

Compendium of Research Project Summaries on Rockfalls - 1994 to 2017

MHSC
Mine Health and Safety Council

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MHSC

Mine Health and Safety Council

CEO's Foreword

The 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 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, 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).

In order to accelerate the attainment of Zero Harm in the sector, through the implementation of risk-specific activities, the 2014 OHS Summit Milestones were collectively developed by the tripartite stakeholders and agreed upon by the SAMI principals in November 2014.

The milestones provides a roadmap to assist the sector to eliminate fatalities, injuries, occupational lung diseases and noise induced hearing loss 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, serious injuries shall be reduced by at least 20% year on year by December 2016, whilst lost time injuries shall be reduced by 20% per year from January 2017 onwards.

Falls of ground (FOG) are still the major cause of mine fatalities and injuries, ahead of Machinery and other causes. Improper and insufficient barring is the major contributor to fall of ground accidents throughout the mining industry.

This publication highlights research work already completed by the MHSC to address the challenges of FOG in the mining industry.

The MHSC continues to conduct research in order to develop Knowledge and Technology which will assist the South African Mining Industry to eliminate FOG incidents, thus Ensuring every mine worker returns from work unharmed every day.

Thabo Dube
Chief Executive Officer
Mine Health and Safety Council

December 2017

1. Multiseam design procedures

By Hill R.W., CSIR Mining Technology, Ref: COL 026, 1994

SUMMARY:

This report summarises world-wide and South African multiseam design guidelines. Design suggestions regarding multiseam mining for South African conditions are given.

Many collieries in South Africa contain more than one seam which is economical to mine. If the seams are in close vertical proximity, stress concentrations may arise during mining, which can result in difficult mining conditions when subsequent seams are extracted and which may impose restrictions on layouts. Marketing constraints and coal quality have meant that, from a strata control point of view, the ideal sequence of mining in a descending order has been difficult to achieve.

In order to maximise reserves, pillar extraction will become more common in the Witbank Coalfield. The No. 2 Seam has been extensively mined by bord and pillar methods. The No. 4 Seam has been mined on a small scale with pillar superimposition being carried out depending on the parting distance. To prevent destruction of the No. 5 Seam, pillar extraction has not generally been carried out.

In Natal, high grade anthracite and coking coal are mined using total extraction methods. Coal reserves are usually limited and it is quite common to mine in areas that have been previously under or over mined.

Multiseam mining has been carried out in many South African collieries in the past and continues to the present day. Multiseam mining is likely to continue since the current trend is for brownfield rather than greenfield expansion. Since 1976, guidelines were formulated from field investigations and finite element modelling. Although the guidelines have been used successfully, they only apply to bord and pillar mining and do not consider total extraction methods.

Table 1 shows the potential safety hazards associated with different multiseam mining sequences and extraction methods.

METHOD OF MINING		
Upper (U) Seam	Lower (L) Seam	SAFETY HAZARD
B&P	B&P	Spalling on pillars and parting collapse if P is thin and no superimposition.
PE (2)	B&P	Roof falls in L seam, parting collapse if P is thin.
B&P	HE (1)	Tensile zones and spalling in U seam when mining over solid/goaf boundaries. Floor collapse over incomplete goafs. High safety risk if P/h ratio is low.
Remnant Pillar (1)	B&P (2)	Intersection collapse in L seam when mining under remnant from HE panels.
PE (1)	PE (1)	Simultaneous mining in both seams roof falls in working area of L seam.
HE (1)	HE (2)	Preferred method of mining safety hazard minimal, except where remnant pillars exist and water in U seam.

Table 1: Safety Hazards in Multiseam Mining

Key:

B&P	Bord and Pillar
PE	Pillar extraction
HE	High extraction pillar extraction or longwall
P	Parting
(1)	Seam mined first
(2)	Seam mined second
U	Upper Seam
L	Lower Seam
H	Seam Height

CONCLUSION:

The safety hazards associated with multiseam mining have been recognised and will depend on the mining sequence and method. Multiseam mining will continue in South Africa for the foreseeable future in order to maximise reserves at collieries.

The Salamon and Oravecz guidelines have proved successful for bord and pillar mining although they can be conservative. They are suitable for average South African colliery conditions. The limiting distance concept is dependent on stress as well as pillar size. It is recommended that, for borderline cases i.e. where the parting approximates 0,75 the pillar centre distance, numerical modelling be carried out to determine if pillar superimposition is necessary. For layouts beyond average conditions (> 100 m depth) numerical modelling should always be used.

2. Analysis of rockburst and rockfall accidents in relation to class of stope support, regional support, energy of seismic events, and mine layout

By Dr Cichowicz A., Wits University, Ref: GAP001, 1994

SUMMARY:

GAP001 was a small project that performed a number of statistical analyses on records of rockbursts and rock falls, with no distinction made between the two. Data from South African mines in the Carletonville area and from the South African Reportable Accidents Statistical System Associated with Rockbursts and Rockfalls were used. A number of different analyses were performed.

Multivariate analysis was carried out on production in volume per month on a longwall, production delay due to Falls of Ground (FOGs) and accidents. Correlations and partial correlations are shown for different situations. Only production figures, such as active face length and volume of rock mined, correlate well with one another.

The number (N) of FOG expressed as a delay in production (d) could be described by the following relationship :

log N(d) = A - Bd,

Where A and B are constants.

This is illustrated in Figure 1.The number of FOGs and the fatal accidents and injuries as a function of the associated production delay was tabled. The 60% of FOGs which caused production delays of two days or less were responsible for 94% of injuries and fatalities.

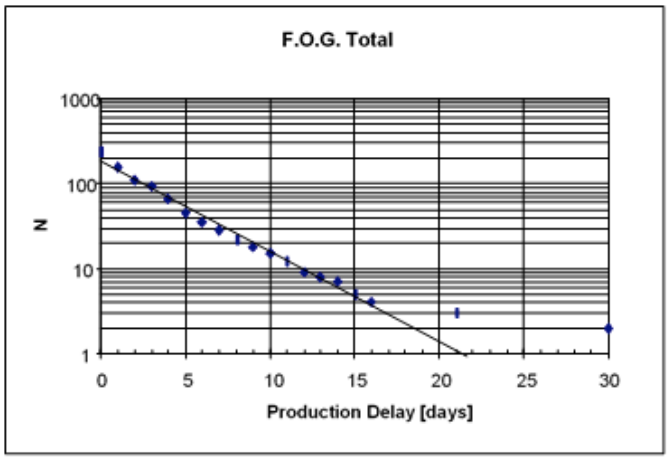


Figure 1. Plot of number (N) of FOGs as a function of production delay

The number of FOGs (N) was plotted as a function of the linear extent of damage (L) (Figure 2) and studied according to the relationship appropriate for fractal geometry :

Log N(L) = A - D log L
where A and D are constants and fractal theory has D=0 for points, D=1 for lines, D=2 for planes and D=3 for volumes.

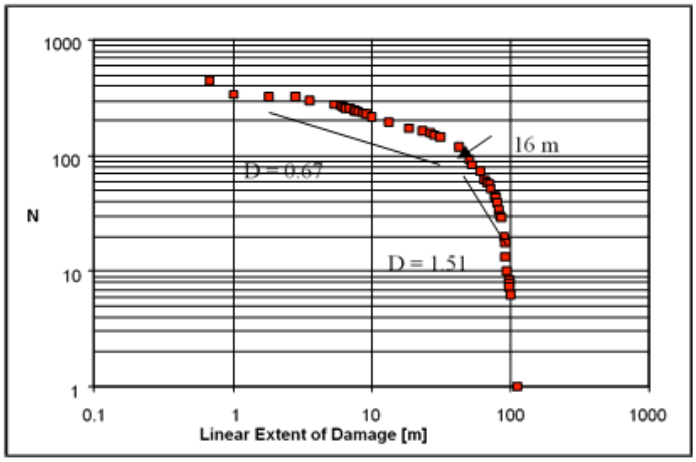


Figure 2. Plot of cumulative number of FOGs as a function of the linear extent of FOG.

The periodicity of reportable accidents was shown as a spectral plot.

CONCLUSION:

There was no clear correlation between volume mined in a month on longwalls and either production delay or the number of reportable accidents. (Note : this suggests that faster face advance is no more hazardous per volume mined. However, this statistic might be biased by the production delay resulting from rockbursts. There is a clear power-law relationship between the numbers of FOGs and the number of days delay per FOG.

3. A reassessment of coal pillar design procedures

By Madden B.J., Canbulat I., Jack B.W., Prohaska G.D., CSIR Mining Technology, Ref: COL 021A, 1995

SUMMARY:

The project investigated several factors influencing coal pillar strength.

These studies included:-

- A case study of long term pillar determination
- Laboratory strength tests of coal blocks, and a comparison with Salamon's and Australian field data
- Numerical modelling
- Pillar classification rating
- Collection of case histories of pillar collapse
- In situ field trials
- Pillar foundation failure

The laboratory results were encouraging by showing a tight range in blocks from the No. 2 Seam, Witbank Colliery, where eight of the coal blocks were extracted.

A comparison between Australian and South African collapsed pillar cases was conducted by the University of New South Wales, Australia. Their conclusions are of significance and are summarised in this project report.

The characterisation of pillars by a pillar classification system was undertaken. Over 300 panels were visited and the pillars rated. Collection of information regarding collapsed pillar cases was undertaken. Three in situ field trials were conducted, but unfortunately, the results were not as productive as anticipated. The influence of the surrounding strata on pillar strength was recognized and examined. Stable workings require all elements in the system to be in a stable condition. Conversely, failure of one element can result in a system failure.

Salamon's design method is based on empirical data. In the collection of that data, pillar collapses due to foundation failure were excluded, as were cases where weak roof may have induced failure. Similarly, Madden's reassessment of coal pillar design conducted in the mid to late 1980's also excluded these types of collapsed cases. The result is that the Salamon design formula works well where the coal pillar is the weakest element in the system. However, of the 81 known collapsed pillar cases reported on up to 1988, only 44 were included in previous analysis. Put slightly differently only 54 per cent of known pillar collapses have been adequately taken into account in current pillar design practices!

CONCLUSION:

Individual seam strength

The classification system indicated that there was a variation in effective strength within collieries and across the coalfields.

Pillar Foundation Failure

Given the potential seriousness of foundation failure on pillar stability, a collection of index tests on suspect strata should be initiated. While this mode of failure may only occur some time after mining, the failure has the potential to result in the loss of access ways as experienced by the trapping of miners at Emaswati

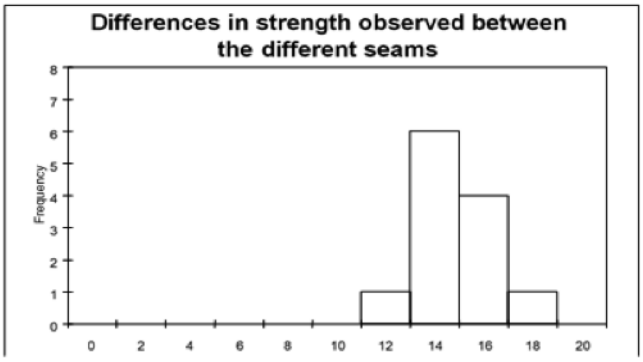


Figure 1: Differences in strength observed between the different seams

4. Develop guidelines for the design of pillar systems for shallow and intermediate depth, tabular, hard rock mines and provide a methodology, for assessing hangingwall stability and support requirements for the panels between pillars

By Haile A.T., Jager A.J., CSIR Mining Technology, Ref: OTH 002, 1995

SUMMARY:

The initial objective of the project was to provide guidelines as to a methodology for designing pillar systems for stope support in shallow, hard rock, tabular mines. During the course of the project an additional objective, 'to provide a methodology for assessing hangingwall stability and support requirements for the panels between pillars', was added. The consequence was that neither objective could be completed but resulted in a continuation project, GAP 034, dealing with the interrelated problems. The project summarized here, therefore, deals with the initial work required to fulfil the objectives.

This project, being the first to look into the issues outlined above, needed to collate much information on the then current practices and mining conditions and relied to a large extent on data compiled in project GAP 027. It was also necessary to identify gaps in knowledge that required further research. The premise on which much of the work was based, was that, to improve design procedures for pillars and support, a better fundamental understanding of factors which influence pillar constitutive behaviour and rock mass behaviour around stopes was required. To this end a number of basic issues were addressed, however not all of the 15 areas of uncertainty identified by the survey of platinum, chrome, manganese, asbestos and gold mines could be investigated.

The main areas of work included the following:- a major laboratory testing programme investigating the influence of w:h, size, rock type and contact conditions on model pillar strength and behaviour; evaluation of the applicability of the codes FLAC, FLAC3D and UDEC to the modelling of the non-linear and post failure behaviour of pillars; the development of ways to improve the calculation of Average Pillar Stress for small irregular pillars, in the codes MINSIM and BEPIL; the development of two back analysis techniques for estimating the in situ strength of pillars; a detailed underground pillar monitoring programme and associated laboratory test programme consisting of standard, various width to height tests and punch tests.

The absolute values of pillar load or stress and pillar strength determine whether a pillar will fail or not. A number of factors influence the calculation of each of these parameters. These factors in turn have a variability or range in values. To account for the uncertainty brought about by this variability, a factor of safety is usually applied to empirically derived design methods. Stochastic methods can be applied to reduce this uncertainty. A preliminary investigation into the use of probability theory in pillar design was thus undertaken. Data on the variability of a limited number of parameters was compiled; a methodology to combine the effects of the variables was produced; and test cases were run. The initial results did not correlate well with observed conditions of pillars but by arbitrarily modifying the influence of one of the parameters, a reasonable comparison was obtained.

As far as panel support requirements are concerned, an initial survey of mines in the Bushveld Complex was carried out to ascertain the variability in hangingwall conditions and to identify typical modes of failure. Six characteristic collapse mechanisms were observed. In addition two stopes were comprehensively monitored. This involved detailed geotechnical mapping, many closure ride measurements, footwall and hangingwall extensometer measurements, borehole camera observations of fractures and detailed recording of falls of ground. The critical span between pillars was determined to be 30 m at one of the sites having a poor rock mass rating but which was strongly supported on grout packs, and where larger than planned pillars were left.

CONCLUSION:

The problems associated with the design of pillar systems for shallow, hard rock mines have been identified.

Strength increase relationships with increasing width to height ratios have been derived from laboratory testing of a suite of rock types from the Merensky Reef horizon.

5. Strata control in tunnels and an evaluation of support units and systems currently used with a view to improving the effectiveness of support, stability and safety of tunnels

By Haile A.T., Jager A.J. Wojno L., CSIR Mining Technology, Ref: GAP026, 1995

SUMMARY:

The GAP026 project represented the initial 3 year period of the tunnel support design project, between 1993 and 1995, which is being finalised under the GAP335 project. The initial project proposal was to examine problems associated with tunnels in the Bushveld Complex. This was subsequently expanded to include all gold and platinum mines and also to conduct a study of the shear characteristics of typical rock bolt types as used in the industry.

Initial investigations into problems associated with tunnels in the Bushveld Complex and also the gold fields indicated the lack of a rational design process, particularly for highly fractured or jointed rock mass conditions, and under dynamic loading. This led to the development of a conceptual design methodology for tunnel support systems based on an improved understanding of the interaction between the rock bolts and the rock mass and a more engineering approach to the design process. This formed the basis of the future research strategy for the continuation of this project under GAP335.

A survey of accident data indicated that approximately 18 % of all rock related fatalities are associated with off reef excavations. With regard to tunnels the distribution of fatalities is shown in the Figure below.

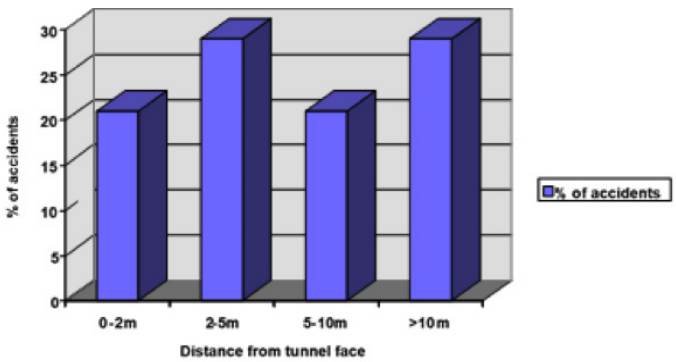


Figure 1:Distribution of tunnel accidents from face

Analysis of Figure 1 indicated that, of the 71 % of accidents that occur within 10 m of the tunnel face, approximately 70 % of the accidents are rock fall related. Of the 29 % of accidents that occur greater than 10 m from the tunnel face, approximately 70 % are associated with rockbursts. Analysis of this data may be indicative of the need for improved areal coverage support systems, primarily capable of preventing rock falls, in the immediate face area. Greater than 10m back from the face, full areal coverage support systems may be implemented but are perhaps insufficient under dynamic loading conditions.
Underground investigations and reports of support system damage during the project period indicated the failure of rock bolts under shear loading. From site visits this was found to be most prevalent in the hangingwall of tunnels sited in highly bedded rock mass structures. A laboratory investigation into the shear capacity of typical rock bolts as utilised in the gold and platinum mining industry was conducted under quasi static and dynamic loading conditions. Typical results for this laboratory evaluation are shown in Figure 2.

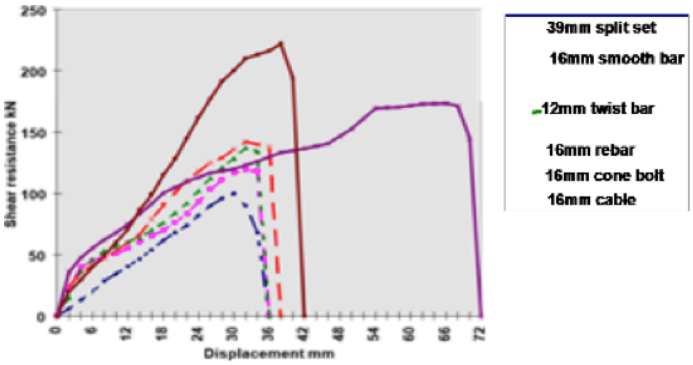


Figure 2: Quasi static shear capacity of typical industry rock bolts

An important finding of this investigation was that the improved shear capacity (load and deformation) of the support unit was dependent on its yield capability. Thus, under quasi static shear loading, rope cables and yielding tendons such as the cone bolt were indicated to have far greater shear capacity.

However, under dynamic loading conditions the shear capacity was also a function of the ability of the support unit to transfer the dynamic shear forces to the yielding mechanism. Thus, with regard to the cone bolt, its ability to absorb dynamic shear was a function of the location of the cone relative to the shear plane.

CONCLUSION:

The requirement within the mining industry for a more rational approach to support design for tunnels under highly discontinuous and rockburst conditions. Attention to ‘making safe procedures’ and the need for suitable support systems for the face area of development tunnels. Consideration of shear deformation, particularly in the hangingwall of tunnels in bedded stratigraphy, under dynamic loading conditions. The use of rope cables or yielding tendons under quasi static shear deformation. The use of rope cables under dynamic shear loading.



6. Rock mass condition, behaviour and seismicity in mines of the Bushveld Igneous Complex

By Haile A.T. Jager A.J., CSIR Mining Technology, Ref: GAP027, 1995

SUMMARY:

This project was initiated to give an overall view of the rock mass environment of the Bushveld Complex and its influence on mine design practices. A general evaluation of the geology of the Bushveld Complex indicated the variability in rock types and properties and this was considered to have important implications in excavation and support design practices. An initial data base of rock properties was established of rock types in close proximity to the reef horizons in order to assist in preliminary design as indicated in Table 1. This data base also caters for the variability in rock properties.

Rock type	Avg. UCS (MPa)	Modulus E (GPa)
Anorthosite (mottled)	196	80
Anorthosite (spotted)	180	74
Leuconorite	189	81
Melanorite	137	74
Norite	209	96
Pyroxenite (Bastard Reef)	167	104
Pyroxenite (Merensky)	133	104

Table 1. Average rock properties for typical rock types of the Bushveld Complex.

The dominant joint sets prevailing in the Bushveld Complex are generally orientated parallel to the strike of the reef and major geological features. The presence of rogue joint sets, on the reef horizon and in footwall development, may be indicative of the proximity of potholing on the reef horizon.

Analysis of all stress measurements available at the time within the Bushveld Complex was conducted. This again indicated the high degree of variability in the stress field across the mining districts.

Pillar design within the shallow and intermediate depth tabular mines of the Bushveld Complex is based on empirically derived design procedures. Although these appear to cater well for the majority of the rock mass environments there is concern that they are primarily based on coal mining experience and may not cater well for the higher degree of variability of the rock mass within the Bushveld Complex. This may result in either over design or under design, particularly when moving from an established geotechnical environment to one in which there is little previous experience. In addition, the application of a constant factor of safety in pillar design may not be appropriate in an area of high rock mass environment variability and adoption of the concept of probability of failure may be more applicable.

A significant aspect of the project was an initial investigation into seismicity of the Bushveld Complex and its comparison with that of the gold fields. This work would have important implications for the applicability of seismic parameters and procedures derived from gold mine seismic research to that of the Bushveld Complex.

General analysis of seismicity indicated the significant variation in seismicity between the mining districts which were investigated. This was considered to be primarily associated with differences in the geotechnical environment, but was also influenced by differences in the pillar and mining systems adopted in these districts.

The difference as noted at Bafokeng 12 shaft between 1991 and 1993 is a change in the pillar system design from 5x5m (elastic) pillars to 3x6m (yield / crush) pillars. This change was associated with a transfer of seismicity from pillars in the back areas of stopes to seismicity occurring on major regional pillars or potholes. Thus a change in the stiffness of the mining system resulted in a transfer of seismicity which must be considered with regard to stope and off reef excavation support design.

In addition, a more detailed seismic study was conducted for a Merensky reef area utilising yield / crush pillars. This study indicated that seismicity was associated with the active mining faces, mining abutments and the formation and initial loading of pillars. The larger events were associated with pillars in the back areas, and the majority of face events were relatively small and were initiated during the blasting period. Very limited seismicity was associated with major geological structures in the area.

7. Controlled Fault Slip: Water Injection

By Lightfoo N., Goldbach O.D., CSIR Mining Technology, Ref: GAP 030, 1995

SUMMARY:

Mining adjacent to geological features such as faults and dykes has always entailed a significant risk of rockbursts resulting from induced slip on the fault surface. Although it is not always possible to prevent a fault from slipping due to the changes in the local stress field brought about as a consequence of mining adjacent to the fault, it may be possible to control the time and location at which the slip occurs. If this is possible then it may be practical to control fault slip in such a manner as to reduce the risk to miners.

Significant evidence exists in the geotechnical literature to suggest that the injection of high pressure water onto fault surfaces can cause them to slip. The mechanism is believed to be one of effective stress in which the normal stress acting across the fault surface and opposing slip is reduced by the water pressure allowing the fault to slip. A number of experiments were carried out underground on natural fault surfaces in an attempt to induce Controlled Fault Slip. Typically the mining scenario chosen for the experiments was in a scattered mining environment where remnants left adjacent to faults were being extracted (Figure 1).

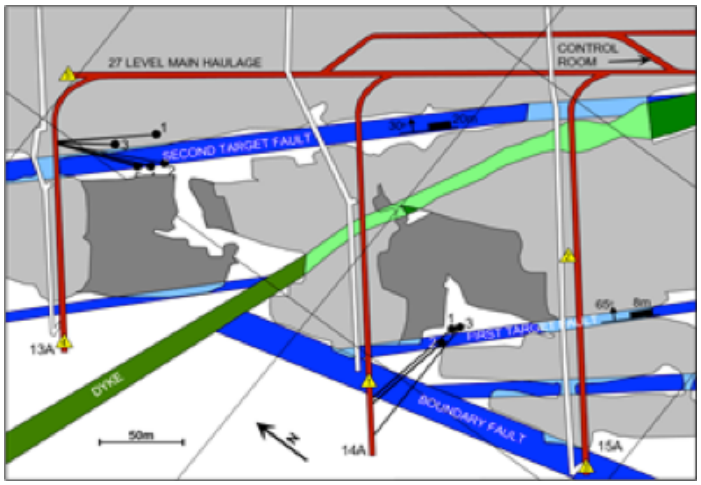


Figure 1: Mine plan showing the location of two Controlled Fault Slip experimental sites.

During the course of this series of experiments, a mobile and modular pumping and monitoring system was developed. This allowed for detailed analysis of fluid behaviour (Figure 2) and fault response to be undertaken. Several boreholes were pressurized simultaneously. Although it was possible to induce microseismicity (M < -1) as a result of pumping high pressure (up to 30 MPa) water at flow rates of up to 120 l/min (Figure 3), it was not possible to induce events of sufficient magnitude (M > 0) to have sufficient impact as to reduce the potential for uncontrolled fault slip that could result in a large magnitude seismic event.

The reason for this inability to produce sufficiently large events lies in the complexity and tortuosity of the fault surface, the relatively high viscosity of water and the limited amounts of water that could be injected onto the fault surface in practice.

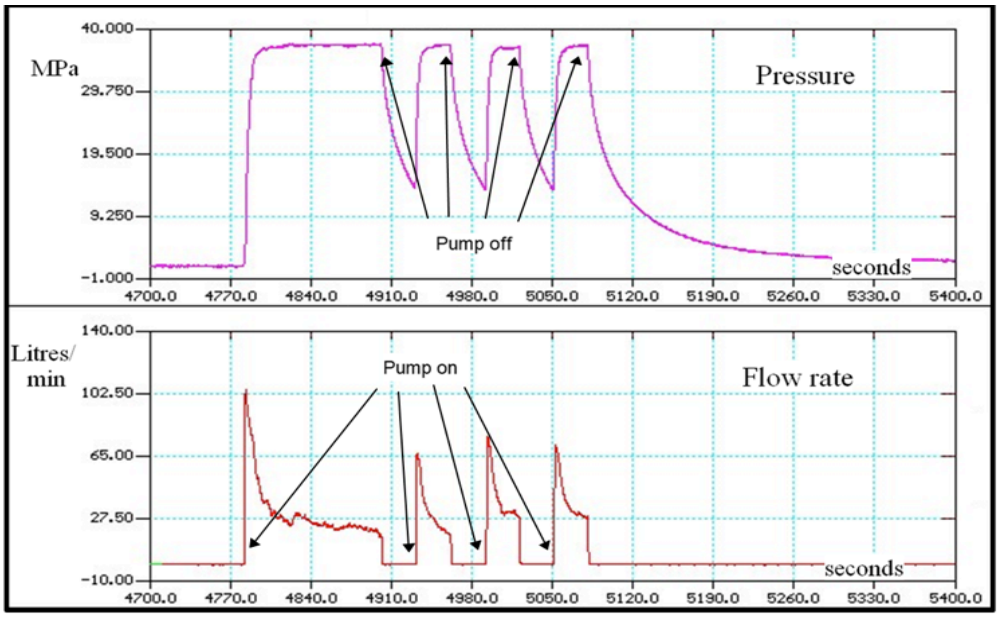


Figure 2: Typical pump record showing the correlation between pressure and fluid flow rate as the pump is switched on and off.

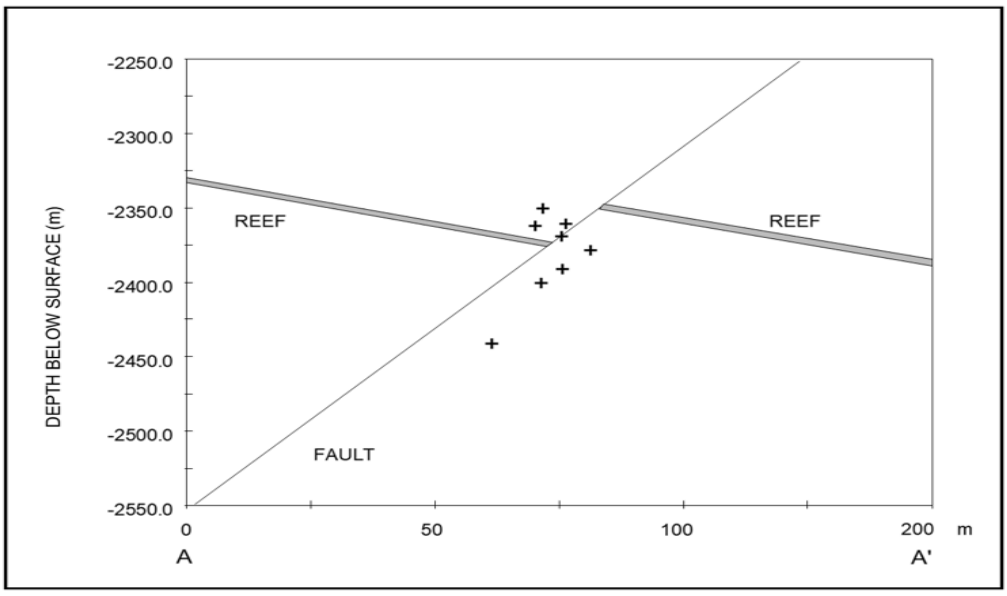


Figure 3: Schematic section showing the location of induced events relative to the fault surface.

CONCLUSION:

Fluids can be used to induce seismic events, however, large magnitude seismic events are not easily induced in unsaturated natural joint systems. Fluid flow in unsaturated natural joint systems is through distinct channels. There is little interconnectivity between relatively closely spaced wellbores in systems dominated by channel flow. For fluid injection to induce seismic events of the required magnitude, pressurisation radii must be of the order of 15 m or more. Necessary pressurisation radii cannot be achieved with water as it is too viscous. Water is not a viable medium for the practical implementation of Controlled Fault Slip by fluid injection. Controlled Fault Slip by low viscosity fluid (gas) injection offers an alternative rockburst control technique.

8. Efficient stope and gully support system design

By Roberts MK.C., Eve R.A., Squelch A.P., Taggart P.N., CSIR Mining Technology, Ref: GAP 032, 1995

SUMMARY:

In South African mines with tabular orebodies, the design of stope support systems has been largely empirically based. A total of 130 different support systems were identified in current use in the industry with clearly , only a number of these systems being optimal. Two main objectives were addressed by the research, namely develop a rationale for the design of stope support systems, and determine criteria for the support of stope gullies for both static and dynamic loading and develop improved support systems for gullies.

With respect to the first objective, a methodology for the design of support systems was developed for both rockfall and rockburst conditions. The methodology requires the setting of criteria in terms of the required support resistance and energy absorption requirements. The support system is then tested against the criteria as shown in the Figure 1. This involves determining the amount of stope closure imposed on a particular line of support by knowing its position relative to the stope face and the closure rate in mm/day. The closure is found from the bottom half of the graph and is extended to intersect the support resistance or remaining energy absorption curve in the upper portion of the graph. The value of energy absorption available (example in the figure) is then read off the y-axis and compared to the criterion.

To determine the support resistance and energy absorption capability of support systems, many laboratory tests, including dynamic tests on packs, were carried out and compiled into a catalogue. In addition the influence on these parameters of stoping width, closure rate, including dynamic, and the spacing of units was accounted for. The criteria for rockfall and rockburst conditions use the thickness of falls of ground as one of the determining parameters. To obtain such data, a data base of relevant parameters taken from all DME fatal accident inquiries dating back to 1990 was compiled.

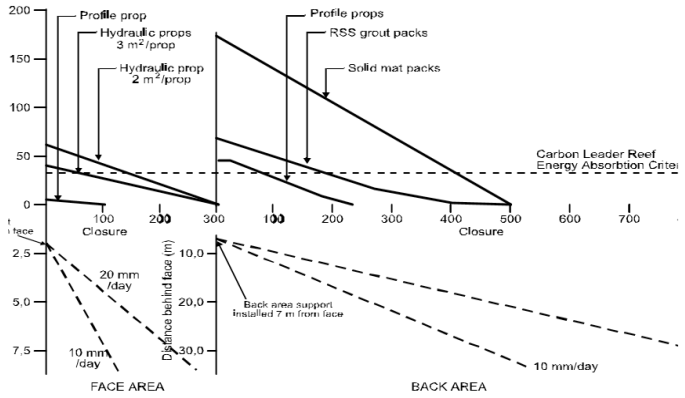


Figure 1: A support design evaluation graph involving energy absorption - deformation curves which are a function of stope closure and of the distance behind the stope face for various support systems and design criteria.

The issue of optimising support spacing was addressed by the development of a numerical model to simulate the interaction between support and a fractured, bedded hangingwall. Some insight was gained with respect to how inter-unit-span stability varies with different support resistances and the spacing of units. Further work is however required in the development of the model and the interpretation of results before the model can be used for design purposes.

The support of stope gullies was addressed in two ways, firstly the problem of gully pack stability and foundation stability was investigated by underground monitoring. This resulted in an understanding of the in situ force - deformation behaviour of gully packs with time and stope closure and the forces at which instability in fractured foundations initiated. From these results a recommendation was made with respect to a desirable force deformation curve for gully packs. This was considered as an important output from this project, and packs constructed from steel or wood fibre reinforced low mass concrete have been developed to conform to the performance specifications and are being implemented in the industry.

The determination of gully hangingwall fallout thicknesses between the gully packs, in order to evaluate support resistance criteria and energy absorption criteria for the prevention of rockfalls and reduction of rockburst damage, was the second part

of the work undertaken. The work concentrated on the Vaal Reef, the Ventersdorp Contact Reef and the Carbon Leader Reef and gully support criteria have been derived for cumulative fallout percentages of between 90 per cent and 100 per cent. Some rockbolting support requirements have been given which would meet these criteria.

CONCLUSION:

A design methodology for stope support applicable to gold and platinum tabular orebodies has been developed.

Support resistance and energy absorption criteria, for rockfall and rockburst conditions respectively, for the methodology are based simply on fall of ground thicknesses.

Values for the criteria were obtained for three reefs not taking into account variations expected in different geotechnical areas. The values obtained were lower than previously recommended but do not include a factor of safety.

Performance specifications for gully packs, where foundation failure is a problem, were developed. Packs conforming to the specifications are available from suppliers.

Recommendations for yielding tendon spacing in gully hangingwalls varied between 0.8 to 1.5 tendons per m² at the 95 per cent fall out thickness level.

Significant development of a discontinuum model to study the interaction between supports and a blocky rock mass was achieved and promising qualitative results were produced. Further work in this field is necessary, particularly to study the influence of dynamic loading.

9. Deep level layout design

By Adams D., Vieira F., Selfe D., Maccelari M., York G., Esterhuizen E., Spottiswoode S, Berlenbach A., Johnson R., CSIR Mining Technology, Ref: GAP 034

SUMMARY:

The aim of GAP034 was to provide the industry with improved mine layout design criteria, an objective continued into GAP223.

The urgent need for criteria for the design of bracket pillars has been recognized by the mining industry. Previously, only rudimentary design rules were available for the use of pillars to stabilise geological structures. It was established during the research conducted that a dyke and fault database was needed that would assist in identifying properties and parameters that influence the stability of bracket pillars. This has been initiated. The development of an easily applied design methodology for stable bracket pillars has ultimately been researched.

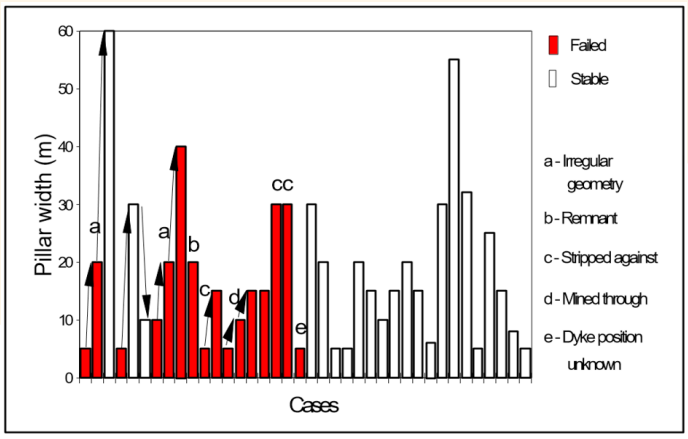


Figure 1: Studying bracket pillar stability against dykes.

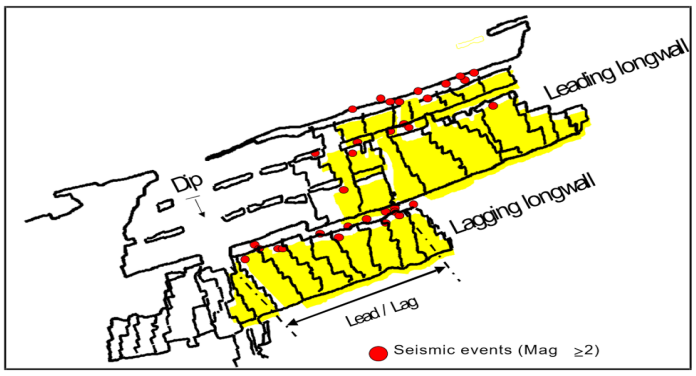


Figure 2. Studying the effect of large leads / lags in regional stability.

The performance of stabilising pillars as regional support structures required enhanced understanding. To this end, fieldwork which involved extensive and regular monitoring of the behaviour of the rock mass in their vicinity was carried out. Design considerations were investigated after the performance of current stabilising pillars were evaluated.The replacement of stabilising pillars with concrete pillars was studied, an attempt directed at promoting new regional support strategies that would allow for full extraction of reef. Theoretical justification and physical underground testing of a concrete support structure were conducted. Modelling studies comparing the behaviour of concrete and rock stabilising pillars indicated that they can be designed to provide identical regional performance. Initial cost effectiveness estimates were made.

Back area caving in the deep level hard rock environment has been successfully practised in at least one mine. The gold mining industry in particular has seen potential in this regional support approach and, thus, work has been carried out to investigate and quantify the rock engineering parameters as well as the practical implications of using such a method.

The development of new criteria for engineering assessment of layout design and performance has been required for some time due to, primarily, the uncertainty of ERR (Energy Release Rate) and ESS (Excess Shear Stress) for consistently characterising the seismic risk of layouts. Both these concepts have been reanalysed resulting in new concepts such as AESS (Area-ESS) and UJESS (Ubiquitous-ESS). The latter have developed, more recently, into a new combined design parameter, termed Rock mass Hazard Index (RHI).

CONCLUSION:

A literature and industry survey of criteria used in the design of stabilising and bracket pillars demonstrated where these criteria were used successfully or not. A database for dykes and faults has been established to assist in determining relationships between their characteristic properties and in situ behaviour. Hazards associated with some structures were identified.

Numerical analysis results were produced that demonstrate the difference in behaviour of various geological features in terms of bracket pillar integrity. Studies of failed bracket pillar cases indicated situations which are likely to be hazardous. Parameters such as face shapes, leads and lags and mining configuration that are critical to layout stability were identified. A comprehensive study on the adherence of current stabilising pillars to the traditional design criteria was conducted.

In situ behaviour of stabilising pillar was assessed through systematic monitoring, using techniques and processes such as PSS,GPR, seismic refraction, wire extensometers, stress measurements, fracture mapping, closure-ride stations and precise levelling). Caving mechanisms in hard rock were researched. Quantification of caving behaviour produced. Determination of caving benefits and conditions for application attained.

Th effectiveness of ESS and ERR in layout design was assessed. Incremental Area-ESS and Ubiquitous-Joint-ESS applied to rock engineering assessment of layouts. Field behaviour of concrete pillars were assessed through a trial concrete block cast underground. Numerical modelling results showed that the effectiveness of concrete pillars was comparable to that of rock stabilising pillars.

10. Reliable cost effective technique for in situ ground stress measurements in deep gold mines

By Stacey T.R., (Steffen, Robertson and Kirsten Institution). Ref: GAP 220

SUMMARY:

The proposed primary output of this research is the following:

- literature survey of existing methods of in situ stress measurement including an evaluation of the applicability of these methods in deep level gold mines.
- a theoretical feasibility study of a potential method, which will be cost effective and practical, and applicable to the gold mine situation.

