Rock-engineering strategies to meet the safety and production needs of the South African mining industry in the 21st century

by N.C. Gay*, A.J. Jager*, J.A. Ryder*, and S.M. Spottiswoode*

Synopsis

The South African mining industry is at present planning to extract orebodies at depths of up to 4 km. To do this cost effectively and, at the same time, ensure safe working conditions for mining, the personnel will require both the development of new strategies and the implementation of current rock-engineering technological developments on a wide scale. This paper covers most of the technological developments likely to be required for mining at very high stresses, but it is clear that improved regional support systems will need to be developed to ensure effective stability of the rockmass during mining operations. Similarly, it is clear that improved strategies for mining in large-scale geologically disturbed areas, where slip on faults or dykes can result in wide-spread rockburst damage, will have to be addressed.

* *Rock Engineering, CSIR Miningtek, P.O. Box 91230, Auckland Park, 2006 Gauteng.*

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Introduction

During the past two to three years, the South African mining industry has experienced the most difficult years of its existence. These difficulties have arisen because of poor commodity prices, rising costs, low grades, and productivity and safety problems. The safety and, to a lesser extent, productivity problems arise largely as a result of the high stresses imposed on deep-level excavations. Fracturing of the rock generally occurs in a controlled manner, but sporadic violent failures can suddenly disrupt mining activities and occasionally injure or cause the death of workers. For example, up to 67 per cent of the fatalities and 30 per cent of the injuries on mines are due to rock-related accidents. These problems are endemic to mining at moderate depths, but are likely to be particularly severe at very deep levels and to aggravate the already high costs associated with deep mines if advances are not made in the design of improved stress-tolerant layouts, better support systems, and more-efficient mining methods.

This paper addresses the potential for current and future developments in rock engineering to enable the gold- and platinum-mining industry to mine more productively and safely at depths ranging from the current average depth of about 1600 m to future depths of 4000 m. Examples of mining projects that have been delayed but would benefit from improved rock-engineering technology include the following:

- Mines of shallow to medium depth:
  Weltevreden Gold Mine
  Northam Platinum Mine
- Deep to ultra-deep mines:
  Target Project
  Sun Project
  Moab extension to Vaal Reefs
  South Deep
  Strathmore area.

The reefs to be mined in these examples are generally tabular but can vary in width from about 1 m up to 40 m. Thus, as the reefs are mined out, the rockmass will be subjected to large changes in stress and resulting energy release, which, unless suitable methods to control the rockmass are available, could lead to severe seismic and rockburst problems. These would be aggravated in the wider reefs by an increase in potential volumetric closure and by a reduction in effective rockmass strength with increasing excavation height.

Rock engineering strategies for safe and productive mining

Table I lists eight strategies that, in general, need to be addressed if mining at great depths is to be carried out safely. To address these strategies, the paper is divided into two main sections.

The first section details strategies 1 to 3, covering areas in which the technology is sufficiently developed to be implemented, at least in part, on mines. This is particularly true for strategy 1.

Table I

Rock-engineering strategies available to the South African mining industry to enable safe and productive mining at all depths

1	Implement current knowledge and technology to reduce rock-related accidents and improve productivity.
2	Develop improved regional support layouts that will enable safe mining at great depths and/or under high stresses while permitting maximum extraction of ore.
3	Develop an improved understanding of mining-associated seismicity, especially with respect to geological structures, and counter-measures.
4	Develop an understanding of rockburst mechanisms so as to be able to devise effective rockburst control measures that would also reduce accidents, production costs, and losses.
5	Develop support systems to meet the requirements of ultra-deep mines and high stoping widths.
6	Develop improved seismic and other methods for monitoring the condition of the rockmass, particularly with respect to its potential for failure.
7	Develop and implement mineworthy technology for the mechanization of mining operations so as to reduce the number and exposure of workers in the stoping area.
8	Develop effective non-linear techniques of solid-mechanics analysis to allow the exploitation of computer-aided modelling and design in the evaluation of mining strategies.

A further major area of research that needs to be tackled more aggressively than it has been in the past few years is the assessment of seismic risk during stoping operations. This, in turn, could require that the number of workers at any one time in, for example, the stope-face area be minimized. To ensure this and still maintain the required production outputs calls for increased mechanization of the mining process and, in turn, a higher level of technical competence among stope workers.

For the strategies covered in the second part of the paper, considerable research has still to be carried out before the technologies can be introduced onto mines.

Developed technology and its implementation

Implementation of current knowledge and technology to reduce rock-related accidents (strategy 1)

Current technology, which was to a large extent developed by the Chamber of Mines Research Organization (COMRO)—now the Mining Technology Division of CSIR (Miningtek)—under an industry-collaborative work programme sponsored by all the mining groups, which would bring significant cost and safety benefits to mines, includes the use of the following.

- Lightweight prop-headboard systems for close-in-face support (about 15 000 units in use), together with similar props coming onto the market, should lead to a significant reduction in falls of ground and accidents, most of which occur at the face.
- Cone-bolt yielding tendons (approximately 30 000 units have been sold to mines) should significantly improve the tolerance of tunnels to otherwise damaging rockbursts.
- The improved 3 m/s rockburst prop-headboard system (about 100 000 units in use) should reduce dynamic stope closure and hangingwall shake-out.
- Backfilling systems for local support of the stope face area should stabilize the hangingwall close to the stope face and reduce ground vibration during seismic events.
- Local and mine-wide seismic systems for the monitoring of mining in hazardous areas should permit the identification of potential seismic problem areas, and facilitate careful planning of extraction sequences and support requirements.
- Rockburst control procedures to reduce bursting of the stope face by preconditioning blasting should enable preconditioning blasting of the stope face to be implemented as part of the mining cycle, thereby increasing the safety and productivity of the workers.
- Prototype ground-penetrating radar systems for the assessment of the rockmass condition and the detection of geological structures should allow timeous detection of geological problem areas and quantification of the effectiveness of rockburst control measures.
- Elastic and non-elastic mine-layout design programmes should allow the better planning of regional support layouts and quantification of their effectiveness.
- Waterjet-assisted face cleaning should result in safer and cooler conditions for workers through the integration of face-cleaning operations with close-in-face support.
- Continuous scrapers for more efficient transportation of ore should provide significant increases in worker productivity.

All these developments have been tried, or are in use on some mines. However, their penetration into mines has generally not been significant, primarily because of cost. While this is understandable, it is important that their implementation should continue. The costs of purchasing and implementing a product could be small in relation to the benefits derived from its use. As an example, the cone-bolt has the potential, if properly installed, to provide significantly better tendon support than the methods currently in use, and its installed cost is only 20 per cent more than that of conventional tendons, yet it provides a considerably better rockburst-resistant tunnel support and a significant reduction in tunnel rehabilitation costs. Similarly, the close-in-stope system of face support should pay for itself within months as the result of a reduction in waste dilution.

In the implementation of any product, the training of mine personnel at all levels in the use and maintenance of the new technology is crucial. This should be done together with the product developers, work-study personnel, and human-resource staff to ensure staff acceptance of the need for and viability of the product.

Also required are appropriate staffing and better supervision than are currently practised. This would result in significantly greater productivity, with corresponding gains in profitability. For example, as highlighted by Noble and Van der Krog[1], the planned routine maintenance of rapid-yielding hydraulic props by properly trained staff is essential if the safety and productivity benefits of these expensive support units are to be achieved.

Development of improved regional support layouts (strategy 2)

Regional support systems for deep mines are aimed at reducing the frequency of rockbursts by stabilizing the mined-out areas through the use of support of such strength and stiffness that closure in the back area is reduced, together with a reduction in stope-face stresses and mining-induced seismicity. The current practice is to utilize a system of stabilizing reef pillars, which are designed to keep the levels of stress and energy-release rate within 'safe' limits during mining, and which are normally implemented with longwall mining methods. However, as the reef plane is mined out, the load on the stabilizing pillars increases, causing the foundations of the pillar–hangingwall–footwall system to fail. Large seismic events are generated during this failure, and footwall excavations can be damaged. Abutments also represent areas of high stress concentration and thus are prone to failure, releasing seismic energy. This is particularly likely to occur close to geological structures (Lenhardt[2]). To control foundation failure, either the percentage of reef extracted must be reduced, or backfill must be placed, although the effectiveness of backfill has not yet been fully assessed (Hemp[3]). A similar method, which is being used on steeply dipping reefs, is the use of crush pillars embedded in backfill.

With respect to the effectiveness of stabilizing pillars for regional support, Hobday and Leach[4] showed that, following the establishment of a stabilizing-pillar system at Western Deep Levels South mine, there was a marked reduction in seismicity. However, with time, the number of seismic events increased and, three to four years after the pillars had been established, the seismicity was significantly higher than before their introduction. Moreover, foundation failure of the pillars occurred in the back areas, damaging roadways and haulages. Thus, current knowledge indicates that, in the Carletonville district, stabilizing-pillar systems do not always fulfil their envisaged functions of reducing seismicity levels and controlling energy-release rates effectively. This does not appear to be true of other deep-mining districts, such as the East Rand, where pillars of greater width, at ERPM, appear to be more stable, as do the pillar systems at Kloof on the West Rand.

A major disadvantage of pillar systems is that they contain considerable quantities of locked-up gold; for example Hobday and Leach[4] estimate that the pillars at Western Deep Levels South contain approximately 5 t of gold, which is equivalent to R175 million in terms of revenue. This has considerable relevance to the design of regional support layouts for mining at depths of 3,5 to 4 km, where the capital cost of establishing mines is likely to be so great that all the ore that can be extracted safely should be mined.

The implications of this will affect all mining operations. For example, early extraction of the shaft pillar will be imperative. This is because, at depths of 3000 to 4000 m, the dimensions of the shaft pillars designed according to current criteria and at a critical stress level of 100 MPa would need to be about 1000 to 4000 m in radius at depths of 3 to 4 km respectively (McKinnon[5]). Pillars of that size could lock up as much as 1500 t of gold, which would have to be extracted at the end of the mine's life. Thus, to protect the shaft environment, a system of satellite pillars would need to be established.

