Newsletter

AUSTRALIAN CENTRE FOR GEOMECHANICS

CSIRO | Curtin University | The University of Western Australia
Joint Venture

Australian Centre for Geomechanics | Volume No. 41 | December 2013

Vale's Copper Cliff Mine, courtesy of Vale Canada Ltd., Canada

Performance of dynamic support system in highly burst-prone ground conditions at Vale's Copper Cliff Mine – a case study

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Introduction

Copper Cliff Mine is located within the Copper Cliff Offset in the limits of the City of Greater Sudbury, Ontario, Canada (Figure 1). The Copper Cliff Offset extends about 8 km south from the Sudbury Igneous Complex into the footwall rocks.

Geology

Of all the major geological structures present at Copper Cliff Mine, two structures are known to be seismically active (Yao et al. 2009), (Figure 2):

1. The 900 orebody (OB) cross fault, which strikes east–west and dips at about 55° towards north.
2. The Quartz Diabase Dyke (trap) located between 100 and 900 OBs striking east–west and dipping steeply towards north.

Rockburst history at Copper Cliff Mine

A review of the rockburst/seismic event history over the past 13 years at Copper Cliff Mine revealed that there were approximately 40 rockburst/significant seismic event incidents in total that occurred in four different orebodies. Of all these incidents, 35 of them (roughly 87%) occurred within the 100/900 OBs, and the remaining five incidents (almost 13%) took place in the 120 and 880 OBs.

Of all the rockbursts, the 3.8 Mn event that occurred on 11 September 2008 in the 100/900 OBs following a crown blast was considered to be the most significant. Although the location of the major event was on 3050 L in the 100 OB the damage was extended across nearly a 300 m vertical block starting from

2700 to 3710 L. Approximately 3,000 t of material were displaced at five locations on different levels. The damage was mostly associated with either the trap dykes and/or 900 X-fault. The support system at the damage locations mainly consisted of resin grouted rebars, and mechanically anchored bolts in the back, and anchored mechanical bolts on the walls to 1.5 m above the floor installed through #6 gauge welded wire mesh. At some locations, shotcrete and cable bolts were used as a secondary support system. The installed ground support system was too stiff in nature and it did not provide much yielding capability. Accordingly, the support system that was employed at the damaged locations was incapable of taking the impact of dynamic loading caused by the 3.8 Mn event.

It should be noted that a central blasting system was used and the Copper Cliff Mine re-entry protocol after major seismic events was followed. No personnel injuries occurred due to these events.

It has been concluded that the trap dyke and the 900 OB X-fault are major contributing factors for elevated seismicity in the 100 and 900 OBs (Yao et al. 2009). The rationale for this kind of thinking could be better explained with the help of the layout shown in Figure 3:

- Since all the stopes along the 900 OB x-fault were mined out on the mining front between 3500 and 3050 L, the natural confinement that the orebody provided to the fault plane was taken out. As a result, a major displacement might have occurred along the fault-plane and caused the 3.8 Mn event after taking the crown blast in the 94561 stope between 3050 to 3200 L on 11 September 2008. By all means, the crown blast could have triggered the slip and caused the large magnitude event.
- As it can be seen in Figure 3, there is a trap dyke between 100 and 900 OB, which is very strong material and highly brittle in nature. As mining