A review of the extensive literature on the subject of in situ stress measurement has been carried out, and a substantial bibliography is included in the report. The majority of methods of in situ stress measurement which are available were immediately rejected as being inapplicable for the deep gold mines. Several methods were identified as having potential applicability from a technical viewpoint and were dealt with in more detail. These methods are:

- borehole overcoring methods, including the CSIR doorstopper strain cell, the CSIR triaxial strain cell, and the CSIRO HI cell
- large diameter overcoring in a bored raise
- Hydrofracturing
- back analysis from monitored deformations around excavations

It is considered that the applicability of these methods in the deep gold mines is poor. Reasons for this include their cost, their requirement for significant assistance from mines in terms of services and personnel and their reliability in general and under the harsh conditions in the mines. Only two of the approaches are considered to have limited application. These are a simplified hydrofracture test in which only the magnitude of the minimum principal stress is determined, and back analysis from monitored deformations, when suitable excavations are made for other purposes and economical monitoring can be used.

Based on the review of the available methods of in situ stress measurement, and experience in stress measurement programmes using numerous techniques, the following have been identified as requirements for a reliable, cost effective technique for in situ stress measurement in deep gold mines:

- the technique must be undemanding on the requirement for services and personnel provided by the mine.
- the technique should be low cost with regard to all aspects - cost of preparation, cost of installation, cost of instrumentation, cost per measurement, and economical in terms of time requirements
- the technique should allow many measurements to be made in a single mining shift. This will allow the results to be treated statistically and therefore avoid localised effects.
- the technique should not be sensitive to high stress effects such as spalling and microcracking of the rock.
- the technique should be flexible. It should be possible implement at very short notice, require a minimum of preparation, be possible to apply in excavations of limited size.
- the technique should preferably not require laboratory testing to determine the deformation properties of the rock or rock mass.

Based on these requirements an in situ stress measurements technique, which will be practically applicable in the deep mines, has been developed conceptually. Referring to the figure, this method involves:

- a borehole-based system, using jackhammer drilled boreholes
- creating the enlargement, or stress/strain change, not by overcoring, but by drilling parallel, overlapping holes
- control of the direction of adjacent holes, by means of a special borehole guide
- measuring deformation changes in the first borehole, as a result of the drilling of the third borehole, using a borehole deformation meter
- sequential deformation measurements down the length of the borehole, by alternately moving the deformation meter and advancing the third borehole
- measurements carried out in three different borehole orientations to provide sufficient data for the complete state of stress to be determined
- calculation of the in situ stress field by back analysis

of monitored deformations using an appropriate numerical stress analysis program
- optimisation of results making use of the redundancy in number of data

The theoretical feasibility of the method has been proved by carrying out two - and three - dimensional stress analyses. These showed that:

- deformations induced around the first borehole by the drilling of the third borehole are of sufficient magnitude to measure easily
- the effect of stress induced failure around the boreholes on the deformations is probably less than the variations due to experimental errors and local rock material variations
- approximately five measurements per metre length of borehole can be carried out.

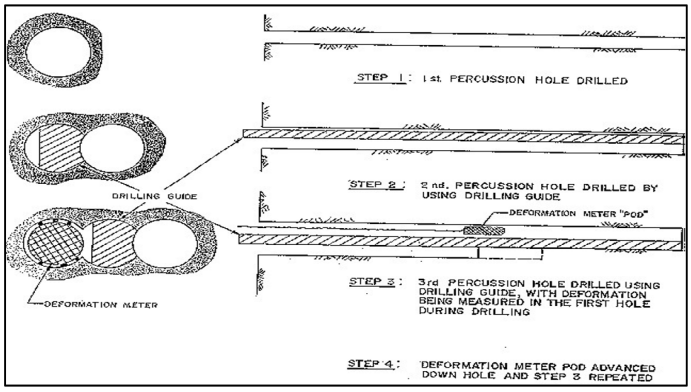


Figure 1: Ground Stress Measurement Technique

11. Experimental investigation of fundamental processes in mining induced fracturing and rock

By Napier J.A.L., CSIR: Mining Technology , Ref: GAP 601B

SUMMARY:

Additional work on joint creep behaviour has been completed and a numerical model of friction mobilisation has been formulated to explain some of these results. Further investigations of underground creep behaviour have been carried out allowing the formulation of descriptive functions of closure rate in relation to face advance rate. This provides practical input to the evaluation of support duties. Significant advances have been made in acoustic emission monitoring studies of rock samples and comparisons of sample size seismic scaling relations in relation to mine-scale behaviour. Testing of cubic shaped samples and x-ray analysis has revealed the fracture structure near lead-lag edges. An example of the x-ray reconstruction for one sample is shown in Figure 1.

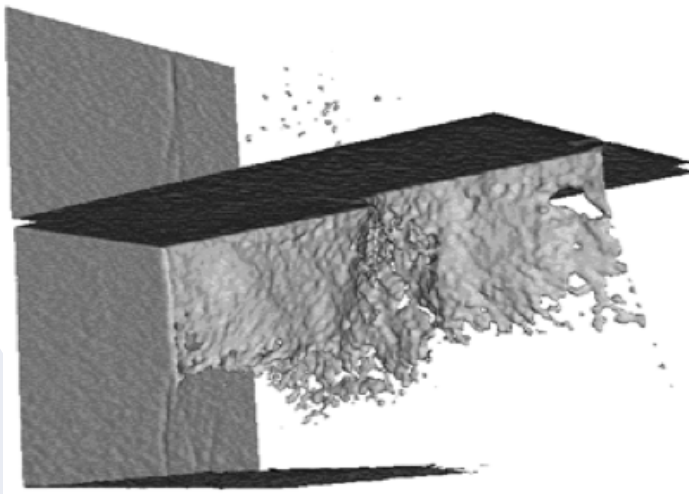


Figure 1: Fracture structure near a lead-lag edge in a cubic, laboratoryscale sample.

Modifications have been made to the DIGS code to allow slip and tension weakening behaviour on fractures. This allows size effects to be modelled as well as a more quantitative approach to the analysis of excavation stability. Additional studies of time to failure have been completed including an analysis of fracture clustering at the point of failure. This suggests that it is necessary to determine both the damage structure and to monitor seismic event sequences if improved prediction of failure time and location is to be achieved. Some attention has been given to developing improved mesh-free methods for the simulation of three-dimensional fracture propagation and fault slip, together with a simplified method to simulate blast-induced fracture growth in the DIGS computer code. An extensive study of the interaction of seismic waves with underground openings and the analysis of wave propagation through fractured rock has been completed.

An improved stressdependent joint stiffness model has been included in the WAVE code. Special techniques to quantify dispersion behaviour have been devised. Mechanisms for surface wave loosening of hangingwall blocks have been identified and studies of mechanisms leading to velocity amplification have been carried out. Figure 2 shows contours of maximum velocity induced by a slip event near a longwall stope. These velocities are found to be several times larger than values estimated from standard source-distance relationships. Induced tensile stresses are also observed and should be used as a complementary criterion to maximum particle velocity in the assessment of stope stability.

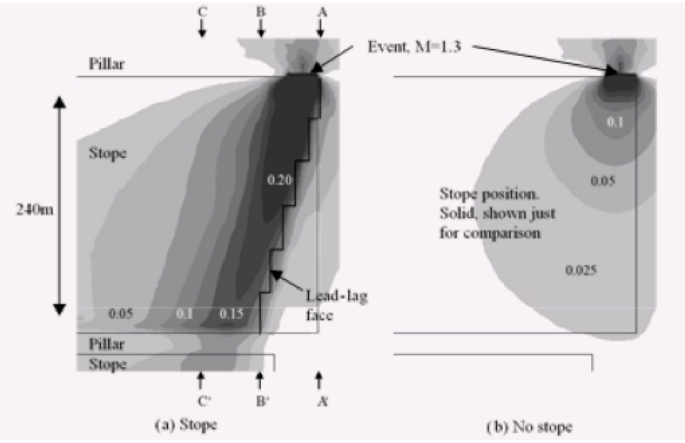


Figure: 2 Wave amplification near a simulated longwall stope

CONCLUSION:

Descriptive time-dependent closure rate functions in relation to face advance rate have been formulated to assist support performance evaluation. Acoustic emission studies from small-scale samples have shown self-similar behaviour to mine-scale seismic behaviour. X-ray analysis of laboratory samples has provided some insight into fracture structures near lead-lag excavation edges.

Slip and tension weakening rules have been included in the DIGS code to simulate size effect behaviour and stability. Time to failure studies suggest that information on damage structure is required to supplement seismic observations if any predictions of time to failure are to become possible. Simulation of critical slip weakening suggests the possible use of this parameter to rank the relative stability of proposed layouts. This has not been evaluated.

A simplified model of blast fracture propagation has been formulated. An application of the model shows a possible explanation for smooth blasting mechanisms. A stress-dependent joint stiffness model has been developed for the dynamic WAVE code. Studies of the interaction of seismic wave propagation with fractured rock have been completed indicating the importance of surface waves in leading to velocity amplification and key-block stability.



12. Improved support design by an increased understanding of rockmass behavior around the Ventersdorp Contact Reef

By Roberts M., Güler G., Quaye G., Schweitzer J., Eve R., CSIR Mining Technology, Ref: GAP 102, 1996

SUMMARY:

The Ventersdorp Contact Reef (VCR) contributes about 15 per cent of South Africa's total gold production and will continue to be a major source into the future. The reef is unique in the Witwatersrand Super Group stratigraphy in that it has a lava hangingwall. Prior to this project little formal rock engineering research had been carried out on this important ore body.

The primary objective of the project was to obtain an understanding of the rockmass behavior around VCR stopes in order to design support for stopes on a more rational basis. Several approached were used in the research, each of which is discussed in detail in 9 appendices. These include the delineation of geotechnical domains based on combinations of either 'hard' or 'soft' lava with various footwall rock types. Six domains were delineated.

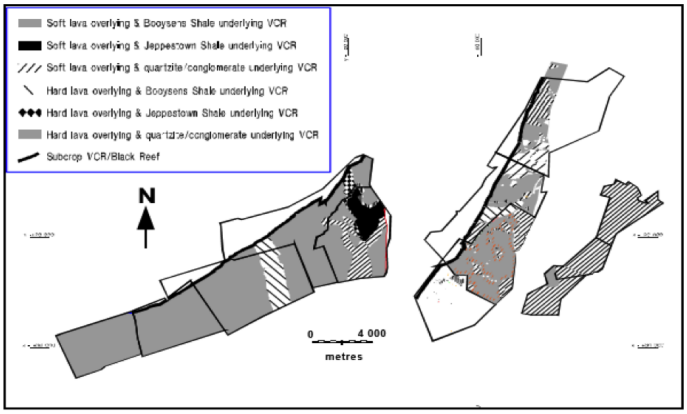


Figure 1: Geotechnical plan of the VCR mines under consideration in the West Rand and Carletonville Goldfields

Sites were established in most of these domains to study and monitor the response of the rockmass to mining in terms of deformation, fracturing and types of falls of ground. Where these were significantly different geotechnical areas are defined.

A comprehensive laboratory test programme was conducted involving over 500 tests, and a database of rock properties established. No consistent differences in strength were obtained when tests were conducted normal or parallel to bedding.

Very different fracture patterns were identified for different geotechnical conditions resulting in diverse block sizes, fall of ground thicknesses and hence support requirements. Detailed analysis of closure and seismic data in different geotechnical areas was conducted. Basic models for numerical analysis of support/hangingwall interaction were developed using FLAC UDEC codes and some initial analysis using J Block was carried out.

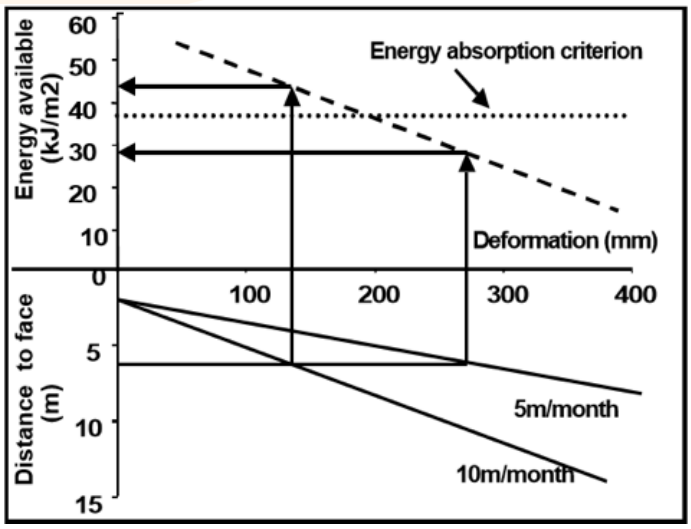


Figure 2: The effect of different face advance rates on the amount of closure experienced at the particular distance behind the face and hence amount of available energy to be absorbed by support system

CONCLUSION:

Sites were established in most of these domains to study and monitor the response of the rockmass to mining in terms of deformation, fracturing and types of falls of ground. Where these were significantly different geotechnical areas are defined.

A comprehensive laboratory test programme was conducted involving over 500 tests, and a database of rock properties established. No consistent differences in strength were obtained when tests were conducted normal or parallel to bedding.

13. Human computer interaction in rock engineering

By Squelch A.P., CSIR Mining Technology, Ref: GEN 203, 1996

SUMMARY:

Current training methods, which rely heavily on the conventional lecture/classroom style of instruction, were criticised for not being effective and being in need of improvement and made more meaningful to the work force. It was therefore proposed that the use of an additional training method be investigated which could help rectify the current shortcomings. Virtual reality (VR) was suggested as the technology to meet this need.

A review of current literature revealed VR to be a young, rapidly developing computer technology that was already being applied to training situations in various industries and showed distinct potential for application in the mining industry. Features of VR which make it particularly attractive for training applications and which have advantages over traditional methods were identified as:

- low cost alternative to creating full-scale physical training scenarios
- flexible configuration and amenable to on-going modification and customisation
- opportunity to create a wide variety of scenarios including ones rarely, or never previously, encountered in 'real life'
- once constructed the training scenarios can be run repeatedly and to order
- provide appropriate levels of trainee interaction

Perhaps the most unique and attractive feature of VR for hazard related training applications is that trainees can learn from mistakes made in the virtual world without being exposed to any real danger. The implications of their mistakes can, however, still be brought home to the trainee via suitable use of audio and visual effects.Hazard awareness training is catered for by the inclusion of three fall of ground hazards and one moving scraper hazard. The trainee must identify each hazard and indicate the appropriate corrective action via clickable on-screen icon buttons. The trainee's failure to correctly identify and treat any of the hazards is recorded and the implications of this failure are indicated to the trainee using graphical and auditory effects.

The VR stope simulator was developed and tested with the aid of evaluation trials at the Elandsrand gold mine training centre. These trials indicated an encouraging level of relevance, understanding and acceptance of the VR generated mine environment among the trainee target group, ranging from stope workers to team leaders.

The main desirable features of future VR training simulators were identified as: modular design, refresh rates exceeding 6 frames per second, use of textures and sound effects and an in-built evaluation and training record facility. A number of VR applications other than those explored within the scope of the project were also identified and documented for future consideration. These suggestions include: a stope co-ordinator training/evaluation system, equipment (e.g. loco, winch and drilling machine) operator training, and individual support installation procedures.



Figure 1: View of VR tunnel model

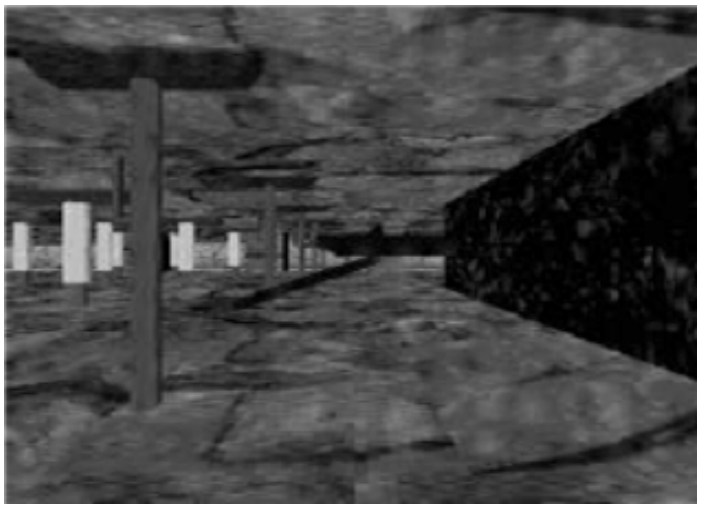


Figure 2: View of stope in VR simulator

CONCLUSION:

Successful development of a VR underground simulator has been achieved which has demonstrated the potential for the application of VR in the mining industry's training systems.

An encouragingly high level of acceptance and understanding was shown by the target group of underground workers for the concept and the imagery of the VR underground simulator.

Indications are that VR based simulators can be successfully applied to raise the level of hazard awareness training in the mining industry, and thereby reduce underground accident and fatality rates.

VR based simulators can be integrated with conventional training techniques and thereby provide an enhancement to current training methods.



14. Develop improved strategies for mining highly stressed areas

By Johnson R.A., CSIR Mining Technology, Ref: GAP 033, 1997

SUMMARY:

The aim of this long-term project was to focus on the extreme highstress end of the mining spectrum. Such high stress conditions prevail in certain ultra-deep mining operations of the near future, and are already being experienced in extracting remnants, stabilising pillars or shaft pillars at more moderate depths. Mining in such ground poses major problems including high rates of closure and ground mobility, difficulties of siting and support of access tunnels, and, above all, severely enhanced hazards of rockbursting.

An industry wide questionnaire, a comprehensive literature review and a high level industry workshop assisted in identifying priority areas of work for this project (Johnson, 1994a ;1994b). These were to develop new and improved layouts and regional mining strategies to satisfy indicated mining and geotechnical conditions, to study remnant stability with a view to reducing remnant associated problems and to develop improved seismic techniques.

Anticipated mining conditions, using current methods, at depths between 3000 and 5000 m were investigated and indicate excessive energy release rate (ERR), stope closure rates, face area stresses and excess shear stress values.

Considerable information on the geotechnical environment of orebodies potentially mined at ultra-depth was collected, much in a database form. This information includes reef information such as channel width and grade; hangingwall and footwall composition and associated properties; fault and dyke distribution and projected future production trends with depth. These provide essential information for future mine layout planning and assessment (Schweitzer and Johnson, 1997).

Laboratory work has been carried out to quantify the post-failure behaviour of brittle quartzite alone as well as with the confining effects of backfill (see Figure 1). This work is in anticipation of the use of crush pillars and backfill as a useful support medium and mining method for application at ultra-depth.Several new mine layouts and approaches were proposed for ultradepth conditions indicated in a geotechnical classification. New regional and local layouts and mining methods are described that provide increased worker safety at ultra-depth. These offer significantly reduced ERR, increased extraction, reduced stope closure and face stress compared to the best methods currently employed (Spearing, 1993; Johnson and Schweitzer, 1996).

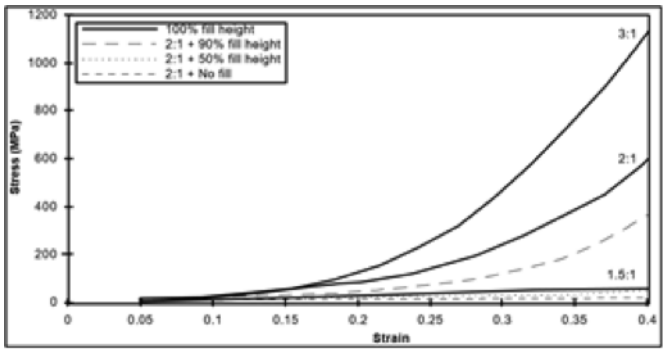


Figure 1: Post-failure behaviour for varying w/h ratios and fill heights.

A detailed assessment of the stability of a number of problematic remnants yielded inconclusive results. Seismic software enhancements offer substantially improved locations and event classification (see Figure 2). This has significant potential in advance identification of hazardous geological features and will be of benefit to ultra deep level mining (Milev et al, 1995; Spottiswoode and Milev, 1995).

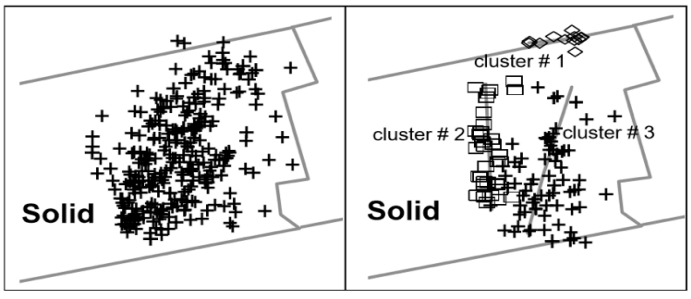


Figure 2: Improved location accuracy and event clustering illustrated.

CONCLUSION:

A classification of geotechnical conditions at ultra-depth was compiled. This extrapolates current information to greater depth and will assist in mine planning. New and improved mining strategies suited to indicated ultra-depthconditions were developed.

While further work is required, these offer improvements over current methods. Seismic techniques were developed which improve location accuracy and event classification. These have the potential to assist with advance identification of seismically hazardous geological features.

15. Develop a more reliable means of assessing safety risk due to rockbursts and rockfalls as a managerial decision support technique

By Dr Cichowicz A., Wits University, Ref: GAP112, 1997

SUMMARY:

A statistical analysis of falls of ground at ERPM mine was performed on data from September 1991 to June 1994 from the K1 shaft area. From a total data base of 1650 seismic events and 344 falls of ground, only 64 falls of ground could be reliably and uniquely linked to a seismic event. Most records of FOGs were excluded due to an inaccurate time estimate. For example damage during the blasting shift was usually allocated a time in the following shift.

A magnitude-distance plot of damaging events showed that most damaging events had peak velocity, vmax, > 10 mm/s and < 1 m/s, as derived from distance, stress drop and corner frequency. This excludes any local amplification, or site, effects.

Damage was studied in terms of stopes that were being mined in geologically disturbed ground and those that were not. The cumulative number of rockbursts were plotted against time of occurrence. A linear plot would indicate random time of events (a Poisson distribution). Rockbursts in geologically undisturbed ground agreed with the Poisson model, whereas rockbursts near geological features deviate from linearity, indicating a non-Poisson distribution. It is not clear from the analysis whether this results from the nonrandom spatial distribution of geological features or that rockbursts are grouped in time.

The Analytical Hierarchy Process (AHP) provides a decision framework to assist in breaking decisions down into their less complex components and quantifying their relative importance. This was applied to data at a number of levels.

Ten rock mechanics experts were surveyed regarding the factors most likely to lead to rockbursts. Comparison between data and the experts was quite close.

Neural network classification, was performed using a selforganised, or Kohonen, network. Inputs were classified by the network into two groups, that were called “high risk” and “low risk”. Figure 1 shows one of the relationships from the report.

The results from the expert system study were included in a user-friendly Windows program that was tested at two mines.

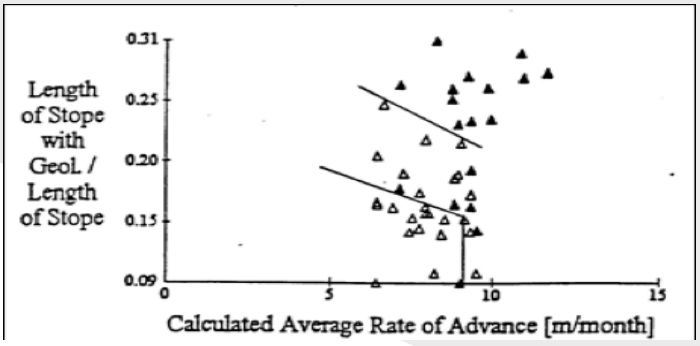


Figure 1: Solid triangles represent the “high risk” data and open triangles are the “low risk” data. Each triangle represents a month of production (Note : the seismic risk has not been normalised for the amount of mining, and therefore penalises stopes with high rates of face advance)

CONCLUSION:

The average seismic stress drop of all seismograms was 6.8 MPa, while the average stress drop of a damaging event is 17 MPa. Therefore, for the same magnitude, high stress drop events are more likely to cause damage than low stress drop events.

The peak ground velocity exceeded 10 mm/s for most areas of rockburst damage, exclusive of local site effects. (Note. Low values should be considered as "shake-out" damage.) Geologically disturbed panels were three times as likely to suffer rockburst damage. Rockbursts next to geological features are non-randomly distributed in time over periods of months.

16. Testing of tunnel support - Dynamic load testing of rock support containment systems (eg wire mesh)

By Ortlepp W.D., Stacey T.R., (Steffen, Robertson and Kirsten Institution), Ref: GAP221, 1997

SUMMARY:

In 1993 a series of tests were conducted on conventional and yielding rock bolts, which act as the retaining components of a support system. Under seismic conditions it is also necessary to have an understanding of the role and capacity of the containment component of the support system such as mesh and lace, and shotcrete. The GAP221 project was formulated to address the deficiencies in the understanding of containment support systems.

The objective of the project was to determine the performance characteristics of containment, typical of those used in the South African mines, under simulated rockburst conditions. The apparatus used in these tests comprised a drop weight, which impacted on a simulated rock mass structure supported by the containment support system suspended from yielding rock bolts. Tests could be conducted up to input energies of 70 kJ and impact velocities of 8 m/s. The panels of containment support were approximately 1,6 m x 1,6 m, tensioned by stay ropes to provide suitable "infinite" boundary conditions. The rock bolt suspension was on a one metre square pattern.

A summary of the results, indicating the trends and limits of performance of the containment systems, is shown in the following figure..The evaluation of the mesh fabric indicated the superior performance of the diamond mesh compared to standard weld mesh. This is considered to be due to the increased yieldability of the diamond mesh, and thus energy absorption, under the given loading conditions prior to failure. In these tests it was noted that the failure of the mesh fabric was mainly due to guillotining of individual strands at the edge of the rock bolt face plate.

This failure mechanism led to the inferior performance of the diamond mesh when combined with lacing within the containment system. The failure of an individual strand of the diamond mesh resulted in unravelling of the mesh fabric and spillage of the contained rock mass. The stiffness and structure of weld mesh, however, meant that the overall integrity of the containment system was maintained even after initial failure of individual strands.

Yieldability in the lacing system, and thus energy absorption capacity, could be improved by the development of novel yielding systems. The inclusion of loops within the lacing was evaluated and proved successful. However this system may not be practical due to sensitivity to clamping pressure and the operational limitation of allowable deformation of the contained rock mass.

The attachment of lacing to the rock bolt retaining elements was also examined and the use of eyed face plates and novel chain link systems was also evaluated.

Shotcrete panels were also evaluated as components of containment support systems. In general the steel fibre reinforced shotcrete could absorb impact energy of up to 15 kJ/m2. Significant cracking of the panel was associated with this initial impact loading and a second, generally smaller impact was sufficient to cause failure of the panel. Concern is also expressed with regard to the long term stability of cracked steel fibre reinforced shotcrete panels due to corrosion of exposed fibres.

Observations of the displacement velocity of the containment support surface by high speed video camera were made on one test. This indicated that

the surface of the containment system, and immediate simulated rock mass, can gain velocities in excess of the initial impact velocity, and may be twice that of the simultaneous impulse “driver” velocity. This observation correlates well with inferred velocities from underground observations.

In general, the observations of the damage sustained by the containment support systems, under the dynamic loading test conditions, were considered to correlate well with underground observations. The testing method was thus considered to be reliable, relevant and realistic.

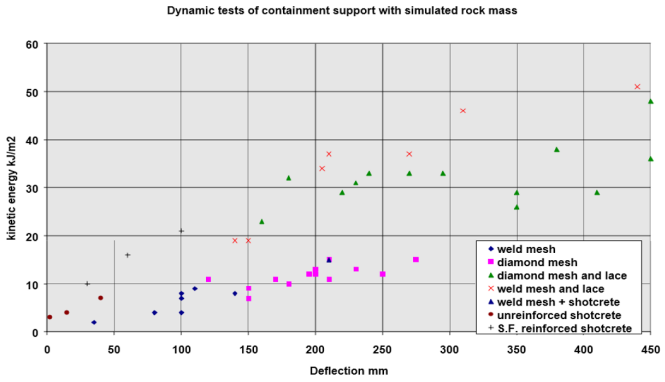


Figure 1: Kinetic energy vs deflection for typical containment systems.

CONCLUSION:

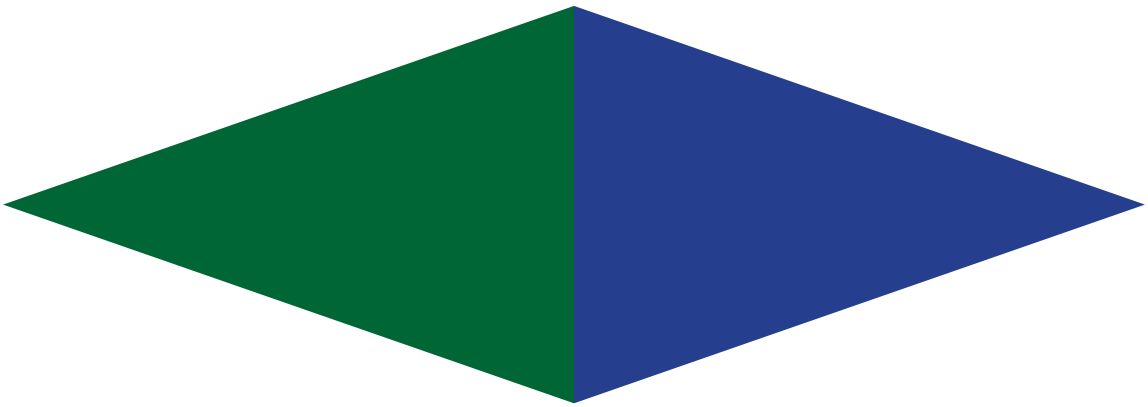
The lacing is the most important component of the containment support system but should be limited to a maximum diameter of 12 mm based on current typical rock bolt capacities.

Yieldability within the lacing system improves its survivability, particularly of small diameter ropes. Failure of mesh is initiated at the face plate of rock bolts (if used), but may be limited through improved design of the face plate.

100 x 4,0 mm diamond mesh over 16 mm diameter yielding bolts may be used as containment support for areas of relatively low intensity seismic risk: 15 kJ/m2. 100 x 3,5 mm weld mesh over 20 mm diameter yielding bolts with 45 kN lacing may be used for areas of moderate intensity seismic risk: 30 kJ/m2.

100 x 3,5 mm weld mesh (extensible) over 22 mm diameter yielding bolts with 120 kN lacing may be used for areas of high intensity seismic risk: 60 kJ/m2.

Current applications of shotcrete may have limited survivability where seismic impulses are in excess of 10 kJ/m2. Under more severe conditions the application of external lacing may improve its operational viability.





17. Literature survey on the advance detection of dykes in underground coal mine workings

By Prof. Fourie G.A., Laboratory for Advanced Engineering University of Pretoria, Ref: COL 503B, 1998

SUMMARY:

To minimise risk, a more effective system of mine planning and operation is required which calls for advance knowledge of structural and geological conditions ahead of the coal working face. This research project attempts to identify, by means of a literature survey, different exploration and geophysical methods that could be used to detect dykes ahead of underground coal mine workings.

In this study a range of geophysical and exploration methods have been identified which are generally used in the mining industry. Each different method exploits a different physical property of the earth's crust, e.g. seismic and acoustic methods measure the velocities of sound waves, while magnetic methods distinguish the variance in magnetic susceptibility of different rock types. Gravitational survey methods detect and measure lateral variations in the earth's gravitational pull that are associated with near surface changes in density.

Dykes generally display a high level of magnetic susceptibility and magnetic survey methods are therefore commonly used by most coal mining companies to identify the presence of this geological feature. The sophistication of aeromagnetic survey techniques has increased significantly over the past decade and is now routinely used by local and overseas coal mining companies to explore relatively large target areas fairly successfully.

Horizontal, in-seam drilling was first tested in the United States during the late 1950's. This technique was initially developed to degasify the coal seam ahead of mining operations. Similar trials were conducted in Australia during 1980 and by mid-1981 the Australlian Coal Industries Research Laboratories Limited (ACIRL) reported that "600 m longholes can be achieved on a regular basis". In-seam drilling is now practiced on a number of South African collieries and hole lengths of up to 1200 m are standard. During the early 1990's Amcoal introduced incline-directional drilling from surface as a means to detect dykes and other geological features ahead of mining. Average borehole lengths achieved with this method are now in the order of 1200 metres, however, development work is in progress to increase the effective reach to 2000 metres. Similar work is currently in progress at Sasol Coal.

The use of ground penetrating radar as a means to detect dykes has been tested on various South African gold mines. Initial reports indicate that the presence of dykes and faults could be detected successfully up to a distance of 40 metres ahead of the mining face. The same level of success could, however, not be achieved on coal mines.

At present most South African coal operations use a combination of vertical drilling, aeromagnetic surveying and in-seam drilling techniques to detect and identify the presence of dykes ahead of the mining face with a relative high degree of accuracy.



18. A methodology for the definition of geotechnical areas within the South African gold and platinum stoping horizons

By Guier G., Quaye G.B., Jager A.J., Dr. Schweitzer J.K., Grodner M., Reddy N., Andersen L., Dr. Milev, A., Dr Wallmach T., CSIR : Division of Mining Technology, Ref: GAP 416, 1998

SUMMARY:

Variable rock mass behaviour is present in Witwatersrand and Bushveld mining districts. Geotechnical classification of the stoping environments is therefore essential. However, a sound methodology that classifies these environments had not been established before 1997. This is the major focus of this investigation. A detailed literature review, in addition, reveals that current rock mass classification systems do not adequately address the basic requirements for defining geotechnical areas in South African gold and platinum stoping horizons.

It was found during initial surveys carried out under this project that various definitions were assigned to the term "geotechnical area" across the South African gold and platinum mining industry. To provide a common basis of understanding across the entire industry, this investigation refers to a geotechnical area as an area exhibiting a specific rock mass behaviour and associated hazards in response to mining activities, where the same rock engineering strategies can be applied.

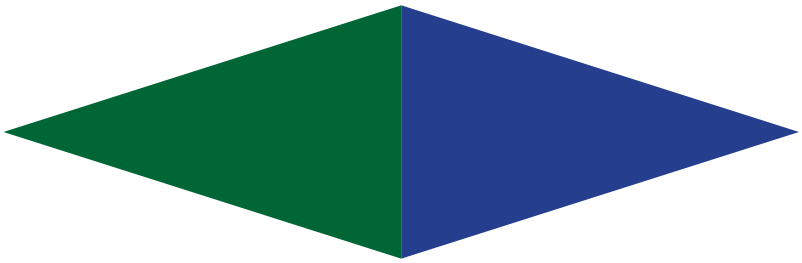
Generally, mine design strategies are adopted on both regional and local scales. The regional mine design strategies, such as mining method/layout and sequence, and regional support, are decided upon approximate values for a few critical parameters, which control the overall rock mass behaviour. Once the regional strategies are determined it is necessary to identify local strata control strategies, both providing a safe and productive mine. Local strategies depend upon a more detailed knowledge of the rock mass response, and are controlled by combinations of a wider range of parameters than the regional strategies. It is thus logical that the geotechnical subdivision of a mine has to be carried out at two levels.

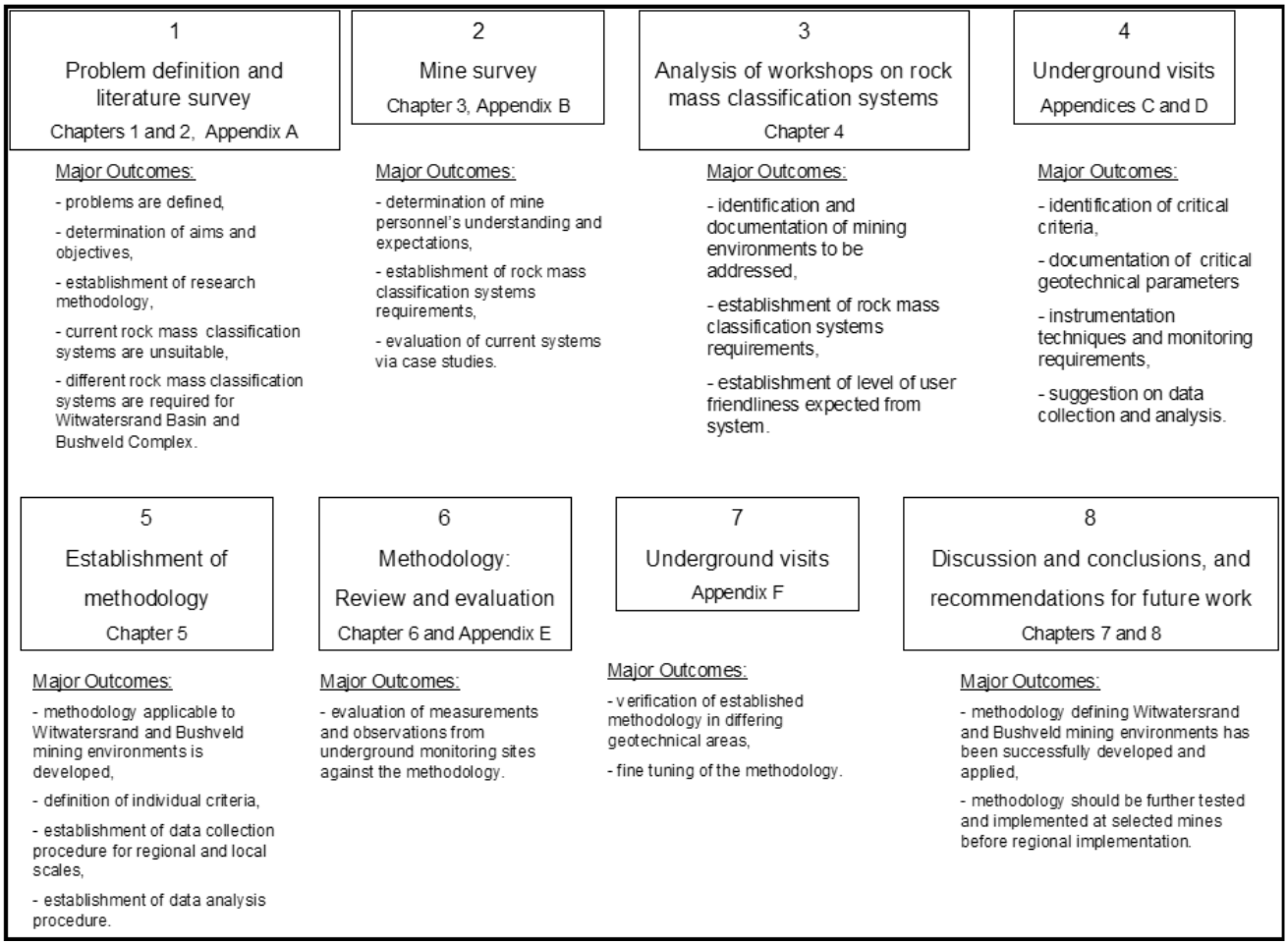
These are referred to as Regional Geotechnical Areas, or Mining Environments and Ground Control Districts (GCDs) in this investigation to correspond with the two levels of strategic decision making.

Regional geotechnical areas are ideally delineated before the onset of mining. The input parameters that are required for delineating these geotechnical areas are derived from orebody information, depth, stratigraphy, major geological discontinuities, and regional hydrology. By combining this information, appropriate mining methods can be selected, for different areas the likely rock mass behaviour can be ascertained. Areas of similar rock mass behaviour will be considered as a particular Regional Geotechnical Area, where a specific regional support strategy can be defined.

Once mining commences more detailed information concerning the rock mass response and the contributing parameters will become available. GCDs that have different rock mass responses can then be delineated. The procedure to define the GCDs will include identifying, quantifying, and classifying the critical geotechnical parameters, establishing the rock mass response and demarcating areas of similar behaviour, and implementing appropriate rock engineering strategies.

The methodology was developed in close interaction with representatives from the mining industry. The various criteria that form part of this methodology were tested and evaluated during several underground investigations. It is found that the newly established methodology adequately defines a large variety of geotechnical settings. Selected practitioners should however test it further. This will facilitate any required modifications and implementation.