The siting of service excavations, ore-pass systems, etc. is also a matter for concern and, at depths of 3 to 4 km, the stresses acting in a pillar are sufficiently large to cause extensive damage to excavations. If the shaft pillar is extracted and the voids are backfilled, the stresses are unlikely to exceed the levels of virgin stress, and the potential for damage to the service excavations due to high stresses or seismic activity could be greatly reduced.

With regard to the support of the mining areas beyond the mined-out shaft reef, a particularly attractive option is the use of a system of concrete pillars, instead of conventional stabilizing pillars. Work by Adams *et al.*[6] has indicated that 20 GPa concrete pillars could provide an alternative to reef pillars, and recent work by Ryder[7] indicates that a layout of concrete pillars with a stiffness of only 1 GPa could provide support resistance for the control of stope closure equivalent to that provided by current stabilizing-pillar layouts. The concrete pillars would need to be laid out more closely than are current pillars, but would allow total extraction of the reef and would adequately control closure so that the energy release is kept to a safe level. Moreover, if concrete pillars of this nature were to replace stabilizing pillars, the average pillar stresses would drop considerably and seismicity due to foundation failure could be virtually eliminated.

The last area of regional support to be addressed concerns the design of bracket pillars in geologically disturbed ground. Currently, the following two criteria are used in the calculation of the width of the bracket pillars that are to be left against a potential seismically active geological structure for the control of sudden seismic slip:

- the ratio between the stress acting on the structure, a dyke say, to the strength of the dyke rock. If this ratio is greater than 1, then bracket pillars of unmined reef are left adjacent to the structure and are designed to be of sufficient width to reduce the stress:strength ratio to less than 1

- the excess shear stress, which can be used in the estimation of the size of a bracket pillar required to eliminate positive zones of excess shear stress on seismically active faults or dykes.

In addition, the size of the bracket pillars required to prevent sudden slip on geological structures can be estimated from the occurrence of rockbursts precipitated by mining close to such structures. These analyses indicate that pillars 20 to 50 m wide are required to ensure safe mining away from the structure (Van der Heever[8] and Gay[9]). The seismic events that are associated with geological structures are generally large (magnitudes of greater than 3) and, because of the length of the structures, can result in rockburst damage over wide areas unless bracket pillars are established systematically.

Development of an improved understanding of mining-associated seismicity (strategy 3)

This strategy addresses problems of mining in seismically active, geologically disturbed areas, to permit the safe extraction of the maximum amount of reef adjacent to a geological structure without the disturbance of any existing state of unstable equilibrium.

The tools required to implement this strategy include seismic networks that are able to define, in real time, a seismic event by providing its location and time of occurrence, as well as providing an estimate of the energy released and the seismic moment. Most seismic networks installed on South African mines are able to meet these requirements. However, for the monitoring of specific areas, such as an isolated block of reef that is being mined out adjacent to a fault or dyke, a relatively close-in network, such as COMRO's PSS or ISS's cluster network, is likely to provide more useful data than a minewide or regional network.

Time-dependent deformation of the rock strata on either side of the structure, as well as on the fault or dyke surface, should also be monitored with the seismic data in real time, so as to enable a history of the structure's response to mining activities to be built up. This could allow the implementation of counter-measures to control or release the available energy in a stable manner. However, to achieve this requires an understanding of the morphology of the contact surface between the fault and the country rock. In addition, routine repetition of control measures requires a fairly detailed mapping of the surface geometry of the fault. Tomographic methods or ground-penetrating radar could possibly provide this type of information (e.g. Young[10], and Young and Maxwell[11]).

An understanding of the *in situ* stress field, particularly with respect to the *k* ratio in the vicinity of geological structures, is also required, although little information is normally available. Young and Maxwell[11] used tomographic methods to identify a highly stressed zone of rock by correlating the velocity structure of the *P* wave with an area of mining-induced seismicity, and confirmed the existence of high stresses by drilling into the zone and observing intense discing of the core. Gay[12] also encountered discing when using strain-relief techniques to measure the *in situ* state of stress in a large dyke at ERPM. From the stress measurements, deviatoric stresses of the order of 100 to 160 MPa were calculated—sufficient to induce the observed discing in the very strong dyke rock (with a uniaxial compression strength of 360 MPa).

Similar information with respect to the stresses acting on fault planes is limited. In the Klerksdorp mining district, Gay and Van der Heever[13] found that large horizontal stresses (48 to 56 MPa) act at high angles to fault planes, enhancing the strain energy stored in the quartzite adjacent to the faults. Moreover, modelling by Brummer and Rorke[14] showed that these structures have the potential to generate large (magnitude 4+) seismic events.

The values of the *k* ratios determined from the *in situ* stress measurements cited vary from 0,6 at the ERPM dyke site to 1,0 in the Klerksdorp district. The lower the value of the *k* ratio, the greater the potential for sudden seismic slip on a structure. Significantly, the minimum *k* ratios (i.e. σ_{hmin}/σ_v) encountered in deep South African mines range between 0,3 and 0,4; *k* ratios of this order probably indicate that normal fault slip could readily occur during mining, supporting Brummer and Rorke's[14] conclusions.

An estimate of the *in situ* coefficient of friction, which effectively limits the gross strength of rocks in the upper crust, is also required. At present, because of the difficulty of measuring this *in situ*, estimates of this parameter are obtained from laboratory experiments. However, a stress-inversion technique for the determination of friction from observations of the orientations of joints or small faults across which shearing has taken place, has been developed by Reches[15]. The assumptions made for this technique to be viable are that (1) the slip on the fault occurs in the direction of the maximum resolved shear stress, (2) Amonton's friction law holds, and (3) slip events occur under relatively uniform conditions. Statistical analysis of 27 fault sets comprising more than 500 faults resulted in an average friction coefficient of 0,58 ± 0,37, which agrees with the results obtained from laboratory tests (0,6 to 0,85). In addition, as observed in laboratory tests, the friction coefficient was independent of the type of rock.

The strategies used by rock engineers to devise mining sequences adjacent to geological structures are primarily based on numerical modelling. The most commonly used models are of ERR, ESS (Ryder[16]), and volume closure (McGarr[17]). ESS is based on the assumption that many large seismic events occur by shear slip in response to stresses acting on geological structures. To quantify this, Ryder[16] proposed that the shear stresses acting on a fault that are greater than the assumed dynamic frictional resistance to movement on the fault plane can be used in the evaluation of the potential for sudden shear slip and, in particular, of the seismic moment. Spottiswoode[18] confirmed this in a study of seismicity and rockbursting in 30 regions at Blyvooruitzicht mine by assuming values of 7,5 MPa and 0,6 for the rock cohesion and coefficient of dynamic friction respectively, and comparing calculated ESS values with the number of seismic events, seismic energy released, rockburst incidence, and incidence of geology. By contrast, the correlation of these parameters with ERR was relatively poor, although Napier[19] showed, by means of a generalized ERR for off-reef deformations, that large energy release can occur during fault slip. Moreover, McGarr[17] has shown that the cumulative seismic moments can be correlated with the volume of elastic convergence in the stopes of deep mines, and hence to the average energy-release rate, which is the most commonly used criterion in the design of layouts for deep mines.

Both ERR and ESS are used by industrial practitioners. Henderson[20], for example, used ESS to evaluate different regional support layouts and strategies for mining adjacent to seismically hazardous structures. The results of the analyses were compared with the known seismic history of a dyke and showed that, in general, ESS values greater than 10 MPa were sufficient to activate large seismic events (e.g. with a magnitude of 3,5). However, when a 20 m wide bracket pillar was left along the dyke, the ESS levels were reduced to 4 MPa.

Similar calibration studies utilizing McGarr's[21] relationship between volume closure and seismic moment have not been well documented, although Spottiswoode[22] and Webber[23] were able to show good correlations between observed and modelled cumulative seismic moments. In addition, the use of the method for the design of bracket pillars and the extraction of remnant pillars adjacent to faults has been reported by rock-engineering personnel at various seminars, and Holmes and Reeson[24] used both ESS and volume closure to estimate the sizes of potential events during the mining out of 10 000 m^2 of reef and to compare them with historical seismic records. In particular, they were able to show a good correlation using the ESS model between the length of fault exposed by mining and the release of seismic energy, and were able to construct graphical relationships between the length of fault exposed, the area of slip, and the seismic moment.

Syratt[25] reported on a specific *in situ* case study involving the extraction of a large remnant cut by two seismically active dykes that intersected at a high angle. From the rate of microseismic emission, a critical remnant width of 30 to 34 m was defined, following which the rate of microseismic events diminished as the remnant softened and was mined out. Simultaneously, a reduction in the number of large events (magnitude 1,4 to 3,4) was observed. The decreases in stress associated with these events were larger than those observed for events of comparable magnitude, and an analysis of the events showed that repeated slip occurred along a 300 m segment of the dyke, resulting in a total closure of 0,9 m, compared with a stoping width of 1,0 m.

With regard to the use of ESS as a pro-active design tool for layouts and extraction sequencing in faulted ground, Napier[26] showed that, to limit the magnitudes of excess shear stress to 10 MPa, unacceptably wide bracket pillars (wider than 100 m) would be required for faults dipping at 45 to 60 degrees. At steeper dips (75 degrees), the sizes of bracket pillars were more acceptable but, in general, the modelling showed that a system of 30 m wide bracket and dip pillars provided the optimum solution.

Better solutions for the planning of stoping sequences in deep mines can probably be obtained by the use of Spottiswoode's[27] concept of volume excess shear stress (VESS), a quasi three-dimensional model that calculates the total variable excess shear stress (i.e. the volume excess shear stress) in a block of ground by numerical integration of the excess shear stress values across regions of positive excess shear stress. This allows an estimate to be made of the maximum release of seismic moment in a seismogenic region at any time. It also allows the average rate of seismicity in a given time period, due to increases in positive ESS values, to be determined.