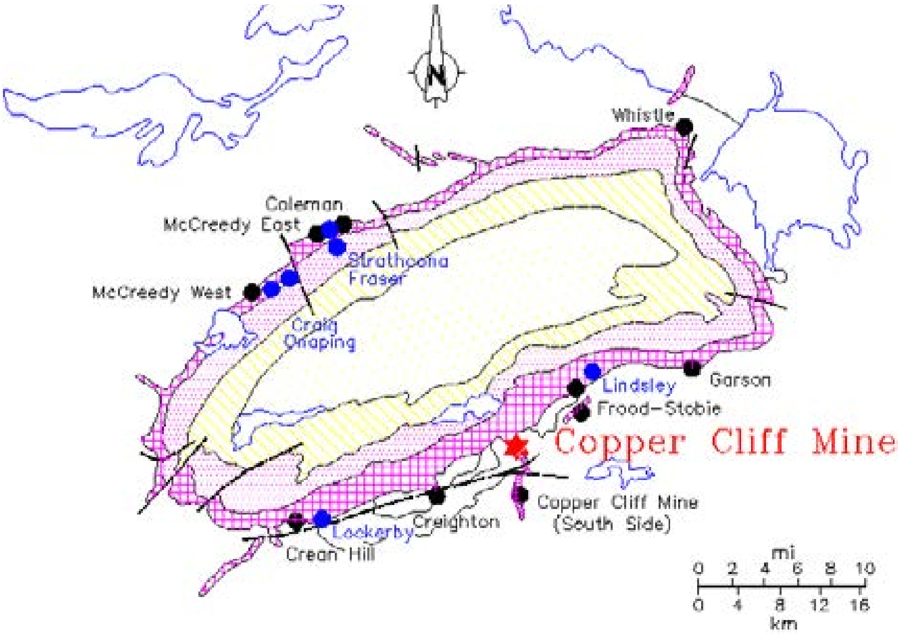

Figure 1 Location of Copper Cliff Mine in the Sudbury Basin of Vale Operations

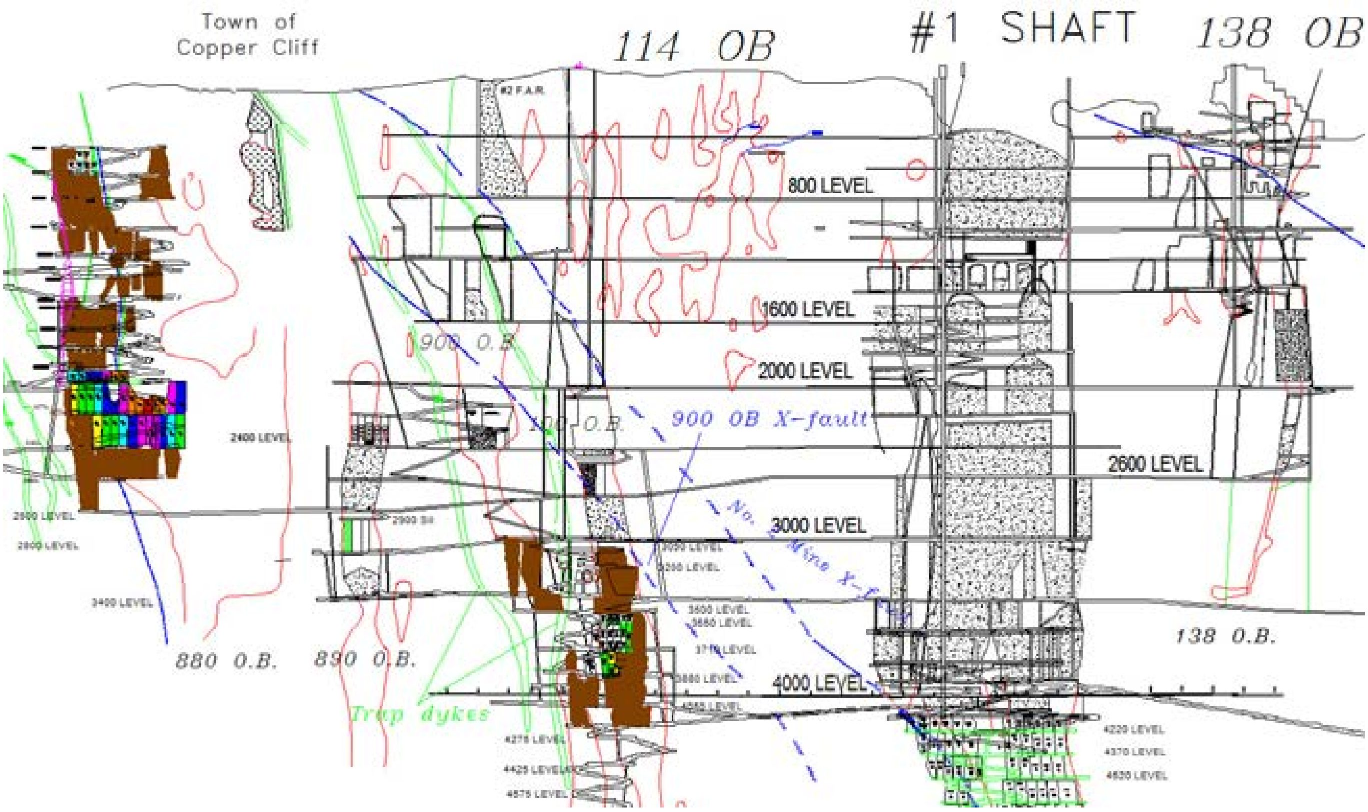

Figure 2 Location of 900 OB X-fault and trap dykes with reference to 100 and 900 OBs

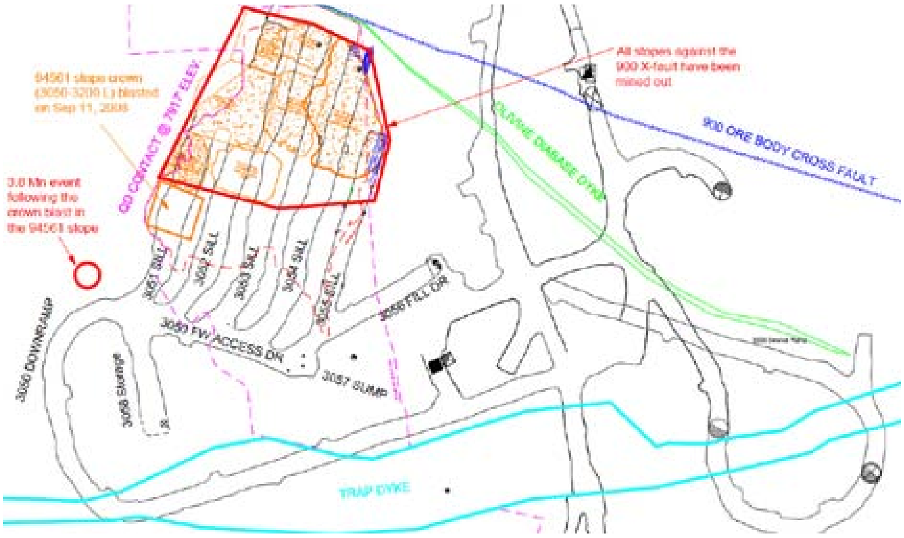

Figure 3 Layout showing the location of trap dyke and 900 X-fault in relation to mining between 3050 and 3200 L

Figure 4 Final product of burst-resistant (dynamic) support system used in one of the drill sills at Copper Cliff Mine

progressed in the 100 and 900 OB, the trap dyke is loaded up, which can lead to significant seismic events/rockbursts.

The 3.8 Mn event on 11 September 2008 was considered to be a result of mining the 94561 stope between 3050 and 3200 L. The location of the stope in relation to the major geological structures, i.e. trap dyke and 900 X-fault is shown in Figure 3.

Since large magnitude events are associated with damage to underground excavations and the installed ground support systems, mining in the burst-prone ground conditions pose a greater challenge both in terms of safety and production. The 3.8 Mn rockburst triggered a series of rockbursts within the limits of the 100/900 OBs and caused damage at multiple locations on different levels. In order to rehabilitate all the damaged areas, considerable time and resources were spent, and production was significantly impacted.

Introduction of burst-resistant support system in burst-prone ground conditions

A system was introduced in all the burst-prone areas at Copper Cliff Mine, with a view to minimise or completely eliminate the damage to the installed ground support and/or the underground excavations in the event of future occurrences. A rating system was developed to identify the burst-prone areas (Yao et al. 2009).

Burst-resistant support elements used in burst-prone ground conditions at Copper Cliff Mine

Based on the guidelines outlined in the 'Canadian Rockburst Support Handbook' (Kaiser et al. 1996), the following ground support elements were identified and used in the burst-prone ground conditions at Copper Cliff Mine.

For walls: 1.95 m long FS-46 split sets on a 1.2 × 0.75 m pattern with #4 gauge welded wire mesh, followed by a minimum 76 mm thick pass of plain shotcrete, and then 2.3 m long modified cone bolts on a 1.2 × 1.8 m pattern with #0 gauge mesh straps. The wall bolting was usually extended to the floor level.

For the back: 2.4 m resin rebars on a 1.2 × 0.75 m pattern with #4 gauge welded wire mesh, followed by a minimum 76 mm thick pass of plain shotcrete, and then 2.3 m long modified cone bolts on a 1.2 × 1.8 m pattern with #0 gauge mesh straps. In addition, 6.3 m long twin cable bolts were used in a ramp, where the depth of failure was almost 5.1 m from the seismic events. The purpose of the cable bolts was to reinforce the rockmass as well as hold the broken rockmass by anchoring them in the solid ground.

The burst-resistant support system that was employed in burst-prone ground conditions at Copper Cliff Mine is shown in Figure 4.

Energy absorption capacity and load-displacement characteristics of various ground support elements (Kaiser et al. 1996) that were used as rockburst resistant support system at Vale's Copper Cliff Mine are given in Table 1.