19. Evaluation and Upgrading of Records of Stress Measurement Data in the Mining Industry

By Stacey T.R. Wesseloo J., SRK Consulting, Ref: GAP 511B, 1998

SUMMARY:

The safety of mining operations is significantly influenced by the in situ stress fields in which the mining excavations are created. Many in situ stress measurements have been carried out for mining and civil engineering projects in Southern Africa over the past 30 to 40 years. Some of these data have been published, but many are contained in mine, company and owner organisation records. A collated and evaluated record of these data will provide a valuable input source for the planning of mining operations, allowing more valid stress calculations to be made.

This report describes such a record, which has been compiled into a computer database. Data on stress measurements and observations have been obtained, evaluated, reanalysed where necessary, and graded. The project described in this report is SIMRAC Project GAP 511b, "Evaluation and upgrading of records of stress measurement data in the mining industry".

A standard spreadsheet program has been used for capturing of the database. The database is available on a stiffy disc in standard DBASE format. In this format it can be accessed using any of the common database or spreadsheet packages such as ACCESS, QUATTRO PRO, EXCEL, LOTUS 123 etc, at least one of which should be available on every mine.

Various GIS and other graphical/map packages were investigated for possible use for visual display and interrogation of data. Ultimately, the package ArcExplorer was chosen. This is an easy to use Windows based package which provides all the functionality considered to be necessary for the project. ArcExplorer has been developed by the Environmental Systems Research Institute (ESRI), the organisation

responsible for the development of the ARC/INFO and ArcView systems. The main reason for the choice of ArcExplorer, however, was that it is freely available on the Internet, and can therefore be downloaded at no cost by any user who wishes to make active use of the database. Instructions for downloading of the system from the Internet are given.

ArcExplorer uses shape files containing the spatial information of each data point in the database, and a DBASE file containing the alphanumeric data of each data point. Copies of these files are provided on the stiffy disc which accompanies the report.

Where possible, the data for each measurement entered into the database include:

- the location (coordinates and depth below surface);
- the mine or project;
- the rock type in which the measurement was made, and its deformation properties
- the geology of the area;
- the in situ stress components in the north-south, east-west and vertical directions, and the corresponding shear stress components;
- the in situ principal stresses and their orientations relative to north;
- an estimate of the overburden stress;
- comments relevant to the measurement;
- gradings of the qualities of the individual measurements and the groups of measurementso on a subjective/quantitative basis.

The purpose of the database is to allow users to extract the information that they require or may have an interest in. It is therefore somewhat superfluous to present any results from the database. However, in the report, some results are presented, which are of interest from the mining viewpoint and also serve to demonstrate the capabilities of the system. The results available from the database indicate that the horizontal principal stresses in Southern Africa commonly trend approximately north west - south east and north east - south west. The major horizontal stress magnitudes are almost always equal to or greater than the vertical stress magnitudes.

Based on evaluation of the information it is considered that there is little to be gained by carrying out in situ stress measurements for the specific purpose of filling in "gaps" in the stress database. Sufficiently reliable extrapolation of data, based on existing trends, can be achieved from the information in the database and from observations of dog earing behaviour in boreholes, shafts and raise bores. It is recommended that data from future in situ stress measurement exercises, if available, should be added to the database from time to time, to maintain the database as up to date as possible.

CONCLUSION:

It is concluded that the availability of the database will contribute to the goal of safer mining conditions in South Africa.

20. Further Validation of Bracket Pillar Design Methodology

By Vieira F., Dede T., Stewart R., Ruskovich A., CSIR Mining Technology, Rock Engineering, Ref: GAP 516, 1998

SUMMARY:

Design charts for bracket pillar design were developed under a previous SIMRAC project GAP 223 to provide rock mechanics engineers with an initial estimate of bracket pillar sizes for clearly identified geological discontinuities, based on mining and geological factors that would be easily measured. A methodology for the application of such a design tool had then been proposed. The novel characteristic of this pillar design approach was the allowance to "tolerate" a certain level of seismic risk along the bracketed feature. Once an informed decision is made regarding the level of seismic risk to be accepted for a given layout, the rock engineer obtains from "design charts" a recommendable pillar width, assumed to keep the risk of "slip type events" below the pre-accepted level. It was felt that designing for an acceptable level of seismicity (e.g. for a tolerable event magnitude) would be less conservative but more realistic an approach than requiring, for instance, zero ESS, as had previously been practiced.

GAP 516, the current project, was concerned with the applicability of the above design approach to real mine geometries and has concentrated, therefore, on studying the behaviour of various closely monitored bracket pillars, including a validation, based on field data, of the bracket pillar design methodology. The methodology has also been reviewed.

To better understand the intrinsic behaviour of geological features as well as the failure mechanism of these when being bracketed, an underground site was identified, and a portable seismic system (PSS) installed. Based on the mining taking place in the area, eight different seismological regions were defined and summaries of mining and seismic information for each area were produced.

Seismic data from the selected site was processed and interpreted. Back analyses were carried out which attempted to compare modelling data and field seismic data for the actual bracket pillar layouts. Seismic moments calculated from seismograms for seismic events in the vicinity of actual geological structures and bracket pillars were compared with theoretical values produced under project GAP 223 from 2-D DIGS-based bracket pillar designs (i.e. the design charts) and from numerical modelling using MAP3D designs. Good agreement between seismic parameters from the field sites and those from equivalent numerical models was obtained. The results from three-dimensional modelling were compared to those from two-dimensional DIGS models, given that the available pillar design charts were obtained from DIGS two-dimensional bracket pillar models, under plain strain conditions. Differences were encountered and are discussed in this report.

In combining the seismic studies of the monitored features with modelling studies, reasonably good agreement was found between the occurrence of the larger seismic events and the positions of slip expected from the three-dimensional modelling. The larger events tended to locate along geological structure planes and, specifically, in the proximity of the expected slip positions on various discontinuities.

The cumulative seismic moment of field events and the cumulative seismic moment obtained from MAP3D modelling agreed well. It was evident that some geological features were better modelled by including the influences from the out-of-plane dimension. A wider scatter and, therefore, weaker association from the 2-D DIGS/design charts results relative to field data was observed. It was found that 2-D DIGS provides values of slip greater than those obtained by MAP3D, for all modelled geological features considered in the study.

CONCLUSION:

It was felt that the results of this study give grounds for confidence in the usefulness of the type of numerical modelling conducted for designing layouts that incorporate geological discontinuities. The bracket pillar design methodology, in particular, developed during the course of SIMRAC Project GAP 223 and refined by 3D modelling as presented in this report, would appear to be a valid tool when applied judiciously by qualified rock mechanics practitioners. Designs derived from design charts alone, however, would appear to be more conservative than designs based on 3D modelling that allow for explicit displacement along discontinuities and irregular geometries. On the strength of this finding.

Recommendation:

Therefore, it would be recommended that 3D modelling codes that allow for explicit displacement along discontinuities and irregular geometries (for example Map3D) should be used when modelling bracket pillar layouts.



21. Develop and implement preconditioning techniques to control face ejection rockbursts for safer mining in seismically hazardous areas

By Toper A.Z., Stewart R.D., Kullmann D.H., Grodner M., Lightfoot N., van Rensburg A.L.J., Longmore P.J., CSIR Mining Technology, Ref: GAP 336, 1998

SUMMARY:

This research report discusses the development of preconditioning techniques to control face bursts, for safer mining in seismically hazardous areas. Preconditioning involves regularly setting off carefully tailored blasts in the fractured rock immediately ahead of a mining face, so as to encourage slip on pre-existing fractures, in order to prevent the accumulation of high strain energy density in the rock mass.

Two different preconditioning techniques have been developed, namely face-perpendicular preconditioning (Figure 1) and faceparallel preconditioning (Figure 2). Although the techniques were developed at different project sites, there seems to be no fundamental rock mechanics reason why either method could not be applied in any given mining environment. However, due to practical limitations, the implementation of the face-parallel method can be difficult under many circumstances, so that its recommended use is generally limited to special areas. Face-perpendicular preconditioning can usually be readily integrated into an existing mining cycle, without any disruption to production. Guidelines for both techniques have been compiled, based on current knowledge.

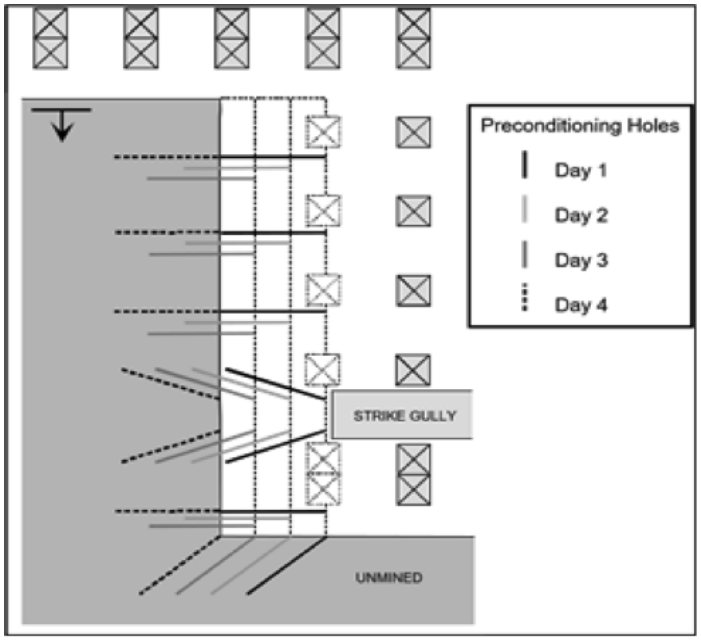


Figure 1: Face perpendicular preconditioning layout, with 3m long holes.

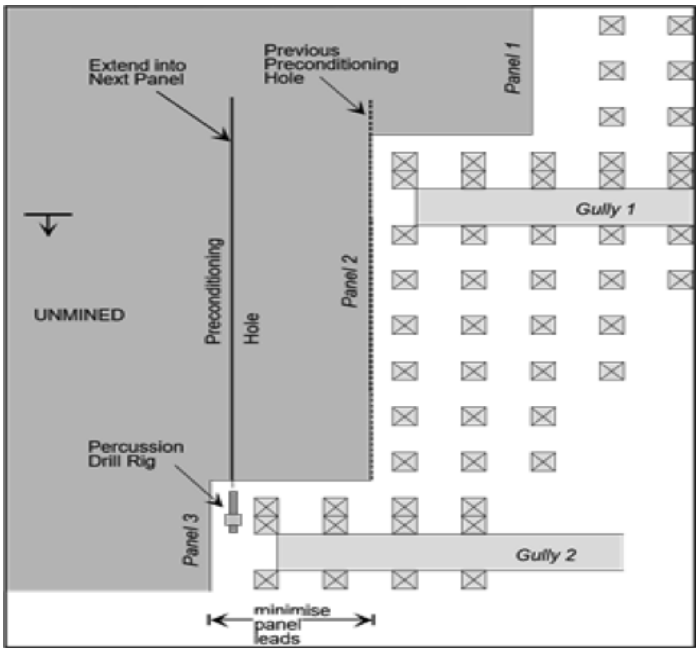


Figure 2: Face parallel preconditioning layout.

In order to assure successful implementation of the techniques into the mining environment, a structured implementation process has been developed. The education of all production personnel and the training of the stope crew are essential. The mine's safety and training departments' personnel should also be included in the process. In addition, the inclusion of preconditioning in the mine's Code of Practice is recommended.

CONCLUSION:

Both face-perpendicular and face-parallel preconditioning have prevented face bursting in areas to which they have been applied. In addition, minimal overall damage was observed in preconditioned panels following several large seismic events which occurred relatively close to the faces in some areas (compared to similarly exposed unpreconditioned panels).

Preconditioning has also provided some protection to face areas from distant events, through the capacity of the preconditioned ground to absorb energy.

An improvement in hangingwall stability and a 40-50% increase in face advance per production blast has also been noted in preconditioned areas. Face perpendicular preconditioning can be incorporated into a mining cycle without significant difficulty. Face perpendicular preconditioning is cost effective, indeed it may increase profitability under favourable circumstances.

Guidelines for the implementation of both preconditioning methods have been formulated. Preconditioning is an implementable tool that has achieved enthusiastic and widespread acceptance from the workforce on numerous mines in the industry. An education and training programme for use when introducing preconditioning to a mine has been formulated.

22. Evaluation of the mechanical engineering design of the rapid pack testing machine

By Kullmann D.H., CSIR Mining Technology, Ref: GAP 338, 1998

SUMMARY:

No large force, high velocity testing facility exists in Southern Africa to evaluate the dynamic behaviour of packs. Rock related accidents (especially those associated with rockbursts) will remain a major hazard in the South African mining industry. Having such a facility would enable an evaluation of new (and current) support systems by simulating rockburst conditions.

The pack testing machine would be a fundamental research tool which will provide information on the dynamic force displacement characteristics of timber, monolithic packs and grout based support systems. The dynamic force displacement characteristics of packs are generally unknown and the evaluation of the dynamic behaviour is essential for the understanding of the potential energy absorption of stope and gully support elements. This testing equipment would permit the current stope and gully support units (for both gold and platinum mines) to be evaluated under rapid loading conditions (ie. Rockburst conditions).
The following mechanical specifications were required from the pack tester

- 300 ton falling mass
- 2,1 m x 2,1 m x 2m packs
- 400 kN pre-load
- 3 m/s impact velocity
- 6000 kN maximum load

The project for the construction of the rapid pack testing machine has been "shelved" at this time for financial reasons. The cost of construction was last estimated at R 5 950 000 in September 1996.

Three primary areas were considered for review. These are a mechanical and engineering review of the machine itself, foundation requirements and environmental issues relating to possible high noise levels.

CONCLUSION:

An independent review of the rapid displacement pack testing machine and foundation have been carried out. A deficiency in the guide system for the falling mass has been identified and corrective actions have been proposed. All other aspects of the machine have been found to be acceptable and necessary.

The foundation must cater for soil settlement and water inflow, which can be expected. Ground vibrations can be dampened by excavating a vertical trench around the perimeter of the facility and filling it with polystyrene.

23. Testing of Tunnel Support - Dynamic Load Testing of Rock-bolt Elements to Provide Data for Safer Support Design

By Ortlepp W.D., Stacey T.R., SRK Consulting, Ref: GAP 423, 1998

SUMMARY:

A range of commonly used tendon types was subjected to dynamic tensile loading at strain rates comparable with those that are believed to occur during rockbursts. The tendon types tested included 16mm diameter rebar shepherds' crooks, 16mm diameter smooth bar shepherds' crooks, destranded hoist rope slings, 18mm diameter compact strand cable, 39mm diameter splitset bolts, and standard diameter Swellex rockbolts. The tendons were grouted into steel tubes, or installed in boreholes formed in simulated rock in steel tubes, the dimensions of the tubes being such that they gave confinement equivalent to that provided by the rock mass. The dynamic loading was imposed using a drop weight system impacting on a swing beam.

The testing showed that, during yield and failure of the tendons, energy was absorbed in rupture of the tendons, elongation of the tendons, and sliding on the interface between the tendon and the grout/borehole. Failure usually involved at least two of these forms of energy absorption. The results of the test programme are shown in the Table 1 below.

Test Series	Tendon Type	Rupture Energy kJ	Elongation Energy kJ/100mm	Sliding Energy kJ/100mm
1	Rebar 16mm	4,8 ± 0,7	-	-
2	Smooth bar 16mm	-	13,9	3,0 x l_{ei} - 0.15*
3	Vaal Reefs hoist rope strand	< 2,4	-	0,8 to 8,2
4	18mm compact strand cable	17,5 ± 2,0	-	-
5	39mm diameter splitsets	11,6 to 15	11,6 ± 1,3	3,0 to 6,3
6	Standard Swellex	4,6 ± 0,5	9,9	3,0 to 5,8

Table1: l_{ei} is initial embedded length in metres

CONCLUSION:

The research project allowed the creation of a realistic and controllable method of testing tendons dynamically. This offers an opportunity for improving the design methodology for the support of excavations under rockburst conditions.

It is now possible to use the same classical design rationale for the design of tunnel support as is employed for the design of structures in more rigorous and precise engineering disciplines. The design values, which are summarised in the Table above, are approximate and have still to be validated against experience. However, they provide a set of data where no data existed previously.



24. Review of Theories of Scaling and size effect In rock

By Sellers E.J., CSIR: Mining Technology, Ref: GAP 601, 1999

SUMMARY:

The experimentally observed size effect in rock strength is investigated by means of a literature survey. The aim is to understand the theories that have been developed to explain the effect, and how it has been included into numerical analysis procedures. There are many results for bend tests on concrete and uniaxial compression tests on rocks, but the search indicated that there was very little information on the size effect in other standard tests. The size effect is introduced when a specific, or representative, volume of material is considered in relation to some intrinsic parameter, such as flaw density, grain size, joint spacing or excavation dimension.

The size effect law, developed by Bazant, assumes that the crack process zone width remains constant and independent of the crack length. The size effect results from the selection of the fracture energy at large sizes as a constant. The theory applies to bending and can be applied to compression, if the correct failure mechanism is identified.

This theory is the one that is most often applied in numerical models. The Weibull yheory is based on the presence of pre-existing flaws, and it's applicability is questioned as it requires that the structure fail immediately on initiation of the weakest flaw. Other theories such as the stress gradient and fractal methods are discussed.

Size effects in jointed rocks are related to the roughness of the joint, relative to the sample size. Planar joints show little size effect as opposed to rough joints. Size effects can be incorporated in models by making the joint roughness coefficient depend on some length parameter.

Few numerical techniques are able to explicitly include a size effect. Most require that the parameters be selected for a specific size. A number of techniques may be modified so as to include a size effect. The literature study has highlighted that the understanding of the size effect is fundamental to the development of material models, to addressing issues of extrapolation of test data to field scale, and to the selection of representative material volumes.

CONCLUSION:

The study has raised important questions about the relationship between the behaviour of the intact material, and the intrinsic length scales such as joint spacing, fracture spacing, and structure size. The size effect links the material and the structural response and is the key to the understanding and definition of failure and of post-failure response.

These issues need to be solved in order to develop numerical representations of rock masses that are discontinuous at different scales.



25. Preliminary Investigation into the Zone of Support Influence and Stable Span between Support Units

By: Daehnke A., Le Bron, Van Zyl M., CSIR: Mining Technology, Ref: GAP 627, 1999

SUMMARY:

The current methodology to design stope support systems in South African gold and platinum mines is based on the tributary area concept. The discontinuous nature of the hangingwall is not adequately addressed and the mechanisms governing the support-rock mass interaction are poorly understood. In the past relatively little progress has been made in quantifying the effect of support spacing on the rock mass stability and, at present, it is the responsibility of the rock engineer to estimate support spacing based upon experience. In order to improve safety and continue mining at increasing depth, it is important to improve the understanding of support mechanisms, zones of support influence and the role of rock mass discontinuities.

The aim of this project is to formulate a basis for quantifying support mechanisms, and specifically to gain insights into the influence of rock discontinuities on stable spans. The work gives a preliminary analysis of the support interaction with a discontinuous rock mass, and the understanding thereof will ultimately result in an optimised support system design methodology.

It is emphasised that, due to the preliminary nature of the work, the support design methodology proposed here is untried in the field. Before its full acceptance, the methodology should be subjected to field trials, and assessed and calibrated under actual rockfall and rockburst conditions. It is highly recommended to conduct further research work focussing on in situ evaluations of the proposed design methodology. This is considered an essential extension to this work.

The project comprises the following four enabling outputs:

EO 1 Identify relevant rock mass parameters to characterise the hangingwall integrity in various geotechnical areas.

EO 2 Using analytical and numerical techniques, establish models to quantify the zone of support influence in a discontinuous rock mass.

EO 3 Establish a procedure to verify and calibrate theoretical models (describing the zone of support influence) by means of underground data.

EO 4 Characterise input parameters and design charts for support methodologies leading to improved support system design for quasi-static (rockfall) and dynamic (rockburst) conditions.

It is found that the critical rock mass parameters governing the rock mass stability are:

i. the number of discontinuities present in the rock mass,
ii. the orientation of the discontinuities
iii. the perpendicular spacing of the discontinuities, which includes the beam thickness of hangingwall,
iv. the interface properties of the discontinuities, and
v. the horizontal hangingwall clamping stresses.

Analytical and numerical models are used to quantify zones of support influence. It is shown that the maximum extent of the zone of support influence is dependent on the bedding plane friction angle and the fallout thickness. The zones of influence of the stope face and backfill follow analogously to the support zones of influence. Various procedures to verify and calibrate the theoretical models are described.

CONCLUSION:

This investigation has shown that mechanisms contributing towards hangingwall stability are:

i. the zones of support influence, and
ii. keyblock stability between adjacent support units. The proposed design methodology combines both support mechanisms, thus potentially providing an improved and further optimised design tool. It is recommended that the design methodology proposed here be evaluated underground and, if applicable, be incorporated in a design program such as SDA II for use in the industry.

26. Safe mining face advance and support installation practice in mechanical miner workings under different geotechnical conditions

By Canbulat I., van der Merwe J.N., CSIR: Mining Technology, Ref: COL609, 2000

SUMMARY:

The recently introduced restrictions on cut out distances sparked major debate in the coal mining industry. There were several reasons for the restrictions, roof stability being one of them. The others included methane and dust control and reducing the total area of unsupported roof in an underground section at any given time. This investigation covered the roof stability aspect only. While it thus did not address the overall issue, it attempted to quantify the effect of cut out distance on roof failure.

The references relevant to cut out distances cover remote-control operation of continuous mining machines, the control of dust and methane, elimination of frictional ignitions, effective ventilation methods, and human factors (worker/machine interaction). Cut out distance ground control aspects are also mentioned in relatively few references. The majority of research into the effects of extended cut out distances on ground control was conducted in the USA by the National Institute for Occupational Safety and Health (NIOSH) during the period 1993 to 1998 by Bauer (1998).

Underground experiments were conducted in 13 sites at six collieries in four seams. These sites were selected by conducting a detailed investigation into the identification of different geotechnical areas. A comprehensive database was used which was collected over 350 panels in South African collieries, and roof and discontinuity ratings were used to identify the geotechnical areas in South African collieries. The results showed that South African collieries may be divided into five groups with the majority of South African coal production originating from one in particular. Therefore, the test sites were concentrated in this area in order to obtain representative results.

A sonic probe extensometer was used to monitor the roof and support performances in the experiment sites. Two holes were drilled and instrumented with sonic probe anchors in each site. The first hole was drilled and instrumented at the face before any mining took place, and the second hole drilled in the middle of the cut out distance. In order to determine the effect of time on roof deformation, the sites were left for 48 hours unsupported, where appropriate. The underground results showed that the most critical parameter with respect to ground control aspects of extended cut out distance is the bord width, which determines the amount of deformation that will take place in the roof. This was also confirmed by the numerical modelling.

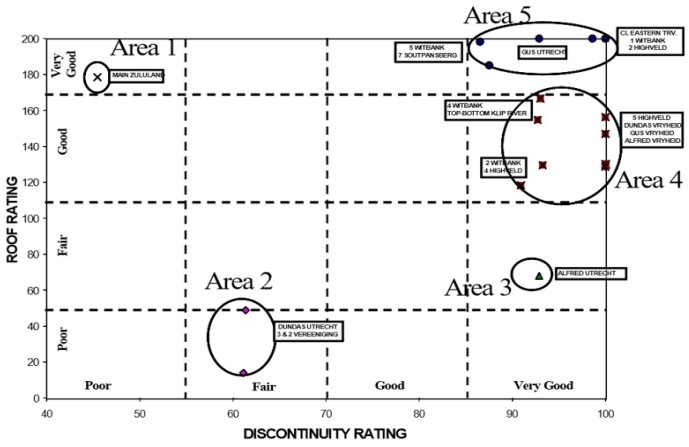


Figure 1: Geotechnical areas identified by discontinuity and roof ratings.

CONCLUSION:

The literature survey yielded little in the form of directly applicable research. It appeared that little work on determining roof failure per se as a function of cut out distances has been done elsewhere. The limitations on cut out distances were mainly due to other issues, like preventing underground workers being under unsupported roof and methane and dust control. Recent work done by researchers in the USA indicated that extending the cut out distance in the USA had little effect on roof stability.

The analyses of discontinuity and roof ratings showed that using one of these ratings alone is not sufficient to identify different geotechnical areas. However, a combination of the two ratings can be used to identify different geotechnical areas in South African seams. Based on the analysis of these two ratings, five different geotechnical areas which have similar roof and discontinuity conditions have been identified.

Underground measurements indicated that once the face had advanced to a distance equal to twice the bord width, there was insignificant additional roof deflection with further face advance. This conclusion was confirmed by numerical modelling and is in line with the analytical beam solutions. Therefore, if it is intended to limit roof deflection by restricting the cut out distance, the cut out distance would have to be limited to less than the bord width.

With regard to the effects of time on roof deflection, it could only be studied for the initial period of 48 hours following roof exposure. It was found that the roof continued to deflect during that period, but that the amounts of deflection were not significant.

27. Assessing and Evaluating Acoustic Techniques to Test Roof Conditions in Coal Mines in Combination with SIMRAC No.9

By Altounyan P., Clifford B. MacAndrew K.M., Rock Mechanics Technology Ltd., Ref: COL610, 2000

SUMMARY:

The objective of this project was to evaluate the use of an Acoustic Energy Meter (AEM) manufactured and supplied by Rock Mechanics Technology Ltd (RMT), to characterise roof stability in coal mines.

To this end AEM surveys were undertaken at eleven coal mines with varying roof lithologies, consisting of weak mudstones, sandstones and coal tops, and particularly targeted at sites where problems associated with “flashing” (slabbing) of the immediate roof occurred which had the potential to cause injury to mine personnel. The Acoustic Energy Meter developed for this project Figure 1 (SIMRAC AEM) was found to be simple to operate and able to detect a variety of roof anomalies in a wide range of roof lithologies.

The AEM was also found to be sensitive to a wide range of observed changes in roof conditions and was warmly and enthusiastically received by mine personnel.

During the period of this research a coal mine has sponsored the further development of the AEM to be a more practical tool for day to day use by mine personnel Figure 1. Further work is required to establish Standard Operation Procedures to survey known roof fall hazards.



Figure 1: AEM

CONCLUSION:

The AEM was found to be sensitive to observe changes in roof conditions and could provide useful information on:

- The effect of slips, in particular if there was a weakness present it could identify the weak side.
- The effect of weathering and time.
- The effectiveness of support on roof conditions.
- Areas of immediate roof bed separation and the extent to which these areas of weakness had propagated. See Figure 2.

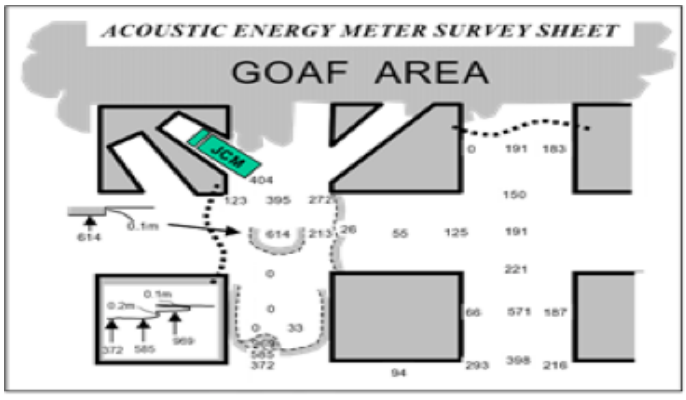


Figure 2: Meter Survey Sheet

28. Coal pillar design procedures

By York G., Canbulat I., Jack B.W., CSIR: Mining Technology, Ref: COL 337, 2000

SUMMARY:

Examination of collapsed pillar cases outside of the empirical limits of Salamon and Munro's in situ database has highlighted the need for additional parameters to be considered in the design of coal pillars. These include the influence of discontinuities, surrounding strata characteristics and the effects of deterioration and time.

Joint frequency, joint condition and joint orientation have an important effect on the strength of coal pillars. A new methodology has been developed to take these effects into account. An evaluation of the field data showed that pillars without any joint structures are likely to be about 10 per cent stronger than predicted by the empirical Salamon and Munro equation.

A simple and flexible technique to incorporate the effect of scaling, or time effects, into pillar design, has also been produced. Pillars may be designed by specifying a probability of survival for a given number of years. Alternatively, it might be required that the probability of survival for an indefinite period should not be less than a specific value. Real rates of scaling for different geotechnical areas are required for the technique to become practical.

An analysis of geotechnical data according to discontinuity and roof rating from a substantial database allowed the identification of eight distinct geotechnical areas in South African collieries.

An in situ experiment was attempted to determine the load/deformation characteristics of a coal pillar designed to crush. A pillar in the centre of a section was reduced in size to approximately 4 m x 5 m, giving it a width to height ratio of about 1.2, and the surrounding pillars extracted in a stooping operation.

The aim of this experiment was to load the pillar to failure and monitor the pillar and strata compression. This was achieved to an extent but the dynamic rate of loading was not captured by the monitoring system. If this exercise were to be repeated, a compromise between reading accuracy and continuous monitoring using a data logging system would have to be introduced to avoid losing very important, if not vital, information during the dynamic stages of the goafing operation.

The size effect at the in situ scale, if present at all, is negligible. Strengths of pillars in Salamon and Munro's back analysis of collapsed cases are shown to be independent of volume. A representative value of in situ critical rock mass strength, Θc, is 6.3 MPa. It has been shown that significant (in the statistical sense) differences in coal material strength detected in the laboratory can lead to a modification of Θc.

The effect of variations of the pillar / roof or floor contact conditions on the width to height ratio (w/h) effect on pillar strength have been catered for by means of a design chart. The 99 per cent confidence interval for the in situ contact friction angle is 21.0° to 24.8°, on a wide range of contact types, although the laboratory tests were restricted to samples from a small geographic area.

The results of these investigations have been assembled to form the basis of a new methodology for pillar design. A new pillar design flowchart has been produced. The influence of Θc, jointing, w/h and roof / floor contact conditions are each explicitly determined. These effects are then used to determine the overall pillar strength. This modular approach to pillar design allows scope for site or geotechnical area specific pillar design. However, this also requires site specific measurements of geotechnical parameters.

There are a number of issues not yet resolved. Among them are:

1. the proposed new design methodology requires underground verification
2. the effect of coal topping has not been determined
3. the factors that form part of the pillar design methodology have been assumed to act in series; this is not necessarily the case.

29. Investigation Into The Causes Of Falls Of Roof In South African Collieries

By Van Der Merwe J.N., Van Vuuren J.J., Butcher R., Canbulat I., CSIR Miningtek, ITASCA Africa (Pty) Ltd, SRK Inc, Ref: COL 613, 2000

SUMMARY:

The main objective of this research was to determine the causes of falls of roof in South African collieries. The research team was assembled from the CSIR Miningtek, ITASCA Africa (Pty) Ltd and Steffen, Robertson and Kirsten Consulting Inc to achieve a balanced view.

The south African Colliery Manager's Association and the members of the South African National Institute of Rock Engineering who are employed by collieries lent full support to the work. Sixteen major roof falls were investigated in detail and their causes established. This data was supplemented by mapping a large number of roof falls on four collieries in less detail. The total number of recorded roof falls was 182. The data bank simulated the data bank of roof falls that resulted in fatalities in terms of the total number of falls recorded and the thickness distribution of the collapses.

It was found that the causes of the falls differed for different thickness ranges of roof falls. The thin falls, classified as "Skin falls", which accounted for approximately 70% of all fatalities, were predominantly caused by ineffective joint support and excessive bolt spacings.

Ineffective or lacking joint support was found to be a dominant cause of roof falls for all thickness ranges, while the influence of excessive bolt spacings diminished with increasing fall thickness. However, it remained a significant cause of roof falls even for the major falls.

Burnt coal, dykes, bad mining practice, weathering and inferior materials were found to be minor causes, but nonetheless contributed to the falls. Horizontal stress was a minor cause, with increasing contribution for thicker falls. It was also found that indications of horizontal stress coincided with areas of reduced roof rock strength.

In general, roof falls occurred in areas where the roof rock was found to be less competent in terms of rock mass rating systems. It was concluded that the Coal Mine Roof Rating (CMRR) system developed by NIOSH in the USA held promise for application in South Africa and should be investigated further.

It was indicated that in South Africa, less bolts are installed for comparable roof conditions than in the USA and Australia. This supports the conclusion that excessive bolt spacing is a major cause of roof collapses in South Africa.

85. Recommendations related to the most appropriate techniques for training and education of workers in the Gold and Platinum industries

By VENTER A.N., ANVITECH. Ref: GAP 609B. 2000

SUMMARY:

The training landscape in South Africa is undergoing veritable metamorphoses as this report is compiled. The Skills Development Bill and its subsequent influence on the National Qualifications Framework (NQF) and the founding of the Mine Qualifications Authority (MQA) has brought a completely new perspective to most training management efforts undertaken by the mining industry. This outcomes based focus has the effect that all of the existing training material has to be reviewed, and re-purposed into modules that focus on definable outcomes. Most of the respondents to our questionnaire commented that this is a time consuming and enormous undertaking.

The MQA specification of Unit Standards will deal with the outcomes which workers must attain to be declared competent, in order to perform a certain task or procedure in the process of mining, it will not deal with the methods employed to attain that competence. The sister project to this project (GAP609B), GAP609A is focussed on this "outcomes based specification for strata control risks" goal.

Even amongst all this change in the outcomes based specification of training curriculum, the researcher has found some organisations that are actively seeking for better methods of delivery, and assessment of training efforts based on the new content structure defined by the MQA. This is a very encouraging sign, for one could expect that such a wide ranging transformation of curriculum could easily engage all the available resources of an organisation, especially subjected to such unrelenting pressures as experienced in the mining industry.

There is a wide spectrum of discovery represented in this report. It ranges from a traditional training approach, with a strong programmed instruction and evaluation structure, to a very dynamic process driven, systems view, employed by training process designers. Some training efforts are clearly not about training people, but rather finding ways to help them learn effectively and perform competently. In the area of safety, the competent performance of individuals, which is the desired outcome of all these efforts will save lives.

Unique solutions like Industrial Theatre and Performance Support Efforts in the work area are clear indications of lateral solutions found to a very challenging problem. Another interesting trend is the recent movement of training efforts to the context of work underground. Helping people be competent in very difficult circumstances is a feat greater than most realise.

There is strong theoretical evidence that a systems approach to learning support (training) has great value. Some of the respondents in the mining and industrial training community have a very clear vision of this approach, and are actively striving to change their current reality to an integrated, system focussed on the quality of human performance. This desired performance in the area of underground safety should impact on safety records in the medium to long term.

Prof. Mbigi and Dr. Mandela not only see the systems picture clearly, but also relate it to indigenous traditions and proven learning systems, which have survived culturally. This is an avenue which combined with the Cognitive Apprenticeship Model described by Sue Berryman are worthy of detailed exploration (Appendix A.4 and A.8).

Findings are derived from analysis of the questionnaire assessment (section 4.5) and the study of the literature survey (section 3) these are categorised as, new guiding ideas, innovations in infrastructure, theories, methods, tools.

30. Identify and Draft Outcomes-based Training Material in Strata Control for Underground Workers

By Johnson R.A., Hagan T.O., Squelch A.P., Jaku E., CSIR Mining Technology, Ref: GAP 609A, 2000

SUMMARY:

Assessments of accident records indicate the need for improved understanding and competence among underground production and service personnel with regard to strata control, and the associated risks, in gold and platinum mines. The underground personnel most directly affected by strata control risks, i.e. shift overseer level and lower, are categorised by the South African Qualifications Authority (SAQA) at job levels 1-4. Appropriate training courses and material are required in order to improve the competencies of employees with respect to the strata control risks to which they are exposed.

Rock engineering related training courses have to date been primarily aimed at rock engineering and senior mine personnel, however, such course material is inappropriate for the current objective. Course material of a simplified nature is required, which focuses on essential information about strata control risks and their amelioration. Training for SAQA levels 1-4 has traditionally focused on the "how" side of things, however it is believed that by also providing an understanding of the "why", such training can significantly reduce worker risk. The outcomes-based education and training (OBET) concept that is being implemented in South Africa by SAQA, and in the mining industry by the Mining Qualifications Authority (MQA), prescribes the framework and direction for the formulation of such relevant and competency-based training material. In this context, the understanding and application of the OBET methodology is required for the successful development of appropriate training material.

In order to formulate appropriate OBET course material, it was first necessary to determine which competencies are required by the underground personnel at SAQA levels 1-4 in relation to the strata control risks that they experience. These competencies were determined by interviewing a representative cross-section of industry, DME personnel and union/worker representatives. On the basis of an analysis of these competences, the identified strata control hazards were then drafted in OBET terminology and format for subsequent use by mine training departments, MQA, DME and unions.

In undertaking this study, input from all sources surveyed has indicated the importance of rock engineering training at these levels. Many individuals from all sectors interviewed have highlighted the potential impact of indicated training on safety. A strong degree of synergy was seen between union and industry representatives.

International input has indicated the efficacy of outcomes based training applied in mining in the United States and Australia albeit at a more advanced level. The outcomes based approach used by MQA/SAQA is seen as appropriate particularly in addressing the needs of the industry at this level.

Appropriate units of learning related to safety and rock engineering applicable to levels 1 to 4 have been drafted. These cover stope support, basic geology, tunnel support and elements within making safe procedures and in drilling of blast and support holes at levels 1 and 2. Similarly stope support, tunnel support, installing support, drilling and blasting and some topics in layouts, barring and geology are covered at levels 3 and 4. The more advanced levels build on competencies demonstrated at lower levels.

CONCLUSION:

A great diversity of detail in training requirements has been indicated in the results of this study. In order to handle this diversity, information is treated in a matrix format, which indicates training requirements specific to the needs of the individual to be assessed. At this level, these would be specific to the mine, mineral, depth, mining method, reef and local conditions.

31. An investigation of methods to combat mudrushes in diamond and base metal mines

By Butcher R., Joughin W., Stacey T.R., SRK Consulting, Ref: OTH 601, 2000

SUMMARY:

Mudrushes as a cause of accidents in diamond and other mines were identified for specific research, and this topic was one of the research projects gazetted for the 1998 SIMRAC research programme. The project commenced in April 1999, with the primary output being defined as follows:

- the assessment of mines at risk from mudrushes;
- criteria for mudrush prevention;
- a set of standards for combatting the mudrush hazard.

p - mud-forming material;

q - water;

r - disturbance, and

s - a discharge point through which the mud can enter drifts and tunnels.

The study has shown that mud can be formed internally in a muckpile or externally through the production of tailings or the production of backfill or both. Mud in boxholes and passes can derive from both of these sources, but also from the accumulation of fines produced during the overall mining process. The role of mud ingress via open bench drawpoints due to slope failure of weak geotechnical horizons is also described. In the case of caving operations, a correlation is given between mudrushes and the occurrence of isolated draw conditions. The presence of shale has been shown to contribute strongly to mudrushes.

The role of mine drainage in mudrush prevention has been shown from the Kimberley mines history. A theory of water retention in cave muckpiles has been proposed. However, the available information provided insufficient information to confirm this theory. It is possible that data from the De Beers mines might be able to provide substantiation in the future.

It has been proposed that mudrushes may be prevented by the implementation of the 3 D's principle - Distance, Drain, Draw. This principle serves to site mud-forming material away from mines, to prevent mud formation by prevention or reduction of water ingress, and to prevent the discharge of mud by ensuring interactive draw conditions. The implementation of this principle eliminates or reduces mudrush triggering by limiting the effects of disturbance and water.

In most cases where mudrushes have occurred, the mine either had a history of mudrushes or the hazard was suspected. In such cases it was found that a situation of isolated draw existed, or that little regard was given to mine drainage or to correct mine waste disposal. Since poor draw control and mine drainage play an important part in mudrushes, the appointment of a competent person to be responsible for such aspects is advocated. The need for properly designed mine drainage and draw control systems, which are regularly reviewed by external parties, is emphasised.

It is recommended that mines in which a significant mudrush hazard exists should be classified as mudrush prone mines and that this is enforced legally. Should a mine be classified as a mudrush prone mine, the following measures should be implemented:

o - the compilation of a mandatory code of practice for the prevention of mudrushes in terms of guidelines which are given in the report;

p - the appointment of a competent person to be responsible for mudrush control.