More recently, Spottiswoode has been working on a three-dimensional boundary-element model, VOLSIM, which allows for inelastic Mohr–Coulomb failure of the rockmass by movement on pervasive fractures modelled by discrete planes only. Using VOLSIM, he simulated the mining out of stopes with spans of up to 270 m, realistically defining zones of fracture growth, bedding-plane movements, and inelastic stope closures.

A newer development is Salamon's[28] random-flaw model, which can fairly realistically simulate seismicity caused by sudden slip on pre-existing flaws. In particular, it is able to model seismicity occurring on reef and in front of the advancing stope face, the effect of increasing depth, the changes in horizontal stress (*k* ratio), and the drop in stress during ground movement.

Thus, it is clear that simulation programs are available that have the potential to greatly enhance the quality of high-stress layout design, including that of bracket pillars and extraction sequences close to geological structures. However, considerable work still needs to be done before such programs become viable planning tools for routine use by rock engineers in designing layouts and reef-extraction sequences that will minimize the potential for sudden releases of seismic energy during mining and that will limit the size of seismic events.

However, there is still a need for improved risk-assessment procedures that will allow management to make informed decisions as to the likelihood of rockbursting in the stope area, where most workers are normally employed. Stewart and Spottiswoode[29] have initiated work on a system that relies, not on one parameter such as the *b* value, but utilizes all the recorded seismic data to define four seismic risk criteria. These are based on the Gutenberg–Richter *a* and *b* values, the fractal dimension of microseismicity, the distance between microseismic event pairs, and the stress drop accompanying a seismic event. To a large extent these parameters are independent of one another and are physically meaningful. An initial back-analysis indicated that, had these parameters been applied at the Blyvooruitzicht site, it would have been possible to withhold worker entry to the stope for 20 per cent of the available working time, during which 80 per cent of the seismic energy would have been released.

Strategies requiring considerable research and development

Development of an understanding of rockburst mechanisms (strategy 4)

The objective of this strategy is to develop and utilize an understanding of rockbursting mechanisms so that improved support and other measures can be taken to minimize the damaging effects of rockbursts.

It is generally accepted that sudden dynamic failure of the rockmass occurs by two mechanisms, resulting in rockbursting and damage to mine working areas. These are (1) 'crush type' events, which are located close to the stope face, and (2) 'shear type' events, which are located off-reef and take place preferentially on geological planes of weakness (Spottiswoode[18] and Ryder[16]). For either type of event to occur, a substantial zone of over-stressed rock must be present in a state of unstable equilibrium (Salamon[30]) that is disturbed by a change in stress, resulting in sudden failure of the rock, a rapid drop in stress levels, and the initiation of a seismic event. A possible cause of the event is the imposition of mining-induced stresses accelerated by time-dependent weakening of the rockmass asperities, which are under load because of their resistance to movement (Rice[31]).

A seismic event generates a sequence of *P* (compressional) waves that radiate away from the sudden compression or tension applied to the source area, and travel through solid quartzite at velocities of about 5800 m/s. Similarly, *S* (shear) waves radiate due to sudden relaxation of the shear stresses in the source area, at about 3600 m/s. However, wave velocities can vary markedly with rock type, and it is necessary to determine the velocity structure of the rock strata for an accurate description of the seismic source and the location. Calibration blasts utilizing explosive charges are suitable tools in the determination of the velocity structure of an area, and can also be used in seismic tomographic studies such as the identification of high-stress areas (e.g. Young and Maxwell[11]). The interpretation is complicated by the refraction of seismic rays at lithological boundaries, and use should be made of ray-tracing procedures such as that developed by Webber[32] for the Klerksdorp mining district.

Seismograms are a record of the ground motion initiated by a seismic event from which ground velocity, ground displacement or strain, and accelerations can be determined. Of particular importance is the peak ground velocity, which appears to control whether or not rockburst damage to excavations occurs as a result of the seismic event. In general, damage occurs at ground velocities of a few metres per second and large values of ground acceler-ations (g) (Spottiswoode *et al.*[33]).

Very few direct measurements of the peak ground velocity have been made. COMRO personnel[34] reported a peak ground velocity of 1,1 m/s at the sidewall of a tunnel after a seismic event of magnitude 2,5, which occurred 70 m distant from the point of measurement. Severe damage, including fracturing and slabbing of the tunnel sidewall and hangingwall and of the hangingwall of the travelling-ways, was observed, particularly in areas that were either poorly supported or not supported at all. The areas that had been meshed and laced were not damaged. Total stope closure was also observed where there was severe footwall lifting and crushing of the brow between the travelling-way hangingwall and the stope footwall. More recently, Hemp[35] reported a peak velocity of 2 m/s, and Adams[36] a velocity of 2,1 m/s, from an on-line closure meter.

At Durban Roodepoort Deep mine, Piper[37] used forensic methods to determine the ground velocity initiated by a seismic event of unknown magnitude that caused considerable but localized damage to a 120 m length of haulage. The rockmass was fragmented into blunt particles, some of which were ejected at high velocities into the water pipes, and Piper's tests using model fragments with the same shape and mass showed that the particles must have been ejected at a velocity of at least 55 m/s, demonstrating how the onset of tensile and shear loading due to the sudden arrival of seismic waves can overload already highly stressed rock and lead to violent failure.

Rice[31] discusses two mechanisms of failure on a fault plane. In the first case, the shear strength of the fault decreases as slip occurs at a rate dependent on the normal stress acting on the fault plane and the stiffness of the surrounding rock. If the stiffness is large, controlled slip occurs but, if the stiffness is small, the system becomes unstable and sudden slip occurs, releasing seismic energy.

Rock-on-rock sliding-friction experiments also give insight into the mechanisms of fault slip and can be used to quantify the influence of normal stress, slip rate, and surface roughness. These experiments (for example, Dieterich[38] and Ruina[39]), show that, as sliding initiates, the shear strength increases, together with an increase in sliding velocity until a constant steady state is reached. If this steady state is perturbed, either by a reduction in stiffness or a change in the velocity condition, the system becomes unstable.

In situ and laboratory experimental observations provide insight into both the mechanisms of rockbursting and the methods for controlling damage. For example, face rockbursts may occur when the highly stressed rock in front of the face is perturbed by mining or dynamic loading due to a seismic event. The occurrence of such an event can be prevented if the rock is softened 3 to 5 m in front of the stope face by preconditioning blasting. Alternatively, the use of rockburst-resistant support, e.g. rapid-yielding hydraulic props, to control ground movements can also maintain the coherence of the rock, particularly in the hangingwall.

The control of tunnel rockbursting due to mining activity in the vicinity of geological structures requires careful planning of the mining sequence of extraction and the installation of good energy-absorbing support where structures are likely to slip. Suitable support methods include mesh and lacing, preferably pinned against the walls by energy-absorbing cone-bolt tendon supports. However, it is expected that improved cladding materials will be required to control the extreme damage encountered after many fault-associated rockbursts. Support of the footwall can also be important for ensuring that haulages stay open during seismic events. The velocities operating during these events probably exceed 1 m/s, which is equivalent to induced stresses of more than 10 MPa (Spottiswoode[40]). Stresses of this order would certainly be sufficient to cause additional rock failure and unravelling of the hangingwall.

Rockbursting also causes severe damage to excavations developed through or in the vicinity of a dyke. This is because dykes are likely to be more highly stressed than the adjacent sedimentary strata, and can therefore be in a state of unstable equilibrium. Mining perturbs this state and can initiate rockbursting, particularly if the dyke material is strong and brittle. A possible control measure is to destress the dyke, either by blasting or by the drilling of holes into the dyke to encourage the development of fractures due to the high-stress concentrations.

Other methods for the control of seismicity and rockbursting due to mining near geological structures require considerable development before being regarded as viable control techniques for seismic-energy release. This is because mining in seismically active districts takes place at gradually increasing depths, and the size of the largest seismic events is likely to increase. For example, the following are the maximum 1992 magnitudes projected by the Geological Survey for the four major mining districts in South Africa: Rand 4,2; Far West Rand 4,6; Klerksdorp 4,9; and the Orange Free State 4,5 (Fernandez *et al.*[41]). In 1991, the equivalent maximum magnitudes were 3,7; 4,2; 4,3; and 3,0 (Fernandez *et al.*[42]). Events of these magnitudes are almost certainly associated with geological structures, and are influenced by the increasing area mined.

The occurrence of larger events will result in the greater release of seismic energy, possibly larger drops in dynamic stress (10 to 100 MPa) and peak ground velocities, and larger areas of damage. Control measures that are available to restrict damage include the installation of suitable energy-absorbing support (such as backfill), 3 m/s 20/40 t rapid-yielding hydraulic props with headboards and footboards in stopes, and yielding tendons such as the cone-bolt, especially in areas where faults and dykes are encountered. The use of flexible-cable cone-bolts in large service excavations in shaft pillars should also be considered, especially in shaft pillars that are located in fault losses.

Pro-active methods for the specific control of the rockbursts that are likely to occur when mining in geologically disturbed or high-stress areas include the following.

- **Methods for inducing controlled slip on fault planes.**
  These methods rely on a seismic network for the identification of areas on fault planes where seismicity is increasing as a result of increasing shear stresses. The stress is likely to build up because of resistance to movement by some form of asperity, and could result in the sudden release of seismic energy.
  A method available to address this situation is the injection of fluid into the region of high stress in an attempt to reduce the normal stress acting on the fault plane and so initiate slip, preferably in a controlled manner. Attempts at the use of fluid injection to induce slip and release seismic energy have been partially successful (Board *et al.*[43]) and, in more recent field trials, small seismic events have been triggered in a controlled manner. However, current work[44] by Miningtek staff has shown that the method is unlikely to be practical owing to the irregular nature of fault surfaces and the high clamping forces acting on them.
- **Controlled fault slip by reduction of the the stiffness of bracket pillars.**
  A possible technique for the mining of remnants in areas where bracket pillars are used to prevent sudden slip on faults is suggested by Syratt's[25] observations on the seismicity released during the mining of remnants adjacent to a large dyke. Syratt observed, as did Legge and Spottiswoode[45], that the rate and magnitude of seismic events decreased once the remnant had been reduced to a minimum critical size, at which the stiffness of the hangingwall–remnant–footwall system was sufficiently soft to allow energy to be released in a controlled manner.