Based on the above energy absorption values, the total energy absorption capacity that was employed in the burst-prone ground conditions at Copper Cliff Mine was calculated anywhere between 20 and 48 kJ per m^2. The required energy absorption capacity was determined based on the 3.8 Mn event.

Performance of dynamic support system

After introducing the burst-resistant system at Copper Cliff Mine, mining in the 100/900 OB was resumed. Four stopes were mined out successfully without any significant damage.

With the resumption of mining in the 100/900 OB, Copper Cliff Mine once again started to experience elevated seismic activity, particularly while mining the stopes surrounding the trap dyke. Several seismic events/rockbursts, ranging from 1.2 to 2.9 Mn, occurred while mining the 9551 and 9281 stopes. The chronology of the seismic events/rockbursts that occurred while mining the stopes in the vicinity of the trap dyke is given in Table 2.

It was interesting to observe that there was no damage, after the 2.9 Mn event that occurred on 18 February 2009, while mining the 9551 stope. In fact, the event was located within 20–30 m from the top and bottom sills, respectively. This has demonstrated that the rockburst resistant support system that was installed after the large 3.8 Mn event had sufficient energy absorption capacity to withstand the

Table 1 Energy absorption and load-displacement characteristics of support elements (Kaiser et al. 1996)

Description	Peak load (kN)	Displacement limit (mm)	Energy absorption (kJ)
19 mm resin-grouted rebar	120–170	10–30	1–4
46 mm split set bolt (FS-46)	90–140	80–200	5–15
16 mm modified cone bolt	50–100	100–200	10–25
16 mm cable bolt	160–240	20–40	2–6
#4 gauge welded wire mesh	34–42	150–225	3–6 per m^2
Shotcrete and welded wire mesh	2 × mesh	< mesh	3–5 × mesh*

** at displacements below 100–150 mm.*

Table 2 *Seismic events/rockbursts occurred while mining the stopes in the vicinity of trap dyke in 100/900 OB*

Date	Magnitude (Nuttli)	Orebody	Level (stope)	Remarks
18 February 2009	2.9	100	3710–3880 (9551)	No damage to the mine openings
30 September 2010	1.9	900	3710–3880 (9281)	No damage to the mine openings, but 80–100 t of trap dyke material was sloughed into the open stope
2 October 2010	1.6	900	3710–3880 (9281)	No damage to the mine openings
4 October 2010	1.4	900	3710–3880 (9281)	No damage to the mine openings
4 October 2010	1.2	900	3710–3880 (9281)	No damage to the mine openings
5 October 2010	1.9	900	3710–3880 (9281)	Some minor surface cracks in the shotcrete
15 October 2010	2.3	900	3710–3880 (9281)	Minor damage to the installed ground support system and some floor heaving

Figure 5 *Minor cracking and bulging of the support system following a 2.3 Mn event in 9280 top sill*

impact of a 2.9 Mn event.

While mining the 9281 stope, the installed burst-resistant support system was repeatedly subjected to seismic event impacts and showed some signs of negligible damage. Although it is difficult to assess the impact of previous seismic events in a quantitative manner, the ground control engineer identifies whether there are signs of support yielding based on their observations, and/or field instrumentation monitoring, if any. If so, it may be prudent to install extra support in an effort to compensate for any potential loss in the safety margin (Kaiser et al., 1996).

Although the support system was subjected to repeated seismic loading, the burst-resistant support system showed its first sign of damage only after the 2.3 Mn seismic event (see Table 2 for the order of events). However, the level of damage was very insignificant (Figure 5).

Conclusions

Even though many seismic events occurred in the 100/900 OBs while mining in the burst-prone ground conditions, no significant damage was associated with such events after introducing the burst-resistant support system at Copper Cliff Mine. It was evident from the underground observations that a well designed dynamic support system will cope very well in the event of large and repeated seismic events, by sustaining the impact of dynamic loading with no, or negligible damage to the underground excavations and/or the installed ground support system. Four stopes were mined out successfully without any significant damage after introducing the burst-resistant support system in the areas at Copper Cliff Mine.

Please contact the ACG for the full paper that was published in the Deep Mining 2012 Seminar proceedings.

Please click here for article references.

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ACG launches GSSO: a new international research project on Ground Support System Optimisation

writes ACG director, Winthrop Professor Yves Potvin

Traditionally, rockfalls have been the main cause of fatalities in underground hard rock mines worldwide. During 1980 to 1997, Western Australian mines were no exception to this rule. Lang (1999) reported that up to 45% of all fatalities in the state were caused by rockfalls during this period.

A number of mitigation measures can be applied to reduce the risk of rockfalls. Arguably, the most effective measures target the reduction in exposure of mine personnel to this risk through the implementation of highly mechanised mining methods. This is reflected in the significant differences in the rockfall injuries and fatalities statistics from countries applying highly mechanised mining methods compared to countries where labour intensive mining is employed.

Once the mining method has been decided, a second level of risk mitigation can be implemented by devising a mining sequence which manages mine induced stresses by funnelling the potential stress into other areas where the consequences can be subsequently minimised and controlled.

The third level of rockfall risk mitigation is the installation of the ground support system. This is an area where Australian mines have focussed their effort over the last 15 years to achieve a step change in rockfall related safety statistics. According to data published by the Minerals Council of Australia, the last rockfall fatality in an Australian underground hard rock mine to date occurred in 2006.

Many factors contributed to this eradication of rockfall fatalities, but amongst them, the industry-wide systematic installation of cut-by-cut reinforcement and surface support played a major role. In Australian mines, access to areas which are not fully supported by rockbolts, mesh or shotcrete is generally not permitted. In most cases, the ground support systems have a very high Factor of Safety (FS).

The sustained success of ground support in mitigating rockfalls has generated enormous benefits to the local mining industry but it has come at a cost. In many Australian mines, ground support is the largest component of both overall development mining costs, and the time consumption required in the development mining cycle.

The question arises as to whether it is possible to optimise ground support systems, with the aim to maintain, if not improve safety aspects, whilst reducing the cost and/or time components. This is the ambitious objective that the Ground Support System Optimisation (GSSO) project is set to achieve.

The project was developed based on extensive consultations with senior rock mechanics engineers from around the world. The first meeting was held at The University of Western Australia in early 2012. There was a unanimous view amongst industry leaders that the first priority was to develop a new comprehensive yet practical ground support guide. Considering that the last textbook written (in English) on the topic, by Hoek, Kaiser and Bawden (1995) will be over 20 years old by the time this new guide is printed, an updated guide is well overdue.

