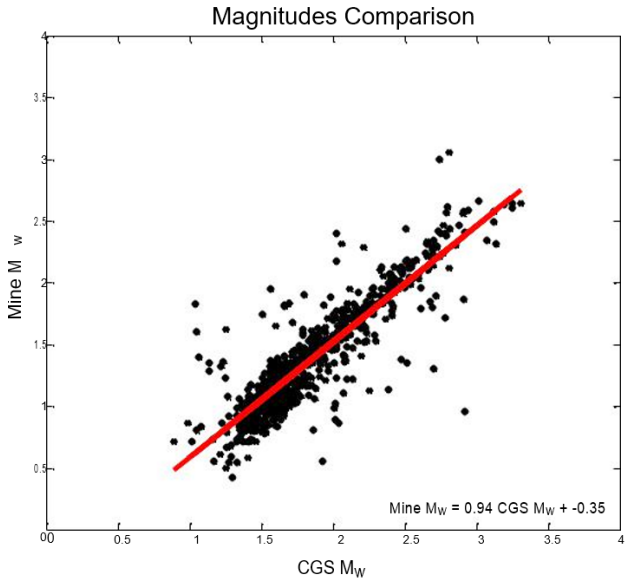


Figure 1: Relationship between Mw magnitude estimation obtained by the two networks in the magnitude range from 0.8 to 3.5.

CONCLUSION:

The KOSH network is fully operational and has been well received by the stakeholders. The importance of the network was highlighted after the magnitude 5.5 earthquake on 5 August 2014, where the data was available and was merged with other mining data in order to better understand the mechanisms of the earthquake.

When comparing the data with the mining networks, it was very rewarding to notice the similarities between the two sets of data. The stakeholders were very positive about the project and encouraged similar studies in other mining regions, such as Platinum and Free State.

Recommendation:

Phase 2 was successfully completed and it is highly recommended that the project expands to other mining regions, such as the Platinum and Free State mining areas and include further research into the observed differences between the networks.



29. Training extension for the adoption of a risk-based approach to support design in Bushveld underground mines

By Walls J., SRK Consulting (South Africa) (Pty) Ltd (SRK), Ref: SIM 14 02 01, 2016

SUMMARY:

The SIM140201 project was commissioned by the MHSC in 2014 as an extension to the SIM120201 project, "Technology Transfer for 'A Risk Based Approach to Enhancing Support Design in Bushveld Underground Mines'", which was completed in 2013. An opportunity was identified as an outcome of SIM120201 to continue and expand the skills development of rock engineering practitioners (REP's) within the industry. The project commenced in August 2014.

The training programme intended to equip REP's with the knowledge and skills to carry out risk-based support designs for fall of ground risk mitigation using the technology on their operations. The programme that was applied for the SIM120201 project was restructured for SIM140201 to promote improved understanding and execution of the method by the participating practitioners. Five training courses consisting of six training days spread over a period of five weeks for each course were carried out from January 2015 through to May 2015. Out of 100 REP's provided for by the training programme, a total of 74 combined rock engineering (RE) and geology practitioners attended the courses.

To date, a total of 17 project reports have been received from a combination of 30 RE and geology practitioners. 42 course participants have not yet submitted a training report.

Overall results from this project, SIM140201, have shown a marked improvement from the outcomes achieved for the SIM120201 project. This was achieved in spite of the full complement of training report submissions not being received for this project (SIM140201).

A formal assessment of competency was not specifically incorporated into the outcomes for the project; however, each report submission was reviewed by the project manager with feedback provided to the author(s). The risk evaluation process comprising each stage of the analysis was successfully carried out by the overwhelming majority of REP's, most of whom demonstrated sufficiently competent understanding of the principles, processing and interpretation to continue developing and applying the risk-based support design approach without direct assistance for the respective operations.

A small proportion of REP's and geology participants will require further assistance to ensure a suitable level of independent, competent ability to apply the technology within the industry.

A number of software application and hardware challenges were identified for further research and development. Amongst these the most significant issues include developing the JBlock application to troubleshoot limitations pertaining to the interpretation of input parameters and developing the RiskEval application further to suitably address the evaluation of risk for bord and pillar operations. More general running issues include compatibility with Windows 8 and/or newer versions as well as the Apple platform.

Recommendations

Going forward, the industry has been provided with a number of REP's with the necessary skills to apply risk-based support design. However, the application of technical solutions to operational challenges remains a challenging objective within the operational environment with the outcome that several REP's still require further practice and guidance to correctly and successfully apply the technology. A strategic initiative is recommended to focus the application of technical solutions to operational challenges from within the operational environment, i.e. by on-mine REP's. This may include independent as well as on-mine input. As such there remains an opportunity to continue development of the applied technical skills of on-mine REP's.