Two possible mechanisms can be envisaged to achieve this softening process in bracket pillars on faults without reducing their size by mining. One of these is by preconditioning blasting of the pillars, utilizing holes drilled parallel to the fault plane to soften the pillar gradually and to encourage controlled slip on the fault over time. A second method is to cut a narrow slot into the bracket pillar by use of a device such as a diamond wire saw[46], or by the drilling of a series of closely spaced holes as used by Adams[47] when experimenting with reef boring, to propagate fractures into adjacent holes under induced high stresses, or by normal back blasting. This method could have an advantage over preconditioning blasting in that it is unlikely to initiate large seismic events on the fault plane, as can happen during a preconditioning blast.

➤ **Preconditioning for the mining of remnants and other highly stressed ground.**

Preconditioning, i.e. the destressing of the rockmass in front of an advancing stope by blasting so as to prefracture the rock and prevent rockbursting in highly stressed ground, has been experimented with in the USA, Canada, and South Africa, but has probably been most successful in the Coeur d' Alene mines in Idaho, where it is used routinely for the destressing of large stope pillars if further mining is likely to result in rockbursts (Blake[48]). However, significant problems are encountered, including the following:

- delayed destressing blasts because of difficulties in the drilling of longholes until the pillar is as small as possible, resulting in exposure of workers to on-shift bursting
- misfires during destressing blasting
- timing of the blast—early destressing can result in subsequent on-shift bursting, and waiting too long exposes workers to the risk of on-shift bursting
- delays in production because of the time required to drill destressing holes and to initiate the destressing blast
- sometimes ineffective destressing of the pillar by the destressing blast.

These problem areas are applicable to all destressing or preconditioning procedures, which, in South Africa, have been mainly used to reduce the potential for face bursts in longwall stopes (Roux *et al.*[49]). Subsequently, Rorke *et al.*[50] used preconditioning to extract a small pillar adjacent to a large faulted dyke at Blyvooruitzicht gold mine. In this trial, the preconditioning holes were drilled at right-angles to the advancing updip stope face. Some drilling problems were encountered owing to the collapse of holes, but it was shown that the blasts did enable the release of relatively large amounts of stope closure and seismic energy (at a maximum magnitude of 0,9), together with an improvement in hangingwall stability due to the steepening of the orientations of the stress-induced extension fractures. Subsequent monitoring of the mining of remnant pillars at Blyvooruitzicht has confirmed the benefits of improved hangingwall conditions (Adams *et al.*[51]). The trials also gave rise to a definition of a routine procedure for the integration of preconditioning for rockburst control into the normal mining cycle, as follows.

(1) The layout of the stope face is that normally used for longwall breast mining with an up-dip to slightly underhand configuration. The length of the face should be short, i.e. 10 to15 m.

(2) The preconditioning holes are drilled parallel to the breast face in fractured ground, 3 to 6 m ahead of the face. The drilling rates are highest when the holes are located in this position, and blockages are reduced. Since the holes are short (10 to 15 m on average), they can be drilled in one shift, thus fitting in with the mining cycle.

(3) Anfo-type explosives are most suitable for preconditioning because of their high gas content and the fact that they maintain higher accelerations for greater distances from the blast than do emulsion-type explosives. This allows gases to penetrate the stress-induced face fractures, causing them to open up and propagate further, thus relaxing the high stress zone close to the stope face.

(4) The success of the blast depends on the stemming of the hole. Too little stemming, or the use of the wrong stemming, results in ejection of the stemming and a reduction in fracture propagation. To overcome this, at least 5 m of a mixture of clay and gravel stemming is recommended to impart sufficient explosive energy into the zone of fractured rock.

(5) Monitoring of the effectiveness of the preconditioning blast requires a dedicated mini-seismic-system such as the COMRO PSS, which can be moved as necessary with the advance of the mining face. Evaluation of the seismic data enables ongoing assessment of the effectiveness of the preconditioning, as does monitoring of the stope closure and ride. Closure pegs are installed in the hangingwall and footwall so that these parameters can be monitored, and continuous closure meters record changes in stope width in real time. Mine personnel use this information to assess the suitability of support systems in maintaining the stability of the stope hangingwall during a blast or a rockburst.

(6) The seismic network also allows the accumulation of seismic data over a long time, which permits better evaluation of the destressing blasts and the build-up of a history of their effectiveness. It also provides information on when preconditioning blasts need to be carried out (i.e. if they are not done routinely) and on the definition of criteria that can be used to withhold worker entry to the stope. Blyvooruitzicht is, for example, currently using the *b* value as its criterion for withholding entry.

Examples of the use of seismic networks to indicate dynamic failure mechanisms that have resulted in rockburst damage have been reported by both Western Deep Levels and COMRO researchers. These include failure by shear slip on contacts between dykes and country rock (Syratt[52] and Lenhardt[53]); pillar foundation failure (Lenhardt and Hagan[54], and Adams *et al.*[55]); and failures in shaft pillars (Cross[56]) and on faults (Van der Heever[8]). In all these instances, the main failure mechanism was the gradual build-up of stress on the retaining structure (for example, a pillar, dyke, or abutment), resulting ultimately in sudden failure of the strata along either existing or newly generated shear surfaces, closing stopes over large areas, and damaging footwall or hangingwall excavations.

The reverse situation also occurs; i.e. where the structure is effectively weaker than the hangingwall and footwall strata pillar, crushing can occur. Thus, in addition to affecting the shear strength of potential failure surfaces, the effective strength of pillar systems can also affect failure mechanisms and should be taken into consideration in relation to the dynamic stability of mine layouts. This is particularly important since pillar failures are generally large, having magnitudes of 3,4 to 5 on the Richter scale, and each is capable of repeatedly generating one or more large events (Lenhardt[53]).

Development of support systems (strategy 5)

Wide-reef mining is generally accepted as taking place where the width of the reef to be mined exceeds 1,8 to 2,5 m. At least three current major projects will be mining reefs that exceed this in width: the Sun, Target, and South Deep projects. In addition, the Carletonville mines are now mining the Ventersdorp Contact Reef, which varies in width from very narrow to 5 m. In general, depending upon the depositional setting, reefs deposited closest to the point of influx into the depositional basin have an average width of 13 m, which thins to between 2 and 4 m at the point where the gold concentration disappears (Viljoen[57]). Subsequent to their deposition, the reefs are frequently disturbed by tectonic activity such as faulting and folding. For example, the strata being considered for mining in the area of the Sun Project (Anglovaal Ltd[58]) are both faulted and folded into a syncline with a steeply-dipping western limb and a shallow-dipping east limb. Mining of these reefs at the depth envisaged (about 3500 m) will almost certainly be accompanied by severe strata-control problems, seismicity, and rockbursting. To control these hazards, novel layouts, and novel methods of support and extraction, will be required if maximum reef extraction and safety for workers is to be ensured.

Similar depths and problems are envisaged for the Moab (Bennets[59]) and South Deep (Tregoning and Barton[60]) projects. However, in both these instances, the new mines have the advantage of being able to access the reef horizons from adjacent sister mines. This facilitates the removal of the shaft reef pillars and the backfilling of the mined-out area prior to shaft sinking. Thus, the sinking of the shaft and the cutting of large service excavations can take place in destressed ground.

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The use of the destressing principle is important for mining at great depths, where gravity-induced stresses are very large, or in areas, such as O'Kiep, where large horizontal stresses (in excess of the vertical stress) are encountered. In the latter instance, it may be necessary to cut vertical destressing slots at right-angles to the major stresses (Nangle[61]).

This highlights the need for an understanding of rock-strength properties and of the ambient *in situ* stress state where mining is to take place. Wagner[62] defined a deep orebody as 'an orebody where the maximum stress value of the primitive or pre-mining rock stress tensor exceeds a half of the rockmass strength'. This condition is usually sufficient for excavation rock walls to fail, and is particularly relevant to wider-reef orebodies.

Thus, for the mining of wide-reef horizons at great depths, the fundamental information required for the planning of stope-support systems includes a knowledge of the virgin pre-mining stress state, and details of the strength and deformation properties of the rock strata in which development and mining activities will take place. A knowledge of the frequency of major and minor geological structures, such as dykes, faults, joints, and bedding parting planes, should also be obtained.

This information is essential for the quantification of the rockmass condition likely to be encountered, by use of a failure criterion such as that of Hoek and Brown[63] or Wiseman[64], and also for the determination of suitable support design criteria to maintain the integrity of excavations. Moreover, an understanding of the orientation of the horizontal stresses, together with the potential damage mechanisms in the haulages, is required to ensure that tunnels and roadways are laid out so that they experience minimum damage due to high induced stresses.

An important parameter to be considered in the planning of the extraction and support of wide orebodies at great depth is the geometry of the orebody, since this dictates the mining method and the required service excavations. In other words, the question arises as to whether the orebody is tabular, as is usual in the layered Witwatersrand strata, quasi-linear or pipe-like, or three-dimensional with approximately equal lengths of side. Wagner[62] emphasizes the benefits of three-dimensional geometry with respect to the support and stability of the final excavation, since the mining-induced stresses depend only on the pre-mining stresses and the shape of the excavation, and not on size. Size, however, affects the stability of the walls of the excavation since, in general, the strength of the rockmass, and hence the potential for failure, deteriorates with increasing excavation size. Service excavations adjacent to three-dimensional orebodies are also subjected to relatively small mining-induced stresses provided they are sited at distances greater than 0,2 of the radius of the main excavation. More important is ensuring the stability of the walls of the mined-out orebody, which in turn depends on the 'wallrock strength' and the presence of geological weaknesses. Slippage along structures, such as joints or faults, is generally confined to the immediate wall of the excavation, and large-scale slip, such as on faults, is rare.