Furthermore, the view was that there is an abundance of excellent work available that needs to be collected, compiled and presented in a format that is useable by engineers at mine sites. This was seen by participants as a critical initial step towards optimising ground support systems in their mines. A second meeting was held in Canada, taking advantage of the presence of world mining practitioners at the MassMin Conference held in Sudbury, in June 2012. The research proposal was then refined based on this consultation and three other research sub-projects were confirmed for inclusion in the guide.

1. Probabilistic ground support design.
2. The use of numerical modelling for ground support design.
3. Benchmarking of current ground support design practices.

Probabilistic design offers a unique opportunity to progress from the widely used FS approach, to a more sophisticated statistical-based approach. In particular, FS combines all the uncertainties and inaccuracies related to the input data and the design method, as well as the degree of conservatism of the design, into an arbitrarily chosen single number. Probabilistic design allows for a variety of design outcomes to be assessed. Ultimately, it will provide the basis for a non-biased decision-making process based on the cost associated with a certain probability of failure versus the cost of risk reduction measures. Sub-project one on probabilistic design will be conducted by SRK Consulting (SA) Pty Ltd, South Africa, under the leadership of William Joughin and Dr Johan Wesseloo, ACG.

The so-called conventional design methods, whether they rely on empirical systems or analytical approaches, including limit equilibrium equations, generally offer a poor representation of the complex interaction between ground deformation

and reinforcement and the surface support response. Only numerical modelling can handle this level of complexity. Previously, a number of numerical modelling codes were applied by consultants and mine practitioners using different approaches for the design ground support systems. The calibration processes often consist of crude comparisons between model and past observations (when available), and the reliability of these models are rarely established. The objective of this sub-project is to develop a methodology based on currently existing numerical modelling tools, to design ground support systems. GSSO seeks to shed light on the limitation, relevance and accuracy of using various modelling approaches, in different contexts and situations. This second sub-project will be led by Winthrop Professor of Geotechnical Engineering Phil Dight, ACG, to be assisted by a new ACG position (a numerical modelling expert).

The third sub-project will simply involve the collection of data at sponsoring mines and other operations of particular interest for ground support system design. In addition, to establish what are the current practices and to collect some cost and cycle time data, this project will allow the collection of case studies to be used in the guide, as ground support design examples. Professors John Hadjigeorgiou, University of Toronto, Canada and Yves Potvin, ACG will conduct this research. The results from the three sub-projects will feed into the writing of the guide.

GSSO commenced its activities on 1 November 2013. It has a budget of more than AUD 1.7 M, with a 3.5 year duration. Currently GSSO has the support of six major sponsors: Glencore Xstrata Zinc, Independence Group, Codelco NML, MMG Australia Ltd, MERIWA, and the ACG, as well as five minor sponsors: DSI, Jenmar Australia Pty Ltd, Fero Strata Systems Pty Ltd, Golder Associates Pty Ltd and Geobrugg Australia Pty Ltd. The sponsors not only offer significant financial contributions but also a large pool of expertise that provides extremely valuable input into the research projects and development of the guide. The project will culminate with the publication of a new ground support design guide in mid 2017.

Please click here for article references.

Winthrop Professor Yves Potvin
Australian Centre for Geomechanics

Guidelines for deriving alarm settings based on pre-determined criteria using the movement and surverying radar

by Alex Pienaar and Anton Joubert, Reutech Mining, South Africa

Introduction

Mining companies have a moral and legal obligation to ensure the safety of their personnel by making use of industry leading best practices. In the case of open pit mining, where people and machinery are constantly working at the base of steep man-made slopes, industry best practices require the continuous management of risks relating to potential slope instability and related hazards.

One of the methods for effectively managing these risks is slope stability monitoring. A properly implemented slope stability monitoring programme will ensure that (Call & Savely 1990):

1. Additional geotechnical information about the slope behaviour is provided to engineers.
2. Safe operational practices are maintained.
3. Advance warning of impending instability is given so that actions can be taken to minimise the impact of the slope failure.

It is Call and Savely's (1990) third point, with regards to slope stability monitoring programmes, that is the particular discussion area of interest in this article.

Advance warning mechanisms

Radar as an advance warning mechanism

Reliable advance warning mechanisms give an indication of imminent collapse and provide a realistic lead time to evacuate problem areas. Radar technology has become entrenched within the mining industry and its implementation as an advance warning mechanism is widely regarded as best practice. A radar-based slope stability monitoring system irradiates a slope with electromagnetic waves, a portion of which is reflected back to the radar. By processing this received signal, the radar can then detect movement in the slope walls. Radar provides the user with the ability to rapidly and accurately scan large areas of a mine wall without the need to install any reflectors. It is also able to work in foggy or dusty conditions that would render laser-based systems blind.

In order for a radar system to be considered an effective early warning mechanism it needs to operate in virtually real-time and should be linked, on a 24/7 basis, to the mine's key emergency communication system. It also needs to be capable of accurately detecting slope movement of as little as 1 mm over distances of 2,500 m. It is important to note that however accurate and reliable the radar surveillance equipment may be, its effectiveness is greatly dependent on the selection of the correct advance warning limits, also known as alarm thresholds.

The importance of selecting appropriate alarm thresholds

Modern day radar surveillance equipment available to geotechnical engineers all provide advance warning by means of a given alarm mechanism. This mechanism will typically raise an alarm when a certain pre-determined alarm threshold is reached. At first glance this all seems simple enough, however Sjöberg cautions that the difficulty is not one of measurement technology, but rather of what these alarm thresholds should be (Sjöberg 1999).

Studies conducted by Zavodni and Broadbent (1978) reinforced this point and found that the magnitude and velocity of an unstable failure varies widely from mine to mine (Zavodni & Broadbent 1978). It logically follows that alarm thresholds used at one mine cannot simply be transferred and used at another mine (Kostak & Rybar 1993).

It is evident that the effectiveness of radar monitoring, as well as the larger slope monitoring programme of which it forms a part, hinges on the fact that the appropriate, mine specific alarm thresholds are selected.

This article provides guidelines to assist geotechnical engineers with deriving alarm settings based on pre-determined criteria, such as required warning time and expected slope failure profile. Field data from the Reutech Mining Movement and Surveying Radar (MSR) provides further insight into the fundamental radar concepts that are critical to understanding and applying this approach.

The action taken when an alarm is exceeded is typically to evacuate the affected work area; this is a critical alarm. However, in practice it is common to define a second threshold level that will trigger some time earlier; a geotechnical alarm. When this earlier alarm is raised, geotechnical engineers are alerted to investigate the movement, which they may not have been aware of, and decide on the requisite action. Defining multiple levels of alarms is simply a matter of selecting different warning times.