In mudrush prone mines, it is recommended that the competent person responsible for mudrush control be appointed by the manager in terms of the Code of Practice for the prevention of mudrushes. Owing to the complexity associated with mud ingress, it is recommended that, in large scale operations, this person be in possession of an appropriate qualification as determined by the Mining Qualifications Authority. It is recommended that an appropriate qualification would be a tertiary qualification in one of the following disciplines: mining engineering, geology, engineering geology, geotechnical engineering, and hydrogeology. A person who holds a Chamber of Mines rock mechanics certificate could also qualify for this

position, provided that this person has at least 2 years of experience in that particular mining environment. It is important that the appointed person has at least 4 years experience in the mining industry.

In the case of small scale operations, the appointed person must have extensive experience in that type of mining, and also have experience with draw control and mine drainage. For these smaller operations, a review must be carried out on a quarterly basis by an independent person/consultant in possession of the qualifications and experience as stated above.

All codes of practice for mudrush prone mines should be reviewed annually by an external party to ensure independent evaluation and updated as required. This party should be in possession of the experience and qualifications indicated above. It is important that the review consultant has experience with mine mudrush incidents, investigations or research projects.

The incorrect design and siting of tailings dams has been seen as a major cause of inrushes. Recommendations with regard to tailings disposal include the following. The disposal of tailings, slimes or any other waste which could behave as a fluid should not be conducted above current mining operations nor where, should the impoundment fail, there is a direct flow path of material to the underground workings. The disposal of tailings should be prohibited in areas that may undergo subsidence due to caving or failure of mine structures (eg failure of crown pillars). The disposal of tailings, slimes or any other waste that may behave as a fluid into open cast mines/open cuts which are situated above current operations should be forbidden. It is important that tailings and slimes dams and their foundations are correctly designed, and the importance of correct dam construction and management is also highlighted.

For underground mines that are extensions of open pit operations, the excavation of open pit bench slopes in mud-forming soils or weak soft rock is critical to the prevention of mudrushes. These slopes should be designed according to established current geotechnical best practice. The effects of variations in rainfall and groundwater regimes, mining sequences, cave break back zones and blasting practices should be taken into account for slope-induced mudrush prevention purposes.

Recommendations:

It is recommended that all mudrush incidents are included in the list of reportable incidents. They should be reported to the Department of Minerals and Energy and recorded in the SAMRASS database of accidents and incidents. The following information should be forwarded to the Department of Minerals and Energy:

- date and time of inrush;
- location of mud rush (location indicated on a plan);
- how far the mud pushed and the quantities discharged;
- percentage extraction for the discharge drift and the drawpoint;
- mine pumping and rainfall records;
- draw control records for caving mines.



32. Realistic Dynamic Stope Support Testing

By Ortlepp W.D., SRK Consulting, Ref: GAP 611, 2000

SUMMARY:

The development project GAP 611 comprised essentially four phases: design, construction, proof testing and a brief programme of testing of actual supports.

The project was driven by the perception that existing support testing equipment could provide only a quality control function. To stimulate development of innovative types of support and support accessories that would overcome some of the limitations of existing systems, required the development and construction of a completely different type of facility. The two most important differences compared with conventional testing procedures would be the provision of a discontinuous roof surface which would be subjected to dynamic loading.

Since nothing like it existed anywhere else the total design process had to start from scratch.

The performance parameters were set at providing an energy impulse of 300kJ with displacement rates of several metres per second.

The impulsive energy would be generated by dropping a mass of 10 000kg from a height of 3 metres to impact at a velocity of 7.7 m/s upon the target 'rock mass'.

The target mass was represented by a discontinuous but compact arrangement of steel-clad concrete blocks which would transmit and disperse the concentrated impact load downward on to a collapsible roof. Carefully designed, shaped concrete blocks would represent the fractured hangingwall layer of the stope, the stability of which would be determined largely by the support system under test. The other determinants of stability would be the energy of the impulse and the initial boundary conditions. The boundary conditions could be varied to some degree. Depending on these conditions and on the areal effectiveness of the support system, the hangingwall layer would have the potential to collapse between support units.

By August 1999, the design was completed in most details and construction of the reinforced concrete structure that constituted the stope was commenced. The steel superstructure was completed before the end of November but commissioning of the hoisting arrangements incurred significant delays.

Initial testing and proof-testing revealed the need for improvements in the techniques for managing the assembling and disassembling of the test structure. Improvements have been effected but it appears improbable that a complete test cycle can be completed in less than 5 days.

The three full tests on hydraulic props, timber elongates and linked timber elongate systems demonstrated that such testing can provide vital understanding of the behaviour of the support units under dynamic loading. The video-cam monitoring gives unique visual insight into the detailed response of support units to the dynamic forces that operate for a very brief period of time. This is obviously something that would be quite impossible to do in a real underground situation.

The linking of the elongates in the final test gave a most encouraging indication of how the facility can be used as a ' testing ground' for innovative developments. In particular, techniques for preventing collapses between support units and over the work area between the front line of support and the stope face can be explored.

There can be little doubt that the facility could also serve a vital training function for specialist support teams.

CONCLUSION:

The conviction is expressed that it is important that opinions and suggestions be solicited from the production people on the mines as well as from rock engineering staff in order that maximum benefit be obtained from the facility in the next few years.

33. Investigation of factors governing the stability/instability of stope panels in order to define a suitable design methodology for near surface and shallow mining operations

By Swart A.H., Stacey T.R., Wesseloo J., Joughin W.C., le Roux R., Walker D., Butcher R., SRK Consulting, Ref: OTH 501, 2000

SUMMARY:

Using the proposed design methodology, rock mechanics practitioners and mine planners should be able to identify and quantify the critical factors influencing the stability of stope panels. The critical factors should then be used as input to the design of stable stope panels that will provide the necessary safe environment for underground personnel working in stopes.

Although rock mass classification should form a fundamental part of the process of designing stable stope panels, particular emphasis should be placed on identifying the most likely failure planes and potential modes of failure. Appropriate analysis techniques should then be carried out in order to assess the stability of stope panels. It is for this reason that the research project has been extended to include structurally controlled analysis techniques such as beam and wedge analyses.

The design methodology proposed in this report identifies the factors governing the stability/instability of stope panels, predicts the response of a rock mass to mining under different geotechnical conditions and, identifies the significant hazards and assesses the significant risks and, if necessary, ascertains the support required to maintain stability during mining and as long as required after mining. It is recommended that the proposed design methodology is used during all stages of the mining process, from pre-feasibility to final design and implementation, and when compiling codes of practice to combat rockfall accidents in shallow mines. Each failure mode requires different analysis techniques in order to assess the stability of an excavation. For example, rock mass classification techniques could be used to assess rock mass stability in rock masses with no major weakness planes. Unfortunately, rock masses with no major weakness planes are rare and failure normally occurs because of unfavourable geological structures. Beam and wedge analysis programs should be used to assess the stability of structurally controlled stope panels.
The proposed design methodology for stable stope panels provides procedural guidance to the design of stope spans subject to all failure mechanisms. It consists of the following main components:

- estimation of in situ rock mass properties;
- identification of potential failure modes;
- stability analyses;
- risk assessment;
- geometric optimisation;
- determination of support requirements;
- support design;
- evaluation of design;
- monitoring.

From the above, it is concluded that the design of stable stope panels should be a process of defining the means of creating stable stope panels for the safety of underground workers and optimum extraction of the orebody. Therefore, a method is required whereby all rock properties, their variability, and an understanding of all rock mechanisms affecting the stability of stope spans are used as a fundamental base. A procedure for identifying the mechanisms and rock properties relevant to the specific problem is then required. In this way, existing knowledge should be used in an optimal way to design site specific stable stope spans.

Hence, it is proposed that the design methodology for stable stope panels is a process consisting of the following steps:

1. Define objective.
2. Rock mass characterisation.
3. Estimation of in situ rock mass properties.
4. Consider an “ideal” stope panel.
5. Identification of potential failure modes.
6. Stability analyses.
7. Identify all significant hazards and assess the significant risks.
8. Geometric optimisation.
9. Determination of support requirements.
10. Design of support.
11. Evaluation
12. Recommendation and implementation
13. Monitoring of excavation and support behaviour to validate design and permit modifications.

Design of stable stope spans should be iterative process, initially looping back from step 8 to step 6. After implementation, stable stope spans should be monitored and rock mass characterisation should continue. Rock mass properties should be adjusted and, if necessary, the design process should be repeated.

CONCLUSION:

Although most FOG incidents/accidents are associated with failure along geological structures, most mines do not use any design methodology based on structural analysis. In a few cases, complicated numerical analysis programs are used on an ad hoc basis to assess structurally controlled panel stability. In most cases, panel lengths being used are based on local experience and equipment requirements.

Valuable information regarding panel stability are often lost because FOG incidents are not investigated, recorded or back-analysed.

Beam stability analyses are applicable to several mines and should be considered as one of the potential failure modes.

Rock mass classification forms an integral part of stope panel design. Although it cannot be used directly for stability analysis purposes, it should be used to estimate rock mass properties required for analytical designs.

More than one rock mass classification method should be used to assess ground conditions. More than one analytical design approach should be used to carry out stability analyses. A systematic design approach should be followed to design stable stope panels.

Analytical methods and failure criteria should be used that can model the anticipated or identified failure mechanism and mode of failure most accurately. All failure modes should be considered during stope panel design. This also applies to FOG investigations.

34. Influence of surface topography on the loading of pillar workings in near surface and shallow mines

By Swart A.H., Keyter G.J., Wesseloo J., Stacey T.R., Joughin W.C., SRK Consulting, Ref: OTH 501, 2000

SUMMARY:

Mining often takes place in areas with steep and variable surface topography. Variable surface topography could be due to natural features such as valleys and hills, or man-made features such as excavations caused by surface mining operations or surcharging by dumping of spoil material, or tailings from the metallurgical plant. The research carried out for this project shows that the stability of underground excavations could be adversely affected by the proximity of such topographical features, especially in near surface mining operations of less than 100 m below surface.

The main objective of this research is to quantify the influence of surface topography on the stability of pillars and to describe a design methodology for pillars in areas of variable surface topography. Consideration of the critical factors identified in the study will improve the design of stable pillar systems, which are required to alleviate the hazard of catastrophic pillar collapse in areas under influence of varying surface topography.

The conventional method of calculating factors of safety for pillars represents the factor of safety for the pillar in compression only, and implicitly assumes that the shear stresses acting on a pillar are negligible. However, due to the proximity or orientation of stopes relative to topographical features, significant rotation of the principal stresses around pillars may occur. As a result, both the normal and shear stresses acting on a pillar have to be taken into account to allow rigorous design of the pillar dimensions and to determine the allowable extraction ratio.

It is therefore necessary to calculate the overall factor of safety for the pillar in terms of the proximity of the Mohr circle, representing the stress condition in the pillar, to the actual failure envelope for the pillar. As a result, the overall factor of safety for the pillar will always be less than the factor of safety for the pillar in compression, especially where significant shear stresses are acting on the pillars as a result of the influence of surface topography, principal stress orientation and/or the actual stope orientation.

In order to overcome the limitations of the standard approach to pillar design, more rigorous approaches using Mohr-Coulomb and Hoek-Brown failure criteria were developed as part of the research. The basic steps of the proposed design methodology, which is described in Section 6 of the report, are as follows:

1. Assess the rock mass conditions and collect relevant geotechnical data.
2. Consider the structural complexity of the surrounding rock mass. If the surrounding rock mass is structurally too complex, a site specific design will be required.
3. Determine a rock mass rating for the rock mass and orebody.
4. Calculate the strength and deformational parameters for the rock mass and orebody.
5. Establish a typical cross-section through the proposed mine, as well as the horizontal to vertical stress ratio.
6. Use typical design charts or carry out numerical modelling to determine the major and minor principal stresses before mining, as well as the major principal stress orientation with the vertical at the point of interest.
7. Calculate or select a maximum allowable stope span, Lw, and pillar height, H.
8. For the type of pillar required, select or calculate a suitable stress to strength ratio, SSR (Table 6.1 or Equations 25 or 33).
9. Calculate the maximum allowable extraction ratio, e, by solving Equations 25 or 33 numerically, or by calculating the SSR for various e values.
10. If the mining operation is financially viable at the calculated extraction ratio, calculate the corresponding effective pillar width, Weff from Equation 17.
11. Assess the potential for bearing failure of the hanging- and footwall using Equation 34.

The proposed design methodology has been validated by back analysing a regional pillar collapse in a near surface mine under influence of surface topography, as well as stable areas down dip of the collapsed area. It is calculated that, using the standard method of analysis based on uniaxial loading only, 89% extraction would have been allowed under the conditions found before the collapse. This compares with a maximum allowable extraction ration of 80% using the Hoek-Brown failure criterion. The corresponding factors of safety are 0,59 and 1,33 respectively.

Thus, it is clear that an overestimation of the allowable extraction ratio is obtained if the shear stresses due to topographical effects are not taken into account. The amount by which the allowable extraction ratio is overestimated depends on the pillar orientation with respect to the major principal stress as well as the magnitude of the minor and major principal stresses. The higher the shear stress after extraction, the less conservative the results will be that are obtained using the simplified method of analysis. Overestimating the allowable extraction ratio could result in regional pillar collapses and safety hazards.

CONCLUSION:

The procedure for the design of pillars in areas of variable topography caused by natural features or man-made excavations is complex. It requires stress analyses of the topographical feature to determine the major and minor principal stresses and the orientation of the major principal stress with the vertical. Also, calculating the maximum allowable extraction ratio requires special mathematical skills. It is recommended that these problems be overcome by:

- compiling a series of diagrams for typical topographical features, indicating the major and minor principal stresses and the orientation of the major principal stress with the vertical;
- compiling a simple and user friendly computer program for calculating the maximum allowable extraction ratio.

The conventional method of calculating factors of safety for pillars in inclined orebodies represents the factor of safety for pillars in compression only, and implicitly assumes that the shear stresses acting on a pillar are negligible. However, due to the orientation of inclined pillars, significant rotation of the principal stresses around the pillar may occur. As a result, both the normal and shear stresses acting on pillars have to be taken into consideration. It is recommended that the proposed design methodology for pillars under influence of natural topographical features or man-made excavations be extended to include pillars in inclined orebodies without influence from surface topography as well.

Recommendation:

It is recommended that this study be extended to include the effects of underground mining on the stability of surface excavations such as open pits.

35. Stability and support requirements for stope backs in the shallow depth mining of steeply dipping vein/tabular deposits

By Johnson R., Quaye G.B. Roberts, CSIR Mining Technology, Ref: OTH 603, 2000

SUMMARY:

A number of smaller mines in South Africa have orebodies, which are steeply dipping vein or tabular deposits. A variety of mining methods are used to extract these orebodies but many have common stability problems with stope backs and crown or sill pillars. Many of the mines have little rock engineering expertise available to address these problems. This investigation has identified mines which fall within the defined "steeply dipping vein or tabular deposit" as those which have orebodies that dip in excess of 40o and have a width of less than 8 m. During the course of the investigation it was apparent that in some cases the use of an inappropriate mining method was the main cause of the rock engineering hazards. Other rock engineering hazards that were identified were:

- fallout of large structurally controlled blocks crown/ sill pillar failure use of poor, inadequate or no support failure to modify mine layout as spans, depth or conditions change
- inappropriately located haulages with respect to the ore body

A detailed literature review was undertaken in order to identify world best practice covering the scope of the project and this was reviewed by Professor Peter Lilly from The Western Australian School of Mines and Dr. Nick Barton of the Norwegian Geotechnical Institute. Following the recognition that inappropriate mining methods have been used on some of these South African orebodies it was considered that a methodology for the selection of a suitable mining method or for testing if an existing mining method was appropriate, needed to be set out in this project.

Guidelines for the support of stope backs and sidewalls have been addressed by the methodology defined by the empirical stability graph method. A second alternative approach, which is semi-empirical, has also been described.

A method of designing crown and sill pillars to suit the prevailing conditions is described. The support of these pillars draws upon the same support methodology used to support stope backs.

Even with regular systematic support installed in stope backs and pillars, structural features can require additional support. These would typically be large structurally controlled wedges and an approach to identifying such unstable wedges is described as is a method of determining the correct support requirements to stabilise such wedges.

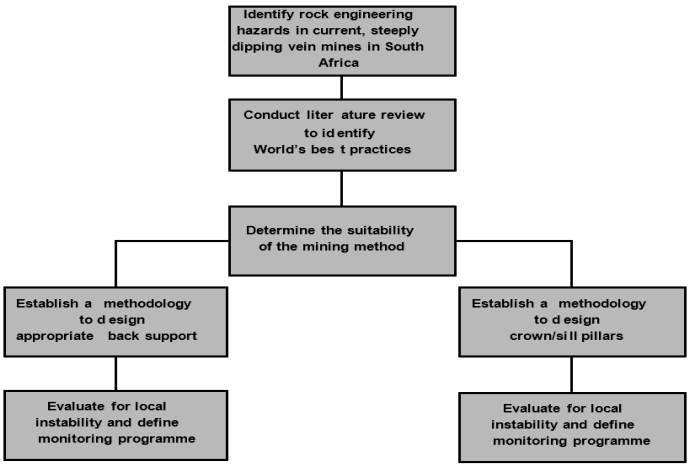


Figure 1: Research methodology adopted in the project

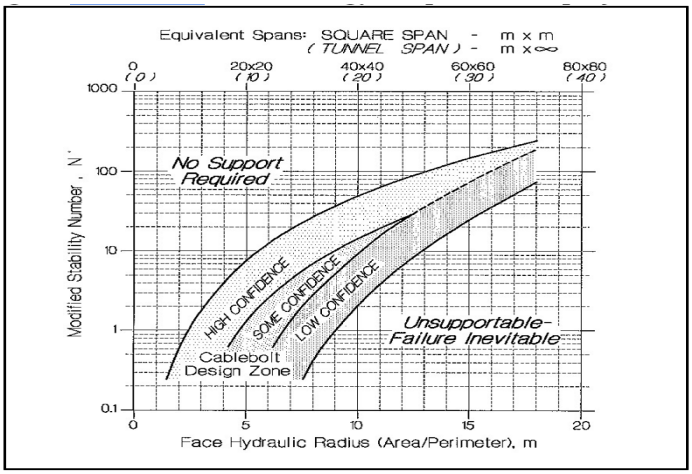


Figure 2: Stability graph showing cablebolt design zone for

CONCLUSION:

In view of the lack of rock engineering expertise on these mines much of the content of this report has been summarised in a small, easy to use handbook, which will have a role in assisting the mine official with respect to the support of stopes, the design of pillars and the identification of unstable ground. Importantly however it is emphasised that in some situations specialist expertise will be required.

36. Current practice and guidelines for the safe design of water barrier pillars

By Rangasamy T., Leach A.R., van Vuuren J.J., Cook A.P. Brummer R., Itasca Africa (Pty) Ltd., Ref COL 702, 2001

SUMMARY:

Naturally occurring and stored underground water poses a threat to the safety of underground employees on coal mines. Often current workings are sited close to an underground reservoir, in particular old abandoned workings that have been flooded. Rather than drain these reservoirs, it is normal practice to leave a barrier pillar between the new mining operations and the reservoir. At present, the barrier pillars that bound flooded underground compartments are designed primarily to ensure mechanical stability although their 'life of mine' function is to additionally prevent the inrush of water into current workings. No adequate local or international design addresses the hydraulic function of barrier pillars.

Underground instrumentation of roadways adjacent to barrier pillars was conducted to ascertain the relationship between compartment water head, barrier pillar width and flow rates for combinations of roof bound, coal bound and floor bound flow. The results obtained from the instrumentation sites were used to calibrate numerical models (UDEC) used to determine the full inter-relationship between pillar size, pillar loading, rock mass geological condition, permeability, water pressure (head) and hence the degree of leakage through barrier pillars. A survey of South African collieries revealed that water leakage associated with barrier pillars can be classed into seven predominant geotechnical flow categories.

Through extensive numerical modelling and case history matching, barrier pillar design charts (hydraulic) for the seven predominant geotechnical flow regimes have been established. These charts will enable an engineering estimate to be made of either the barrier pillar width, tolerable water compartment head or roadway pumping rate required to manage water ingress into workings. An easy to use guidebook containing a series of hydraulic design charts have been developed for coal measure strata to aid anyone involved in the management of water on South African underground coal mines.

37. Suitable long tendon technologies and practices

By Altounyan P., Clifford B., Rock Mechanics Technology Limited , Ref: COL 704, 2001

SUMMARY:

The use of tendons is the primary means of support in underground collieries. Therefore, in order to introduce safer and more effective tendon support systems, the performance of tendon reinforcement has been assessed.

In the main South African Collieries have relatively stable roofs compared to some of the other major coal mining countries. This is documented most clearly by the large cut-out distances that can be tolerated in many situations underground.

However, there are situations in most, if not all South African coal mines where long tendon support is necessary. Long tendon support systems currently in use in South Africa were reviewed and a number of shortcomings identified.

The objective of this project was to identify safer and more effective long tendon support systems suitable in South African coalmines. Appropriate test procedures for the evaluation of the performance of long tendon systems were also determined and are detailed in this report.

Long tendon systems currently in use in South African coal mines were reviewed and a number of shortcomings identified. Methods of improving the performance of these systems, bearing in mind the roof failure mechanisms identified in South African coal mines, are also suggested.

Internationally, a wide range of long tendon support types are in use. These systems were reviewed for their suitability for South African coal mine conditions and the selected system assessed through laboratory

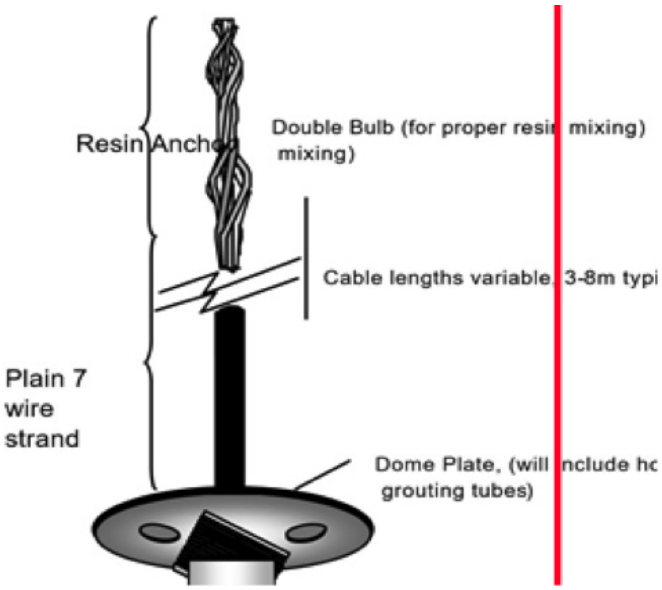
performance testing and compared with the existing South African cable anchor system.

Laboratory test results indicate that, in general, there are two long tendon reinforcement methods in use worldwide which offer improved performance over the cable anchor system currently used in South African collieries. These are the bulbed or birdcaged cable bolt system, installed in high strength grout and the flexible bolt system which adopts high strength capsule resins and small diameter boreholes.

Improvements can be made to the performance of the existing cable anchor system but this will involve changes to the current installation practice and/or to the design of the cable anchors themselves.

Underground field trials were established to compare existing and proposed systems in terms of ease of installation, quality control and system performance. The system chosen for the underground trials was the flexible bolt, a stranded tendon installed like a rockbolt through capsule resin in small diameter boreholes. Previous application of flexible bolts have shown system performance to be similar to fully encapsulated resin bonded rockbolts offering a high strength and high stiffness system.

Underground trials highlighted the benefits of the flexible bolts in terms of support installation and support effectiveness. Flexible bolts proved quicker and easier to install than existing cable anchors and provide better control over the installation quality. The flexible bolts are installed in smaller diameter holes than cable anchors, hence reducing drilling time, and, as they are installed in capsule resin, they can be installed during the normal drivage support cycle, hence speeding up drivage through dykes and difficult areas. Existing cable anchors have to be installed separately and slow down developments considerably. In addition, the flexible bolts provide improved support effectiveness as they provide a stiffer support reaction.



CONCLUSION:

It was concluded that the adoption of the flexible bolt system in South African coal mines could result in a significant improvement in support safety and effectiveness. Guidelines for the installation of flexible bolt systems and for improved quality control of existing systems were also drafted under this project.

38. Develop international links for co-operative research in coal mining with special emphasis on interaction with international experts for developing industry guidelines for the design of pillars in coal mines

By Van Der Merwe J.N., Research Enterprises At UP , Ref: COL 815, 2001

SUMMARY:

This was a unique SIMRAC project in the sense that it did not address an expressed technical need explicitly. Rather, it afforded the opportunity for scientific discussion and exchange of ideas that could result in the identification of future research needs and enhance the general level of research execution in South Africa.

The bulk of the technical issues that were discussed are already in the public domain and as such accessible through literature searches. However, the possibility to incorporate highwall design results into underground production bord and pillar design methodology, would probably not have arisen had it not been for attendance at the workshop. Furthermore, it was of value to discuss technical issues in a forum without the constraints that are normally present at open meetings. This was an occasion where time could be spent to discuss the finer details that are avoided at open meetings.

The question arises whether it is worth while to fund meetings that do not address identified and active SIMRAC projects. This question can only have a positive answer if SIMRAC approves funding to pursue ideas that were gained at the workshop. At the moment, for instance, there is no SIMRAC project on the topic of coal pillar strength and if no projects to investigate this issue further are approved, the knowledge gained will be of little benefit.

The original intention with this project was to sponsor a number of non-researchers to attend the workshop. In this particular case, that would have been of little benefit as no definite knowledge in a practically usable form was released. The benefit was restricted to research as opposed to application, and in that sense it was beneficial to sponsor a researcher to attend. However, there could well be other events in the future with a different slant, that should also be attended by non-researchers.

39. Pre-feasibility investigation to provide an early warning of roof falls prior to support installation

By Kotze G., du Plessis J.A., ISS International Ltd, Ref: COL 703, 2001

SUMMARY:

ISS International was contracted by SIMRAC to conduct a pre-feasibility investigation regarding the association of electromagnetic emissions and acoustic/ultrasonic emissions with rock failure. The presence of such emissions as a result of material or rock failure could have tremendous potential for the development of techniques to provide early warnings of rock failure. In the mining industry, falls-of-ground do occur frequently because of roof failure in underground excavations, causing safety and production problems.

In-situ or laboratory measurements of such emissions from rock failure are difficult to conduct because of difficulties in eliminating external emissions from interfering in the experiment. Measurements were conducted at two sites in underground coal mines where seismic activity, electromagnetic, and acoustic/ultrasonic emissions were monitored for a period in a seismically active area. Electromagnetic emissions were measured using an ISS International multi-seismometer with a Rohde & Schwarz professional short wave receiver. Electromagnetic frequencies between 190 kHz and 30 MHz were monitored. Acoustic/Ultrasonic emissions were monitored with an ISS International multi-seismometer with 10 kHz accelerometer, an off-the-shelf microphone, and a custom built ultrasonic detector.

Results from the electromagnetic emission test site showed that some electromagnetic emissions were measured in the same time window in which seismic events were also measured. The results do suggest that the observed electromagnetic anomalies may be electromagnetic emissions that are emitted during roof failure. Measurement of background electromagnetic radiation also suggests that the electromagnetic anomalies are not considered to be artificial or background electromagnetic emissions. However, the results do not prove without any ambiguity that the observed electromagnetic anomalies can in fact be associated with roof failures. In order to make definite quantitative conclusions, sufficient data need to be collected on which a quantitative statistical analysis needs to be performed and equipment and data processing algorithms designed to minimise ambient noise levels.

Rock failure processes fall in the category of physical systems that exhibit the phenomenon of Self-Organised Criticality (SOC, please refer to literature study for more details). Such systems are characterised by power laws e.g. the Guthenburg-Richter cumulative number vs magnitude type distribution in mining and earthquake seismology. Since magnitude can be translated to corner frequency this power law also implies that a power law governs the cumulative number of acoustic and ultrasonic emissions versus frequency (refer to literature study). This power law (figure 2.3.3) behaviour implies a large number of high frequency (ultrasonic) acoustic emissions are present while a smaller number of low frequency acoustic emissions are associated with a process such as roof failure.

The acoustic/ultrasonic trial, however, did not measure any ultrasonic emissions associated with seismic roof activity. It is expected from the SOC theory that ultrasonic emissions are present, although their amplitudes were too low to be measured above the background noise level in this pre-feasibility study. (Alternatively, the background noise level that comprises noise from the underground environment, equipment and internal instrument noise were too high to enable measurement of the ultrasonic emissions.) No definite conclusions can be drawn at this stage whether or not acoustic/ultrasonic emissions can be measured with a system during or prior to rock failure. However, results suggest that improved system design could lower instrument noise levels resulting in reduction of total noise levels.

Recommendations

It is proposed that further measurements be conducted using a multi-array accelerometer/acoustic sensor based continuous seismic monitoring system (sensors coupled to rock surface and into holes) to quantify seismic events, electromagnetic, and acoustic/ultrasonic emissions in terms of location, amplitude, and source characteristics. Continuous monitoring technology (Standalone QS developed by ISS International) will provide bulk information during shorter monitoring periods and could improve the understanding of precursors. This access to bulk data can assist to further investigate the potential for using electromagnetic/acoustic/ultrasonic emissions as early warnings to roof failure. (Bulk data often reflects more about the processes involved than newly developed theories based on smaller amounts of data.) The access to a larger pool of data could also allow confirmation of the power-law behaviour of seismic/acoustic/electromagnetic events that are caused by failure of the roof of the excavations (SOC systems). Work that is funded internally is currently in progress at a site in a deep level gold mine to demonstrate the feasibility of continuous monitoring.

It is also proposed that further funding is made available to develop a low noise, sensitive ultrasonic receiver to be able to determine the presence of low intensity ultrasonic emissions.

40. Implementation of state-of-art mining knowledge and technologies in design and operation of a safe and efficient deep gold mine stope for 21st Century

By van der Merwe, J. N., Wojno, L. and Toper, A. Z., CSIR Division of Mining Technology, Ref: GAP 712, 2001

SUMMARY:

Several new mining, rock engineering and environmental control technologies aimed at improving safety have been developed in the past decade or so. Since several of these developments have either not been implemented properly or only implemented at isolated instances, they have not led to any significant improvements in safety and health and efficiency on the mines.

The purpose of this project was to evaluate the suitability of currently available and potential mining technologies for integration into a mining system that would have superior safety, health and efficiency. The ultimate aim of the investigation is to implement and evaluate the derived mining system in a trial stope which could evolve into a benchmark stope for the most common geotechnical conditions in the Witwatersrand goldfields and for certain Bushveld mines. The underground evaluation of the proposed system is envisaged as a follow-up project

To design the conceptual, integrated mining system, a broad range of expertise was required. Individuals or groups who could provide this expertise were identified and their views and recommendations are included in this report.

The optimal mining system was designed principally for the following conditions: mining depth - intermediate to deep level, thus catering for moderate to severe fracturing and potential seismicity; stoping width - from 1.0 m to 1.7 m; and faults in the stope face with throws up to 2.5 m and rolls where the slope angle changes by up to 20.

The performance targets set for the system are a reduction in stope accidents of at least 50 % compared to the industry average, greater than 15 m face advance per month, dilution and gold losses reduced by 15 % compared to adjacent stoping, consistent provision of recommended environmental conditions and finally cost effectiveness.

The main areas of concern with current mining and which are attempted to be addressed by innovative combinations of appropriate technologies in this project are the following:

Less than adequate support in the immediate face area particularly during the cleaning shift and making safe operations,

- Inadequate area coverage by support in the working area of panels and over gullies,
- Low cleaning rates in long strike gullies leading to accumulation of rock with resultant difficulties in access for men and materials and restricted ventilation flow,
- Timeous supply of materials to stope face area not ensured,
- No separation of water from rock in the cleaning process,
- Congestion in cross-cuts, and
- Recommended environmental conditions not ensured.

Other areas of concern where no reliable improved technologies were found are barring and face drilling. Mine wide issues such as training, appropriate reward schemes, organizational structures are not addressed in this project but preliminary investigations would be carried out in any implementation project.

The proposed conceptual mining system comprises the following equipment and technologies:

- Water powered, hand-held rock drills.
- Reliable sequential detonation blasting system – final decision of what system to be used will rely on latest information.
- Rapid yield hydraulic props with 0.8 m load spreaders. Lines of props aligned 700 to face to enhance drilling accuracy and limit interference with drilling.
- Safety nets suspended on the props where necessary.
- Roof bolts installed less than 0.5 m from face.
- Yielding mechanical props as temporary support during roof bolting.

- Diagonal blast barricades to facilitate water jetting.
- Fines barricade.
- Routine face-normal preconditioning to prevent face bursting and enhance advance per blast.
- Water jet assisted scraper face cleaning. Assists with preliminary barring and dust suppression.
- Gully shoulder support – constant yield force, low mass reinforced concrete packs or Fill-packs in backfilled stopes.
- Gully hangingwall support – roof bolts, shepherd's crooks or yielding tendons (depending on conditions) with lacing over membrane support to provide good areal coverage.
- Strike gully cleaning – scrapers, double-separated-scraper system for long pulls.
- Centre-gully – up dip rock transport by means of continuous scraper to a single, long box-hole with adequate bunkerage. Water separated from rock.
- Separate men and material access. Mono-rope from x/cut storage bays – delivering to top gully/face intersection in overhand stope.
- Environmental control systems and equipment appropriate for local conditions.
- Initial geological and rock engineering assessment of ground control district. Geophysical probing of ground conditions ahead of the stope face where deemed necessary.

Developing technologies that could be added to the system once they have been finalized include the support system initiated under GAP708, mechanised rock breaking, the quiet rock drills and the plasma hole maker if permission is granted by the principals.

The above mining system remains a mere concept until implemented and evaluated underground. As part of this project consideration was given to the practicality and approximate costs of running a field trial of the proposed system. Three crucial issues were identified that need to be resolved before such a trial should proceed.

The first is to identify a willing mine that would enthusiastically champion the trial. Following discussions with Anglogold, it was concluded not to combine this proposed trial with that of Anglogold, as those results will not necessarily be made public to protect Anglogold's competitive position. Enquiries at Mine Manager's level indicated support for the project pending Head Office approval.

The second is to recruit or second necessary staff to the project. CSIR: Miningtek has the experienced researchers to design, monitor and evaluate the trial but no longer has sufficient production personnel nor technologists to run and service the site; suitable people to fill these positions would have to be either supplied by the mine or outsourced.

Thirdly, the cost, funding and revenue associated with the project need to be addressed by consultation and negotiation between the three parties concerned.

CONCLUSION:

It is believed that none of the above problems are insurmountable and that an initial small project could be set up to finalize tripartite agreements. The successful outcome of the trial, benchmark stope could be of immense benefit to the industry in terms of technology transfer leading to significant improvements in safety, health and productivity.



41. Influence of regional support systems (pillars and backfill) on local areas and internal support requirements adjacent to that regional support

By Squelch A.P., Milev A.M., Acheampong E., Dlokweni T., van Rensburg A.J., Watson B.P., CSIR Miningtek, Ref: GAP 615, 2001

SUMMARY:

The general influence of regional support on local stope conditions is not always well understood and a number of anomalous conditions, both positive and negative, have been reported in the vicinity of regional supports. Extensive ground motion and closure monitoring, studies of hangingwall conditions and numerical modelling analyses were carried out for a selection of different regional support (pillar and backfill) layout and reef horizon combinations. Although evidence arose from this work of differences in conditions possibly requiring different support design, it was felt that no absolute values could be provided for relevant design parameters. Rather, it was decided that a modified support design methodology be proposed that accounts for the additional influencing factors pertaining to the zone adjacent to regional support (Figure below).

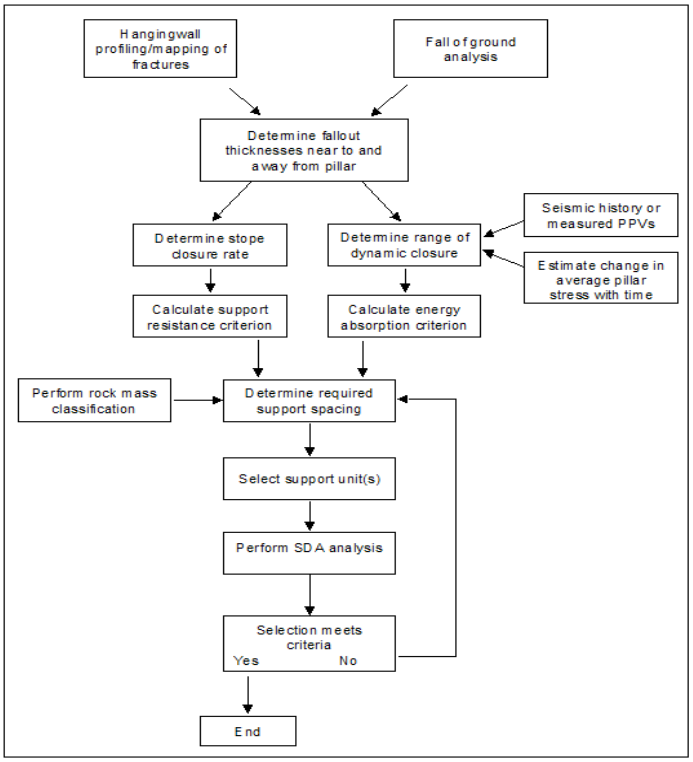


Figure 1: Methodology for design of local support in vicinity of regional support

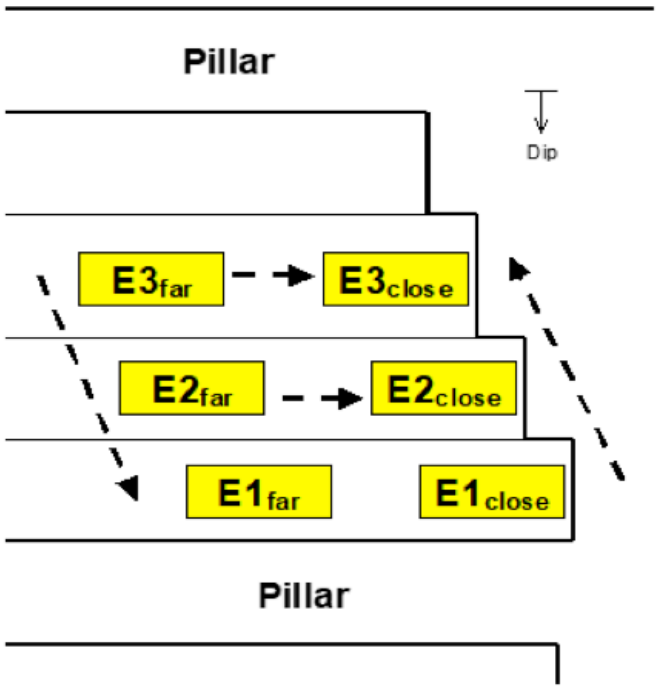


Figure 2: Schematic of relative magnitude (dashed arrow = increase) of PPVs as a function of proximity to face and strike stabilising pillar at Tau Tona mine

Principle findings

- Peak particle velocities (PPVs) increase with increasing perpendicular distance from the pillar (cf. Figure 2).
- PPVs recorded close to the pillar increase with the increase in pillar stress resulting from the face advance (cf. Figure 2).
- The reduction in PPVs from the face to the back area varies for different geotechnical areas.
- Generally, well placed backfill improves conditions in face areas if it is kept close to the face and conventionally designed working area support that fits in well with the backfilling/mining cycle is implemented.

CONCLUSION:

Dynamic closure, resulting from events generating PPVs, is reduced in the vicinity of regional support (i.e. pillars and well consolidated backfill).

Support units in the vicinity of strike stabilising pillars will be required to withstand less dynamic and quasi-static closure than units in panels further away from the pillars.

A greater relative increase in PPVs will be encountered in the vicinity of strike pillars than in areas closer to the middle of the stope.

Areas close to strike pillars, particularly gullies, have to sustain the cumulative effects of nearby events emanating from the pillar for the lifetime of the gullies. Conditions in gullies adjacent to backfilled panels are generally considered to be at least as good as if not better than those where conventional support is used, particularly under rockburst conditions.