With respect to wide, tabular deposits (i.e. reefs wider than 2,5 m), three mining methods are used: conventional mining, pillar mining, and massive mining. However, at depths of 3000 to 4000 m, all these methods are problematical because the stresses acting on the stope faces are sufficiently large to cause severe fracturing and strata-control problems, including the support of the high stope faces and the hangingwall roof, and face bursting.

The reef can be destressed by the cutting of a narrow slot using either conventional narrow-stoping techniques or, as is planned for the South Deep Project, mechanized-mining techniques (Tregoning and Barton[60]). The width of the orebody at South Deep to be mined after destressing in this way varies from 2 to 40 m, but it is planned to extract the orebody as an 8 m wide multiple-reef package, with all development taking place on the reef plane by means of room-and-pillar mining.

Vertical-crater mining is planned for the extraction of the wide-orebody zone and, after the completion of this massive mining, it is planned to remove the pillars in the room-and-pillar mined-out areas.

Backfill is the primary support to be used for the mining out of the orebody. In the narrow-reef stopes, the proposed stope-face support consists of yielding timber props and hydraulic props, and rockbolts, mine poles, and cable trusses are proposed for the roadways (Espach[65]). In the room-and-pillar mining, pillars laid out on a herring-bone pattern provide the initial support, together with pre-tensioned resin rebar bolts in the roof, while the pillar removal will take place on retreat after the vertical-crater slots in a specific block are mined out with backfill as support.

Finally, the vertical-crater mining of the wide-orebody zone is to be laid out according to the same plan as that for the room-and-pillar mining. After the mining out of each slot, cemented crushed-waste backfill will be placed as stope support. However, it may be necessary for the integrity of the stope sidewalls to be maintained by the installation of mesh and/or lacing support after a stope has been completely drained.

The South Deep project is important for the future planning of mining at great depths since it is likely that other wide-reef or multiple-reef projects will be extracted in a similar manner. However, most wide-reef mining at present is being carried out in the Far West Rand mining districts on the Ventersdorp Contact Reef, where closure rates are relatively low and stope widths vary from 2 to 5 m. Under these conditions, the reef is generally extracted in two cuts. The first cut is a narrow upper cut in which mechanical or hydraulic props with headboards are used to protect workers close to the stope face, and rapid-yield props, also with headboards, are used in the back area of the first cut. The narrow first cut reduces the danger of face bursting and of falls from the high face wall. Rockbolts are used to support the hanging behind the hydraulic props and over the most recently mined second cut. Cementitious or conventional backfill is used as permanent support to replace timber packs, with considerable cost and safety benefits, including a significant reduction in lost blasts and hangingwall overbreak (Gürtunca *et al.*[66]).

With regard to the general support of the stope-face area, current support technology is probably able to meet most support requirements for wide stopes provided the reef is extracted in two or more cuts. This includes the new-generation hydraulic props with their elongate headboards; pre-stressing devices such as the packsetter and propsetter for increasing the stiffness of timber-based packs; and barrier props with headboards, especially in stopes prone to face bursting. However, when the reef is extracted in a single cut or at relatively high stoping widths, serious support problems arise that have to be addressed if falls of ground and face bursting in the face area are to be contained. It is also not clear how seismicity will be affected by multiple-cut mining in wide reefs.

Face bursting is a particularly difficult problem to solve so that workers in the face area are protected adequately. A possible solution is the development of strong, lightweight curtains that are anchored to the hangingwall and footwall through which the driller can operate his machine.

With regard to temporary support of the face area, the use of rapid-yielding hydraulic props 2 m or more in length is severely hampered by the intrinsic mass of the prop and its potential buckling instability. Also, in the stopes in the Ventersdorp Contact Reef, undulating footwalls and friable jointed hangingwall can cause major problems. A solution to the hangingwall problem could require the development of a new generation of skin coatings, an example of which is at present being evaluated by Miningtek. Improved shotcrete mixtures, preferably with fibre reinforcement, should also be considered.

To facilitate the handling of hydraulic props, COMRO staff have experimented with a linkage system that would allow props in the back row to be moved forward in sequence while being supported by an adjacent prop. However, problems have arisen owing to undulations of the footwall and the jamming of moveable parts by rust and dust. Nevertheless, it is recommended that these types of support systems should be re-assessed for use in wide stopes where significant areal coverage and energy-absorbing support are required to protect workers. This coverage could be achieved by the extension of the elongate-headboard concept to lightweight shield-type supports.

A further alternative technique for hangingwall support in wide stopes (Roberts[67]), is the installation in the face area of conventional tendons on a staggered pattern to which are attached elongate plates to provide areal coverage. A variation of this method would be to install yielding timber beneath the plates as the face advances, and so provide permanent support.

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Gully support is a problem that is probably best addressed by the placing of backfill permanent support as close to the gully sidewalls as possible. This could require a reinforced-earth system to limit the movement of the backfill into the gully and also to increase the stiffness of the backfill. In addition, the gully hangingwall should be supported, ideally with a skin coating, but also with mesh and lacing, or trusses where conditions are particularly poor. Movement of the gully sidewall into the gully needs to be controlled. This is best achieved by the use of cone-bolts, which are able to yield under conditions of high deformation.

Development of improved methods for the monitoring of the rockmass (strategy 6)

The purpose of this particular strategy is to identify suitable methods and techniques for the monitoring of the rockmass condition and, hence, its response to both static and dynamic load conditions. The areas to be addressed include the following:

- determination of the *in situ* state of stress
- seismic and other continuous monitoring methods
- assessment of rockmass condition and its interaction with support systems
- monitoring of excavation stability and support effectiveness.

Determination of the state of in situ *stress*

Considerable information on the state of stress at depth can be obtained during the drilling of exploration holes. In particular, the orientation of the horizontal principal stresses can be obtained from the direction of elongation of the boreholes as determined by the use of borehole callipers or borehole camera. The magnitude of these stresses at a particular depth can be determined through the sealing off of that portion of the borehole and pumping fluid into it to induce hydrofracturing. The vertical stress is then assumed to be equivalent to the mass of the overlying strata (Gough and Bell[68]). This technique can thus provide valuable information at a very early stage in the planning of a new mine project. In addition, analysis of the borehole core with respect to the presence of discing and the morphology of the discs can provide further information on the *in situ* stress levels (Maury *et al.*[69] and Guenot[70]).

Alternatively, borehole core can feature in the estimation of the *in situ* stress by the use of acoustic emissions through the 'Kaiser effect' (Chunlin *et al.*[71]). In essence, the Kaiser effect (Kaiser[72]) is the absence of detectable acoustic emissions in the rock specimen under load until the applied stress level exceeds the maximum previously applied stress levels. Thus, the method could have potential in the determination of the maximum *in situ* stresses experienced by the rock. Price[73] documents some data on the applied load at which acoustic emissions first appeared during uniaxial compression tests on a variety of rock types. These ranged from 25 per cent (for granite) to 75 per cent (for basalt) of the uniaxial compressive strength of the rock types tested. With regard to absolute values of stress, Hughson (Chunlin *et al.*[71]) recorded a pre-existing maximum stress in a sample of gabbro of 20 MPa. Although the mechanism of the Kaiser effect is not fully understood, nor is its application to rock-engineering problems, the method has potential in the assessment of stability (for example, of a stope face). With a knowledge of the previous maximum load or stress acting on the rock in front of the stope face, as well as the uniaxial compressive strength of the rock, the onset of acoustic emissions in the rockmass in front of the stope face could provide an indication of the build-up of load on the stope face, together with an indication of the likelihood of rockmass failure.

Alternatively, acoustic rock jacks have been used in the measurement of the previous maximum stress in either cyclic loading tests or controlled stress-rate tests. Chunlin *et al.*[71], using the latter test procedure, measured a Kaiser effect stress of 3,35 MPa.

Devices that are used routinely in mines for the measurement of the *in situ* stress state are generally based on Leeman's[74] method, in which a sensor is used to monitor changes of displacement or strain in a sample of the rockmass when the stresses acting on the sample are relieved. This is generally done either by overcoring of the monitoring device or by the cutting of a stress-relieving slot. Strain gauges or similar devices are then used to measure the resultant displacements, which in turn are used in the calculation of the magnitudes of the relaxed stresses (for example, Jaeger and Cook[75]). Commonly used devices are the CSIR doorstopper biaxial-strain cell and triaxial-strain cell, the CSIRO triaxial hollow inclusion cell, and the USBM borehole deformation gauge. Procedures for the use of these cells are outlined by Herget *et al.*[76] and the Chamber of Mines catalogue of instrumentation for use in geotechnical investigations in mines[77].

A new device for measuring *in situ* stress is the borehole slotter stress meter (Bock and Hartkorn[78]). The method involves the measurement of the two-dimensional stress state in the rockmass by the cutting of 3 radial slots oriented at 120 degrees into the borehole wall with the blade of a small diamond-impregnated saw. The consequent tangential strains experienced by the borehole surface adjacent to the slots are measured with a recoverable contact strain sensor. These data are then analysed by use of a computer program that calculates the principal stresses and plots them in graphical form. The method provides a high degree of redundancy, which allows rapid assessment of the reliability of the measurements.

Changes in stress in the rockmass are usually monitored with inclusion stressmeters. These are generally stiff devices with a modulus of rigidity greater than that of the rock. They are normally grouted into the borehole under a high pre-stress, and the subsequent changes in stress are monitored by hydraulic devices, photo-elastic plugs, and magneto-strictive materials or, more commonly, by electric resistance strain gauges. Perhaps the most commonly used and reliable device for the monitoring of stress changes is the vibrating-wire stress meter (Hawkes and Hooker[79]). This instrument has the advantage of being suitable for both hard and soft rocks, and can be used in boreholes up to 100 m deep.

Seismic and other continuous monitoring methods

This section considers monitoring systems for capturing, in real time, information on the stability of excavations and the rockmass in general. The technology currently being used is that of analogue or digital seismic networks that utilize a layout of geophones and other monitoring sensors to provide information on the stability of working areas, geological structures, mine regional support, local support, stope closure, and ride.