Independently, further development of the technology is required in particular for JBlock and RiskEval to improve compatibility, relevance and usability of the interface. Research initiatives are recommended to address these two applications respectively.

30. Fluid-Induced Seismicity in the Central Basin Area: Ground Motion Prediction and the Development of an Early Warning System for Risk Reduction

By Cichowicz A., Birch D., Kgaswane E., Kijko A., Mangongolo A., Ngobeni D., Midzi V., Rathod G.W. Smit A., Council for Geoscience, Ref: SIM 14-02-05, 2016

SUMMARY:

The last active mine in the Central Rand Basin (CRB) area (50 km x 15 km), ceased operations in 2008. This resulted in the closure of the pumping stations, which previously maintained the underground water level in the mining voids. As a direct consequence of the water being allowed to flood the mine voids, seismic activity has increased directly beneath the populated area of Johannesburg.

Methodology

The proposed project addresses the problem of the risks posed by fluid-Induced seismicity in the greater Johannesburg area by focusing on accurate quantification of the most important factors that contribute to the estimation of this hazard.

Results of the study

The level of seismicity in the CRB is not showing signs of decreasing with time since the number of strong events with moment (Mw) magnitudes above 1.4 is still as high as it was when monitoring began over five years ago (see Figure 1).

Most events are located within historical mine boundaries. The seismicity pattern shows a strong relationship between the presence of the mining void and high levels of seismicity; no seismicity migration patterns were observed outside the areas of old mining. 3D modeling showed that residual mining induced stresses are being exploited by the water and also confirmed that the mined out reef is controlling seismicity. Strong lineaments in the seismicity pattern were observed that supported the result of the modeling.

Probabilistic seismic hazard analysis showed a maximum expected magnitude of 5.13 and a maximum value of peak ground acceleration of 0.15g. The highest levels of acceleration were observed in the eastern parts of the CRB. This was expected, since this area produces most of the seismic activity. Low rise buildings with unreinforced masonry, load-bearing walls are the most vulnerable to damage with an expected damage percentage ranging from 5 to 7%. The estimated expected damage is small when compared to the damage experienced in more seismically active areas.

Over the last five years, the total cumulative seismic moment released in the CRB was 9.0x1014 Nm. This is equivalent to a single earthquake of magnitude Mw 3.9, which is significantly less than the largest earthquake experienced during mining times.

The temporal evolution of the inter-event time confirms that the fluid-induced seismicity follows a clustering pattern and is not random.

Static stress drop heavily influences ground motion characteristics, which, in urban areas, affects the risk assessment. The observed static stress drop varied from 0.05MPa to 10MPa. It was found that large static stress drops could be associated with both small and large events.

CONCLUSION:

The nature of seismic events triggered by mine flooding, as well as the damage potential of surface ground motions associated with such events, needs to be monitored and investigated.

Recommendation

Temporal and spatial changes in the rock mass of the Central Rand Basin imposed by fluid related seismicity should be evaluated continuously for the next few years. Characterization of the dynamic processes in the rock mass is mainly achieved by estimating the values of the spectral parameters of the seismic source. The methodology described in the report should be implemented in all areas where fluid-induced seismicity is observed and a seismograph network on the surface is available.