The PPVs recorded close to the face adjacent to the recently placed backfill are higher than the PPVs recorded further back in the areas where the backfill is already consolidated and provides a better support. To achieve the benefits of backfill, strict adherence to a well-established set of standards for both backfilling and working area support must be ensured.

It is not possible to provide a generic design criterion for local area support in the vicinity of regional support, due to the variable nature of conditions that exist. It is, however, relevant to apply a methodology to the process of determining the criterion for each situation.

42. Determine the effect of repeated dynamic loading on the performance of tunnel support systems

By Güler G., Kuijpers J.S., Wojno L., Milev A., Haile A., CSIR Miningtek, Ref: GAP616, 2001

SUMMARY:

The main objective of this project is to determine the support capacity of various tunnel support components and systems under dynamic loading conditions. The findings of rockburst investigations (GAP201 and GAP530) were used to identify support failure mechanisms in tunnels after large seismic events. These findings allowed for a qualitative evaluation of support requirements and support influence and were used as a basis for the actual testing programme.

Deficiencies in the performance of fabric support were highlighted in the investigations of rockbursts. Adequate criteria, or methods, for the design of fabric support do however not exist and this is considered to be a serious shortcoming.

A special test rig was redesigned in such a way that a tunnel support system could be incorporated into a fairly realistic rock mass environment. The effect of repeated impact loads, representing the effect of subsequent seismic events, was evaluated by monitoring displacements and velocities in response to each impact. This evaluation showed the effects of lacing, reinforced shotcrete and Evermine on the energy absorption capacity, deformations and local instabilities. Tendons, supporting the specimens, did not appear to be subjected to permanent deformations subsequent to impact loading. All other components were however prone to failure after sufficient deformation was imposed upon them. The artificial rock mass can be observed to accumulate a relatively large amount of this deformation, which is associated with irrecoverable strain due to sliding and crushing of blocks.

The differential behaviour between the tendons, which were not subjected to yielding, and typical fabric support components appears to reflect the mechanisms involved in a deep-level tunnel in highly fractured rock. In sufficiently fragmented rock, the loading demand on the tendons may be limited to the (lower) capacity of the fabric support. The reason for this is the fact that the fabric support has to transfer most of the dynamic loads into the tendons. A highly fragmented rock mass will have a severely limited capacity for load transfer. In such a scenario, the capacity of the entire support system may be restricted to the capacity of the fabric support. It is in effect the fabric support capacity (shotcrete and membranes with and without lacing), which was evaluated in these impact tests.

In addition, individual support components such as tendons, mesh and lacing, were evaluated separately in different laboratory tests. One of the more revealing findings from these tests was the fact that the capacity of lacing can significantly be reduced by the clamping devices used. Similar observations were made in the impact tests. In addition to that it was also noted, in some impact tests, that the lacing was cut along

the sharp edges of the tendon washers. This had a strong effect on the test results. These observations may represent realistic limitations on actual support performance. Nevertheless, the presence of lacing always contributed strongly to the performance of the support system.

A model for impact testing is also presented in this report. With this model, various influencing parameters can be quantified and the monitored impact tests can be analysed in more detail. One of the most important parameters in this model is the resisting force, which is made up of an internal rock mass resistance and a support resistance. As long as this resistance is not overcome, no permanent deformations can be induced into the system.

The relationship between induced displacements, which are non recoverable, and impact energy can be represented with this model. It is thus possible to simulate relationships between accumulative energy and accumulated displacements as well. This relationship is typically non-linear, as can be observed from the impact tests. The results from the impact tests can be interpreted with more confidence with the use of this model, as the influence of various parameters can be assessed directly.Underground monitoring at a site on Tau Tona, to determine the effects of seismic events on support systems has been completed. The effects of the seismic events to which the site was subjected during the monitoring period, were relatively small, compared to the impact testing results, as maximum velocities of only 67 mm/s were measured at this site. Typical laboratory testing velocities from even the smaller impacts were an order of magnitude larger. Nevertheless, it was possible to establish a relationship between induced velocities and tendon support loading from the underground monitoring data. By means of extrapolation, an attempt has been made to relate velocities to induced strains and associated damage. Such relationships should be extremely useful for the design of support in relation to seismic activity.

The importance of correct design of fabric support has been highlighted in this project and a conceptual design methodology, addressing this issue, has been formulated. With the use of this methodology, support components can be designed in such a way that compatibility is ensured, resulting in more efficient and effective support systems. It is considered important that further testing of fabric type support elements should be carried out, while the design methodology should be developed and evaluated in more detail as well.



Photograph showing damage to an artificial rock mass supported by a membrane, lacing and tendons. Loading was by multiple impacts from a 1 ton weight.

43. Develop tele-controls for self- thrusting percussion drilling machine and associated interface

By Ottermann R.W., von Wielligh A.J., Burger N.D.I., Dr. Kononov V.A., RE @UP, University of Pretoria, Ref: GAP 702, 2001

SUMMARY:

The purpose of this project was to develop tele-controls such that drill operations can be effected from a distance. The controls were fitted to the prototype quiet rock drill and demonstrated to be working.

Noise is a generic hazard common to all commodities and, to a greater or lesser extent, all operations within mining. More people are exposed to the risk of noise-induced hearing impairment than to any other hazard. The pneumatic percussion drill is a major contributor to noise-induced hearing impairment in mines. The design and development of tele-controls for drills so that the operator can be positioned away from the drill will reduce this risk. The tele-control of rock drills has the added advantage of increased safety to the drill operator by his positioning away from the rock face. During this project such a system was developed, tested and demonstrated.

After a literature survey was conducted a functional analysis was done from which a specification was drawn up. Different concepts were generated and evaluated. The preferred concept was presented to the SIMRAC technical committee for approval after which a detail design was done. An experimental development model (XDM) was built and commissioned. The XDM was successfully tested and demonstrated on surface.

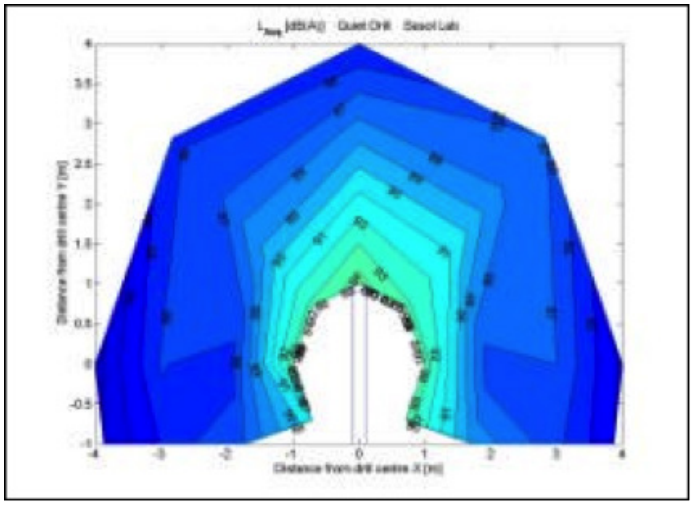


Electronic control unit inside assembled XDM quiet rock drill with telecontrol



Hand held controller

The design of the quiet self-thrusting rock drill, developed during project GAP 642, was used and the tele-controls incorporated into it. The tele-control system consists of a hand held controller and the electronic unit in the drill unit, with which different valves are controlled. Radio control is used as communication between the hand held controller and the drill unit. A generator, powered by an air motor, is incorporated in the drill unit to charge a battery, which supplies electricity to the electronics.



Sound levels around the prototype quiet rock drill tested on surface

CONCLUSION:

Tele-controls were successfully developed and demonstrated for the quiet, self-thrusting blast hole drilling system (GAP642). The specified design criteria as set out in the system specifications were met. A considerable reduction in sound levels was achieved with comparable penetration rates to standard pneumatic drills. With the tele-control rock drill the operator can be positioned away from the drill where the noise levels are acceptable. This has the added advantage of increased safety to the drill operator by standing away from the rock face.

As this was an experimental development model (XDM) no durability tests were performed and the electronics were also not built to withstand underground conditions. In order for this technology to be used underground the electronics have to be designed and packaged to withstand underground conditions. As a tendency exists for the introduction of drill rigs in mines it is also recommended that tele-controls be incorporated in drill rigs.

44. Best practice rock engineering handbook for "other" mines.

By Stacey T.R., SRK Consulting, Ref: OTH 602, 2001

SUMMARY:

This handbook provides the means of evaluating stability and support requirements simply and quickly. When the application of these simple and quick methods is considered to be insufficient, appropriate "best practice" methods are outlined. In addition, a process for identifying rock related hazards is provided. This process, and the evaluation of stability and support requirements, will serve as direct inputs to the Code of Practice to combat rockfalls. Example codes of practice, and a standardised system for auditing the implementation of a code of practice, are included.

The scope of application of this handbook includes "other mines" (both underground and surface) and officially excludes gold, platinum and coal mines. However, the material contained in the handbook is considered to be applicable, at least in part, to these mines as well. The consideration of open pit mines and quarries includes both rock slopes and slopes in weathered material.

The aim is to provide a means of ensuring that sound rock engineering practices, which will minimise rock related accidents, are being followed. Whilst the philosophy of preparation of the handbook has been to preserve the "best practice" concept, it is not a textbook - material has deliberately been summarised and simplified to make the handbook as user friendly as possible. Only material that is considered to be directly useful is included in the handbook. Background information such as basic rock mechanics principles, details of laboratory rock testing, rock material failure criteria, etc have not been included since they can easily be sourced from numerous rock mechanics text books. It is hoped that this handbook will be used by managers and others who have not specialised in the rock engineering field.

The approach to excavation design and stability evaluation follows a straight forward path:

i. The purpose of the excavation determines its geometry and size, for example:

- development excavations such as haulages and crosscuts must accommodate vehicles and equipment, drawpoints must accommodate loaders etc;
- service excavations may be large to accommodate crushers, hoists, workshops etc;
- the mining extraction excavation geometry is dictated by the orebody shape and the chosen mining method.

ii. The practicality and stability of the excavation must then be evaluated in relation to the quality of the rock mass in which it is located:

- is it, or will it be, stable?
- what is the mode of identified instability, if any?
- can the instability be overcome by modifying the geometry and location of the excavation?
- what support, if any (quantity and type), is necessary to ensure that the desired stability is achieved?

The above process can be complicated in cases when instability is a requirement such as in caving. However, the approach remains unchanged - to cave, the opening must be larger than a critical dimension, and to be stable it must be smaller or be supported.

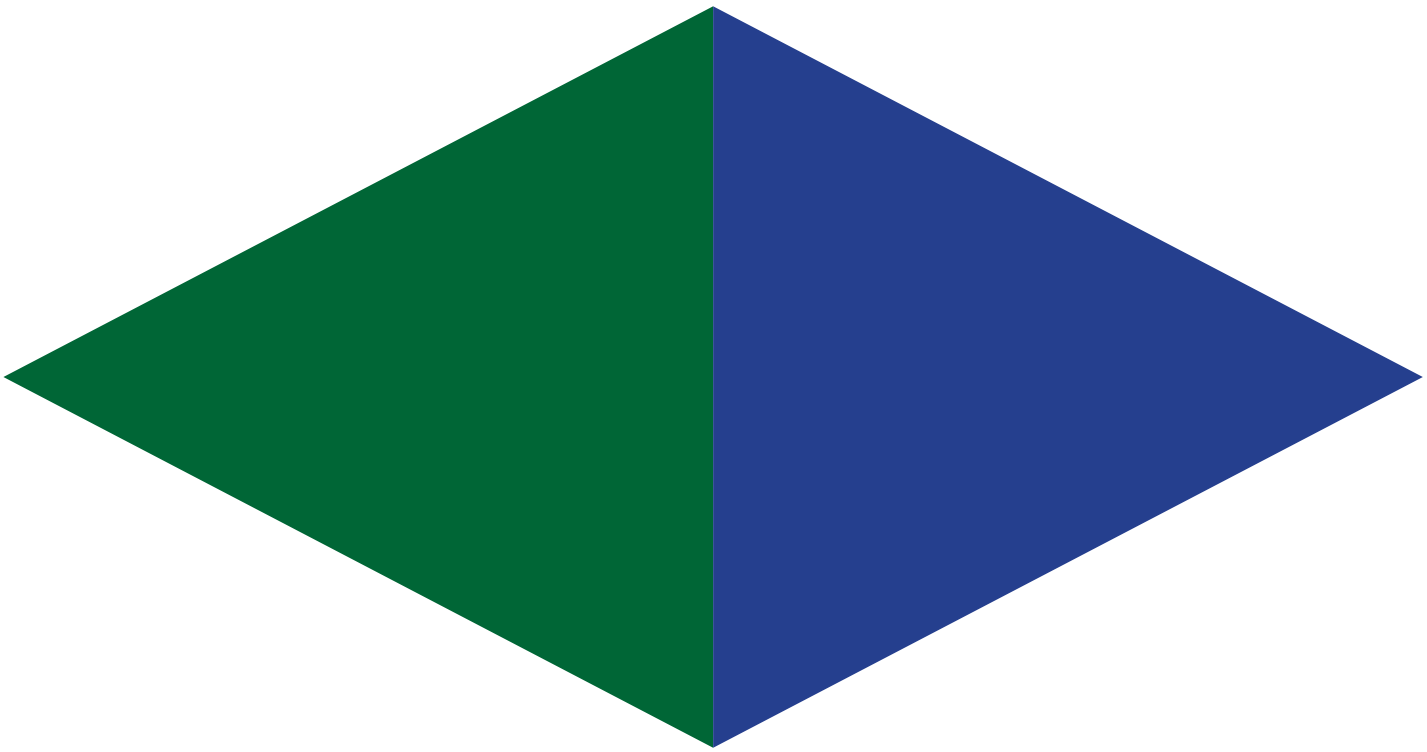
Appropriate support will depend on the risk associated with the excavation - when there is workforce access then stability is of the utmost importance. Conversely, when there is no workforce access, the important criterion may be the efficiency of the excavation process.

The handbook has been structured logically to deal firstly with rock mass behaviour. This is followed by stability evaluation (which takes into account a range of excavation geometries), and support considerations. Since rock mass behaviour is fundamental for the evaluation of stability of all types of excavations, rock mass characterisation is dealt with first. The recommended geotechnical data to be collected, the methods of collection, the recording of the data, the interpretation of the data, and the required outputs are all described. Rock mass characterisation provides the definition of geotechnical areas or domains, which is an essential requirement in a Code of Practice to combat rockfalls.

The evaluation of excavation stability, the rock engineering design of excavations, and the design of solid and artificial support, are dealt with separately for:

- tunnels and other horizontal operational excavations (eg drawpoints);
- service excavations, and vertical excavations (shafts, orepasses);
- underground mining extraction excavations (stopes), and
- surface mining excavations (pits, quarries).

Advanced techniques for design and analysis, such as stress analysis, kinematic analysis and probabilistic approaches, are described briefly. In addition, factors of specific influence are also dealt with, such as the effect of blast damage on stability of underground stopes and pit slopes, and the effect of groundwater on stability of slopes.



45. Development of a method to estimate coal pillar loading

By Roberts D.P., van der Merwe J.N., Canbulat I., Sellers E.J., Coetzer S., CSIR: Mining Technology, Ref: COL 709, 2001

SUMMARY:

The primary goal of this project was to determine the accuracy and validity of the Tributary Area Theory (TAT) and to provide better estimates of pillar load using numerical modelling and other tools.

A detailed literature review was conducted on pillar loading and parameters effecting coal pillar loading. An underground test was conducted to assess the magnitude of changes in pillar stress. Various stress measurement devices were installed in test pillars just prior to mining.

Numerical modelling was employed to determine the influence of mining parameters on pillar loads. The overburden stiffness and seam stiffness were varied using the LAMODEL boundary element code. Finite element models were generated to assess the representivity of the LAMODEL boundary element code, which assumes no friction on stratigraphic parting.

A formula for calculating pillar load was determined through a numerical modelling procedure. The formula contains overburden stiffness and the panel width-to-depth ratio as parameters.

In light of the observed over-estimation of pillar loads using TAT, the implications for the Salamon safety factor formula were examined. Analysis of the roof showed that failure of the immediate roof beam occurred in nearly all failed cases in the database. The resulting discontinuous beam results in full load distribution to individual pillars, exactly the conditions required by tributary area theory.

This implies that pillar load estimations were accurate and that the safety factor formula, as it stands, is valid. Pillar loading on irregular geometries was also investigated as part of the study.

46. Rating system for coal mine roofs

By I Canbulat and T Dlokweni, CSIR: Mining Technology, Ref: COL 812, 2001

SUMMARY:

This project attempted to evaluate and document all the existing roof rating and hazard systems that are used in South Africa and others that have been developed in other countries. In addition, this project proposed the way forward for the development of a system that could be used universally on South African collieries (Phase2). Alternatively, the information presented in this report could be used to enhance local systems.

The review of all existing systems highlighted that in the mining industry the Q and RMR classifications form the basis of many empirical design methods as well as the basis of failure criteria used in many numerical modelling programs.

Application of these systems in South African mines, specifically in coal, have been limited due to the lack of acceptance of the techniques from the industry, which can be attributed partially to the lack of personnel trained in the use of such systems and/or the generally good coal roof conditions in South Africa compared to the rest of the world. Where these systems have been used it has been mainly for portal design or feasibility studies, and there has been little documentation on their use in South Africa.

It is considered that the CMRR could overcome most problems associated with the application of rock mass classification systems in coal mining. However, due to the fact that the system is based on case histories from the United States, certain modifications need to be made to the system to cater for the different conditions in South African collieries. From initial research carried out so far in using the CMRR in the context of South African coal mining, it appears that appropriate modifications can be made.

The investigation into the rating systems being used in South Africa highlighted that roof rating systems are applied mainly for planning purposes, and not for

determining any change in conditions underground. However, rating systems have also been developed in South Africa by Ingwe Coal Rock Engineering (a division of BHP Billiton Energy Coal), in which support systems are changed based on the ratings obtained for underground conditions.

Impact Splitting Tests, Section Physical Risk Rating and the Section Performance Rating systems developed in South Africa are described as being most effective and appropriate for South African conditions. They can distinguish different roof conditions necessary for initial planning and support design.

They can also be used for identifying changing conditions while mining and determining the best response to the different conditions. These systems are also found to be the best systems to address the risk (Variable Step Model) used by sectors of the coal mining industry.

CONCLUSION:

Investigation into various rating systems that are being used in South Africa showed that, although many collieries have hazard plans, these plans do not readily quantify the mechanistic behaviour of the roof strata, they are mostly descriptive and are subject to different opinions. Furthermore, there is no uniform methodology behind the development of these plans, which makes it difficult for another person to apply them.

It was considered that the CMRR could overcome most problems associated with the application of rock mass classification systems in coal mining. However, due to the fact that the system is based on case histories from the United States, certain modifications would have to be made to the system to cater for the different conditions in South African collieries.

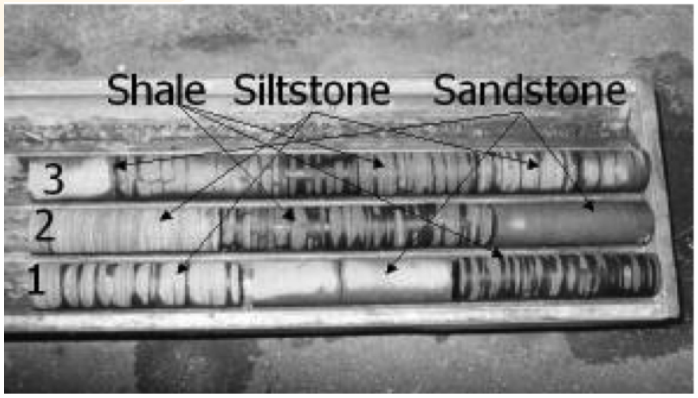


Figure 1: Cores used for CMRR and impact splitting testing

47. Survey of Horizontal Stresses in Coal Mines from Available Measurements and Mapping

By Dr. Frith R., Strata Engineering (Australia) Pty Ltd, Ref: COL 802, 2001

SUMMARY:

Project COL802 presents the outcomes of a detailed review of the role of horizontal stress in coal mine roadway roof stability and control. This has been an issue that has received a significant amount of attention in the recent past in South Africa and generated much debate as a result. This project has attempted to provide an overall assessment of its significance to benefit both mine operators and strata control professionals alike.

The fundamental technical basis for the project has been the experience base in terms of both understanding and ultimately, dealing with adverse horizontal stress effects, from the Australian underground coal industry. Adverse horizontal stress has been a significant geotechnical constraint to mining for many years in Australia. As a result, specific ground support techniques, assessment and design methods as well as operational controls have been developed and proven over time. These are now considered as standard practice.

The project work consisted largely of site inspections at a representative cross-section of underground coal mines in the Witbank and Highveld coalfields, supplemented by the assessment of a number of actual stress measurement outcomes that had been undertaken at several mine sites.

Issues addressed included the extent to which horizontal stress effects were evident in underground mining operations, its operational significance, views on the potential origin of horizontal stress in the coal measures (this being an item of significant debate amongst local strata control professionals) and consideration of the need to improve stress mapping techniques in use at mine sites.

The major conclusion from the study is that whilst horizontal stress is undoubtedly present within the coal measures, it is not a significant geotechnical constraint to mining, this being well supported by the efficiency of the cut and flit development operations that were inspected. Such operations would be constrained to a much higher degree if horizontal stress were a primary controlling factor of roadway roof stability.

The one area whereby horizontal stress is clearly at work is in the guttering and ultimately, instability within the skin of the immediate roadway roof. Whilst this is not a major production constraint, it does represent a potential safety threat, especially in high seam workings whereby even small pieces of falling material can inflict serious injuries.

A possible method of reducing the potential for guttering and associated skin instability has been put forward, this taking the form of "slotting" of the immediate roof. This represents a possible control that would be both effective and compatible with the mining methods in use. It would however, require further development work for it to become a practical mining option.

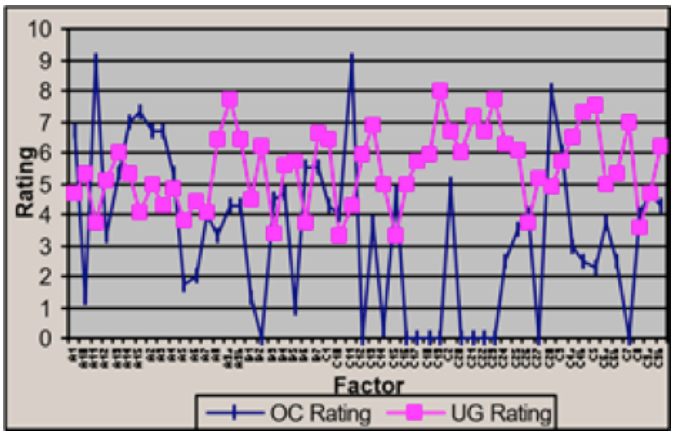
48. The impact of geotechnical factors on the secondary extraction of coal in the Witbank and Northern Highveld Coalfields, specifically related to safety

By Jeffrey L.S., CSIR: Division of Mining Technology, Ref: COL810, 2001

SUMMARY:

The literature review identified geotechnical factors impacting on unplanned secondary coal extraction. These factors are grouped into nine broad classes of factors; namely, stratigraphy, rock /coal engineering properties, spontaneous combustion, discontinuities, igneous intrusions, collapse of previous workings, hydrology, stress environment and the primary mining parameters. For underground extraction, the stratigraphy, igneous intrusions, collapse of previous workings and the stress environment have the most impact on the rock mass behaviour, roof support and explosions/ignitions. For open cast extraction the stratigraphy, igneous intrusions, collapse of previous workings, the stress environment and the primary mining parameters had greatest impact on the rock mass behaviour and explosions / ignitions.

Selected collieries were then visited in order to quantify the impact the factors may have on resource/ reserve estimation and safety. This resulted in the identification of additional geotechnical factors with impact on secondary extraction, as well as a tenth class of factors – secondary mining parameters. The factors impacting on safety, as well as which exploitation phase (i.e. mine planning and/or production) they affect, were identified. These factors belong to all ten classes; those with most impact are primary and secondary mining factors and stratigraphy. A quantification of 10 implies the factor has a high negative impact, while a quantification of 0 implies the factor has no negative impact. The factors were evaluated considering the selection of a suitable mining method (open cast or underground), as well as the negative impact on underground and open cast extraction (results are shown below).



Standing water bodies, spontaneous combustion, roof competency, pillar condition, time elapsed since the primary extraction, and the presence of dykes and sills are the most critical geotechnical factors in terms of safety, mining method selection, open cast and underground operations and with regard to potential adverse effects on the resource / reserve classification.

It would be misleading to present a list of factors that are critical for each operation, as every site is unique. However, this report identifies the factors most likely to be important and describes the effect and impact geotechnical factors can have on secondary coal extraction.

49. Develop specifications for a portable counting seismometer to be implemented routinely in mines underground

By Lynch R.A., ISS International Limited, Ref: COL816, 2001

SUMMARY:

The SIMRAC project COL816 investigated the feasibility of a small, independent device monitoring local seismicity and extracting similar activity information to what can be recorded by a temporary network of seismometers. To do this seismic signals from a uni-axial geophone mounted on a roof-bolt stub were continuously logged while a small seismic network simultaneously operated. Various algorithms were developed and applied to the continuous data to find the one that most closely reflected the network data. A satisfactory correspondence was found, paving the way for development of such a Counting Seismometer.

50. Stope gully support and sidings geometry at all depths and at varying dip

By Naidoo K., Leach A.R., Spencer D., Itasca Africa (Pty) Ltd., Ref: GAP 602, 2001

SUMMARY:

Stope gullies provide the main access route into stopes for men and material and removal of broken ore. From a rock engineering perspective they are recognised as being a particularly hazardous area in tabular mine stopes, requiring specific recommendations for excavation geometry and support to counter the effects of local ground conditions, stress, and seismic activity.

The project provides a compendium of industry knowledge concerning the optimal design of gully excavations and support, and has been carried out under four broad study areas. These include a review of the extensive published information, and a review of mine opinions on best practice including a review of mine standards. The third study area was a review of actual conditions and applied practice in over 100 gullies at 22 gold and platinum mines, effectively an assessment of the effectiveness of supposed best practices.

The final study area used numerical models to assess the stress orientations and likely stress fracture patterns around a wider range of gully geometries. This modelling was used as a basis for quantification of the merits of different sizes and positions of headings, sidings, and gully depths.

The project identifies criteria for adoption of sidings and headings to counter the effects of adversely oriented stress fracturing, based first simply on depth, and secondly on a rock stress to strength ratio criterion that can be easily assessed. Recommendations are provided for optimal dimensions and support for each gully geometry. Cognisance is taken of the effect of reef dip and the variation that rockmass strength has on gully stability. All recommendations either are, or have been, successfully applied somewhere in the industry.

Practical aspects for ensuring good conditions underground are highlighted, such as drilling and blasting practice, gully marking and direction, and the importance of correctly installed support. A checklist is included for use by underground staff when examining a gully. In addition to the main project report, recommendations are summarised in a short guideline booklet, for use by rock engineers and mining personnel.

51. The impact of soft loading conditions on the performance of elongate support elements

By Daehnke A., CSIR: Division of Mining Technology, Ref: GAP 613, 2001

SUMMARY:

In a discontinuous, jointed rock mass, loose blocks can, under certain conditions, rotate and move obliquely, thereby subjecting elongate support elements to different loading conditions than those experienced under conventional axial loading. Since many panels of Bushveld and shallow gold mines are supported by elongates only, it is vital to understand and quantify the elongate interaction with rotating and obliquely moving blocks of rock. An assessment is required to determine if elongates are suitable and the optimum support type to stabilise such unstable blocks.

This project investigates the mechanisms of the rotation and oblique movement of blocks, and assesses the capabilities of elongates to support such unstable blocks. To this end, theoretical, laboratory and underground studies are conducted to ensure meaningful and practically relevant insights and solutions.

The primary outputs of this project are (i) a confirmation as to whether or not it is kinematically possible for jointed blocks to rotate or move obliquely downwards, and (ii) an assessment of the capabilities to support such unstable blocks and their suitability for use in most Bushveld mines.

Based on underground investigations, as well as analytical and numerical simulations, it was found that the probability of blocks failing by rotating out of the hangingwall of a stope depends on the shape of the block. Blocks with shallow dipping sides rotate more readily than blocks with steeply dipping sides. Design charts, based on discontinuity orientation and block size, have been developed to determine if block rotation is kinematically possible.

An important output of the project is a computer program to quantify optimum support spacing in geotechnical areas where keyblock rotation is likely (see Figure 1 for an example of the program interface).

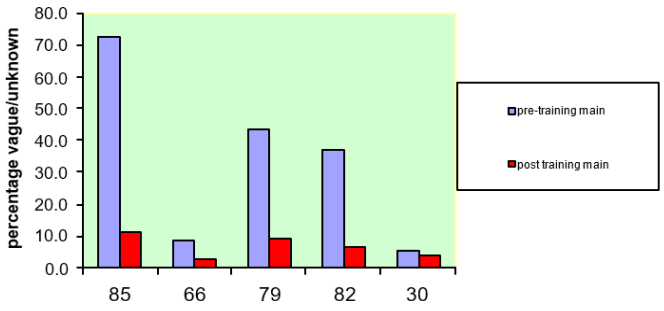


Figure 1: Support design example based on 150 kN props and a 10 m x 7 m x 1 m keyblock.

A laboratory press was modified to simulate the loading conditions induced by rotating and obliquely moving blocks. The press was used to investigate and quantify the performance characteristics of mine poles and Profile Props subjected to non-axial loading. It was found that installing elongates at an oblique angle to the loading surface has little effect on the elongate performance.

Based on the results of extensive underground tests, a modified (empirically based) relationship is proposed to determine the anticipated underground mine pole performance from the performance of mine poles quantified by means of laboratory compression tests. The relationship downgrades the laboratory curves in terms of (i) initial mine pole stiffness, and (ii) peak mine pole load.

The work conducted as part of this project has led to the conclusion that, in general, elongate support is considered suitable for use in shallow gold and Bushveld mines. To ensure rock mass stability, however, the rock engineer needs to determine if block rotation is likely (guidelines for determining if block rotation is possible are given in the report). If block rotation is likely, the tributary area design methodology needs to be modified and the software developed as part of this project should be used

to optimise support spacing. Furthermore, if block rotation is likely, elongates should always be pre-stressed (preferably to 200 kN). If block rotation is kinematically impossible, the standard tributary area method for support design is appropriate.

Finally, a video was produced, giving details of the salient findings of the project. It is recommended to distribute the video to the relevant rock mechanics practitioners on the platinum mines to increase the awareness of the hazard and special support requirements associated with hangingwall keyblocks with a propensity for rotation and oblique movement.

CONCLUSION:

It is preferable for elongates to be installed perpendicular to the dip of the strata. The ends of timber elongates should be cut parallel to the loading surface, or suitably wedged to maximise the timber contact area with the foot- and hangingwall.

Elongates subjected to simultaneous rotational and axial loading have reduced load carrying capacity. The reduction in capacity is relatively minor at angles of up to 50. At rotation angles approaching 10o the load-bearing capacity is significantly compromised. It is thus important to design active support systems providing sufficient initial pre-stress such that any block rotation is prevented.

Mine poles failing in the brushing mode offer improved yieldability compared to mine poles failing due to splitting or buckling. Brushing failure is thus the preferred failure mode. In mines where increased yieldability is required, it is preferable to use poles with an engineered brushing mechanism (e.g. Profile Props).

52. Understanding and determining the variability of the primitive stress environment

By S ellers E.J., Coetzer S.J., Kamstra R., CSIR: Division of Mining Technology, Ref: GAP 707, 2001

SUMMARY:

The primitive stress state is an important boundary condition for the design of underground excavations. However, it is well known that the stress state varies considerably from place to place. The aim of this project was to determine the main causes of the variability to be able to improve mine design procedures. The understanding of stress variability was addressed initially in a review of the literature on the stress state in South Africa.

Descriptive summaries of the stress state in the gold and platinum mines were also provided. The tectonic history, geological structures, folds and surface topography influence the stress state. Faulting causes the major principal stress to rotate towards a perpendicular orientation to the fault. Dykes can behave in a similar manner to faults if the contact is sheared, but may also contain residual stresses that are significantly different from the surrounding stress state. A single stress measurement cannot be considered to be representative of the complete primitive stress state within a mine or mining region.

The problem of quantifying the variability was addressed by re-evaluating the stress measurement database provided in SIMRAC project GAP 511 to determine statistical trends and variation between the major mining regions. Methods are presented for the correct determination of the average and standard deviation of a number of stress tensors. Often, the stress measurements taken within a local mining region are relatively consistent.

In the Bushveld Igneous complex the horizontal principal stresses are aligned parallel and perpendicular to the curve of the outcrop and averages must be determined in regions where the outcrop direction is more or less constant.Improvements were made to the SIMRAC three-borehole stress measuring technique initally presented in GAP 314. A new measuring device was developed. A statistical method was developed to determine the stress state using combinations of the eight borehole diameters and to identify errors in the measurements. Underground testing proved that the drilling technique was practical as long as the drill was set up accurately.

Testing was undertaken at sites on Western Platinum Mine, Impala Platinum Mine and Tau Tona Mine. The measuring device was shown to be able to measure the borehole deformation to sufficient accuracy in underground tests. However, the measurements that appeared to be successful underground contained significant deviations, possibly due to the locator hitting the measuring device, and could not be used to determine a complete stress tensor. The three-hole method was found to be highly sensitive to variations in the borehole diameter.

The method is not very sensitive to changes in the stress component aligned parallel to the line connecting the centres of the three holes. The three-hole method will not be economic in lower stress regions where the CS IR and CS IRO strain cells operate well and only require a single hole for a full three dimensional stress tensor determination. The benefit of the three-hole method would be that only the measuring device needs to be purchased and can be used many times, whereas the strain cells are discarded after use. The three-borehole technique as developed does appear to be a step towards a device that is able to make stress measurements in highly stressed ground, but more testing and further modifications are required.

Region	Average (Std Deviation)		
	major ratio (kH)	minor ratio (kh)	bearing (deg from N)
Central	1.05 (0.28)	0.58 (0.16)	98 (61)
Free State	0.80 (0.17)	0.60 (0.21)	106 (64)
Klerksdorp	0.99 (0.31)	0.65 (0.26)	112 (41)
Mpumulanga	1.06 (0.45)	0.35 (0.95)	82 (48)
North West	2.28 (1.28)	1.00 (0.39)	93 (46)
West Rand	1.13 (0.32)	0.63 (0.19)	95 (60)

Table 1: Stress variability in different regions

The effect of geological structures on the stress state was determined using measurements at two sites. At the Modderlaag thrust fault on the Western Platinum Mine the major principal stresses are aligned in an East-West direction, sub-parallel to the outcrop of the Bushveld Igneous complex, and perpendicular to the approximately North-S outh trending dyke sets.

Numerous attempts to measure the stresses in and near the S peckled Dyke on Tau Tona mine were hampered by quartz veins and the faulted nature of the dyke. The single reliable stress measurement that was obtained indicated that the principal stress was consistent with the activation of the dyke contact as a normal fault.

Finally, dealing with the variable stress distribution is the most important task from the point of view of the rock engineer. Ten measurements per borehole is the optimum number required to obtain a statistically meaningful average stress. Thirty to fifty average stress tensors would be required to determine a stress field. This number of tests would be uneconomical for most mines.

However, analysis of the stress database suggests that the stress state can have a representative mean value within a similar tectonic environment or geotechnical region. Thus, the number of measurements required to determine an average stress state would depend on the geological variability of the mine and where the mine is located in relation to the Bushveld Complex or Witwatersrand basin, the presence of large structural events such as thrust blocks, the Pilanesburg, and the relationship to dykes and faults.

The basis of a design methodology is presented, using flowcharts, that assists in developing a stress measurement progamme and permits the formal application of variable in situ stress conditions in rock engineering designs within the context of risk-based design procedures. The methodology describes the selection of the appropriate input data for application in analytical design formulas or numerical models to determine the probability distribution of the failure criterion as a function of the variable stress state. A simple example is given to illustrate the technique. The application of probabilistic design methods is becoming more popular, but consideration still needs to be given to the selection of appropriate confidence levels and to set acceptable risk levels.

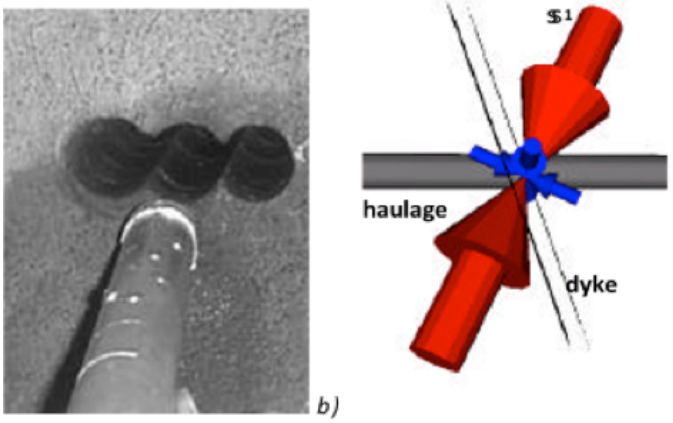


Figure 2: a) new three-borehole measuring device being tested underground and b) stress state near S peckled Dyke

53. Software tool for managing rock related hazards in South African mines

By Brink A.V.Z., Hagan T.O., CSIR: Division of Mining Technology, Ref: GAP 714, 2001

SUMMARY:

The objective of the project was to develop a software tool for the assessment of rock engineering (RE) risk based on the concept proposed in GAP 608 and to evaluate the software with real mine data from a deep level gold mine and a platinum mine.

Hazard is a physical situation, object or condition, which, under specific circumstances has the potential to cause harm, whereas risk is a measure of the likelihood that some specific harm, arising from an incident (a particular hazard), will occur.

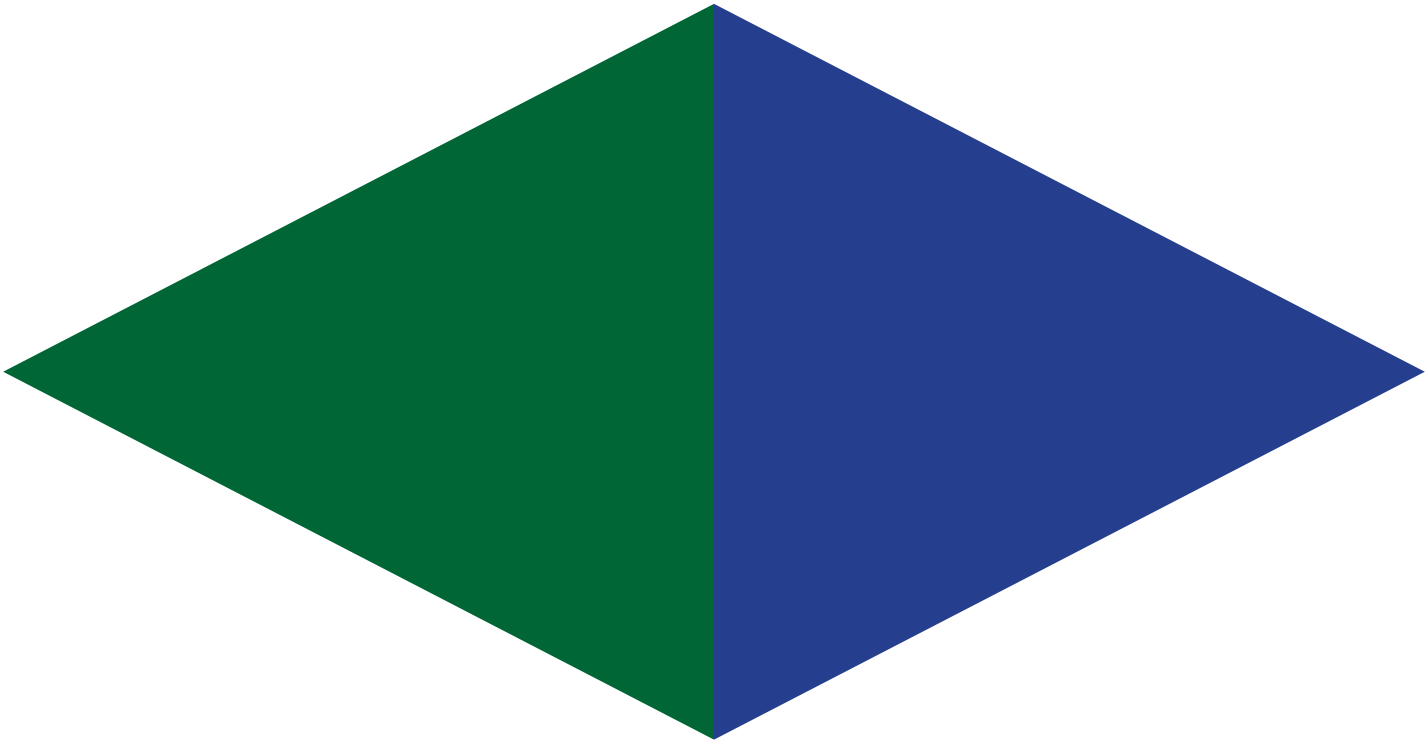
The generic equation:
Risk = Hazard * Vulnerability was, in this case, extended to:
Risk = Seismic Hazard * Induced physical hazard * Systemic Vulnerability
(Induced physical hazard = triggered by the ground motion; support failure; fall of ground.
Systemic Vulnerability = Exposure of people; economic vulnerability; quality of information.)