Two systems are in general use on South African mines: Anglo American Corporation's Integrated Seismic System (ISS)[80] and COMRO's Portable Seismic System (PSS)[81]. The ISS is being used mainly for mine-wide or regional coverage, while the PSS is being used for the monitoring of mining in relatively small, particularly hazardous areas or other experimental sites. Thus, to a large extent the systems complement each other, and are designed to incorporate future developments in monitoring technology and so ensure upgrading as necessary.

It is important to note that these systems can be used to monitor static deformations, as well as dynamic events. The instrumentation available for this real-time monitoring include load cells, extensometers, strain-gauged tendons, closure ride meters, multipoint borehole extensometers, and hydraulic pressure cells. These instruments are generally used to monitor slow movements or changes in the rockmass with the purpose of recording significant changes that may be precursive to sudden dynamic movements. Dennison and Van Aswegen[82] describe a particularly thorough study of stability changes on the Tanton Fault in the Orange Free State Goldfield during the mining out of an extensive area of reef at a depth of about 2400 m. This work involved not only real-time monitoring of the deformation or movement on the fault plane, but also correlation of these strain changes with numerical models such as UDEC[83] and MINSIM-D[84].

Instrumentation used in the long-term monitoring included the ISS regional system to provide seismic data for events greater than 0,7 in magnitude, and a COMRO PSS microseismic network for events in the magnitude range –2 to 1,0. For each mining step, seismic events located near the fault were plotted, together with the numerically determined stress values on a model of the fault surface. In this way, it was possible to correlate the seismicity with the stress distribution on the fault, the argument being that the stress state is a function of the mining sequence, regional support, fault geometry, roughness, and friction.

Measurements of stress and changes in stress were carried out with CSIR doorstoppers to determine the *in situ* state of stress on the fault, and vibrating-wire stress meters were used to monitor subsequent changes in stress due to mining in the vicinity of the fault. These stress changes correlated well with increases in the stress determined by both MINSIM-D[84] and UDEC[83].

Creep on the fault was monitored with mechanical gauges installed at distances of up to 50 m from the reef, together with electronic tiltmeters. Relatively small movements were measured, although in one region where positive excess shear stress was predicted, some creep was observed and the tilt meters recorded a co-seismic slip of 5 mm during the largest seismic event to occur on the fault. The topography of the fault itself was fixed by the extrapolation of observations at 27 intersections over an area of approximately 3,5 km^2. Interpolation of these data allowed an estimation of the roughness of the undulatory fault surface, and calculations of angles of friction and excess shear stress for the interpretation of the seismic data.

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Assessment of rockmass conditions, and its interaction with support systems

The primary purpose in the monitoring of the rockmass surrounding deep excavations is to enhance the safety of workers during mining operations. This applies, not only to sudden dynamic deformations, but also to time-dependent deformations, which can result in gradual damage to excavations. The seismic and other methods described in the previous section are obviously applicable to this work. However, there are particular techniques that also need to be considered.

➤ **Seismic imaging (or tomography)**
This is a particularly powerful tool for identifying areas where stress-related phenomena are occurring. Young[10], with his co-workers (for example, Young *et al.*[85], and Talebi and Young[86]), has been the major promoter of this technique, and in a fairly recent paper (Young and Maxwell[11]) used the technique to define the geometry of a highly stressed zone at the Lockerby Mine, Canada, and to document the transfer of stress from areas failing due to either mining-induced seismicity or blasting of adjacent areas, where the total stress levels increased as a result of stress transfer.

The geology of the area in which the experiment took place comprised norite and granite. Subsequent mapping of the tunnels that traversed the area showed minimal damage had occurred in the uniform coherent norite excavations, but that damage in the granite areas was severe owing to mining-induced fracturing and the opening of natural joints. The authors concluded that an initial large mining-induced tremor and resultant rockbursting had destressed the area where the tremor occurred so that it became a region of low seismic velocity and negligible mining-induced seismicity, whereas the area of high stress identified by the seismic imaging became a region of high seismic velocity and a focus for mining-induced seismicity. Thus, the implications are that a low-velocity aseismic zone has little rockburst potential, whereas high-velocity zones have significant potential for seismic energy release and rockbursting.

Three types of imaging surveys can be carried out: active imaging utilizing controlled explosive sources; passive imaging utilizing induced seismicity for a simultaneous inversion procedure for source location and velocity structure; and repeat surveys using either sequential active or passive surveys.

➤ **Acoustic emissions**
A similar approach to the monitoring of small-scale time-dependent failure of the rockmass is the monitoring of microseismic or acoustic emissions occurring in the rockmass. Perhaps the most encouraging work in this area is that of Van Zyl Brink[87] and his team of workers at Western Deep Levels mine. Using a system of single-station triaxial accelerometers, they clearly detected changes in rockmass condition, especially in situations of high stress such as abutments and on fault boundaries and pillars. These changes in a limited number of situations, were followed by a larger event. More recently, Li and Nordlund[88] used acoustic emission to assess the damage caused by blasting of the rock surrounding a tunnel, and were able to relate the damage to the Kaiser-effect onset stress. Talebi and Young[86] have also monitored acoustic emissions to measure failure in a shaft wall and to correlate it with zones of high stress.

Currently the US Bureau of Mines[89] is developing advanced acoustic-emission acquisition systems to provide better methods of ground control

➤ **Ground-penetrating radar**
The use of ground-penetrating radar as a tool for the monitoring of the rockmass condition is relatively widespread, despite the fact that no mineworthy unit is yet available. However, the prototype unit that is currently being used has allowed for the development of a measurement procedure by which fracture patterns can be assessed before and after preconditioning blasting in stopes and development ends. It has also been used to define the deterioration of excavations, such as ore-passes, as a result of high-stress conditions, and also to detect the presence of potentially hazardous structures, such as faults, dykes, and risky hangingwall conditions in front of the mining face (Frankenhauser and Berlenbach[90]).

Monitoring of excavation stability and support effectiveness

The maintenance of the stability of long-life excavations is essential to ensure effective productivity in a mine, and it is therefore important that routine procedures should be implemented to ensure the integrity of an excavation or, if necessary, to provide remedial action when monitoring systems indicate excess ground deformation or overloaded support systems.

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A good example of the importance of monitoring is given by Kersten, Piper, and Greeff[91], who describe the deformation of a large refrigeration complex in a shaft pillar at Hartebeestfontein gold mine. The excavations comprising the complex were cut in 1981/1982, the largest excavation being 100 x 8 x 8 m in size. During the excavation, it was found that significant deformation, including stress-induced fracturing of the rockwalls, had occurred. This was despite the use of two types of standard active support, one comprising 180 kN mechanical anchors 3 m in length on a 1 m grid together with mesh and lacing, and the other 7,5 m grouted cables pretensioned to 200 kN and supplemented by 5 m long cables pretensioned to 400 kN on a 2 m spacing.

To monitor the long-term deformation of the refrigeration complex, convergence monitoring and determinations of the depths of fracturing by petroscope observations were carried out routinely. On the basis of these observations, 8 m long cables were grouted into the sidewalls and hangingwalls, and 6,0 m long 400 kN cables were grouted into the footwall, in an attempt to reduce or control the ground movements. These remedial measures appeared to be successful and, in particular, the sidewall deformation was almost totally reduced.

However, over the following nine years, time-dependent deformation of the excavations continued to occur as mining around the shaft pillar took place, resulting in an increase in sidewall movement from 1 mm per month to 8 mm per month at the end of 1991. Extrapolaton of this to 1996 indicated that, by then, closures would have caused significant interference with the equipment in the chamber (Grave[92]). Moreover, ground movements are likely to exceed the maximum elongation of the cable anchors (by about 15 per cent), resulting in failure of the support system.

A second monitoring case study is that of Hepworth[93], who compared the performance of 13 different support systems during the overmining of a tunnel at field stresses equivalent to those encountered at depths of 3,5 km. His monitoring instrumentation comprised wire extensometers with anchors spaced at intervals up to 7,5 m into the sidewalls, rod extensometers at depths to 2 m into the footwall, and convergence meters to monitor the sidewall and roof movements. In addition, he used Glotzl cells to measure the build-up of load on the different support tendons, and he installed geophones and an accelerometer to measure the ground velocities and accelerations.

The outcome of this work included an assessment of the effectiveness of the various support systems, and also an evaluation of the effect of support tendon length and support stiffness in controlling the amount of deformation or dilation that occurred with respect to each support system. The depth zone at which the dilation occurred was also determined, and the amount of dilation was predicted. An important factor realized by Hepworth was the need for tendons to be able to yield without breaking so as to allow the rockmass around excavations to deform or dilate in a controlled manner. This principle ultimately led to the development of the cone-bolt.

These case studies were generally concerned with normal, stable, controlled deformations of the rockmass, the dynamic movement being estimated from the seismic record. However, it is important that instrumentation is available to directly monitor dynamic movements and measure the rate of ground movement. To address this need, COMRO staff developed real-time, potentiometer-based convergence meters to record both slow and rapid movements of the rockmass. The first of these was developed by Gibbon *et al.*[94], who determined real-time sidewall-movement velocities of 0,25 to 1,1 m/s, and in a haulage noted that damage occurred only where the velocity exceeded 1 m/s. Subsequently, potentiometer-based closure meters were used to monitor stope closures during both seismic events and preconditioning blasts. To date, the maximum closure rate recorded is 2,3 m/s. To provide this type of information more reliably, a robust ground-motion monitor was developed that is able to routinely record the ground velocities of rockwalls under dynamic loading in stopes and tunnels during mining activities. The utilization of this 'black box' device should greatly assist in the development of improved rockburst-resistant support systems.

A further development currently being undertaken is a continuous monitoring device to record hangingwall and footwall ride movement in stopes in real time so that the patterns of the ride movements preceding rockbursts can be identified. Observations in preconditioning stopes indicate that sudden reversals of ride between hanging and footwall may be predictive of impending failure or rockbursting.