Fundamental considerations

Generic slope failure movement profile

Initially, when setting alarms in a mine without any experience, the threshold

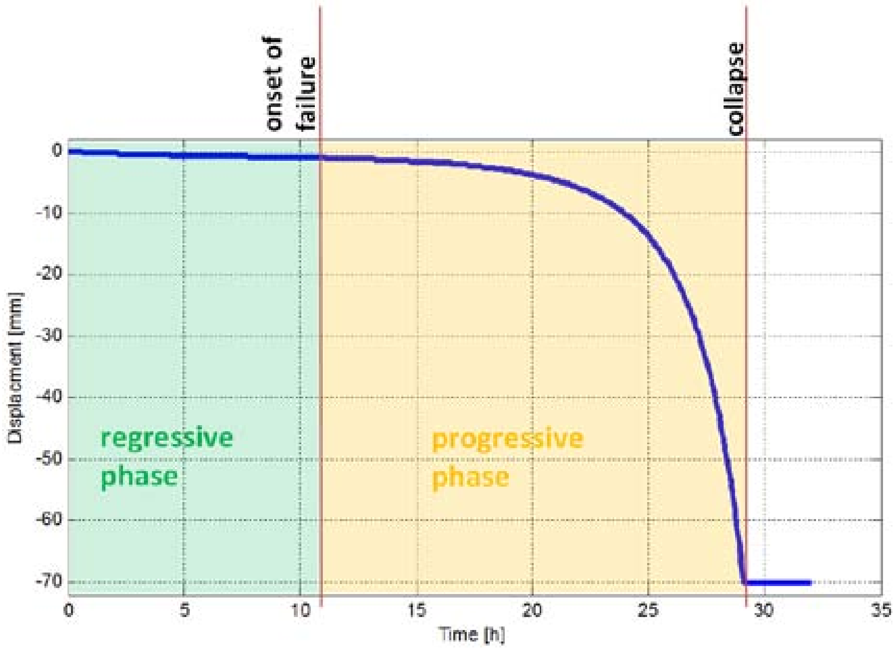


Figure 1 Generic slope failure movement profile showing phases

should be set above the error band for the device in the prevailing conditions. This will minimise the number of false alarms. With time, these alarm triggers can be adjusted to meet the geotechnical requirements.

When selecting alarms for a certain area of concern, it is extremely important to determine the type of failure and to be able to predict the failure's behaviour (Sjöberg 1999). These two factors have major implications on the selection of alarm thresholds and in turn affect the mine's productivity and safety. In most cases, mining activity could continue despite slope displacement assuming that the failure is not accelerating, the failure mechanism is well-defined and that the monitoring is performed continuously (Call et al. 1993).

A study by Zavodni and Broadbent found that all large scale failures occurred gradually and that the slope failure movement profile can be defined by two distinct failure stages known as a regressive and progressive stage (Zavodni & Broadbent, 1978), (Figure 1).

The regressive failure stage is characterised by zero or constant velocity whereas the progressive failure stage is characterised by an increasing velocity that eventually leads to slope collapse (Sjöberg 1999). Others (Zavodni & Broadbent, 1978) have a wider definition of the regressive phase, including episodic events with sudden acceleration and subsequent deceleration, without collapse.