RE risk as defined in GAP 608 is a function of the potential ground motion; the vulnerability of the workplace; exposure of people; and a general term – quality of information.

The deliverable of this project is a prototype software package that is a combination between an expert system approach for determining overall risk assessment, a risk control recommendation and a basic GIS (Graphic Information System) approach for graphically overlying input parameters and the output risk.

The expert system uses a Bayesian probabilistic approach to combine various workplace-related information, a quantified seismological environment and exposure of underground staff to provide an overall RE risk rating. The risk assessment is dynamic in that it is recalculated each time any input parameters change, for example each time a seismic event is recorded in the area of interest.

The code was tested on a data set from Savuka Gold Mine and Northam Platinum Mine. As risk assessment is not an absolute prediction it is therefore not possible to relate to individual case studies and the reliability of this method would only be determined after an extensive period of use.



54. Performance of various types of containment support under quasi-static and dynamic loading conditions

By Kuijpers J.S., Milev A.M., Jager A.j., Acheampong E., CSIR: Division of Mining Technology,
Ref: GAP 810, 2001

SUMMARY:

Seismicity causes the most severe loading conditions for tunnel support systems in deep-level South African gold mines. The design of such support systems should therefore incorporate the potential effect of seismicity on support requirements. Unfortunately, this is presently not possible as such effects have not been adequately quantified. Within the current project, an attempt has been made to address this shortcoming.

The main objective of this project is to determine the support performance of various tunnel support systems and components under dynamic loading conditions. Deficiencies in the performance of wire mesh were highlighted in the investigations conducted for this project. Investigations were limited to support systems consisting of yielding tendons, wire mesh and steel ropes (lacing). Part I of this project represents the contribution of CSIR: Miningtek while the contribution of SRK is represented in Part II.

A special testing rig, which was designed to incorporate full size tunnel support systems in a realistic rock mass environment, was used for dynamic as well as quasi-static loading tests. While impact loading was used to represent the effects of seismicity as realistically as possible, quasi-static loading tests were conducted in order to obtain accurate load-deformation characteristics of the system and its components. These (calibrated) characteristics could subsequently be used to interpret more complex systems under both dynamic and quasi-static loading conditions.

A conceptual model, describing the fundamental and essential characteristics of cable systems, such as mesh and lacing, is presented in this report. This model enables a direct assessment of the effect of relevant parameters and is therefore of great practical use. The model can also be used to obtain a better understanding of the mechanisms involved in the support function of mesh and lacing.

Various combinations of mesh and lacing with and without tendons were tested. Some examples of their performance characteristics are given in Figure 1.

One of the most revealing findings from this project was the general incompatibility between wire mesh and lacing and the yielding tendons in terms of stiffness and strength (Figure 2). In addition, the effective strength of the wired mesh was found to be severely reduced by stress concentrations associated with the tendons and, to a lesser extent, the lacing. As the effective stiffness of mesh and lacing increases with increasing deformation, the incompatibility is especially evident at the initial stages of deformation. Suggestions to improve the compatibility are made, even though the practical feasibility of these suggestions was not investigated in any detail. A potential problem with providing too much stiffness to fabric support has also been identified.

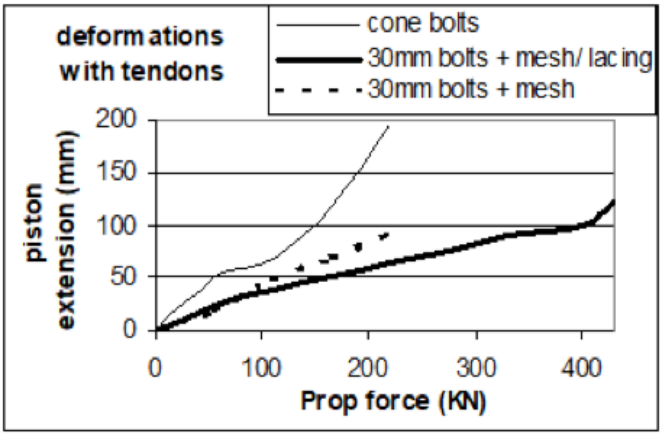


Figure 1: Tests in which tendons were used to support

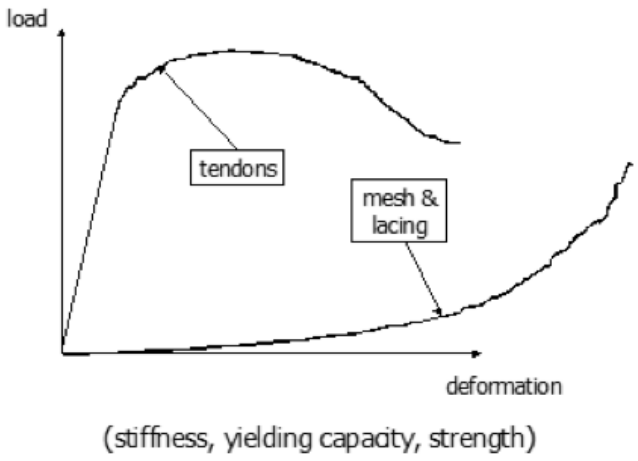


Figure 2: (In)compatibility between support components

The characteristics of the yielding tendons control to a large extent the performance of the support systems that have been tested and it is therefore important that these characteristics match specifications under all loading conditions.

Contrary to previous findings, welded mesh was not found to be stiffer than diamond mesh. A possible explanation for this finding is the full-scale size of the current tests, compared to the small-scale tests previously conducted.

CONCLUSION:

In general, it can be stated that a relatively soft fabric in combination with stiff tendons, will quickly lead to rock mass unravelling, while a stiff fabric with soft, yielding tendons may prevent such unravelling and maintain rock mass integrity. As stiffer systems attract larger impact forces, it is imperative that adequate yielding capacity is available in such systems.

55. Effective roof support for tabular stopes

By Ottermann R.W., von Wielligh A.J., Burger N.D.L., Handley M.F., Fourie G.A., RE @UP, University of Pretoria, Ref: GAP 813, 2001

SUMMARY:

This project follows on from project GAP 708, The Design and Development of an Effective Support System for Tabular Stopes in Gold and Platinum Mines. The primary output of this project is to develop and surface test an effective roof support system and to do a risk assessment of the system. Mines with tabular ore bodies will use the roof support system to reduce fatalities and injuries in the face area of tabular stopes.

The roof support system consists of two similar support units connected to each other by two cranks. Each unit consists of a 210 x 1200 mm headboard supported by a “wishbone” structure (top and bottom leg) and a self-contained loading/yielding mechanism that keeps the legs apart. The yielding mechanism consists of a selfcontained hydraulic cylinder, a pump to load and pre-stress the unit and a relieve valve to accommodate slow yielding of the system. The system yields to both slow and sudden stope closure by a combination of elastic deformation in the wishbone structure and the pressure relieve valve.

Rapid yielding is mainly accommodated by the flexibility of the “wishbone” structure, and this also gives the system the necessary rebound, since stopes do not only undergo monotonic closure during a seismic event. The system is therefore more likely to stay in place, providing support resistance and energy absorption during the critical moments of the rockburst. The yielding mechanism is made from steel while the two legs of the “wishbone” and the headboard are manufactured from glass fibre.

To move the system forward one support unit is collapsed and then hangs from the other by the two cranks. The collapsed unit is then manually cranked forward and re-loaded. Each “step” forward is 300 mm, hence the system must be cranked forward three times to move 900mm (the typical face advance per blast in a tabular stope). Surface tests reveal that each collapse-crank forward-reload cycle takes 1 minute, requiring 3 minutes to move the system forward by 900 mm. Typically, twenty wishbone support units will be installed in a 30 m panel, and will take two persons one hour to move forward a distance of 900 mm.

The support system can be tailored to meet specific mining parameters such as stoping width and average face advance per blast.

The system has undergone the typical engineering design and development process, beginning with a review of the design specifications, and a detailed evaluation of and modification to the concept that appeared in the GAP708 report. The prototype development process was an iterative process that involved detail design, building and testing of sub components and building and testing of the prototype. The testing of the prototype was done on surface in the laboratory and in a mock-up stope. A risk analysis, in which technical, logistical and economical aspects were assessed, was done to determine the critical areas of the system.



Figure: Roof support system in mock-up stope

CONCLUSION:

The tests and risk assessment of the roof support system identified certain areas that need to be addressed in order to implement the new support system safely underground.

For the further development of the support system the areas that need attention are:

- Improve the design taking the results of the tests into account.
- Quality control during manufacture.
- Correct and safe installation procedures.
- Safe operating procedures.
- Develop a different support system, which can be cheaper and lighter for rock fall mines.

56. The Development of an Effective Face Support System for Tabular Stopes

By Acheampong E., Jager A.J., Roberts M.K.C., Habeck M., Ishmail A., Frangakis T., Wells J., Rowe A., CSIR Miningtek, Ref: GAP813A, 2001

SUMMARY:

This report is an interim report of a two-year project, the first phase of which ended at the end of March 2002. The report therefore highlights progress made to date only. In SIMRAC project GAP708; eleven innovative means of reducing the rock-related hazard near the stope face were investigated. A linked prop and headboard system, named the remotely advanced headboard system, was identified as having the highest potential to significantly improve safety in the face area, while seemingly being a practical system that should not be difficult to implement and maintain.

The objective of this project is to develop the remotely advanced headboard concept and test prototypes. The primary outputs of the research are:

- Design and manufacture of a prototype (Phase I)
- A risk assessment of the support system (Phase I)
- Quasi-static and dynamic surface tests (Phase II)
- Underground trials (Phase II)

The design process and the manufacuture of a protype are completed (Figure 1). The final design is a combination of all the original concepts i.e. from the original link-bar system to the crank system, and finally to the crank slider system. Despite the serious practical engineering shortcomings of the previous concepts, their advantages were incorporated into the crank slider system. This headboard system is based on one developed by the Chamber of Mines Research Organisation in the early 1990's.



Figure 1: The Final Concept

The movements are remotely controlled. The sequence of movements results in an advance of approximately 600mm and has therefore to be repeated in order to achieve the required advancing distance. This concept produces a continuous range of 600 to 1200mm advance.

Initial laboratory tests on the beams indicate that plastic deformation is unlikely to occur up to tip loads of 20 tons (Figure 2). The crank slider system is thus more flexible and its expected to be more user friendly. A risk assessment was performed on the system to identify sources of potential hazards that could be associated with the use of the support system. Recommendations on appropriate controls to minimise the hazards are also prescribed.

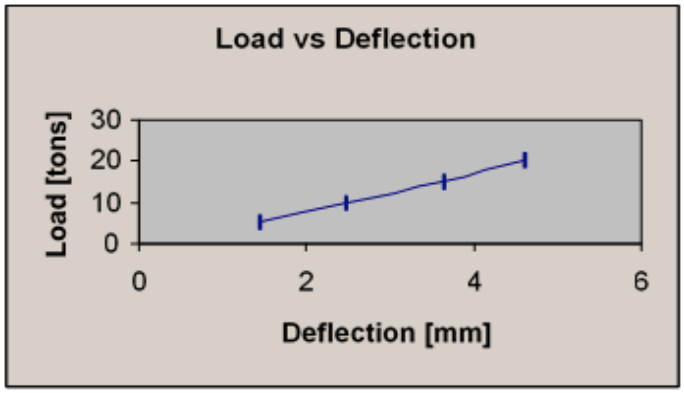


Figure 2: Load vs deflection for the Miningtek headboard from the finite element analysis

The second phase will commence with a surface mock-up trial to be followed by a load-deformation test in Savuka test rig. It is expected that a suitable underground site would have been found after the test to enable underground trials to commence in May/ June 2002.

57. Review of past research areas – stope and gully support

By Stacey T.R., Arnold D.A., James J.V., The University of the Witwatersrand, Ref: GAP 816, 2001

SUMMARY:

The report involves a review of past SIMRAC research into aspects of stope and gully support. The findings are summarised in terms of four broad research categories: Accident Analysis, Rockmass Behaviour, Support Design and Support Performance. Key outputs in these areas are listed in the report, and are flagged as either having found use in the industry, or alternatively, as being of interest but having as yet not found use.

An overview of the knowledge gaps identified is also given for each of the four categories. In the limited space available for this summary, it is not possible to include all of the above information. Only the perceived knowledge gaps identified from more recent projects are therefore included below.

Knowledge Gaps:

Rockmass Behaviour

More detailed geotechnical area mapping of the UG2 and Merensky reefs is required to quantify the influence of geotechnical parameters on support requirements and to delineate areas with varying rock mass behaviour. Underground investigation and back-analysis is required to verify the three different deformation mechanisms postulated for the reef horizons and associated parting planes, and to calibrate the support design methodology and provide appropriate input parameters for support design. [GAP 330].

A better understanding of the statistical distribution of dome joints and their orientation adjacent to potholes is required. [GAP 613].

Further research work is required to assess support design methodologies, based only on preliminary numerical and analytical analyses, in the field. A userfriendly procedure to facilitate continuous gathering of necessary support design input data is required if the proposed methodology is to become practically viable. [GAP 627].

Although a continuous mine-wide closure monitoring system has been shown to be technically feasible, the purpose, benefits, consequences and acceptability of such a system have yet to be established. [GAP 705].

Support Design

A complete industry support database, updated on an annual basis, will ultimately give useful insights into support systems requirements for various geotechnical areas. As mines go deeper, support densities are expected to increase. Research should consider alternatives to current technologies to provide areal support at the required density and address problems of face access and workforce safety upon re-entry. [GAP 330 & 611].

A possible knowledge gap with regard to stope gully support and siding geometry is the identification of a set of achievable standards that should apply to all mines. Any mine that wished to deviate from these standards should be required to specifically motivate and justify its case. [GAP 602].

A succinct, illustrative 'Stope Support Technology' booklet should be produced in respect of support design procedures and the selection of appropriate support types corresponding with identified strata conditions. [GAP 606].

The lack of a sufficient amount of real seismic data for large magnitude events that may be the 'killers' represents a significant gap in the knowledge required for good support design. [GAP 709b].

Support Performance

Investigations to establish whether hydraulic props do in reality provide a superior rockburst-resistant support function than other support types, and whether, as a result, it is, or is not, necessary to attempt to reverse the industry trend away from rapid yielding hydraulic props. [GAP 330 and GAP 442].

In-situ performance results should be obtained to develop some sort of laboratory equivalent test procedure to assess a pack's response to non-ideal loading and end-effect conditions. More research relating to ride, especially in more steeply dipping stopes where this can affect the performance and stability of certain packs such as composite or end grain types, would be valuable. A special laboratory test procedure needs to be established to test grout-based packs that are cast in place and compress during curing, since it is expected that a fully cured pack must be much stronger than one compressed as it is curing. [GAP 508 & 613].

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

Persuing Research Excellence”

58. Infrared thermograthy of loose hangingwalls

By Dr Kononov V.A., CSIR Miningtek, Ref: GAP 820, 2001

SUMMARY:

This project is the continuation of GAP706 “Prefeasibility investigation of infrared thermography for the identification of loose hangingwall and impending falls of ground”. The main concept behind the infrared thermography method is that with an exposed hangingwall surface area, loose sections of ventilated rock should have a lower temperature than solid sections of ventilated rock because the former acts like cooling fins. The temperature gradient between loose and solid rock depends on the thermal conductivity of the rock, the ventilation conditions, the looseness of the rock and, to a lesser extent, the type of rock and age of mining.

The intensity of a perfect radiator (black body) is a function of its temperature only. Rock can be thought of as a so-called “grey body”, for which the true emissivity is always lower than that of a black body. When the IR thermometer is used for remote measurement of rock temperature, a true emissivity for the particular rock should be used to achieve the best accuracy. As it is difficult, in underground conditions, to correct for the rock emissivity during the measurement process, a few typical samples of rock were examined in order to identify their true level of emissivity.

As was concluded in the report for project GAP706, commercially available IR thermometers are unable to provide reliable operation in the hostile South African underground environments. They lack the general protection against dust, moisture and the robustness required of a mineworthy instrument.

Apart from these obvious drawbacks, commercial devices are usually designed to provide a very narrow measurement angle that enables the integration of the temperature from a surface of about 30-40 mm in diameter at a distance of 1,5-2 m. During this project it was proven that when the method is used for the identification of loose rock, a much wide angle or measuring area is required. It is proposed that a measurement area with a diameter of 80-120 mm provides better measurement results.

The best resolution provided by commercial devices is about 0,1 0C within the measuring range from –10 0C up to + 900 0C. It was reasonable to propose that reduction in the temperature measurement range could enable the improvement of resolution. For South African mines, a measurement range from 0 0C to 90 0C is more than sufficient for the purpose of rock temperature monitoring.

To this end, together with Raytek, one of the MX4 IR thermometers was modified to comply with the project requirements.

CONCLUSION:

Underground tests have been conducted at Anglo Platinum Townlands platinum mine and Gold Fields Driefontein 5 shaft gold mine. Results of the prolonged underground experiments, aimed at the evaluation of IR thermography for identification of loose hanging walls, enables the following main conclusions to be drawn:

- The modified IR thermometer provided better resultsthan the commercially available instrument.
- The method of IR thermography for identification ofloose hanging wall is only applicable for primary roofbolting after the face blast, before reinstallation of the ventilation duct.

Recommendations:

In order to make the final decision on the implementation of the method for primary roofbolt installation, the following main recommendations are formulated as an extension of this project (See full project report).

59. The application of acoustic techniques for identifying rock-related hazards in gold and platinum mines

By Piper P.S., Le Bron K.B., van Rooyen H., Goldbach O.H., Clifford B., Groundwork Consulting,
Ref: GAP 822, 2001

SUMMARY:

The project was divided into four areas, viz.: (i) Theoretical evaluation, (ii) AEM calibration under controlled laboratory conditions, (iii) AEM surface field trials, and (iv) AEM underground trials on gold and platinum mines.

The objective of the theoretical evaluation was to assess the use of seismic techniques in order to develop a method that can instantly detect hazardous rock conditions in a quantifiable way.

The focus of the evaluation was on the propagation of seismic energy in the immediate skin around an underground excavation. Based on this approach, the theoretical analysis has shown that seismic techniques have the potential to detect changes in hangingwall integrity in gold and platinum mines. The second part of the theoretical analysis focused on assessing the Acoustic Energy Meter, a device that can determine the position and extent of sub-surface hazards by quantifying the decay time of seismically-induced oscillations, and which has been applied in coal mines with a fair degree of success.

The examination showed that, from a theoretical point of view, the AEM has application potential in the gold and platinum mines, provided that the correct frequencies are introduced into the rock mass and that the technical specifications of the instrument are such, that the required frequencies are measured to correctly interpret the reflected seismic energy. Although the AEM has been successfully applied in coal mines, where the nature of the hazards is much simpler than the complex fracturing expected in deep-level gold and platinum mines, its performance was difficult to predict in the more complex gold and platinum environments. Thus, the degree of success in the various potential applications in gold and platinum mines had to be established in a detailed field evaluation programme.

As a result of the work described in this project report it is believed that the applications for the AEM fall into two main categories; 'hard' applications and 'soft' applications. The 'hard' applications are discussed in Table 1. The 'soft' applications for the AEM are as part of a documentation and management control system rather than placing all the emphasis on the readings themselves. For example, the AEM has the potential to identify the leading edge of domes structures.

In the application of the AEM against the hanging wall the obvious presence of a dome is likely to be recognised by the operator. However, without the obligation to apply the AEM and obtain readings, mine personnel would not necessarily observe the hanging wall. Therefore, a management control system can be put in place which ensure that the operator checks all critical parts of the hanging wall and provides the necessary proof of the work in the form of a completed survey sheet. Furthermore, the AEM could be used as a training tool to assist mine personnel in confirming the presence of loose blocks which could then be identified visually in future.

CONCLUSION:

The accuracy of the AEMs was determined using a calibration table. These AEMs were also used for the surface and underground trials. As part of the surface evaluations, the influence of the following parameters was analyzed: (a) Hammer weight, (b) Distance of hammer from AEM, (c) Intensity of hammer blow and (d) Hammer side. The most consistent results were obtained when the intensity of hammer blow was constant, the distance between hammer and AEM was 200 mm, the ball-point side of the hammer was used and a 0.75 kg hammer was used.

Except for the rock volume and distance between the hammer and the AEM, all the other parameters (which are hammer blow intensity, hammer shape, hammer weight and the orientation of the hammer blow) are a function of the person conducting the survey and the hammer used in the survey. The influence of these factors could be eliminated by introducing a "constant impact gun", which could result in more consistent AEM results. An in-situ calibration of the AEM should be conducted at each site before conducting a survey i.e. an AEM reading, which indicates loose rock could be different for different rock types depending on densities and other rock properties.

60. Trial training in strata control for underground workers

By Johnson R.A., aku E.J., Squelch A.P., Barnett H., Hagan T.O, CSIR Miningtek, Ref: GAP 851, 2001

SUMMARY:

In-depth assessments of accident records in gold and platinum mines indicate that an improved understanding of rock mechanics among production and service personnel will improve worker safety in relation to the rockburst and rockfall hazard. Workers' lack of competence with respect to certain aspects of strata control increases their, and their fellow workers', exposure to such strata control risks. To date, however, rock engineering related courses have been mainly aimed at rock engineering personnel, mine management and senior production personnel. It is, therefore, desirable that a course suited to less senior production and service personnel, up to shift supervisor level, be implemented. An appropriate course for this audience is one that focuses on essential strata control and hazard identification information. At the mine where trial training was done courses for team leaders, shift supervisors and miners had been implemented and these courses were used to further enhance training as part of this project.

Recently completed SIMRAC projects GAP609a and GAP609b provide a sound framework (in terms of material and methods respectively) to perform effective training at the required level in gold and platinum mines. However, the results of the research need to be tested and proven, and effectively communicated before they can be generally accepted. This project, therefore, undertook a trial implementation of purpose-developed training content according to guidelines drawn up under the previous GAP609 projects. To achieve this, a champion mine was selected that had a desire to implement a different approach to training than that currently being pursued.

A further requirement was that a flexible organisational structure and work environment existed that could accommodate the, albeit relatively minor, level of intrusion and disruption caused by the implementation and monitoring process. Topics for instruction were chosen, for their relevance and importance, by the project team in consultation with personnel on the selected mine. A process was then undertaken to 'model' (i.e. identify and describe) the knowledge/expertise necessary for a stope team to achieve the desired level of strata control and hazard identification effectiveness. The experimental training intervention consisted of a selected stope team on the mine being trained and evaluated by a trained facilitator/trainer in respect of the assimilation and application of this knowledge/expertise. A 'control' stope team was also evaluated for comparative purposes.

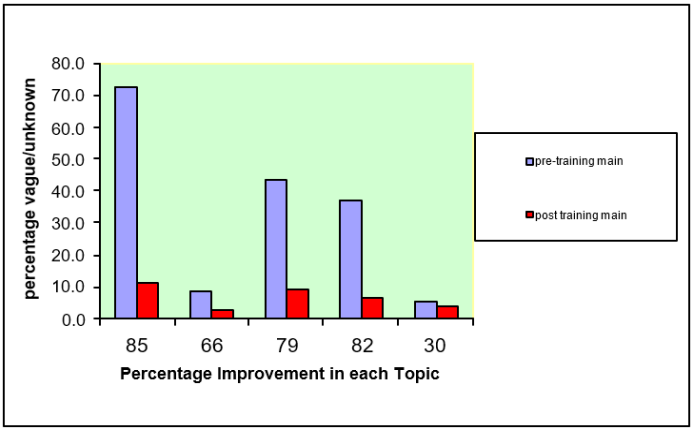


Figure 1: Improvement in five topics after training. Number of vague or "don't know" answers were reduced in all cases.

CONCLUSION:

The introduction of basic strata control knowledge to the team leaders directly and to the rest of the workers indirectly made a significant improvement in terms of hazard recognition and the method of coping with the bad ground conditions.

Positive results are recorded from the implemented training intervention exercise, and the positive impact of this method is evident (Figure 1). Topics selected were appropriately focused and targeted the needs well.

A certain amount of knowledge sustainability was obtained, despite a change of leadership in the tested team. Greater significance can be attached to the findings if the trial method is compared with similar measures of conventional training. The comparison with the control stope provided this.

Setting up a more tightly controlled research project would be difficult to achieve because of the constantly changing work and team environment in underground mining. Training alone (even using an appropriate methodology) is not sufficient for success; positive organisational influences are also required.

Recommendations:

Taking into account the results of this project, it is highly recommended that:

- similar focused methods of training be implemented on a wide scale (the practicalities of doing so may seem onerous, but since the training is delivered through the team leader, and is based in the working environment, this would not require a great deal of additional personnel); and
- work is done alongside this training to ensure that the team leader has adequate resources and authority to implement what has been learnt.

Additional research in the following areas could supplement the contribution made by the current project work:

- The impact of safe work procedures on production output over the medium to long term.
- The impact of management visibility on safe work.
- The impact and efficiency of initial induction and post leave refresher training currently used.

61. Application of indirect stress measurement techniques (non strain gauge based technology) to quantify stress environments in mines

By Stacey T.R., Wesseloo J., The University of the Witwatersrand & SRK Consulting, Ref: GAP 858, 2001

SUMMARY:

Reliable values of in situ stress are essential for the valid modelling of mine layouts. Available non-strain gauge methods are reviewed as potential practical techniques for South African mines. From this review it is concluded that the following methods could be attractive for use in this mining environment:

- Estimation of in situ stresses, using the existing in situ stress database and observations and interpretations of failure around openings, is a very important technique for developing an understanding of the in situ stress state. It is recommended that it should always be the first approach applied to obtain an indication of the in situ stress directions and magnitudes. It should also be used as a check on the results of other methods of in situ stress measurement. This method should however, not be used in isolation, but with other methods to build an understanding of the stress regime.
- Kaiser effect gauging, using the monitoring of acoustic emission to determine the Kaiser effect change point. Considerable development of this method has taken place in the past few years, and it is now being applied successfully to determine the in situ stresses in mines in Australia. It is applicable to both greenfield sites as well as existing mining operations. From orientated core from a single borehole, the full three dimensional in situ state of stress can be determined. All detailed work is carried out in the laboratory, so there is no interference with normal mining operations.
- Sleeve fracturing such as developed by Serata Geomechanics Inc appears to be well suited for routine measurement of in situ stress since 20 to 30 tests can be carried out in a day and the fracture can be induced in the borehole at any desired direction. There are drawbacks to the method for application in South African mines – a smooth borehole drilled to close tolerance is required, and at least three boreholes are required to determine the full three dimensional in situ state of stress. At present the maximum stress that can be measured is 40 MPa.
- Back analysis of in situ stresses using deformations measured around excavations as a result of changes in the geometry of that or adjacent excavations. This is not rated as a routine method of determining in situ stresses, but the approach is valuable since a large volume of rock is involved.

Therefore, if the opportunity arises with suitable deformation measurements being available, the approach is ranked as a method that should be used. The in situ stress results, based on the deformation of a large volume of rock, will be valuable for comparison with the stresses obtained from other methods.

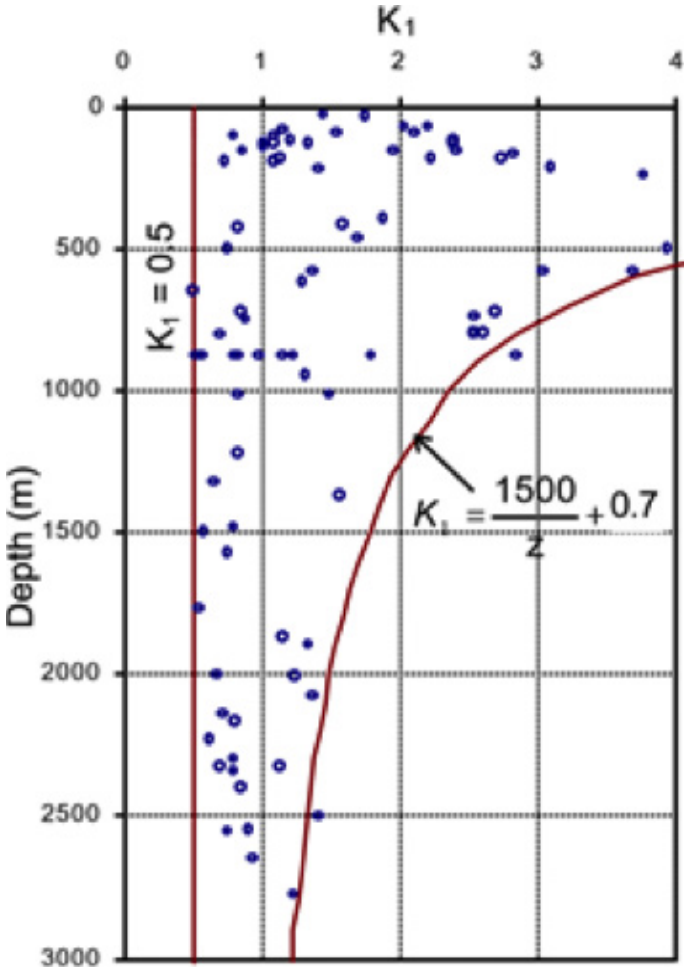


Figure 2: Major horizontal to vertical stress ratio vs. depth

CONCLUSION

It is concluded that the most suitable “new” method of in situ stress measurement is the use of Kaiser effect gauging, with the Kaiser effect change point being determined by means of acoustic emission monitoring. It is recommended that some research and development of this method should be funded to gain the necessary familiarity and experience with its application to make it a routine method of in situ stress measurement in South Africa.

62. Development of an effective pinchbar

By Otterman R.W., Burger N.D.I, Von Wielligh A.J., Handley M.F., BE@UP, University of Pretoria,
Ref: SIM 020201, 2003

SUMMARY:

A significant proportion of rockfall accidents occur during re-entry into a workplace, when the initial inspection and making safe procedures are carried out to stabilise the rock before work in the area begins. The reason is that making safe is one of the most stressful and dangerous activities an underground miner can undertake. The operator often is unable to work at a safe distance and is sometimes forced to work directly underneath unstable rock ilise the rock effectively and efficiently from a safe distance before work begins when attempting to “make safe”.

The equipment currently used is archa ic and there is a need to devise a simple system to enable operators to stab in the area. Studies undertaken through the University of Laval, Canada, have shown that scaling (making safe) with conventional hand-held equipment is difficult, arduous and stressful, with operators needing to rest for rapidly increasing amounts of time after only 8 minutes of activity. This is almost a guarantee that excavations requiring a lot of barring will not be made safe adequately.

During a previous project GEN 801, Investigate a possible system for “making safe”, a literature and international survey on existing systems as well as a problem survey of different mines (gold, platinum and coal) was conducted.

A functional analysis was done from which a specification was drawn up. Different concepts for making safe were generated and evaluated against a system specification.

The following concepts were recommended for further development:

- A “lightweight pinch bar” where the bar is manufactured of composite materials.
- “Mechanical jaws”: A hand held and operated mechanical system, which makes use of hydraulic pressure activated jaws to pry rocks loose.

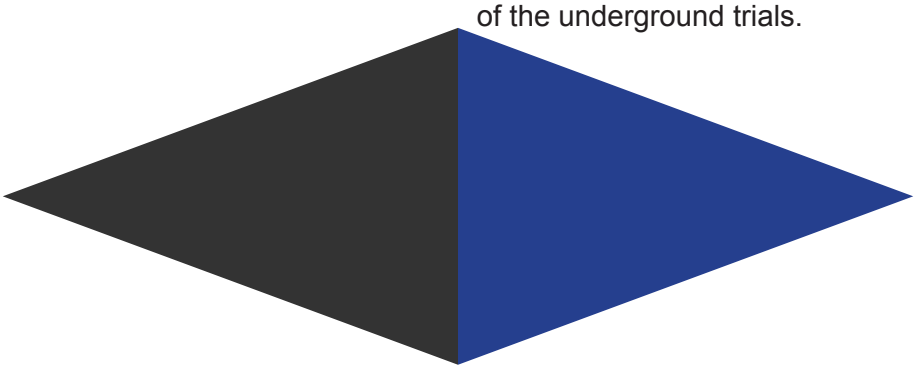
During this project both concepts were developed to a tested prototype. An experimental development model was first designed, built and tested, after which design reviews were held and prototypes built. The prototypes were tested in the laboratory after which the design was again modified. The prototypes were then evaluated underground in platinum, gold and coal mines. The underground evaluations showed that the prototypes were effective but that they could be further improved with certain minor modifications.

The underground evaluations showed that both the lightweight pinch bar and the mechanical “jaws” can be successfully used for making safe. The equipment is designed to reduce exposure to falls of ground, and to assist in reducing operator stress and fatigue.

These tools will have an impact in addressing rock fall fatalities and injuries associated with making safe or barring activities. The tools will also improve the overall quality of barring, especially in excavations requiring a lot of work, and this will reduce the rockfall hazard from loose pieces becoming dislodged in the longer term.

Recommendations:

It is recommended that both making safe tools be manufactured and marketed by an industrial partner once they have been improved, based on the findings of the underground trials.



63. Report to SIMRAC demonstration of overcore stress measurement from surface using the SIGRA IST tool

By Gray I, Sigra Pty Ltd, Ref: SIM 020202, 2003

SUMMARY:

This report has been prepared for SIMRAC and contains the results of a demonstration in South Africa of Sigra's in-situ stress testing system by Ian Gray. The test site originally planned was at Nooitgedagt Colliery but the failure of the drilling contractor to drill and deliver intact core meant that the location trial was switched to Goedehoop Colliery. Here Zaaiman Exploration operated the equipment completely successfully. From first seeing the equipment at 10 am to completed packing up at 5 pm three successful overcores were undertaken at 60 to 63 m depth.

This report contains a description of the stress measurement process and the results of the stress measurements at Goedehoop.

The results of the stress measurements show a very low horizontal stress regime but one in which the stresses are consistent in magnitude and direction. The major principal effective stress is in the region of 4 to 5 Mpa, which is small in the context of a rock UCS of 100 MPa. Of more concern if longwall mining were being considered is the minor principal stress which is in the range of 0.4 to 2 MPa.

This low stress could lead to caving problems. In the context of the bord and pillar mining being used the stress regime is considered benign.



Image source: CSIR

64. Development of a smart rockbolt for underground monitoring operations

By Moema J.S., Paton R., Scheers P.V., Mintek, Ref: GEN 810, 2003

SUMMARY:

Rockbolting is a prerequisite for providing roof support in mining operations. The problem of rock pressure becomes worse as mining becomes deeper. Most of the rockbolts designs are made from simple carbon steel. The related problems of uncertainty as to the quality of installation of grouted steel tendons for reinforcement of rock around tunnels and stope gullies, and the effect of corrosion, rock deformation and repeated dynamic loading from seismic events on the integrity of even well installed tendons, have been long recognised both in South Africa and world-wide.

Support, which is adequate on installation, may be insufficient when loading conditions change. A smart rockbolt was therefore proposed that, when stressed underground, undergoes a microstructural transformation, with its properties changing from a non magnetic to magnetic depending on the degree of plastic deformation, thus leading to a change in longitudinal ultrasonic velocity. These properties can be monitored using a portable magnetic and ultrasonic instrument respectively to determine a change in loading conditions, and to warn of a possible overload condition. The smartbolt acts as a sensor for determination of stresses in mine workings. Also, since the smartbolt alloy could be installed in a mine containing aggressive minewater, the opportunity has been taken to study some of its corrosion properties.

This report detail the research work performed at Mintek for SIMRAC into the development of a sensor material (smartbolt alloy) using metastable austenitic stainless steel. This study was split in to two phases. Phase 1 investigated the use of magnetic methods in measuring the stress or microstructural transformation in an individual bolt in both laboratory and underground environment. The corrosion performance of the smartbolt alloy was evaluated in synthetic mine water and compared with that of AISI 304 and 316 as well as mild steel. Phase 2 investigated the use of ultrasonic methods, modelling of the smartbolt alloy and testing underground.

Phase 1: Mintek identified an alloy composition as being suitable for acting as a smartbolt material under the influence of strain. Laboratory work has shown that there is good correlation between ferritescope (magnetic) readings and those from a magnetic balance. Some problems do, however, exist with the ferritescope method of monitoring, in particular the depth of penetration. However, tests were carried out underground.

Phase 2: Characterisation have been performed under laboratory conditions and show that the smartbolt alloy has a good combination of high strength and ductility; this is important in order to resist extreme conditions of high tensile and shear stress, much as a conventional roof bolt. Comparisons have been made with standard grades such as Type 304 stainless steel. Modelling load-strain characteristics of smartbolt alloy including the work-hardening behaviour have been performed.

The corrosion rate of the 12Cr-9Mn-C steel (smartbolt alloy) is more than two orders of magnitude lower than that of mild steel and one order of magnitude higher than that of AISI 316 (Figure 1). The value close to 0.01 mm/y over a large pH range is quite acceptable, but the corrosion rate increases sharply below pH 4.
A portable ultrasonic (USM 25 DAC) monitoring device was used to measure the longitudinal velocity during the calibration trials. The results show a good correlation between the longitudinal ultrasonic velocity and strain (Figure 2). Potential smartbolt warning guidelines were drawn up based on load/ferrite and ultrasonic velocity readings.

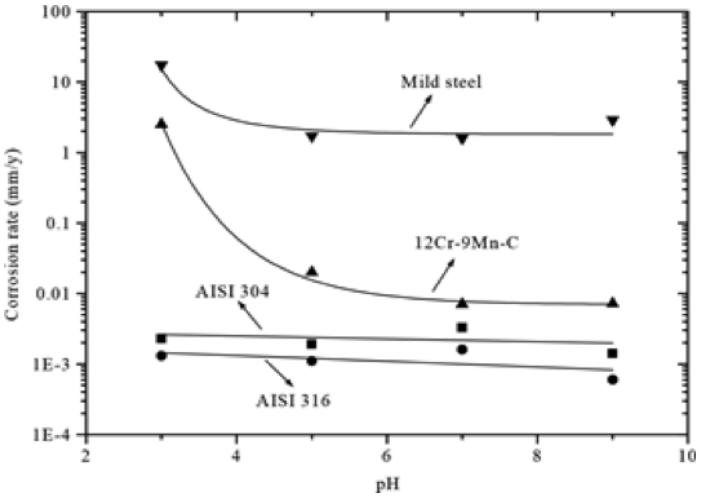


Figure 1: Corrosion rates as a function of pH

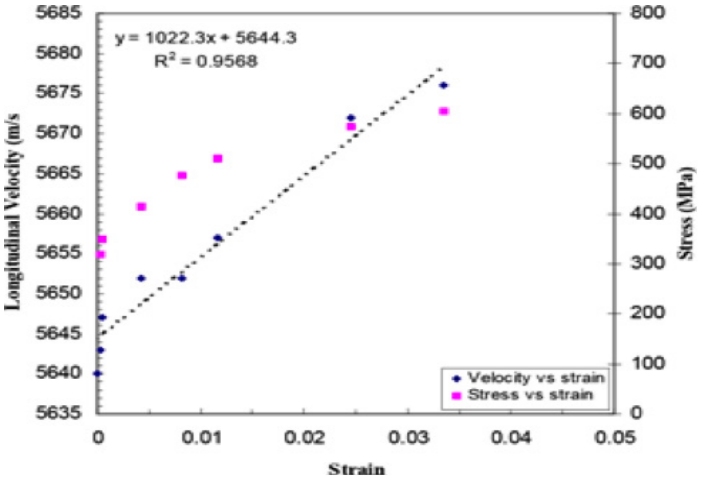


Figure 2: Longitudinal ultrasonic sound velocity versus strain of the smartbolt alloy

CONCLUSION:

An alloy composition has been identified as being suitable for acting as a smartbolt material under the influence of strain.