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Finally, there is a need for a device that is able to monitor the effectiveness of installed grouted tendons. Ideally, this device should be able to detect the length of the tendon as well as the depth of grouting to control the resistance of the tendon against being pulled out during sidewall dilation or sudden seismic loading. COMRO, with collaborators, has carried out considerable research into the development of a suitable monitoring device. Consideration was given to the applicability of various electronic monitoring methods, of which the most viable appeared to be an acoustic 'returned pulse method' and a continuous time-frequency modulation technique. Neither of these methods was successful, partly because of the irreproducibility of the signals and spurious reflections from the rockmass, and also because of interference from other components of the support system, particularly the mesh and lacing.

However, given the fact that mining at great depths is planned by most mining houses and will take place under conditions of very high stress, routine monitoring of support systems will form an essential rock-engineering function and will require suitable instrumentation. Devices to do this effectively will therefore have to be developed. A possible means of achieving this would be to use the support system or units as the monitoring system.

In conclusion, the monitoring of rockmass deformation and support effectiveness will have to be carried out on an ongoing basis, preferably in real-time. This will be necessary, not only to ensure the stability of excavations, but also to provide rock engineers with quantitative rockmass deformation properties so that they can design better regional and local support layouts, as well as safer mining-extraction procedures.

Mechanization of mining operations (strategy 7)

The objective of this strategy is to address the measures that are currently available for mining activities in the stoping area so that optimum extraction of the gold-bearing ore is achieved, together with maximum worker protection, by utilization of newly developed and future technology. In so doing, it is envisaged that the number of workers at present carrying out mining operations in the stoping area will be reduced, and that those required to work in the area will be better protected. This is particularly important because rockbursting could be endemic at the depths envisaged for mining in the next decade and will, as indicated by current statistics[1], result in multiple fatalities. It is also envisaged that the implementation of new technology would lead to significant cost savings and improved productivity.

The approach that has been adopted to date with respect to mechanized mining systems has been directed mainly at non-explosive mining methods, together with layouts that would make better use of the available stope-face area by concentrating the mining activity in limited areas, thus permitting greater rates of face advance.

Potential methods for non-explosive mining include impact ripping, slot mining, waterjet cutting, and electric rockbreaking. Of these systems, the most advanced is impact ripping (e.g. Pickering *et al.*[95], and Haase and Pickering[96]). A similar, but perhaps less versatile, system is the slot-mining concept of Anglo American Corporation (Fenn and Marlowe[97]), which uses abrasive waterjet cutting or diamond-wire sawing to isolate the reef from the hangingwall and footwall. This relieves the vertical stress and allows the reef to be mined selectively, by the cutting of vertical slots into and parallel to the face. However, in some cases the rock may need to be broken out with an impact hammer. An alternative approach could be to cut a horizontal notch in the centre of the stope face. This would cause tensile splitting of slabs in front of the stope, which could be removed by the impact ripper (Napier[98]).

Both the impact-ripping and the slot-cutting methods have an additional benefit in that much of the waste rock can be separated from the reef at source, and can then be packed in the mined-out area to provide a high-quality backfill to stabilize the hangingwall in the face area.

Waterjet cutting with high-pressure water pulses is in a less advanced state. The principle is to drill a series of suitably spaced holes into the stope face, working up-dip from the advanced gully heading, and generate a water pulse at ultra-high pressure, instead of an explosive gas pulse. This has the advantages of greatly reducing the dust problems since no fines are generated, and of being non-toxic. Tests in quarries have shown the method to be feasible, and a prototype mining system has been proposed that comprises an accumulator water-driven high-energy (300 kJ) gun on a mobile drill-rig carriage (Haase and Pickering[96]). However, to be viable, multiple fracturing of the face is required, which can be achieved only with very high fluid pressures (10^3 MPa) and pressure rates (10^2 GPa/s).

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Electric rockbreaking is the least developed of the rockbreaking techniques available, and has been evaluated only in laboratory tests (Andres and Bialecki[99], and Haase and Pickering[96]). It appears to be very efficient, and some understanding of the mechanism has been achieved. Two stages are involved: firstly, in a high electric field, conductive gaseous paths are formed that break down the rock; these are followed by the discharge of stored energy, which generates high expansive forces to disintegrate the rock. Considerable development is required to make the process viable.

The practicability of non-explosive mining has to a large extent been proved by the work done by COMRO staff in field trials of the impact-ripper system, and it is now being implemented as a production mining system that compares well with conventional mining. The essential aspect of the system is its simplicity. For example, the mining machine, controlled by one operator, carries out the functions of rock breaking, rock handling, and advancing the support. Having broken the rock, the machine moves it onto a conveyer, which in turn moves the rock into the gully, where it is loaded onto a belt conveyer and transported out of the stoping area. During these operations, the impact ripper mines the footwall, and the face support is advanced. The system relies, however, on the presence of stress-induced fracturing in the stope face, which increases in effectiveness with depth. Moreover, high-pressure water is used, not only to power the system, but also to provide power for hydraulic prop-support systems, waterjet-cleaning systems, dust suppression, recovery of gold fines, and cooling of the stope-face area.

The optimum method for providing the high-pressure chilled water required for the holistic mining–support–cleaning–cooling system, is 'hydro-power', which utilizes the power available from the hydrostatic pressure in the water feed line in the shaft. The benefit of this resource is that, the greater the depth at which this mining takes place, the more cost-effective it becomes, provided that the run-off water is controlled effectively so that mining operations are not hampered and that the pumping to surface for re-utilization of the water is carried out in a cost-effective manner.

With respect to rock-engineering safety, there are significant cost benefits. These include reliable cost-effective powering of the hydraulic drilling and support systems, including both hydraulic props and tendon grouting, with significant advantages over the current compressed-air and electrically-driven systems of support installation. There could also be considerable benefit for rockburst control procedures, such as the use of high-pressure water guns to destress the stope face and to carry out fluid injection on potentially hazardous fault planes, and so ensure that the release of energy by slip occurs routinely.

However, it can also be argued that, with respect to rockburst control, non-explosive mining methods could have major disadvantages, since a significant release of seismic energy occurs during normal production blasts, reducing the available energy for later rockbursting. An estimate of this potential cause of rockbursting can be made from data reported by Stewart[100], who measured the seismic moment and energy per kilojoule of explosive during production blasts at the Blyvooruitzicht pre-conditioning site. The results of this work suggest that production blasts release energy equivalent to that released by small seismic events of a magnitude of −1 or less.

O'Connor[101], in a discussion of problems that could arise in deep mines using non-explosive mining methods, points out that the seismicity released after a blast is relatively independent of the explosive energy released, and represents the direct near-elastic or time-dependent response of the rockmass to the enlargement of the excavation. If this release of seismic energy were inhibited initially, it would have to be released at some time during continuous mining operations, exposing workers to a greater rockburst risk. However, because fewer workers are required in the face area during mechanized mining, the exposure of personnel to rockbursts is not significantly different from that for normal stoping methods. In addition, methods such as preconditioning blasting could largely eliminate face-bursting.

However, it should be pointed out that non-explosive mining operations have been used so far to mine more than 100 000 m^2 with no fatalities. This can be attributed in part to good rock-engineering support practice, which includes the use of rockburst-resistant hydraulic props very close to the stope face, and the sorting out and placement of waste rock in the back area immediately behind the mining machine. This provides excellent support of the hangingwall, which in turn limits the potential for falls of ground in the area in which the mining machines operate.

Other new developments in mining technology that have the potential to improve productivity and reduce the exposure of workers to risk in the stope-face area include hydraulic rock drilling and waterjet-assisted scraping. These developments can greatly reduce the time taken to prepare the face for blasting, and enable the face to be cleaned after the blast from the back area of the stopes, where the support is generally good. Similarly, implementation on a wider scale of continuous scrapers to remove the mined rock will permit the design of more-stable gullies, which can be supported more effectively, and could also allow the design of more cost-effective layouts and a reduction in the amount of off-reef mining.

Geological structures could inhibit this type of mining system, particularly where the reef is displaced, and as a result there is no infrastructure for the re-establishment of mineable face length. However, Ferreira[102] describes the use of raiseboring and ledging, instead of north-siding, through high-stress ground to re-establish a longwall. This technique has proved to be very cost-effective and safe, and should probably be considered for widespread implementation in all deep, highly stressed mining operations in geologically disturbed ground.

Development of effective non-linear techniques of solid-mechanics analysis (strategy 8)

Implicit in all the strategies addressed in this paper has been the need for geotechnical models that can be used to simulate and indicate the potential effectiveness of particular strategies for the mining out of orebodies. To achieve this effectively, the rock engineer needs to be able to integrate a wide variety of information such as an assessment of the rockmass condition from geomechanical rockmass-rating methods, seismic information, and data on the geological structure, together with numerical methods of stress analysis.

This approach to the design of mine layouts is proposed by Grabinsky and Curran[103] to allow for the fact that mine layouts are based on conditions encountered at depth, such as localized stress magnitudes and rockmass response, and are designed on a *'design as you go approach'*. To allow for this, the initial rock-engineering design, which is based on limited data from geological mapping, borehole cores, and ideally some seismic information obtained by use of conventional or tomographic seismic techniques, is at best a *risk assessment.*

To enhance the quality of this assessment, Grabinsky and Curran[103] identified the following three interrelated approaches to a better design methodology:

(1) generating an improved understanding of rockmass behaviour *in situ* and when reinforced by support
(2) developing calibrated tools for characterizing and monitoring large volumes of rockmass
(3) improving the integration of practical tools of numerical analysis in the design process.

With respect to the practical implementation of these methodologies, (1) requires either the instrumentation and monitoring of the *in situ* rockmass at a suitable underground site (Dennison and Van Aswegen[82]) or the design of laboratory tests that will allow the simulation of the rockmass behaviour under equivalent load-deformation conditions. Seismic technology to carry out methodology (2) is generally available (Mendecki[80], Maxwell and Young[104], and Patrick *et al.*[81]).