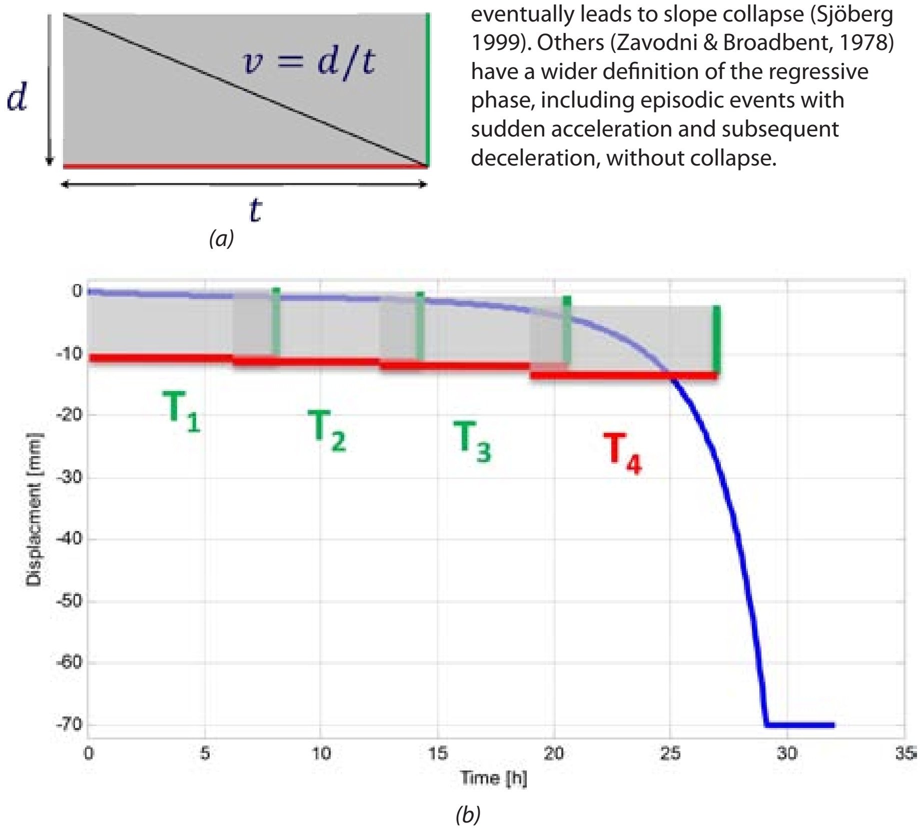


Figure 2 Calculating average velocity alarms

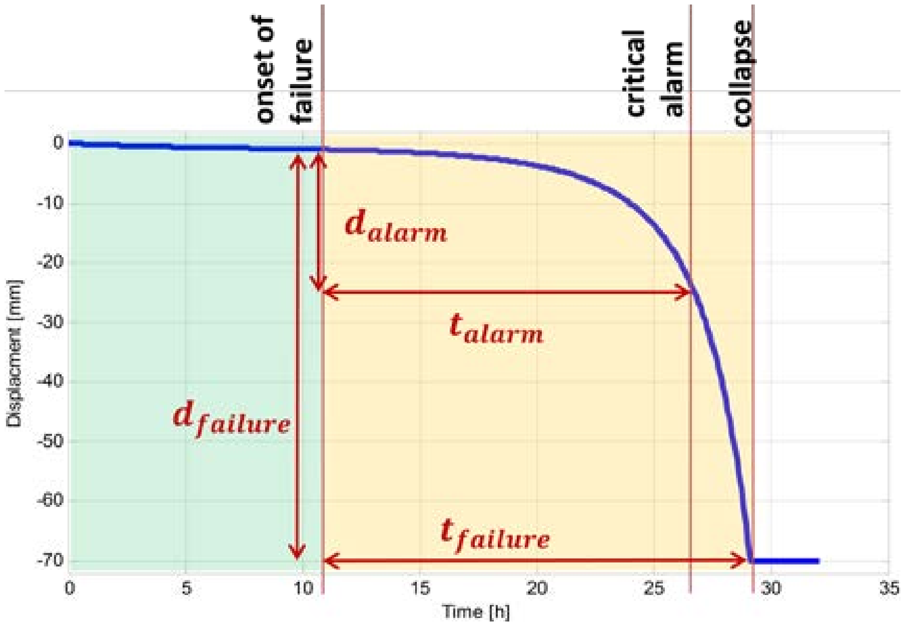


Figure 3 Generic failure movement profile showing alarm parameters

Designing velocity-based alarms

In the previous section, the regressive and progressive failure stages of the generic slope movement profile were defined in terms of displacement. Displacement-based alarm thresholds could be used, however in the case of a constant displacement rate, they may eventually be exceeded, even if a collapse is not imminent. Displacement rate, i.e. velocity, provides a better indicator of imminent failure as it exhibits less variations (Sjöberg 1999). It is therefore fitting that the accompanying alarm thresholds are also based on velocity.

There are various ways of calculating a velocity from the sampled displacement measurements. Two general approaches are:

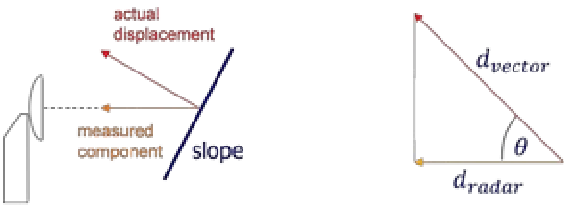
1. Calculating an empirical curve that best fits the slope movement profile; or
2. Calculating average velocity based on displacement over a certain time window.

The first approach entails methods such as using a multiple order polynomial that best represents a known displacement curve for the specific area of concern. The polynomial can then be differentiated to yield a velocity curve for alarm purposes. Fitting a polynomial to the scaled slope movement profile for all measured points is a computationally intensive exercise, and in some cases can lead to errors. The calculation of an average velocity, *v*, based on the total displacement, *d*, over a fixed time period, *t*, is a much simpler way to achieve a similar result, where $v = d/t$.

Using the average velocity approach, the user typically needs to select three parameters; the minimum size of the area expected to fail, the average velocity (or equivalent displacement), and the time period. The radar will then consider all points being monitored, and check if the displacement for each possible area of that size exceeds the average velocity limit.

Figure 2 shows the average velocity calculation graphically. Simply put, the parameters define a box. If the displacement for the current time period exceeds *d*, i.e. the gradient exceeds *v*, then it will exit the bottom of the box (red area), and an alarm will be triggered. Otherwise, it exits the side (green area), and no alarm will be triggered. In Figure 2(b), the box is shown at various time intervals and it is clear that the alarm will be triggered in period T_4.

The minimum size area is based on geotechnical knowledge of the slope and radar point size, given the distance to the slope. In order to properly design the remaining alarm parameters, a slope failure profile relevant to the area of concern (a so-called benchmark profile) is required. In other words, the geotechnical engineer is required to have some idea of the magnitude and time-scale of the actual displacement expected for a failure.



θ	$\cos(\theta)$
0°	1.00
15°	0.97
30°	0.87
45°	0.71
60°	0.50
75°	0.26
90°	0.00

$$d_{radar} = d_{vector} \cdot \cos(\theta)$$

Figure 4 Effect of geometry on component of movement measured by radar

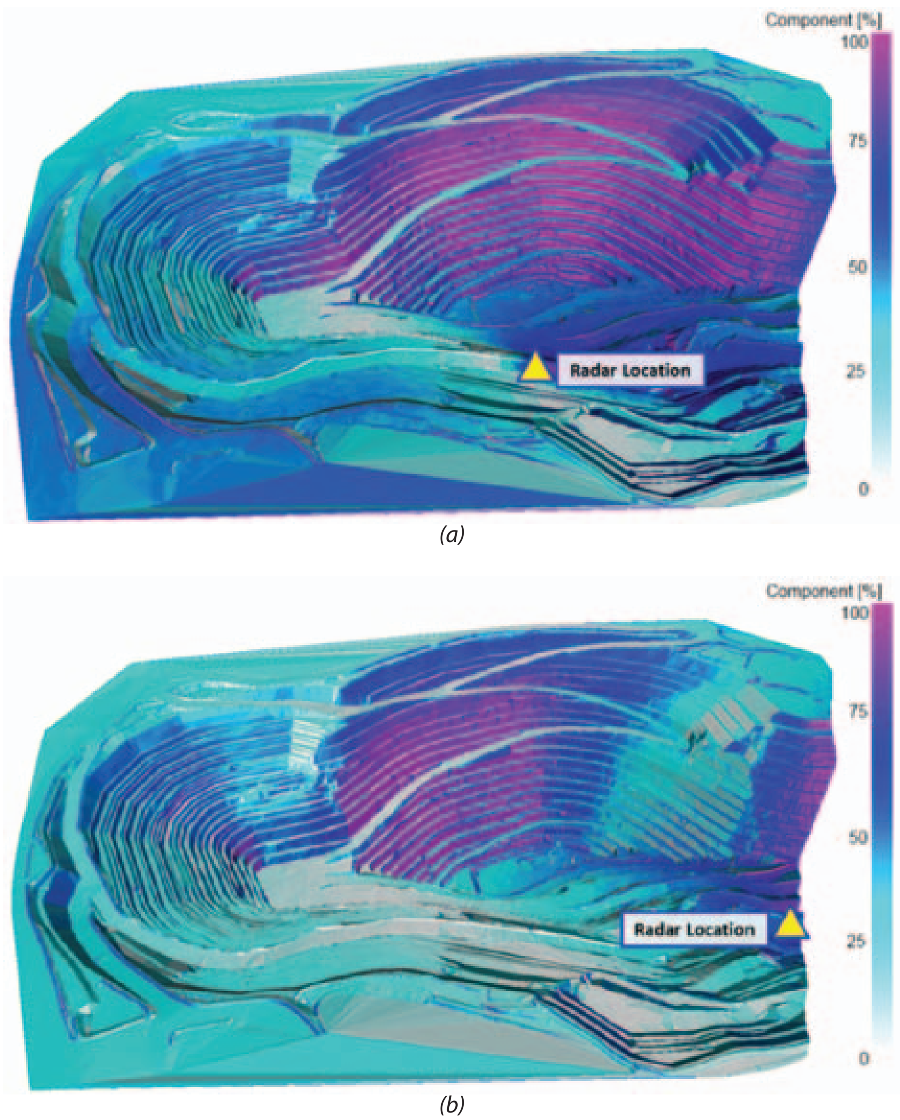


Figure 5 Comparison of geometric effects for two different radar locations

Common practice is to use measurements of previous failures at the mine. Alternatively, a theoretical model could be considered, or even measurements from another mine with near-identical geotechnical conditions, albeit with the necessary caution.