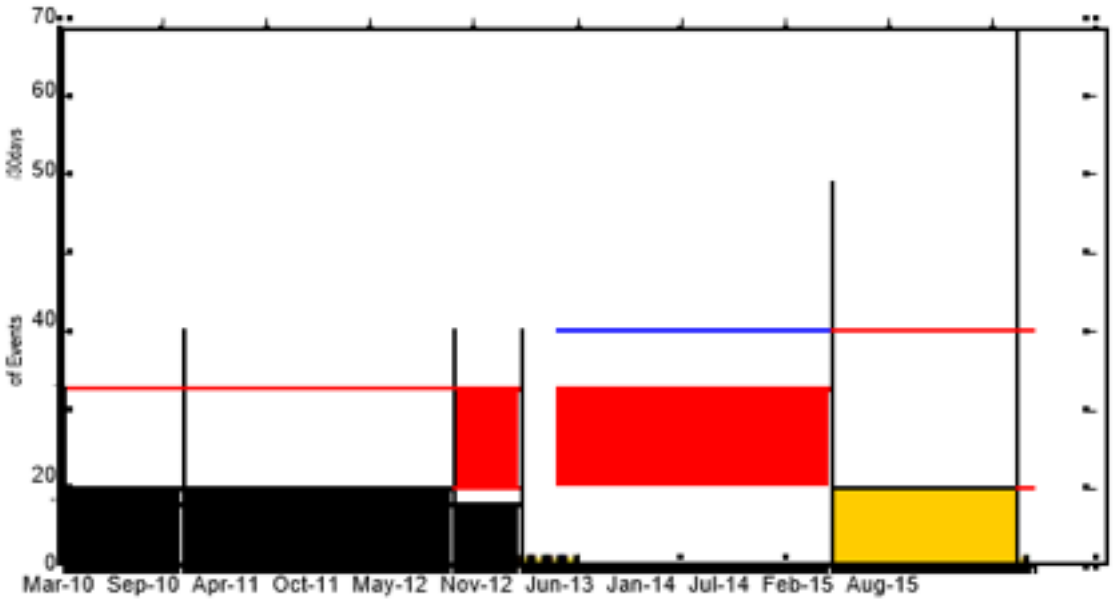


Figure 1: Bar chart of the seismicity rate over a period of 62 months for events with magnitudes larger than 1.4. The seismicity rate does not show signs of decreasing when examining the numbers of events recorded over the last five years. Instead, a variation in the level of seismicity was observed.



31. Technology transfer on minimising seismic risk in the platinum mines

By Essrich F., Hanekom J.W.L., Bouwer J., SiM Mining Consultants (Pty) Ltd, Ref: SIM 14-03-01, 2016

SUMMARY:

Background of the Study

This study stemmed from an earlier project that drew attention to the increasing rockburst risk experienced by PGM mines in the Bushveld Complex. SIM 10-03-01 established common causes and factors contributing to the seismic rockmass response to mining in the BC. It found that panel support in the form of chain pillars are a common source of seismicity and that the rock types associated with the MER reef horizon can be prone to brittle failure.

Another potential source of seismic energy release are irregularly shaped remnants and pillars, often created around potholes or where ground conditions become too difficult to apply standard mining configurations.

The project found indications of poor pillar cutting and a possible lack of appreciation of seismicity related issues among mine personnel and production management.

In order to better manage exposure to rockburst risk, SIM 10-03-01 also recommended to improve the collection of pertinent data on rock types and their properties, rock mass conditions, geological features and seismic response.

Objectives of the Study

The objectives of this project were to enable training of mine personnel in seismic risk mitigation and to improve the practice of data and information collection relevant to this specific risk.

Methodology

The processes used to undertake the study consisted of the development of separate sets of training materials for production personnel (shift boss and denominations above) and for rock engineering personnel (strata control officer and denominations above).

A separate set of outputs created a seismic system audit protocol and applied this protocol to nine participating mines in the Bushveld Complex.

The results of audits, including relevant recommendations for improved data analysis, were shared amongst the operators for the benefit of all participants.

This project experienced a high level of buy-in from the mining industry in the BC throughout all of its milestone activities and through its 2-year duration. The project was completed within budget and three months ahead to time.

Results of the study

The training materials for production personnel consist of 14 animated modules with a length of five to seven minutes each (see Figure 1). The module set includes a user interface to select relevant modules, loop a selected set or to or play once-off as a sequence; it also includes an introductory and a summary module.

The animated modules were rolled out to mine training centres during train-the-trainer workshops in November 2015 and distributed by means of USB memory sticks.

The training materials for rock engineering personnel consist of a new text-based manual (43 pages, see Figure 1) on pertinent issues related to seismicity in the BC; a slide show (93 slides) with linked-in animations, research reports and text books in PDF format; and a manual published by SiM Mining Consultants covering all specific outcomes listed in the Rock Engineering Certificate syllabus that relate to seismicity (130 pages).



Figure 1: Module on gullies and sidings (left), manual cover.

Three hundred sets of the manuals, including CDs with manual, slide show and referenced documents, were distributed to mines in June 2016.

The seismic system audit initiative was joined by nine mines in the Western BC. The average compliance level with the score card was 88% (range: 60-100%). Audit results suggest that ensuring seismic data quality by means of standards is common, but that the ability to replace a faulty sensor within three months is still a challenge experienced by the mines.

CONCLUSION:

The results of audits, including relevant recommendations for improved data analysis, were shared amongst the operators for the benefit of all participants.

Recommendation

The learning materials are well suited for incorporation into strata control training at mine training centres, especially for leave returnees and newly hired staff.



NOTES:

“Every Mine Worker returning from work unharmed everyday. Striving for Zero Harm.”



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