Some problems do, however, exist with the ferritescope method of monitoring, in particular the depth of penetration. The use of the smartbolt concept together with ultrasonic monitoring can be an effective and simple method of monitoring the load in a bolt.

The readings are accurate and the technique should help to promote safety in the industry. Information can be gathered and logged quickly to determine whether bolts are bent, broken, or overloaded.

Ultrasonic methods appear to be a satisfactory method for interrogating smartbolts but the overall hardware package may need to be refined with continuing feedback from mining personnel. Smartbolt alloy is particularly suited to application where corrosion is problematic.

65. Effective face support system to minimise rockfalls

By Ottermann R.W., von Wielligh A.J., Burger N.D.L., Handley M.F., de Wet P.R., BE@UP, University of Pretoria, Ref: SIM 020203B, 2004

SUMMARY:

This project aims to develop an effective face support system to minimise rockfalls in rockfall-prone mines. The primary output of this phase1 of the project was to develop and surface test a technology demonstrator roof support unit and to do a risk assessment of the system. Mines with rockfall problems in the face areas will use the roof support system to reduce fatalities and injuries in the face area of stopes.

The roof support system consists of two similar support units connected via two crank mechanisms to each other. Each unit consists of a headboard supported by a “wishbone” structure (top and bottom leg). A threaded bar with struts similar to a scissors jack keeps the legs apart.

All the components are manufactured from steel. To move the system forward one support unit is collapsed and then hangs from the other to which it is connected. The collapsed unit is then manually cranked forward and pre-stressed. If the required position is still not achieved the other unit is released and moved forward. Different models will be used for different stoping heights.

The specification for the system was determined and presented at a workshop with industry participants. Different concepts were developed and evaluated against the system specifications. A technology demonstrator was then developed and surface tested. The technology demonstrator development process included detailed design, building and testing of components and sub-systems, design reviews and the building and commissioning of the technology demonstrator.



Photo of support system in test stope.

The testing of the technology demonstrator was done in a 500-ton hydraulic press and in a mock-up stope. A risk analysis, in which technical, logistical and economical aspects were assessed, was done to determine the critical areas of the system.

CONCLUSION:

The tests and risk assessment of the roof support system identified certain areas that need to be addressed in order to implement the new support system safely underground.

For the further development of the support system the areas that need attention are:

- Improve the design, taking the results of the tests into account.
- Quality control during manufacture.
- Correct and safe installation procedures.
- Safe operating procedures.

During phase 2 of the project, working prototypes will be developed for underground evaluation.

66. Required technical specifications and standard testing methodology for Thin Sprayed Linings

By Kuijpers J.S., Sellers E.J., Toper A.Z., Rangasamy T., Ward T., van Rensburg A.J., Yilmaz H., Stacey R., CSIR: Mining Technology , Ref: SIM 02 02 06, 2004

SUMMARY:

The main objective of this project was to formulate a standard testing methodology and to develop testing equipment in order to quantify relevant TSL parameters. During the course of this project, a series of international seminars was organised between the Australian Centre for Geomechanics, the University of the Witwatersrand, and the Université Laval in Canada. The testing methodology that is presented in this report is therefore not only relevant to South African applications, but is also broadly supported by the international mining community.

Two standard test procedures were formulated, namely a tension test and an adhesion test. The test procedures have been optimised based on laboratory and underground testing. Guidelines for the standard testing procedures were developed and are presented in the report.

The adhesion test can be used to assess the underground performance in relation to potential performance under laboratory conditions. While it is not the intention here to prescribe detailed requirements for TSL products, it is apparent from this study that only tough and strong bonding materials are capable of providing effective support resistance when large deformations need to be accommodated. However, certain brittle and stiff materials may provide very effective support when they are not subjected to large deformations. In fact, the perception that large deformations need to be accommodated seems to be misplaced in many of the observed cases.

It is therefore important that any potential application is carefully assessed.

A simple mechanical model that quantifies the interaction between a TSL, the substrate, and the retaining support is described in this report. With the aid of this model, the required TSL parameters can, in principle, be quantified.

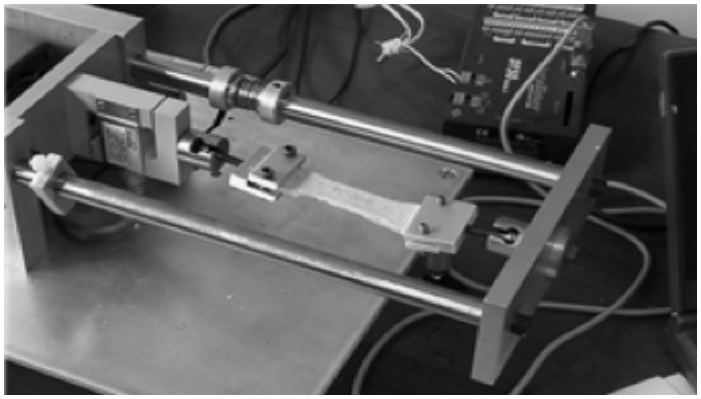


Figure 1:Tensile test



Figure 2: Adhesion test

CONCLUSION:

The aim of this project was to develop standard methods for the testing of Thin Sprayed Liners. The results of many workshops and discussions concluded that only tensile and adhesion tests should be considered at this stage. Guidelines for the standard testing procedures were developed. These tests provide quantification of the two main material properties and are also easily performed. The adhesion test is suitable for both laboratory and underground situations.

To optimise the test procedures, test devices were constructed and a number of tests undertaken. Tensile and adhesion tests were performed in the laboratory on a range of different products. Also, up to 20 adhesion tests were performed at each of five underground test sites. This experience allowed the test procedures to be optimised for ease of use and the quantification of the required measurement resolution.

67. A feasibility study of thermal imaging equipment to identify potential rockfalls

By Oldroyd D.C., Rock Mechanics Technologies , Ref: SIM 040202, 2004

SUMMARY:

In this project both high and low resolution thermal imaging equipment have been used to determine whether or not this equipment is capable of detecting loose rock in the hangingwall and sidewalls of hard rock mines. One surface application over burning coal seams was also examined.

It has been postulated that loose rock in the deeper mines should be of lower temperature because the thermal conductivity between the loose rock and the deeper and warmer rock will have been reduced by cracking. As cool ventilating air is continually passed through mine excavations there is a flow of heat from the warmer rock to the excavations. This should result in loose rock having a lower temperature.

The project was carried out over two stages. In the first stage both a high and a low resolution machines were used. A crosscut and three stopes were surveyed on three gold mines, two at intermediate depth and one at deep level. In the intermediate stope and haulage loose rock and cracks were found to be visible with both machines and the temperature difference between loose and adjacent solid rock was found to be between 1 and 2°C. In the deeper level mine stress fracturing in the stope hangingwall was visible with the high resolution machine. At the other intermediate depth mine the stope hangingwall was found to be of uniform temperature and did not show any significant features. During this stage of the project it was determined that the low resolution device took images of too low a resolution to be useful as a tool.

During the second stage only a high resolution machine was used, the FLIR Thermacam PM695. This imager was used in two platinum mines, one at shallow and one at intermediate depth. In stage 2 the device was also used on surface over an area of burning coal.

In both the platinum mines the imager proved successful in showing loose rock and this was particularly the case in one mine where a venturi had been used to assist ventilation.

The cold water spray accentuated the temperature differences between loose and solid rock.



Figure 1: Photograph of the face area of a stope

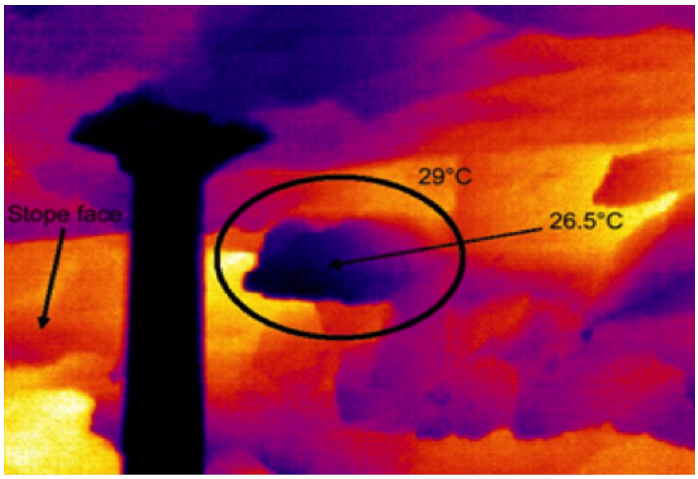


Figure 2: Thermal image of the portion of the stope shown on previous page

Above the colliery areas of burning were clearly seen but it was also noted that the sun very quickly warmed bare earth and led to high surface temperatures and care is needed to ensure the correct conclusion is drawn if used when the sun is up.

CONCLUSION:

The results obtained in these investigations are promising and the technique should be pursued further. Equipment could be used during “making safe” operations in a number of different mines to determine how useful it is. Investigation of the use of water to enhance the technique by cooling the rock surfaces should also be undertaken at suitable sites. Such trials should be done with a qualified rock engineer in attendance to record results obtained.

68. The design and development of an effective support system for tabular stopes

By Naidoo D., Singh N., Jager N., Grave M., Letlotla S., CSIR, Ref: SIM 020204, 2005

SUMMARY:

The single largest cause of injuries and fatalities suffered by the workforce in South African gold and platinum mines is rockburst and rockfall hazards. The majority of these rock-related fatalities (± 56 per cent) occur in the immediate vicinity of the stope face. Relatively few fatalities (less than five per cent) are associated with the back areas. The strike gully is associated with the second highest number of fatalities (15 per cent). This project is a continuation of GAP 708 that was initiated as part of a research thrust to develop an alternative support system technology to combat the hazards of rock mass instabilities.

The objective of GAP 708 was to develop a stope support system concept that addressed the deficiencies in current stope support systems, and that would significantly reduce the fatalities in the stope face area in the short to medium term. Rock engineering and operational specifications were detailed for the purpose of clearly identifying the requirements for a support design concept. These specifications are re-iterated in Chapters 2 and 3 of this report. Eleven concepts were generated and the remotely advanced headboard system was selected during a process of evaluation as the most viable system that could be implemented.

The continuation of GAP 708 entailed developing the remotely advanced support system to prototype level, at which stage it could be trialled on surface and in an underground environment. The system was developed from a basic conceptual format to manufactured prototype, which entailed a rigorous mechanical design process. At its current prototype level the system became known as the "walking beam" system. The system consists of two steel beams that are linked to each other, with each beam mounted onto a hydraulic prop. The combination of beams and props was automated to enable the operator to control the machine safely from a remote area.

The walking beam underwent a serious of surface and underground trials. Design enhancements were made to the system as the trials progressed. The system concept was proven from the successes attained from the functionality tests in surface rigs, notably at the Savuka test facility where the system withstood freefall drop tests. The underground trials were conducted at Randfontein Estates and East Driefontein Gold Mines.

CONCLUSION:

The surface and underground tests performed on the system from the beginning of its development added up to a thorough evaluation of the system. The operational shortcomings were identified and recommendations were drawn up for improving upon the system's implementation with respect to present and future developmental stages. In its current status the system is not suitable for implementation.

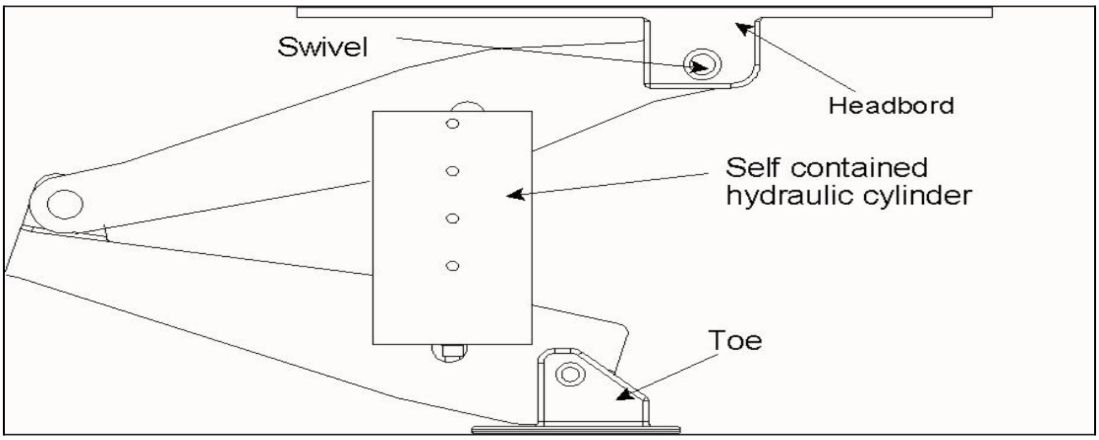


Image Source: CSIR

69. Effective roof support for tabular stopes

By Ottermann R.W., von Wielligh A.J., Burger N.D.L., Handley M.F., BE @UP, University of Pretoria, Ref: SIM 020203, 2005

SUMMARY:

The primary output of this project is to develop and evaluate an effective roof support system and to do a risk assessment of the system. Mines with tabular ore bodies will use the roof support system to reduce fatalities and injuries in the face area of tabular stopes particularly where rockbursts pose a significant risk.

The roof support system consists of two similar support units connected to each other by two cranks. Each unit consists of a 210 x 1200 mm headboard supported by a “wishbone” structure (top and bottom leg) and a selfcontained loading/yielding mechanism that keeps the legs apart. The yielding mechanism consists of a selfcontained hydraulic cylinder, a pump to load and prestress the unit and a relief valve to accommodate slow yielding of the system. The system yields to both slow and sudden stope closure by a combination of elastic deformation in the wishbone structure and the pressure relief valve.

Rapid yielding is mainly accommodated by the flexibility of the “wishbone” structure, and this also gives the system the necessary rebound, since stopes do not only undergo monotonic closure during a seismic event. The system is therefore more likely to stay in place, providing support resistance and energy absorption during the critical moments of the rockburst. The yielding mechanism is made from steel while the two legs of the “wishbone” and the headboard are manufactured from glass fibre.To move the system forward one support unit is collapsed and then hangs from the other by the two cranks. The collapsed unit is then manually cranked forward and re-loaded. The support system can be tailored to meet specific mining parameters such as stoping width and average face advance per blast.

During this project a prototype support system was developed and evaluated underground. Four different development cycles were necessary. Each development cycle included full design review, building of new prototype and testing of the prototype at Savuka. Three systems were manufactured for underground evaluations. These prototypes were evaluated underground at Great Nolingwe (2 locations), No. 4 Shaft of Harmony Gold Mine and No.8 Shaft of West Driefontein. The risk analysis, in which technical, logistical and economical aspects are assessed, was updated. The system was demonstrated to interested parties at No. 8 Shaft of West Driefontein, where a workshop was also held.



Photo of system

CONCLUSION:

The underground evaluations and risk assessment of the roof support system identified certain areas that need to be further addressed in order to implement the new support system safely underground.

Recommendations:

- Design review of the system with cost, weight, maintainability and durability as drivers taking results of underground evaluations into account.
- Redesign of the yielding mechanism with cost, weight, maintainability and durability as drivers.
- Integration of system with mining methods: Drilling, blasting, making safe, scraping and other support (permanent support, backfill, etc.)
- During the follow-up phases of the project it is recommended that the system be devolved further to address the shortcomings and to do a full stope evaluation of the system.

70. An investigation into the support systems in South African collieries

By Canbulat I., Wilkinson A., Prohaska G., Mnisi M. Singh N., CSIR: Mining Technology, Groundwork Consulting (Pty) Ltd , Ref: SIM 02 02 05, 2005

SUMMARY:

Roof bolting can be ranked as one of most important technological developments in the field of ground control in the entire history of mining (Mark, 2002). It is an essential component in the design of underground excavations and has been used to provide an overall ground improvement scheme since the middle of the last century.

Since the introduction of mechanical bolts in the 1940s, the amount of research into the understanding of the behaviour of roof bolts has been significant. Today, almost all coal mine roofs are supported with roof bolts in South Africa.

In the early years, the design of roof bolt patterns was based on local experience and the judgement of mining personnel. The suspension mechanism was the most easily understood and most widely used roof bolting mechanism. However, significant advances have been made over the last 20 years, in particular, the development of resin anchors, tendon elements, and installation hardware. These advances have resulted in an increase in the use of full column resin bolts.

The design of roof bolt patterns has also been improved, and four main rock reinforcement techniques have been developed: simple skin control, beam building, suspension, and keying. The geology and the stress levels determine the appropriate mechanism for a particular application.

Investigations into the causes of roof falls in South African collieries have highlighted that, whilst roof conditions are comparatively better in South Africa, the roof bolt densities are relatively low in comparison with those used in the USA, the UK and Australia. Therefore, the main cause of FOG was found to be excessive bolt spacing, causing skin failures between the bolts.

A study of falls of ground in South African collieries by van der Merwe et al. (2000) concluded that the majority of falls of ground occur under supported roof.

For these reasons it was decided that roof support systems should be investigated for the purpose of obtaining an understanding of the fundamental mechanisms of roof support systems and developing guidelines and design methodologies for their improvement. To this end all of the currently available roof bolt support elements and related machinery were evaluated underground in three different rock types, namely sandstone, shale, and coal.

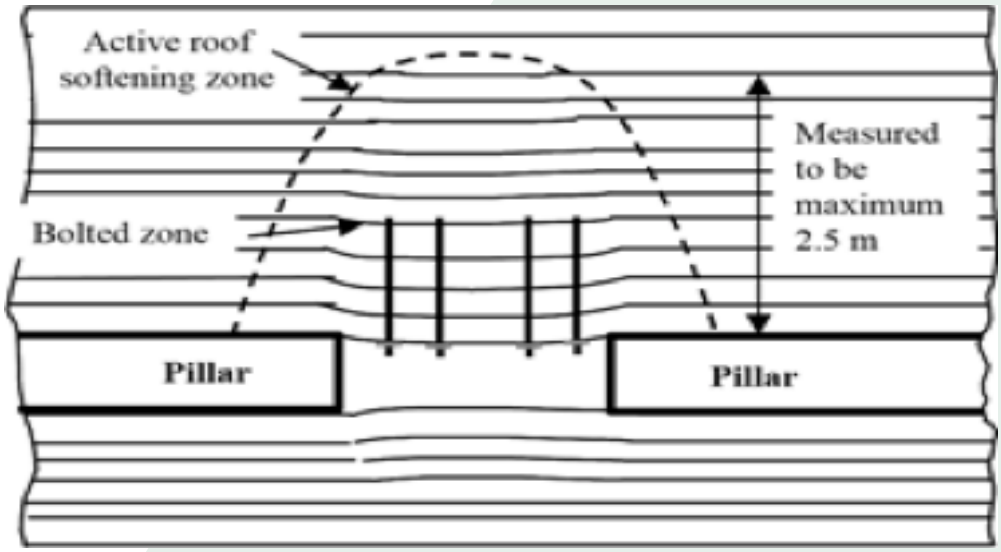


Figure 1: Zone of roof softening

71. Optimisation of support in collieries

By Canbulat I., Groundwork Consultancy (Pty) Ltd , Ref: SIM 02 02 05B , 2005

SUMMARY:

The aim of this project was to gain an understanding of the fundamental mechanisms of roof support systems and to develop guidelines and design methodologies for their improvement. The project scope was aimed at:

- Quantification of the effect of parameters that were identified by SIM 020205 as important in roof support system performance.
- Development of guidelines for testing procedures to achieve the maximum support performances as recommended by SIM 020205.
- An improved roof bolt installation technique.
- Development of new resin quality control and testing procedures.

SIM 02 02 05 highlighted the hot and cold rolling of steel in the manufacture of bolts as an area which should be investigated. Tests showed that the hot rolling process, when used in bolt manufacture, provides a more consistent product. Underground short encapsulation pull tests (SEPT) also showed that hot rolled bolts provided better support performance than cold rolled bolts.

“Finger gloving” occurs when the Mylar cartridge wrapper remains intact around the hardened resin. This prevents the resin from completely bonding to the rock. In order to determine the severity of this phenomenon, a series of test installations were carried out in Perspex tubes to observe the quality of resin mixing when using different installation methods.

These observations revealed that “finger gloving” occurred, to a greater or lesser extent, in almost all installations. It was found that more “finger gloving” was likely to occur if the bolt was pushed through the resin capsule before spinning took place. Whilst underground SEPT identified a slight difference in the performance of bolts installed in this method, indicating a higher percentage of finger gloving, the difference was not great enough to be alarming and still comfortably passed recommended support requirements. It is therefore concluded that “finger gloving” is present in most installations but has little effect on the performance of the system.

The importance of the shear strength of roof bolts that are currently being used in South African collieries was highlighted due to the need for shear strength in beam building. Laboratory shear tests were therefore performed on full column resin roofbolts, indicating that the shear strength of roofbolts is approximately 89 per cent of the ultimate tensile strength of the bolt, which is significantly greater than the previously assumed 50 per cent of tensile strength. It is established that the shear strength of 20 mm roof bolts currently being used by the steel supplied by ISCOR is approximately 213 kN.

SIM 020205 identified the method of roofbolt installation as an area which must be investigated. Four different types of installation were identified, namely:

- Flat end bolt spun through resin;
- Flat end bolt pushed through resin, then spun;
- Angle end bolt spun through resin;
- Angle end bolt pushed through resin, then spun.

Visual observations on the quality of resin mixing in each method were carried out. It was observed that when bolts, either flat or angle end, were pushed through the resin capsule before spinning, then a displacement of the catalyst took place. The catalyst is less viscous than the resin in a resin capsule and so when a bolt is pushed through the capsule, the less viscous material is displaced to the bottom of the hole. This effect is increased by gravitational effects in vertical holes. Mixing of resin and catalyst cannot take place properly therefore reducing the effectiveness of the support.

In addition, underground short encapsulation pull tests were conducted to quantify the effects of the different methods. The results showed that whilst differences in grip factor were minimal, large differences were found when comparing the stiffness of a system. In tests done for 15-second and 30-second resins, grip factors for an angled bolt pushed through the resin before spinning were the poorest for both resin types. The other three installation methods performed equally well. In terms of stiffness, the flat end bolts outperformed the angle end bolts, with a flat end bolt spun through the resin capsule giving the best performance. It is therefore concluded that the best installation method to use

is a flat end bolt, spun through the resin, in order to achieve a stiffer roof support system.

Pull tests were also conducted on resins of various ages to determine the effect of expiry upon performance. It was established that performance of the system is reduced as the time increases past the expiry date. A resin, which had expired by 6 months, gave 33 per cent poorer performance than a new resin.

In order to determine a resin testing procedure, a series of tests were conducted using the testing facilities at Anglo Coal Rock Engineering Laboratory. Resins of different ages were tested. The results indicated that the older the resin, the lower the torque achieved. This indicates that a torque test could be a quick and easy method of determining expired resin. The effects of over or under spinning of resin were observed and the effects quantified. It was established that under spinning of a 15second spin to stall resin still provided results within the acceptable limits of performance. For a 30-second resin, over spinning resulted in reduced performance and it was concluded that a 30-second resin should not be spun for more than 20 seconds no matter how slow the spinning speed.

72. The use of tendon support in tabular stopes

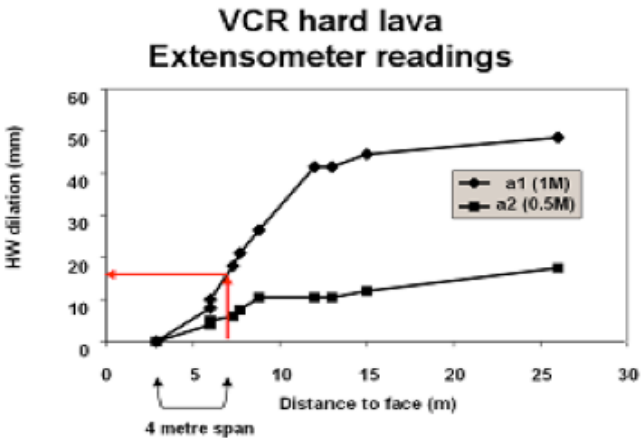
By Sellers E.J., Roberts M.K.C., Lamos R.A., Roberts D.P., Watson B.P., CSIR Mining Technology, Ref: SIM 02 02 08, 2005

SUMMARY:

The aim of this project was to determine whether rock bolt support systems could improve face area hangingwall stability. If the efficacy of rock bolting could be demonstrated, then rock bolt support design methodologies for different generic hangingwall types would be developed. These aims were addressed through underground monitoring and numerical modelling, both involving comparison of bolted and unbolted sites in different generic reef types.

The underground monitoring phase of this project was compromised by various factors that made direct comparison of bolted and unbolted sites very difficult. The monitoring did, however, reveal a number of trends which bear on the design of support systems.

Repeated profiling of the hangingwall on both the Ventersdorp Contact Reef (VCR) and the Carbon Leader Reef (CLR) has provided strong evidence that the addition of rock bolts to the other stope support systems can significantly improve the stability of the hangingwall, particularly in the vulnerable face area. There is also strong evidence that rock bolts, installed in the jointed rock adjacent to faults on the Carbon Leader Reef, stabilise this blocky ground.



VCR hard lava Extensometer readings

One metre long vertical extensometers were installed in the hangingwall of the VCR and CLR. The data derived from this instrumentation indicated a rate of opening in the hangingwall of about 3 to 4 mm per metre of stope face advance, implying that an extension of 12 mm to 16 mm occurs in the hangingwall of the stope face area, as indicated below. This amount of extension could cause tensile failure in a resin-grouted rock bolt and also brings the applicability of cement grouted rock bolts into question. Yielding rock bolts (for example, cone bolts) could be applicable in these areas.

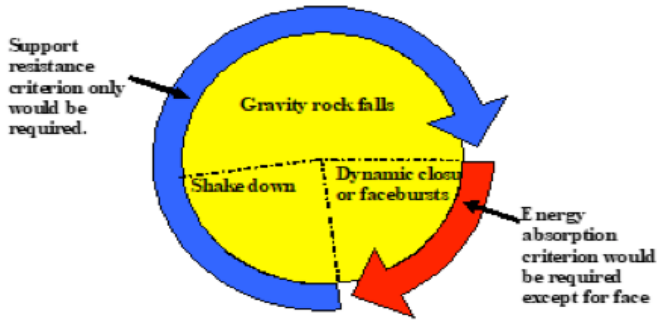
Hangingwall extensometer data obtained from other reef environments (Vaal Reef (VR), Merensky Reef (MR) and UG2 Reef) showed that the vertical extension in the hangingwall is not large and can be accommodated by all commonly used rock bolts.

Continuously monitored load cells indicated that the tensile stress in the bolt decreased by up to 20 per cent during the blast. This reduction in the bolt load may account for the large number of "loose" bolts which appear to have lost their installed pre-tension after the blast. The susceptibility of the different types of rock bolts to blasting effects should be determined and methods for preventing bolt tension loss investigated. It was suggested that some indication of bolt integrity should be provided at the exposed end of the bolt. A modified crimp-nut or shear pin system could be used to indicate the integrity of end-anchored bolts; however, an electronic device or the use of "smart materials" may be necessary to address this problem for full-column resin or grouted rock bolts. Further research into such systems may be necessary.

Significant shear movements were observed in boreholes at the VR site. This aspect of rock bolt behaviour was not studied at all in this project; however, this should make up part of the design criteria for the rock bolt support system. More research is required to quantify the ranges and define the selection criteria. Numerical modelling indicated that some of the underground geotechnical results could be reproduced. For example, the model closure profiles qualitatively approximate those obtained from closure readings underground for VCR stopes. In CLR environments, the numerical models indicated there was an improvement in stability of the hangingwall where rock bolting was installed.

The analysis of a gold-mining-based fatal accident data base has shown that in excess of 75 per cent of the rock related fatalities in stopes occurred by gravity or by "shake downs". "Shake downs" refer to the falls of ground that are triggered by seismicity, but are not accompanied by dynamic closure. It may therefore be stated that a support system which satisfies a criterion based on gravity rock falls would also be effective in more than half the cases that involved seismicity (in which no dynamic closure occurred). This indicates that rock bolt support systems that are designed to cater for rock falls can partially address rock instability caused by seismicity (in which no dynamic closure occurs). This concept is illustrated below. Note that this statement in no way implies that the rockburst criterion does not need to be considered in the rock bolt support design. Where dynamic closure is evident, the energy absorption criterion must be satisfied.A methodology is proposed as a guide for the design of rock bolt support systems for stopes. The methodology is not complete by any means and requires elements of rock engineering judgment in order to determine a suitable rock bolt support system for the prevailing geotechnical conditions.

It was found that the use of conventional rockmass ratings can be used indirectly to determine the spacing and orientation of the rock bolts. However, the overall design requires more detailed analysis of geotechnical parameters and hangingwall deformation characteristics. In addition to the analysis of fall-of-ground databases for ground control districts and the position of stratigraphic parting planes in the hangingwall, probabilistic keyblock analyses or block geometry analyses can be used to estimate optimal rock bolting requirements.



Recommendations:

To obtain statistically significant data on rock bolt performance, it is recommended that;

- the monitoring carried out in this project be repeated at a number of sites representative of the generic reef types.
- further research be undertaken to determine the failure strain of rock bolts typically employed in the South African mining environment.
- the influence of multiple opening discontinuities on fully coupled bolts should be investigated to determine the influence of discontinuity spacing.
- situ pull tests should be conducted on representative samples of resin–grouted bolts installed underground. This will give an indication of both the quality of installation and of how close the bolts are to failure at various positions within the stope.

73. Technology transfer of the risk assessment tool for rock-related risk issues

By Brink A.V.Z., CSIR Mining Technology, Ref: SIM 03 02 01, 2005

SUMMARY:

The outcome of earlier SIMRAC projects, GAP 608 and Gap 714 was a prototype software package that was a combination of an expert system approach for determining overall risk assessment, a risk control recommendation, and a basic Graphic Information System (GIS) approach for graphically overlaying input parameters and the output risk. The expert system used a Bayesian probabilistic approach to combine various types of workplace-related information, a quantified seismological environment, and exposure of underground staff to provide an overall rock engineering (RE) risk rating. The risk assessment was dynamic in that it was recalculated each time any input parameters changed, for example each time a seismic event was recorded in the area of interest.

The objective of this project was to provide clear documentation of an actual application of a rock engineering risk assessment tool as developed earlier. As the validity of risk assessment cannot be shown by relating to specific short term incidents, the project attempted to evaluate the method after an extensive period of use with a large data set. This could not be achieved in real-time because of severe limitations in access to real-time seismic data on mines, as well as in system interfacing to an operational mine's database for rock engineering parameters.

The project ran a case study demonstration over two years on, primarily, seismological data. The continuous risk assessment on dynamically changing input parameters was proven. The case study allowed for correlation between assessed risk and the subsequent hazards experienced.

The case study included a continuous assessment of RE risk on a mining section on the Ventersdorp Contact Reef (VCR) in the Far West Rand. Over a time window of one year, a significant majority of the larger events (Magnitude >1) in the immediate vicinity of the operating panels were preceded by a noteworthy increase in the risk levels. Similarly, a significant minority of all high risk levels did not manifest into a relatively large event. These conclusions were subjective and reached by viewing the risk display over a period of time. An objective and quantified measure of determining success rate was not proposed in this project. Absolute quantification of success rate should be possible, but then again, the implication is that this concept is being presented as a seismic prediction tool, rather than RE risk assessment.

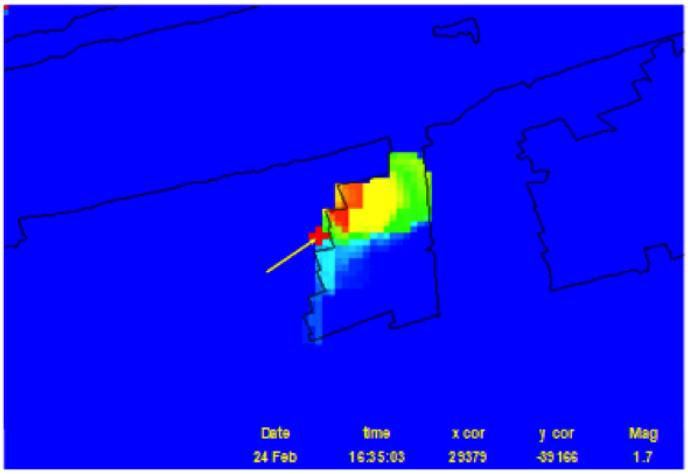


Figure 1: A Magnitude 1.7 event indicated on the risk assessment contours as existed immediately prior to the event.

The project may be criticised in that the non-seismic aspects of RE risk assessment did not contribute to the case study evaluation. The interfacing problems to mine RE data were the cause of this valid criticism. However, the concept of using changing input evidence that could influence the probability of a output hypothesis being true, was well proven.

The software platform used was ideal for development and proof of concept, but an operational tool, interfacing to mine systems and user interfacing, still needs to be developed.

CONCLUSION:

The project output is compiled on a CD, comprising all the evaluation coding, input data and results. A future software implementation of the proven concept should be achievable by a professional software developer, without this person needing to understand the basic principles employed in this demonstration/evaluation version. The important outstanding aspects of this potentially useful tool are mainly the real-time data and user interfacing. It is recommended that such an investment should be made.



74. Testing of stope support systems under dynamic and static loading conditions

By Human J.L., Harvey F., Naismith A., Ortlepp W.D., SRK Consulting, Ref: SIM 020305, 2006

SUMMARY:

Previous work and the original SIM 020305 proposal accepted by the SIMRAC committee focused on the performance characteristics of stope support systems under dynamic loading conditions. During the current project, the inclusion of safety net testing under gravity driven fall of ground conditions made the project applicable to stope support systems in shallow as well as intermediate and deep level mines.

A primary output was to identify modes of failure in the support systems under gravity or dynamic loading and their interaction with a discontinuous hangingwall beam. The interaction between SRK, support suppliers and mine personnel led to improvements being made to the support system through the better understanding of the support system behaviour under controlled conditions. This understanding assisted mine rock engineering practitioners to improve their selection of support systems from those commercially available.

The potential impact of this research project is improved safety in relation to rockburst and rockfall hazards through better control of the discontinuous hangingwall beam between support units in the stope face area.

During the three year project, 122 tests were performed at the drop test facility. Various types of stope support systems were evaluated under dynamic and gravity driven fall of ground conditions. These types covered yielding timber elongates, interlinking systems, safety nets, elongates with headboards and experimental “Walking” stope support units.

CONCLUSION:

With regard to the more specific objectives of the programme the following conclusions can be reached:

- Elongates with headboards are more effective in limiting differential displacement of the hangingwall blocks between support units than elongates without headboards.
- A long headboard does not perform more effectively than a smaller, stiffer headboard with regard to limiting the differential movement between hangingwall blocks.
- Headboards must load the prop symmetrically, i.e. the headboard laggings should not be staggered.

Based on test results, improvements have been made to safety nets to increase their capability to retain gravity driven falls of ground up to 2 tons:

- Safety nets were developed to withstand minor rockburst conditions.
- Fluid filled plate prestressing of timber packs has allowed block rotation and potential hangingwall failure. This has been presumed to be due to the fluid in the plates moving away from areas of higher loading to areas of lower loading of the pack under asymmetrical loading conditions thereby allowing more deformation. The transfer of fluid also inflates the portion of the prestressing plate under lower loading thereby inducing tilting of the pack and allowing hangingwall block rotation.
- Calibration testing done at the facility gave a good indication that the test system behaves in a repeatable manner. This makes it possible to create reproducible conditions under which to compare different support systems.

75. Closure profiles of the UG2 and Merensky reef horizons at various depths, using different pillar types, in the Bushveld Complex

By Roberts D.p., Malan D.F., van Rensburg A.L.J., Grodner M., Handley R., SRK Consulting,
Ref: SIM040207, 2006

SUMMARY:

A large body of data was obtained and analysed with the aim of characterising Bushveld excavations in terms of closure behaviour.

Similar to measurements in the gold mines, it was found that Bushveld closure profiles contain a significant time -dependent component, even in the absence of mining. The characteristics of the closure profiles varied considerably from site to site, and even between adjacent stopes in the same area. Determining the dominant cause of these differences with so many variables in play was a challenging and ultimately valuable exercise.

Criteria for evaluating and comparing closure profiles were developed to simplify the analysis. The rate of steady-state closure (RSSC – expressed in mm/day) captures the time-dependent component of behaviour and provides a consistent parameter for comparing site responses and tracking changes in closure behaviour. The geometric closure rate (GCR - expressed in mm/m) was also found to be useful, but showed greater variability than the RSSC analysis. The closure ratio used to characterize closure in gold mines was difficult to isolate and quantify; it was therefore not used as a criterion for comparison.

A detailed analysis of closure profiles in terms of mining parameters was conducted. A discrete averaging process was suggested for analyzing closure data. This allowed the data to be condensed to a single value representative of the average. The use of discrete averages tended to reduce the influence of outliers and also provided a consistent means of comparing site behaviour.

The initial spans and relative positions of instruments seem to have had little effect on closure rates. Numerical modelling indicated that Young's moduli of between 2. 2 GPa and 8.6 GPa were required to match the measured closure rates. This level of material downgrading (from laboratory moduli of between 70 GPa and 110 GPa) is unrealistic and indicates that elastic boundary element models are not suitable for estimating closure in the Bushveld.

No clear correlation between closure rates and depth could be established, though there was a weak trend towards increased rates with increasing depth. No distinction between closure behaviour on the Merensky and UG2 reefs could be made.

The closure rates in stable and yield pillar sites were comparable, however the geometric closure rate in particular was notably elevated in the crush pillar sites. In general, the average geometric closure rates were 2.15 times greater than the average RSSCs, but in crush pillar sites this factor was around 6.1.

At at least one site, nearly all of the closure was associated with footwall movement. This result indicates that more effort must be concentrated on determining relative footwall and hangingwall movements. Ride values more than two times greater than the vertical closure were measured, which has implications for support and pillar behaviour. No correlation between ride and other parameters could be established.

Rock mass ratings did not explicitly correlate with the closure rates, however there was a strong correlation between the observed panel conditions and the steady-state closure rates. A grouping of sites based on ranges of average steady-state closure rates is presented below in Table 1. Future work should critically assess this grouping and amend or build upon it as appropriate.

Site	GCR ave. [mm/m]	RSSC ave. [mm/day]	Conditions	Characterisation and criterion
Impala 10 # 1766-3N	3.65	0.23	Stable conditions	Low closure RSSC < 0.75
Rowland #	1.30	0.33	Stable conditions	
Impala 10 # 2070b-4N	1.73	0.36	Stable conditions	
Impala 1 # - 17c13 1S	1.47	0.44	Stable conditions	
Impala 9 # - 17c45 3W	1.02	0.48	Stable conditions	
Impala 9 # - 17c45 4W	1.90	1.02	Poor conditions in face due to presence of nearby pothole	Moderate closure 0.75 < RSSC < 1.5
Impala 9 # - 17c45 8W	1.19	0.86	Moderate conditions – FOGs reported	
Frank II # - 30E42	4.06	1.08	Stable conditions	
Impala 10 # - 1969 3S	2.94	1.68	Generally good conditions	High closure 1.5 < RSSC < 4.0
Impala 9 # - 17c45 7W	3.54	2.38	Poor conditions – many FOGs reported	
Site B	7.41	2.61	Poor conditions – large FOG closed panel	
Site A (UG2 under Merensky remnant)	10.18	5.89	Poor conditions	Very high closure RSSC > 4.0

Table 1: Grouping of sites based on ranges of average steady-state closure rates

The following conclusions were reached regarding the use of closure rates to assess and compare closure behaviour:

- It appears that geometric closure rates may be influenced by the pillar type. Geometric closure rates are generally higher (relative to steady-state closure rates) in crush pillar areas than in yield or stable pillar panels.
- The rate of steady-state closure generally provides a good indication of rockmass conditions within the panel, and therefore provides a measure of the potential for instability and fall-of-ground (FOG). Steady-state closure rate, as defined, is also a consistent parameter for comparing site responses and tracking changes in closure behaviour.