With the benchmark profile available, the remaining decision is the desired warning time prior to collapse. As depicted in Figure 3, this will give the alarm time, t_{alarm}, and alarm displacement, d_{alarm}, which define the alarm velocity, v_{alarm}.

Pit geometry and radar measurements: the scaling factor

A radar system with a single receive antenna can only measure the component of the slope's actual displacement in the direction of the antenna – the so-called line-of-sight direction. This limitation is illustrated in Figure 4.

The effect of this limitation on accurately monitoring slope displacements is rather significant as the actual displacement, d_{vector}, is always scaled by a factor of cos(θ). This cos(θ) factor can, in the best-case scenario, have a magnitude of 1, if the actual displacement is exactly parallel to the radar's line of sight to the reflector on the slope. In the worst-case scenario, it can have a magnitude of 0, where the actual displacement is perpendicular to the radar's line of sight.

A practical illustration of this geometric effect is provided in Figure 5. For this example, it is assumed that the actual slope displacement, d_{vector}, is perpendicular to the surface of the digital terrain model (DTM). The radar is deployed at two different locations and the magnitude of the cos(θ) scaling factor is indicated by the colouring. The radar position in Figure 5(a) provides better coverage of the pit, with most benches having a scaling factor well above 70%. In Figure 5(b), a large portion of the wall on the right is very oblique to the radar, so very little movement perpendicular to the benches will be seen from that position.

It is possible that the actual slope displacement may not be perpendicular to the face of the slope, but without any other information, it is a reasonable starting point. If there is some idea of the direction beforehand, then the radar can be deployed in a more optimal location.

Calculating and applying the scaling factor to slope movement profiles

As described previously, the benchmark profile is usually derived from previous measurements. If these were obtained with a prism monitoring system, then the true magnitude of the displacement, as well as the direction, will be immediately available. Other sensors, such as radars and extensometers, will only record a component of the true movement. However, geotechnical knowledge of the failure mechanism should be sufficient to estimate the actual direction of the failure, and thus the true magnitude of the displacement.

Next we look at an example where previous radar measurements are used. Figure 6 shows an example of a slope measured with the radar. For this example, let us assume that knowledge of the failure mechanism allowed the actual direction of displacement to be determined. This is indicated by the red arrow. The radar was positioned as shown, and only measured a component of that. By simple vector calculation, we determine that θ is approximately 45°, or equivalently that the cos(θ) term is 0.7.

In Figure 7, the displacement measured by the radar is illustrated with the solid brown line. As we have determined that this corresponds to a cos(θ) term of 0.7, the measured displacement is simply divided by the same factor to estimate the actual displacement. This is shown by the dashed red line.

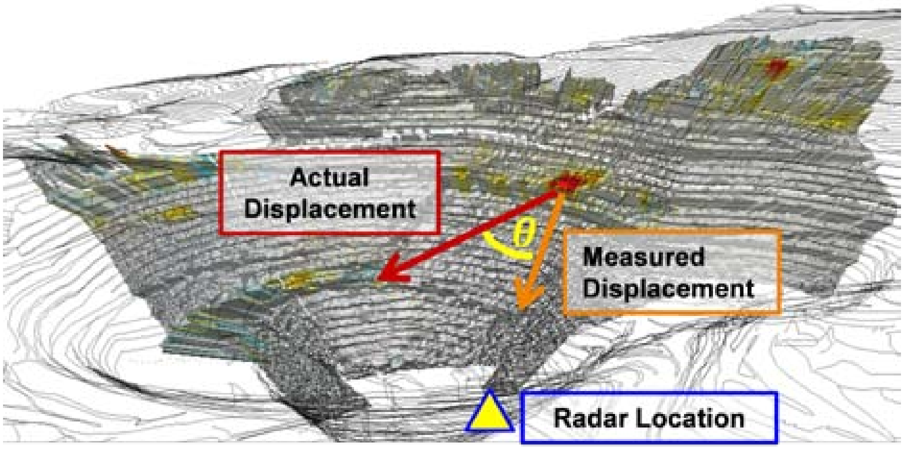


Figure 6 *Radar movement data overlaid on mine DTM showing angle between actual and measured displacement*

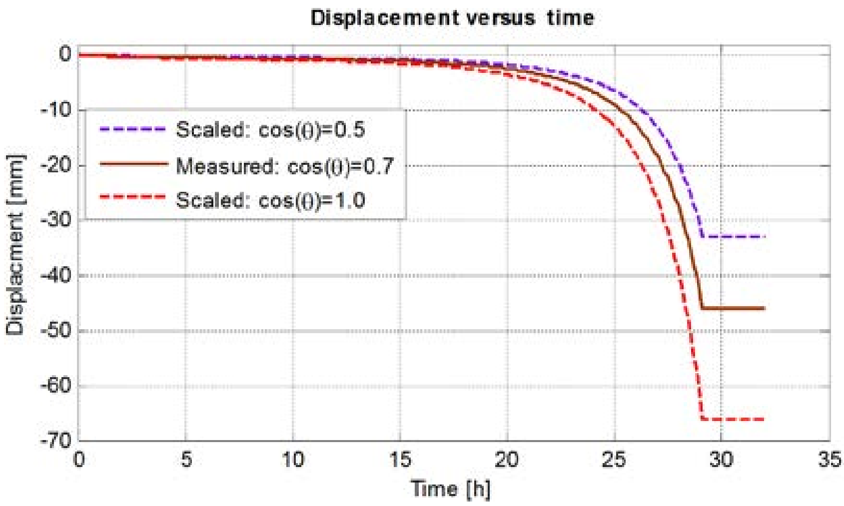


Figure 7 *Applying a scaling factor to the slope movement profile*

The actual direction of displacement for a new failure is not exactly known beforehand. A simple way to allow for this is to reduce the expected movement to what could occur in a non-ideal scenario. A reasonable, practical limit for θ is 60°. Thus, if the radar's line of sight is anywhere within ±60° of the actual direction of movement, then the slope stability alarm will be triggered as designed, or earlier. Depending on the geometry of the particular scenario, other values of θ could be used for this adjustment. However, using a very small scale factor will result in very tight thresholds, which may be impractical due to a high number of false alarms. The designer must make a trade-off between the acceptable number of false alarms and the obliqueness of the viewing geometry.