However, with respect to methodology (3), current non-linear models, although able to simulate the gross deformation of the rockmass at depth, are not able to realistically depict the failure processes operating during the deterioration or failure of the rockmass under either static or dynamic loading. A particularly important deficiency is the lack of suitable numerical models that simulate in detail the growth of fractures in the rockmass as a result of the mining out of reef or country rock. One possible model that can be used for this purpose is Napier's DIGS program[98], which models the initiation and growth of fractures around excavations. To achieve this, Napier places randomly distributed 'seeds' in the model rockmass. These can be likened to Griffiths's 'cracks or flaws', which allow the initiation and growth of cracks under load as mining takes place. DIGS uses displacement discontinuities to simulate crack growth, and thus the potential exists for the integration of the program into three-dimensional models such as MINSIM-D or three-dimensional VOLSIM. Numerical models of this sort are also useful in identifying the gross deformation mechanisms underlying observed behaviour. Alternatively, if the mechanisms suggested by the modelling studies are not observed, some guidance can be obtained for the setting of appropriate boundary conditions, material properties, and geological structures. These steps form a complementary phase to the development of empirical design philosophies and guidelines.

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The incorporation of seismic parameters such as fault-plane solutions, realistic properties of rockmass failure (e.g. friction coefficients, cohesion and failure criteria), and rockmass condition rating could ultimately lead to a design philosophy based on risk management. Grabinsky and Curren[103] argue strongly for this approach as being significantly more feasible than attempts to precisely predict the rockmass response to mining. This approach has considerable potential for the development of models that can be used to guide mine-layout design procedures for the deep-level mines that are to be brought into production in the next ten to twenty years.

Discussion and conclusions

The object of this review of the current status of rock-engineering knowledge and technology has been to address strategies (Table I) that will enable the South African mining industry to mine safely and productively at depths of 4000 to 5000 m into the 21st century.

The first strategy discussed, that of the implementation of *current knowledge and technology,* is perhaps the most important. This is because the mining industry is dynamic in its nature and requires the input of new ideas and technology to meet the needs of changing conditions in depth, environment, geology, and rockmass. Thus, the need for new technology to meet changing mining environments is ongoing, and may well require the development of novel mining systems, methods, layouts, and support systems.

This development can happen only in a strong climate of implementation that is fostered primarily by mine and group managements. However, other essential participants include the Government Mining Engineer, mineworker unions, research-and-development organizations, and equipment manufacturers. It is also necessary to upgrade the quality of the mine workforce, particularly with respect to education and training. This would ensure the better utilization of new developments, and would probably result in cost savings through reduced numbers of personnel working underground.

The remaining seven strategies can generally be regarded as providing a holistic approach towards the finding of solutions to the rock-engineering problems likely to be encountered as mining goes deeper and the nature of the orebodies changes.

Strategy 2, which addresses *regional support systems to enable safe mining and maximum ore extraction in highly stressed ground,* requires urgent attention. This is because recent experience has shown that the pillar systems currently in place no longer provide adequate regional support to the stope-face area, nor to the service excavations in the shaft pillar and access ways into the stope area.

However, current research into the use of backfill, such as combinations of backfill and crush pillars, as well as concrete pillars of varying strength and geometry, appears to provide viable alternatives for regional support, as well as permitting the extraction of greater quantities of reef. This work, together with work on the improved design of bracket pillars for stabilizing geological structures, should receive urgent attention if the ultra-deep mining projects envisaged for the next decade are to be implemented successfully.

Strategy 3, which is aimed at *improving our understanding of mining-associated seismicity and of how its damaging effects can be minimized,* will obviously contribute not only to the work outlined in strategy 2, but will also provide valuable input into other strategies. A distinct advantage is the availability of locally developed state-of-the-art seismic technology, together with an understanding of how this technology can be used in the evaluation of changes in the near- and far-field rockmass. However, an understanding of the quantitative rockmass-deformation parameters is lacking, as are numerical modelling procedures that can realistically simulate failure of the rockmass *in situ.* These aspects also require urgent attention.

However, progress has been made with the development of pro-active methods to control the seismic events initiated during the mining of reef adjacent to faults and in remnant areas. The progress made in the implementation of preconditioning as a means for extracting high-stress blocks of reef is particularly encouraging, and augurs well for mining at great depths.

Strategy 4, *understanding rockburst mechanisms so as to devise effective rockburst control measures,* builds on strategy 3 to a large extent and has as its main objective the quantification of rockmass response to sudden seismic loading. Current knowledge suggests that, during a rockburst, ground movements can happen at rates of 2 to 50 m/s depending upon whether the accelerated rockmass is highly overstressed or not. This information on the magnitude of rapid ground movements is necessary if suitable support systems are to be devised for stopes and tunnels in rockburst-prone areas. Again, seismic-monitoring systems are crucial to provide relevant information on the failure mechanisms operating in the rockmass.

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Strategy 5 addresses *mining at great depths and in wide-reef areas with respect to support and other rock-engineering applications.* Ultra-deep projects that are planned for the next decade include Sun, Target, Moab, and South Deep. In addition, many of the currently operating mines are planning to mine the Ventersdorp Contact Reef at stope widths of up to 5 m. Of these projects, the South Deep appears to be most advanced in its planning and implementation. The orebody to be extracted varies in width from 2 to 40 m, and will be extracted after being destressed by mining-out of the overlying Ventersdorp Contact Reef. Room-and-pillar mining, followed by pillar extraction and backfill placement, is planned for the removal of an 8 m wide zone in the upper part of the reef, followed by vertical crater mining of the lower wide orebody.

Projects of this nature require an initial understanding of the *in situ* stress field, together with the rockmass properties. Such an understanding would facilitate design layouts and numerical modelling to indicate the likely response of the rockmass and the support forces required to ensure excavation stability. Of particular importance is the stability of the stope face and the excavation-wall rock, which is a function of the presence of geological weaknesses. In general, destressing of excavations should be implemented, as should backfilling in the mined-out areas.

With respect to temporary support, face bursting is likely to occur because of high stresses acting on the face, and innovative support or protection methods may need to be devised, as will methods of support for hangingwalls and gullies, particularly if the footwall undulates significantly. Support methods that should be considered include the use of improved shotcrete mixtures, flexible skin coatings, yielding tendons and, if appropriate, hydraulic props.

Strategy 6 is concerned with *seismic and other methods for the monitoring of changes in rockmass condition, particularly with respect to its potential for failure.* The requirements for this are seismic tomographic methods, together with an understanding of the ambient stress conditions and the changes in the stress field with mining and time. This calls for monitoring systems that are able to record the effect of rapid or dynamic stress changes, as well as time-dependent creep or slow stress-induced movements. The techniques available include seismic imaging for the identification of high-stress zones, and ground-penetrating radar for the monitoring of fracture growth around excavations and in front of the mining face.

Also of importance is long-term monitoring of the stability of essential excavations and of support effectiveness. This is especially relevant to service excavations in key areas like the shaft pillar, which, if severely damaged because of slow, time-dependent stress-induced ground movements, may have to be abandoned. Similarly, mining in the vicinity of or over haulages can render support systems in the haulages ineffective, resulting in the need for rehabilitation of the support.

Two specific devices need to be developed in the short term. These are an instrument for the measurement of rapid ground movements in tunnel walls and stope hangings so that better rockburst-resistant support systems can be developed, and an instrument to monitor the quality of the support installation. The latter device will require considerable research and ingenuity, since major efforts to develop prototypes have failed.

Strategy 7 addresses the technology required to enable mining to take place at depth *by means of mechanized equipment so that the use of explosives is eliminated.* The benefits of this approach are two-fold: fewer mineworkers would be required in the stope face, and mining operations could be carried out on a three-shift cycle. Currently, two prototype options are being evaluated: an impact-ripper system, and a diamond-wire slot-cutting system. The impact ripper has an advantage in that it is powered by hydropower, which has additional environmental benefits.

These non-explosive mining systems would also have significant rock-engineering benefits, such as better cleaning of the stope face, sorting of the waste rock at source and its utilization as backfill, hydraulic power for prop installation and tendon grouting, and high-pressure water guns to destress the stope face and provide power for fluid injection on faults.

However, there is the potential disadvantage that, if seismic energy is not released during the blast, energy could be released in the stope face, resulting in casualties and damage to very expensive equipment.

This dynamic response of the rockmass to ensure the release of available seismic energy highlights the importance of strategy 8, which stresses the need for the *development of non-linear analytical techniques to allow modelling of the various mining strategies.* O'Connor[101], emphasizing the hazard of delayed seismic energy release should it not be released during the blast, points out that it represents the inelastic deformation of the rockmass as a result of the enlargement of an excavation. The other relevant aspect of the rockmass response during mechanized mining operations is the need for the rock at the stope face to fracture routinely under the load acting on the rockmass just behind the face. If this fracturing does not occur, the mechanized mining equipment is unable to function as a viable mining system and, again, the use of explosives to break the reef would have to be considered.

Thus, it is clear that, for this strategy, and for virtually all the strategies discussed in this paper, the development of numerical models that enable rock engineers to understand the mechanisms of fracture or failure of the rockmass material in which the mining activities take place is very important to the South African mining industry. To obtain this understanding requires quantitative assessment of the rockmass response to mining by the use of geomechanical rockmass-condition rating procedures, geological mapping of structures, and analysis of borehole cores. Seismic tomographic methods can be used to improve the basic rockmass rating, as can ground-penetrating radar, by identifying stress fractures, geological structures, and sedimentary features. Given this information, together with measurements of the deformation response of relatively large volumes of rock to mining, it becomes possible to calibrate numerical analytical tools and use them to carry out hypothetical studies into failure processes during the loading of the rockmass.

Clearly, this approach of combining the deformation behaviour of the rock, observed either *in situ* or from laboratory tests, with non-linear numerical models will greatly enhance our understanding of the processes operating during mining, and will allow at least partial calibration of the models. However, this will enable rock engineers to design excavations and reef-extraction procedures only on a risk, or at best a 'fail safe', assessment.

This approach, together with the development of improved numerical models that can accommodate seismic information, rockmass-failure characteristics, the effect of support, and realistic deformation-time characteristics, could well provide design philosophies for the mining layouts required to extract orebodies at the great depths at which mining will take place in the years to come. ◆