An exponential function, characterized by the stope closure decay (SCD) parameter, also shows promise as an indicator of deteriorating panel conditions, showing a clear correlation between SCD and rockmass conditions. The data available for assessing this parameter was somewhat limited, however the promise is clear.

Future work should focus on determining how the SCD parameter can be employed as a measure of risk for normal face advance rates.

Arrays of closure-ride stations showed the distribution of closure across panel spans. Generally, the highest values were recorded at centre-span, but in crush pillars sites the highest closure was recorded at the furthest distance from the abutment, near the crush pillars. This indicated that the effective span extends across the entire panel. Distributed closure readings can provide an indication of whether pillars are indeed crushing.

Numerical modelling was employed in an attempt to replicate closure at one particular site, and to better understand the mechanisms governing closure in Bushveld stopes. Detailed site modelling failed to replicate closure at the site in question, despite the introduction of a relatively thin discrete beam above the stope. It was concluded that this layer was not behaving like an intact beam, but that the discontinuous nature of the beam was accounting for the high closure through movement along joints and fractures. Blocky hangingwall modelling showed numerical instabilities

resulting from insufficient model dimensions. It was concluded that the time-dependent component of the numerical results was not analogous to the time-dependent component of real-life closure profiles. It is necessary for the model to achieve a steady state by the end of every stage, with true time-dependent effects modelled through visco-elastic or visco- plastic material and contact models. Despite these difficulties, it was shown that the rockmass geometry (in terms of the orientation and frequency of discontinuities) and the stress regime have a significant influence on closure behaviour.

Some very encouraging results were obtained regarding the use of closure data to warn of large collapses. In some cases, warning of up to a week prior to the collapse could be obtained due to accelerating rates of steady- state closure. However, the closure rates did not always indicate a hazard before a FOG occurred, nor did high closure rates (or changes in closure rates) always result in FOGs. Determining criteria for generating alarms based on closure readings should make up a major part of future research in this area.

Changes in geotechnical conditions can be identified and tracked through closure measurements. Different behaviours in adjacent stopes were captured by both the expanded closure profiles themselves, and by distributions of steady-state closure rates. A methodology was suggested for regular analysis and interpretation of steady-state closure rate distributions. This methodology would allow the rock engineer to quantify current closure levels, adapt support design if necessary, and track changes which may indicate impending collapse or changing geotechnical conditions.

The CIVAT (Closure Investigation, Visualization and Analysis Tool) computer program was developed to analyse real closure profiles with non-constant face advance rates and to produce distributions of closure and steady-state closure rate as a function of face advance. The program should be made available to practitioners for beta testing and developed further according to user needs.

Recommendation:

The potential for continuous closure readings to provide leading indicators of hazardous conditions has been demonstrated. Further work is recommended for quantifying baseline closure values for defining risk envelopes, and on developing autonomous systems for indicating changing risk profiles.



76. A holistic assessment of SIMRAC rock-related research to date

By RJ Durrheim, ET Brown, PK Kaiser and H Wagner, CSIR, Ref: SIM 04 02 06, 2006

SUMMARY:

Since the establishment of the South African Safety in Mines Research Advisory Committee (SIMRAC) in 1991, more than R250 million has been spent on rock-related research, representing some 500 man-years of effort. The improvement in injury and fatality rates was initially disappointing, but the coal sector has shown a marked improvement since 1999 and the gold sector since 2003. This study seeks to assess holistically the scope, quality, and impact of the SIMRAC research programme to ensure that the maximum benefit is derived from past work, and that future research work is appropriately directed. The information, conclusions, and recommendations contained in this report are intended to support SIMRAC decision makers for the next decade, at least.

The Status Report enables stakeholders to derive the maximum benefit from SIMRAC's past investment in rock-related research by structuring and analysing the large body of completed work and work in progress. SIMRAC projects have been reviewed, evaluated, and compared with similar efforts in Australia, Canada, and Europe, as well as several local collaborative research programmes (DeepMine, FutureMine, Coaltech 2020 and PlatMine).

The evaluation by an independent panel of international experts ensures that the assessment is unbiased and benchmarked against global best practice. It was concluded that SIMRAC has:

- identied the major research needs,
- conducted a comprehensive programme of research,
- contributed to the emergence of new technologies, such as systems for seismic monitoring and analysis, dip-pillar mining layouts, preconditioning, and prestressed elongates,
- contributed to the work of the collaborative research programmes, and
- contributed to the formulation of the codes of practice to combat rockfall and rockburst accidents.

The relatively small impact on safety statistics is attributed to: the increasing depth of mining; the increasing proportion of remnant and pillar mining; the long lead-time for new knowledge to be implemented; and shortcomings in the knowledge and technology transfer process. It was noted that implementation is not driven by researchers, but by industry and regulators. It was found that the scope, quality and impact of the research work is comparable to efforts in Australia, Canada, and Europe.

The Foresight Report is intended to enable SIMRAC to deploy funds and resources optimally in future, ensure that any important knowledge and technology gaps are filled, and that research efforts are strategically focused.

CONCLUSION:

Focus areas for research remain the measures to mitigate and manage the rockburst and rockfall risk. Emerging rock-related hazards associated with coal pillar and multi-seam mining, small-scale mining, and deep platinum mining should be proactively addressed. The research portfolio should cover the entire innovation cycle, ranging from basic science, engineering, risk assessment, and human issues, to technology transfer and the assessment of the impact of implemented technologies.

Recommendations:

- SIMRAC's mandate be reviewed to recognise the critical importance of rejuvenating competence.
- Greater emphasis must be placed on knowledge and technology transfer and implementation, though new funds will have to be sought for implementation activities, as the research budget has declined to a dangerously low level.
- It is desirable that universities play a larger role in the research endeavour so that the pool of highly qualified rock engineering practitioners is enlarged.

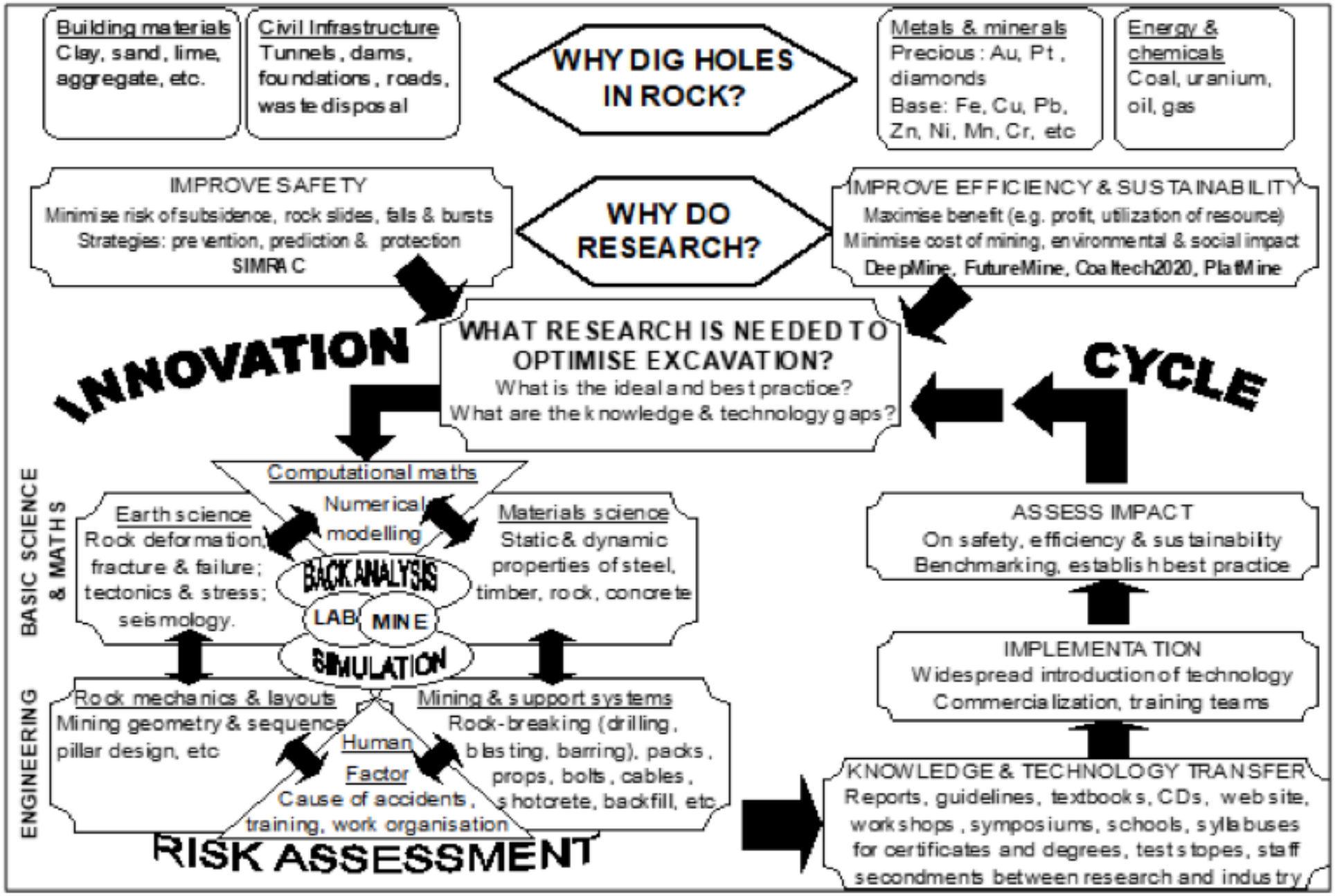


Figure 1: Mind map showing the rationale and methodology for rock-related research

77. Technology transfer of strata control knowledge

By Hagan T.O., CSIR Miningtek , Ref: SIM 02 02 07, 2007

SUMMARY:

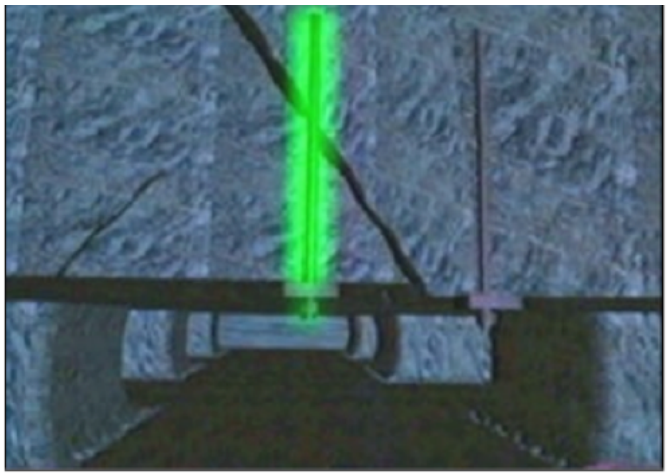
It is clear from several in-depth assessments of accident records that a better understanding of very basic strata control principles and practices, by production personnel, will improve worker safety. Workers lack of knowledge, with respect to aspects of strata control, increases their exposure to such strata control risks.

The primary objective of SIMRAC project 02 02 07 was to develop tools for the transfer of strata control knowledge. Project deliverable 1) below, applies only to collieries. Project deliverables 2), 3) and 4), focussed on illiterate underground mine team members on gold and platinum mines. These are workers who are not yet able to take advantage of the MQA learning process. The tools were designed to equip these workers with sufficient knowledge and ability to work safely underground until such time as they can take advantage of the MQA system. A significant number of workers (up to 65% on some mines) fall into this category and it was felt the most impact would be achieved by focussing on this group.

Project deliverables:

1. *A four-minute video* using animation to illustrate the causes and prevention of falls of roof in South African collieries. The video was based on the findings of SIMRAC project COL618 and was considered to be the most effective means of transferring the findings. A scene from the video is shown below and illustrates correct support positioning.
2. *Software* to demonstrate the capability of reconstructing, interactively, accident scenes and actions that have taken place underground in stopes. Once fully developed the program will enable an operator (say a junior rock engineer on the mine) to interactively set up an accident scene using libraries of support, equipment, accident sites and sounds as well as the personnel involved. Different scenarios can be set up showing how miners can be injured by falling rock and then how this could have been prevented had adequate support been correctly installed. These reconstructions can be used to train all levels of workers.

The potential of such a program for training is huge, especially when one considers it's interactive nature that allows specific accident scenes to be set up and played back. Further development is strongly recommended.



The figure below shows virtual images of miners, support, pumps etc. that can be set up at different positions as they existed at the time of a particular accident.



3. *A training facilitator's booklet.* The material, and methods of knowledge transfer referred to in the booklet, is targeted at illiterate underground mine team workers. The course is designed to equip these workers with sufficient knowledge and ability to work safely underground until such time as they can take advantage of the MQA system. The facilitator's booklet cover is shown below. On it is photograph of one of the wooden models discussed under the next heading.



4. *Small-scale wooden models.* The abovementioned training course requires the use of life-size mock-up facilities. Small-scale models were designed and manufactured upon which to base such a facility. Gold Fields, in particular Gold Fields Academy and Driefontein Mine, have been involved with the project from the start and will test the bridging course. The abovementioned models have been delivered to them and there are plans to build life-size equivalents at their Leadership Centre near Carletonville.

5. *Workshops and presentations.* These were set up to demonstrate the tools for technology transfer.

78. Development of a practical guideline for support quality assurance for different pumped product support systems

By Malan D.F., Groundwork Consulting (Pty) Ltd , Ref: SIM 04 02 05C, 2007

SUMMARY:

The objective of this project was to produce guidelines for the support quality assurance procedures of different "pumped" product support systems. The project is an extension of the SIM040205 project which developed guidelines for pre-manufactured products ready to be used when delivered to the mines (e.g. elongates). The main focus of the current project was on the following product types:

1. Backfill
2. Shotcrete
3. Grout packs
4. Thin sprayed liners

The project focused mainly on the support products and not the human aspects of quality assurance such as non-adherence to support standards.
As these pumped support products consist of a number of different components that are only "assembled" or applied underground, a slightly different approach in the proposed procedures had to be followed compared to the pre-manufactured support units described in the previous SIM040205 report.

The quality assurance procedures for "pumped" products contain the following important components:

- Quality assurance testing for the separate components of the support type before it is transported underground.
- Underground quality assurance procedures of the installed support types.
- Due to the pumped nature of the support, the frequency of testing had to be modified to take the specific attributes of the support into account.

From the data collected in the initial phases of the project in workshops and through questionnaires, it became clear that no standardised quality assurance procedures existed for pumped support products. The aim of the project was therefore to introduce some degree of standardisation. To keep the process practical and affordable, the procedures specify the minimum testing requirements. Additional testing can be conducted if requested by affected parties.

Owing to the time and cost involved in product testing, the intention is also to accumulate data over a period of time rather than to obtain comprehensive data immediately. This will enable the industry to absorb the additional testing costs without significant changes in support costs. For each product grouping, the minimum number of test samples is specified. Although this number may seem small in some cases, additional tests should be conducted if there is a large variability in the results. A statistical procedure is also given in this report to give guidance on the number of samples to be tested in cases where a large deviation in results is found.

Regulation 14.1 (6) of the Mine Health and Safety Act requires that a quality assurance system is in place that "ensures that the support units used at the mine provide the required performance characteristics for the loading conditions expected." As it is very difficult to simulate the underground loading conditions correctly in the laboratory for these pumped products, the procedures in this report had to find a balance between laboratory and underground testing.

Laboratory testing is nevertheless an important aspect of the procedures for pumped products to ensure the different components of the support type perform as expected, while the underground testing gives data on the behaviour of the support when installed underground.

Similar to the original SIM040205 report, the guidelines detailed in this latest report should be seen as a starting point for uniform improved quality assurance procedures within the South African mining industry. This document should be a living document that is updated as more information and experience is acquired from the testing programmes suggested for pumped products.

CONCLUSION:

Quality assurance procedures were developed for backfill, shotcrete, grout packs and thin sprayed liners.

The guidelines detailed in this report were developed as a starting point for uniform improved quality assurance procedures for pumped products within the South African mining industry. This document should therefore be seen as a living document that is updated as more information and experience is acquired from the testing programmes suggested in the report.

This project only focused on the actual support products and not the human aspects of quality assurance such as poor installation of support units or non-adherence to support standards. From the many workshops with industry, it was found that this aspect is crucial and probably the root cause of most of the problems experienced underground. As it was not included in the scope of the current project, it will have to be dealt with in a future project. Quality assurance procedures with more emphasis on support systems and adherence to support standards, rather than purely focusing on support products, are therefore required.

Product types such as mesh and lacing which are "manufactured" on site fell outside the scope of the current project and further work should be conducted to standardise the quality assurance procedures for these products.

79. Evaluation of the performance of shotcrete with and without fibre reinforcement under dynamic and quasi-static loading conditions

By Joughin W.C., Thompson J., Leach A.R., Le Bron K.B., Howell G.C., Chiwaye H., Dube J., Venter J., SRK Consulting, Ref: SIM 040204, 2007

SUMMARY:

The primary output of the project is the quantification of in situ performance of shotcrete under high stress, dynamic loading and deadweight conditions.

Methodology
A literature survey and a survey of the use of shotcrete in the South African mining industry were conducted in the early stages of the project.

Several underground sites were selected for underground monitoring of shotcrete, which cover a range of rock mass and stress conditions. These have been instrumented and monitored over the life of the project. At some of sites large displacements were experienced which caused significant damage to the shotcrete. This enabled the investigation of the interaction of shotcrete with the rock mass.

Non-linear modelling of various rock masses under different stress levels was conducted to quantify the effects of quasi-static loading. A series of ground reaction charts has been developed which can be used to estimate the maximum expected displacement.

A programme of laboratory tests was conducted, which included standard beam and panel tests, UCS tests and measurement of the fibre content.

Yield line analysis was used to quantify the performance of shotcrete. This was applied to the standard test panels and the required shotcrete characteristics have been determined for design. It was also used to estimate the shotcrete capacity at the underground monitoring sites.

A methodology for the design of shotcrete has been determined for application in underground mines. The method is based on the underground monitoring, numerical modelling, laboratory testing and yield line analysis. It summarises the important shotcrete characteristics and rock engineering inputs required for the design of shotcrete. Design charts are included to simplify the methodology.

CONCLUSION:

Underground monitoring
Five test sites were identified, established and instrumented at three different South African mines.

- At the two South Deep Mine sites quasi-static pillar loading led to high deformations of 70 mm and more over the 14 months of testing. The South Deep sites were ideal for the analysis of shotcrete failure mechanisms. The sites were also exposed to strong ground motions from both seismicity and nearby bench blasts, further adding to the contributions they have made.
- The Mponeng Mine 109 site captures the performance of fibre reinforced shotcrete at deep level mining depths where over-stoping or de-stressing conditions prevail.
- The Mponeng Mine 116 test site was established to investigate the influence of strong ground motions from seismicity on the performance of shotcrete.
- The Impala Platinum Mine site captures the effectiveness of un-reinforced shotcrete at controlling spalling ground conditions at intermediate mining depths where moderate stress changes are expected.

Thorough site assessments have led to the following conclusions and improved understanding.

- At Mponeng Mine 116 level induced deformation is related to the calculated PPV of the seismic event responsible for the strong ground motion.
- Instantaneous jumps in sidewall deformation were measured at the South Deep sites at the time of nearby longhole blast events. The vibrational intensity of a blast event has been found to affect the degree of deformation that results.
- In depth analysis of shotcrete crack formation and propagation at the South Deep sites identified that shotcrete fails in two distinct stages.
    - The primary stage of failure is identified by the formation and propagation of individual “primary” cracks throughout the applied shotcrete. This stage of failure is not believed

to result in a noticeable drop in the performance of installed shotcrete.
    - The secondary stage follows once primary cracks have propagated far enough to join or interact. In many cases the joining of primary cracks is achieved through the development of secondary cracking. This stage of shotcrete failure is synonymous with a marked drop in the performance of the applied shotcrete and is accompanied by significant increases in sidewall deformation and damage.
- It is suggested that primary cracks form mainly in flexure. The damage caused by these cracks and its influence on the overall shotcrete performance is localised and thus rather limited.
- Cracking during the secondary stage of shotcrete failure has been observed to commonly occur in flexure. Secondary cracking has a substantial influence on the overall performance of shotcrete.

Non-linear modelling

- A generic model has been created with the discrete element code UDEC, using a voronoi tessellation scheme to permit examination of the effect of rock fragmentation on support requirements.
- The model has been used to examine the relationship between support pressure and rock wall deformation in terms of Ground Reaction Curves (GRC) for a range of common rock mass circumstances and stress regimes that occur in both the gold and platinum/chrome tabular mines. A summary of maximum sidewall deformations is presented. These are related to rock type, rock mass rating in the form of GSI, and an estimate of the maximum stress concentration induced in the tunnel wall. The listed values are derived from models of 3.5 x 3.5 m tunnels. For other excavation sizes, a scaling factor can be applied.
- For selection of appropriate shotcrete systems for a tunnel in a certain rock mass under an estimated stress field the Ground Reaction Curves can be used to supply estimates of required support pressures to limit deformations, plus maximum movements in the tunnel wall. Considerations and modifications to the basic GRCs include scaling for excavation size, potential for extreme squeezing (deformations that are large and relatively indeterminate), and whether the tunnel will be subjected to stress change, including de-stressing.
- With excavation support requirements defined in terms of deflection, estimates of required shotcrete strength, thickness and deformability would be made using yield line theory.

Laboratory testing

- The laboratory test results have revealed that the incorporation of fibres in shotcrete has little effect on the modulus of rupture and the uniaxial compressive strength of shotcrete. However, the incorporation of fibres improves the post peak performance of shotcrete by offering it residual strength and energy absorption capacities.
- The shape of the specimens has an effect on the performance of shotcrete. This is especially the case with EFNARC panels due to the nature of the testing method, which requires the panel base to be flush with the stand on which the panel is placed during testing.
- It has been noted that the performance of fibre reinforced shotcrete depends on the types of fibres used in the shotcrete.

Yield line analysis

- A yield line solution for shotcrete and bolts has been developed
- The demand and capacity of shotcrete support can be estimated using yield line analysis
- The displacements calculated from numerical modelling can be used as a preliminary estimate of quasi-static loading demand, where measurements are not available. It is assumed that the shotcrete will deform by that amount and may enter its residual state, limiting its remaining capacity.
- The rock load demand from deadweight loading is estimated using a roof prism formed between tendons. It is relatively low for the tendon spacing (1.0 m to 1.5 m) used in the underground sites. However, the rock load demand increases with the cube of the spacing and this will become a concern for greater tendon spacing.
- The dynamic loading demand can be estimated by assessing the kinetic and potential energy of a prism of rock formed between tendons. The demand increases with the cube of the spacing and the square of the ejection velocity.
- The deflections at peak load (dp) in both EFNARC and ASTM C1550/RDP tests are highly variable, but can be used as an indication of whether, the peak load is likely to be exceeded. The dp values for unreinforced and polypropylene fibre reinforced shotcrete are much lower than for steel fibre reinforced shotcrete.
- The peak moment capacity of steel fibre reinforced shotcrete can be determined reliably using ASTM C1550/RDP test results.
- The peak loads and moduli in EFNARC steel fibre reinforced tests are quite variable and this is probably due to the non-unique crack pattern.

The moment capacities are however, similar to the ASTM C1550/RDP test results.

- Unreinforced shotcrete and polypropylene fibre reinforced shotcrete have highly variable peak moment capacities.
- Peak moment capacity is proportional to steel fibre density or mesh area and increases with the cube of the shotcrete thickness.
- The residual moment capacities can be reliably determined for both steel and polypropylene fibre reinforced shotcrete. The residual strength increases with increasing fibre content.
- Steel fibre reinforced shotcrete will control the displacement rate more effectively than polypropylene fibre reinforced shotcrete. Polypropylene fibre reinforced shotcrete can accommodate larger deformations.
- Unreinforced shotcrete does not have residual strength and therefore should not be used where quasi-static deformation or dynamic loading is expected.
- Energy absorption increases with increasing fibre content.
- The shotcrete capacity increases with increasing thickness.
- The potential corrosion of steel fibre after cracking should also be considered.
- The large scale panel tests indicate that the capacity of shotcrete does not increase quite as expected with increasing shotcrete thickness and for uniformly distributed loading. This must be taken into account in the factor of safety for design. More work is required to verify the actual relations.
- Actual underground capacities are highly variable and dependant on the local rock surface geometry and shotcrete application.

Shotcrete design methodology

- The design methodology developed can be readily applied to shotcrete design for underground excavations and caters for a variety of rock mass and loading conditions.
- It specifically assesses the quasi-static, deadweight and dynamic loading conditions.
- The required inputs for the design of shotcrete are listed and discussed.
- A required testing programme for determining shotcrete characteristics is described.
- The assessment of demand for each loading condition is described and design charts and tables are provided for each condition.
- The shotcrete capacity is determined from laboratory testing of shotcrete panels. The ASTM C1550 round panel test is recommended for design.
- Factors of safety are recommended for each loading condition, which take the uncertainty into consideration.
- Recommendations for monitoring of shotcrete performance are provided.

Recommendations:

- There is scope for further study into a relationship between PPV and induced instantaneous deformation. The existence of a “no deformation” threshold needs to be confirmed and the relationship after this threshold needs to be quantified.
- There is scope to further study the depth of fracturing in deep level, high deformation environments. Large variations have been noted to exist between depths of fracturing as measured using different types of instruments. It is believed that the intensity of fracturing is probably locally variable and the behaviour is more complex than may be expected.
- The concept of the primary and secondary stages of shotcrete failure deserves further testing. If possible, back analysis of underground failure incidents should be conducted assessing the stage and degree of shotcrete failure at the time of site collapse.
    - There is scope to develop a surface test under controlled laboratory conditions that can effectively demonstrate and repeatedly produce a progression from primary to secondary stage failure as defined in this research.
    - If such a test can be devised then it would be useful to conduct a series of tests that investigate the effect of varying common design parameters like strength, thickness and type of reinforcement on the duration of the primary stage of failure as identified by this work. Can the shotcrete be engineered to effectively prolong the time before entering the secondary stage?
- It is recommended that a further programme of large scale panel tests should be carried out, with varying tendon spacing, shotcrete thickness, different types and quantities of reinforcement for uniformly distributed and point loads. The data should be analysed with yield line theory.
- A small round panel test method should be developed for quality control purposes. This will be easier to transport underground.
- A dynamic test programme should be developed with a basic test configuration that can easily be quantified. This should also include panel tests with varying tendon spacing, shotcrete thickness and different types and quantities of reinforcement.

80. Elimination of Rockfalls – Measuring and monitoring to reduce the rockfall risk

By Brink A.V.Z., CSIR, Ref: SIM 060201, 2007

SUMMARY:

SIM 06 02 01 Track A has been a three year project contracted to CSIR and MIDDindi to develop procedures and methodologies towards the elimination of rockfalls. Track A, specifically, placed the emphasis on measuring and monitoring techniques to reduce the rockfall risk.

The proposed primary outputs of the project and the final deliverables are summarised as follows:

- Interpret the predominant and critical mechanisms involved in rock falls.
- Identification of critical parameters that will assist in signalling an alarm before a rockfall
- Design instrumentation that will allow practical collection of critical data
- Design algorithms that are able to combine critical parameters in order to activate an alarm
- Measurement of precursory indications of rockfalls. Comprehensive field testing of the technique/s.
- Identify new approaches that will stop or ameliorate the effects of rockfalls

In order to achieve the above outcomes, a series of enabling activities were planned, delivered and reported on in this report.

Year 1. Preparation

- Analyse rockfall mechanisms in all mining environments - including risk and observed precursors
- Literature survey of rockfall failure mechanisms
- Establish theoretic precursor sensing requirements
- Design/acquire appropriate sensing equipment
- Install a comprehensive monitoring system in a high risk environment

Year 2. Measuring/monitoring & innovation

Install and operate rockfall monitoring systems at three different mines (ideally in different commodities)

Year 3. Evaluation, reporting

- Develop and validate technology assisted procedure for entry examination in gold and platinum.
- Develop decision support tools for the continuous assessment of rockfall risk
- Install an extensive monitoring system a mining section (at least 6 panels) in a platinum mine.
- Maintain the monitoring system installed in both gold (Driefontein) and platinum
- Report on the implementation of a GoafWarn system at New Denmark

The extensive stationary sensor network per panel did not yield reliable early warning and it cannot be presented as a practical solution towards early warning in hard rock mines. Early warning of goafing in coal, both in longwall and stoping operations are successfully been implemented.

A Personnel Area Safety System (called PASS) employing similar technology and the monitoring of the same parameters, is a very positive outcome. It involves a mini-system with portable microseismic/ acoustic emission sensors and ultrasonic closure meters, to be employed in a localised mini-network prior to the shift and with local alarm recognition.

The most significant outcome of the project, however, has been the validation of thermal imaging and acoustic sensors as the ideal technology for use during entry examination. The most common attribute to falls of ground resulting in injuries or fatalities are small falls happening in the panel within meters from the face.

Recommendations:

Technology to improve the effectiveness of entry examination has the potential to singularly provide a significant positive impact on mine safety. In order to realise this impact, it is recommended that both the "Integrated Thermal Imaging and Acoustic Sounding Device (ITA) and the 'Personal Area Safety System "(PASS) be industrialised to a level of a commercially ready prototype.

Further research should be conducted towards optimising the performance of the in-stope sensors units in a 'noisy' environment. The microseismic sensors, in particular, are ineffective whilst drilling or cutting occurs in the immediate vicinity of the sensors.

81. Technology Transfer for "A Risk Based Approach to Enhancing Support Design in Bushveld Underground Mines"

By Walls J., Maduray C., Govender D., Boaduo A., Joughin W.C., SRK Consulting, Ref: SIM120201, 2013

SUMMARY:

A technology transfer program was carried out to facilitate adoption of research outcomes from "A Risk Based Approach to Enhancing Support Design in Bushveld Underground Mines". The risk based approach to support design enables an improved support system to be selected by comparing different mining support strategies based on the injury and cost consequences of Falls of Ground (FoG's).

Twenty stakeholder operations were selected from Champion Mines in the Bushveld Platinum sector to adopt the technology. The project commenced with training sessions together with Open House Management Solutions (OHMS) during the latter part of 2012, following which the operations were tasked to carry out a risk evaluation for support design.

Adoption of the technology comprises the three main stages of:

- Geotechnical Data Acquisition and
- Interpretation
- Keyblock Analysis
- Risk Evaluation.

SRK carried out reviews of the risk evaluation results at each of the stakeholder operations during February to June 2013. Each of the reviews comprised an underground visit to check data acquisition techniques, and a desktop review to check the data processing, analysis and interpretation of results. Guidance was given where required to address obstacles and improve results.

Outcomes

On completion of the project, it was found that varying success was achieved at each of the stakeholder operations.

Data Acquisition and Interpretation

Data acquisition and interpretation had been carried out with varying success on 80% of the operations. A shortage of tools and equipment on some operations prevented implementation, while on others, data had been acquired through the input of an external consultant.

Keyblock Analysis

The majority of the operations (65%) produced at least one or more support scenario analysis which was available for review to check the methodology and provide guidance to correct errors and improve results. A single operation succeeded in interpreting the comparative outputs from which to select an improved support design for the operation. Licensed, working copies of the keyblock analysis tool had not been obtained at the remaining operations.

Risk Evaluation

A risk evaluation from results of keyblock analysis had not been carried out on any of the stakeholder operations, however an attempt had been made at several shafts. IT restrictions or a lack of data prevented several operations from carrying out the risk evaluation.

Recommendations:

A complete adoption of the technology had not progressed to include the final stage of risk evaluation at any of the operations, however success was achieved to a large measure in data acquisition, interpretation and rockfall analysis which are essential elements in the process. Some of the operations had not succeeded in using the technology to carry out any of the stages in risk evaluation.

Going forward:

- A culture of data acquisition and interpretation, having Rock Engineering practitioners suitably equipped with the necessary tools and equipment needs to be promoted in the industry to obtain the basis for technical solutions to reducing the consequences of FoG's.
- Rock Engineering practitioners need to become better acquainted with the concepts of injury- and cost-based risk evaluation for support design.
- The technology needs to undergo further development to suit the requirements of changing ground conditions and mining methods.

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

By Walls J., Pillay O., Joughin W., SRK Consulting, 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.

83. Identify opportunities to improve the safety of barring-down practices

By Boodoo A., Jermy C.A., Kuppusamy V., Ramsaroop P., Golder Associates (Pty) Ltd, Ref: SIM 140202 , 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.

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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.

CONCLUSION:

The SIM 14 02 01 technology transfer project was successfully completed with substantial improvement over SIM 12 02 01. Potential for further development of the technology and continued training in skills was revealed. The principles of risk based support design were suitably well applied by several practitioners such that fall of ground risk mitigation using the technology will be possible on their respective operations going forward.

This study stemmed from the number of preventable rock related accidents that led to fatalities over a two and a half year period in South African Mines. Between 2009 to May 2011, 94 fatalities occurred. It was suggested that twenty four of these fatalities could have been prevented by remote barring. Mechanical barring had to be investigated as an alternative to intensive manual barring in areas where it could be practicably done. The task of barring is mostly completed manually, even in highly mechanised mining environments in South Africa. It remains an activity requiring considerable physical effort and skill and may be considered high-risk. This is verified by the number of accidents that have occurred during the act of barring.

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.

84. Development of methodologies for determining maximum spans (bord widths) applicable to platinum and chrome mines using bord and pillar mining method in the Bushveld Complex, South Africa

By Joughin W.C, Murphy S.K., Walls J., Zermatten C., SRK Consulting (South Africa) (Pty) Ltd ,
Ref: SIM 150201, 2016

SUMMARY:

Background of the Study
The study stemmed from a knowledge gap, namely the apparent absence of a common and/or structured engineering approach in determining maximum spans in platinum and chrome mines using bord and pillar mining method in the Bushveld Complex, South Africa. This was partly motivated in response to several major falls of ground which had occurred in recent times - some of which had tragically resulted in fatalities.

Objectives of the Study
The objectives of this study were to identify, or if necessary, develop a methodology for the design of maximum spans (bord widths) applicable to platinum and chrome mines using bord and pillar mining method in the Bushveld Complex, South Africa.

Methodology
The approach used to undertake the study was multifaceted. Personal interviews of key rock engineering personnel at a number of mines were undertaken and documentation relating to the design of maximum span was collected. Thereafter, a comprehensive literature review of material in national and international publications was undertaken. An initial design methodology was formulated and presented at workshops for further interrogation. Discussion elements from the workshops informed considerations and the methodology was refined.

Results of the study
There was an expectation that the reports and codes of practice (CoPs) collected would clearly outline the bord design methodology applied at specific mines, however, it was the case that the documents were largely retrospective - for example, it was a common finding that after a fall of ground incident, the span width at an operation was reduced from X to Y - but no explanation was given of the methods used to calculate the new span width are given.

Thereafter, it was found that a common approach was to use a rule of thumb method (taken from bord and pillar practices in the coal mining industry) where the length of support is designed by specifying a support height based on a 95% fall-out height for different ground conditions or ground control areas. The approach has proven to be ineffective in preventing the falls of ground that have occurred.

A methodology adapted from earlier work by Swart and Handley (2005) 1 was developed to address the project objectives. The methodology relates rock mass characterisation, potential failure modes (whether intact rock mass, structural, beam or rock mass failure), geometric optimisation and monitoring and draws on empirical design methods as well as analytical [computational] design methods for validation.

CONCLUSION:

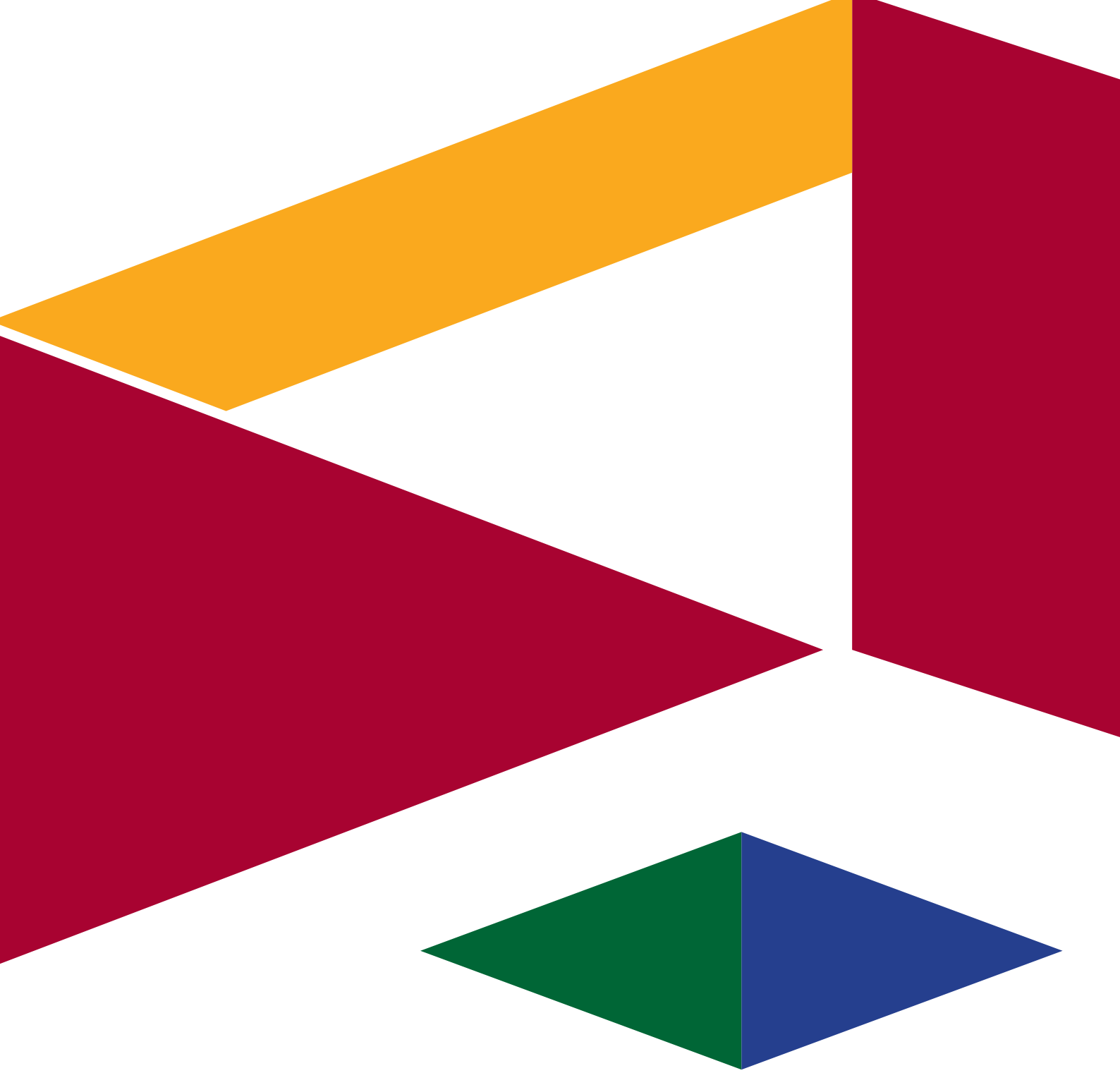
A methodology to determine maximum spans in platinum and chrome mines using bord and pillar mining method in the Bushveld Complex, South Africa was developed.

An e-Book was prepared, which describes the project, theoretical underpinnings and the methodology in greater detail.



Notes:

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



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