The example continues in Figure 7, where the actual displacement (dashed red line) has been scaled by 0.5, due to the ±60° allowance. This final adjusted benchmark profile (dashed purple line) is the one to use when designing alarm threshold as contemplated previously.

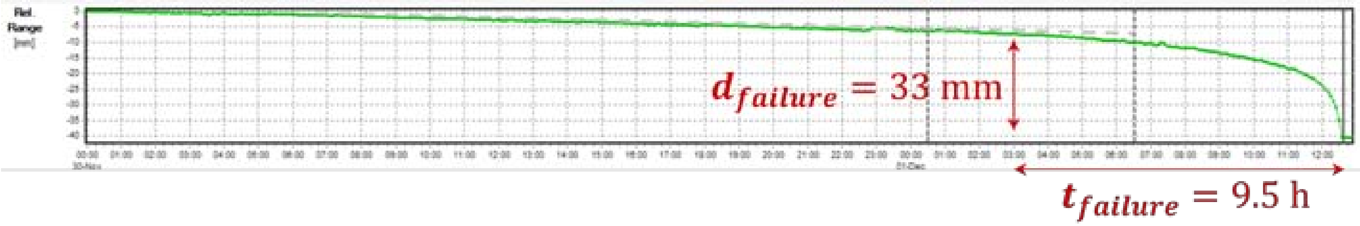
Examples

The following examples illustrate how average velocity alarms are derived based on the guidelines that have been discussed in this article. The data from the examples were collected by the MSR (Figure 8).

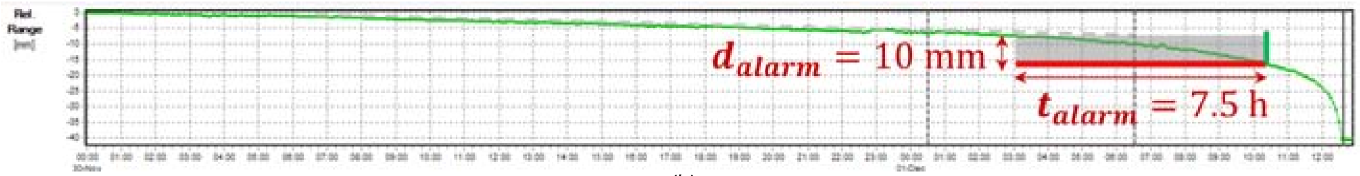
Example 1: medium time to collapse

An MSR200 system was deployed to monitor a wall with a history of failures tending to occur rapidly. An analysis of a

Figure 8 *Movement and Surveying Radar – MSR300*

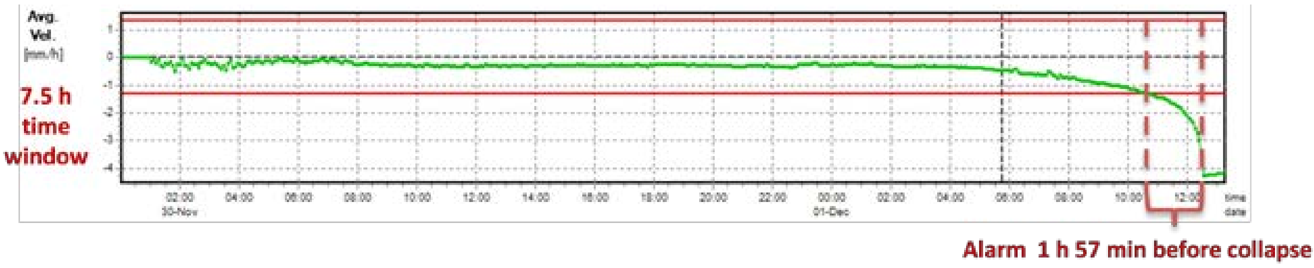


(a)

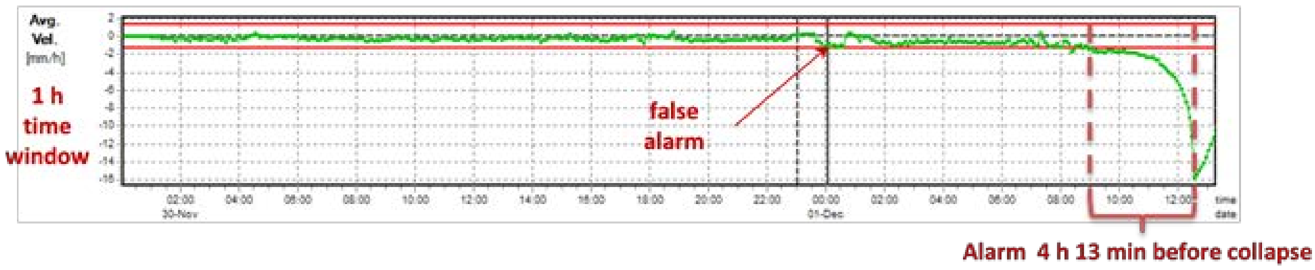


(b)

Figure 9 Scaled displacement benchmark profile of past failure, showing failure and alarm values (medium time to collapse)

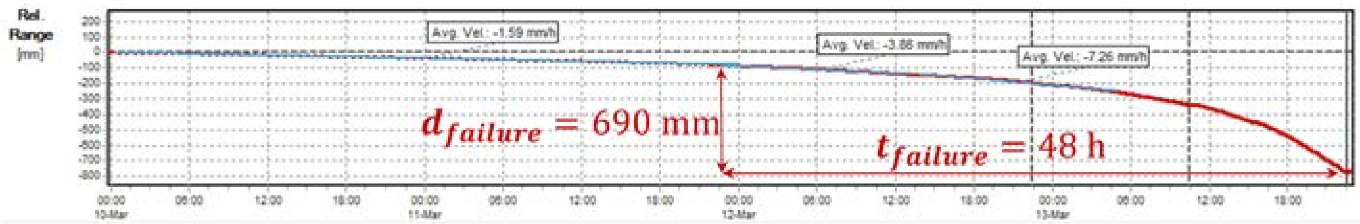


(a)

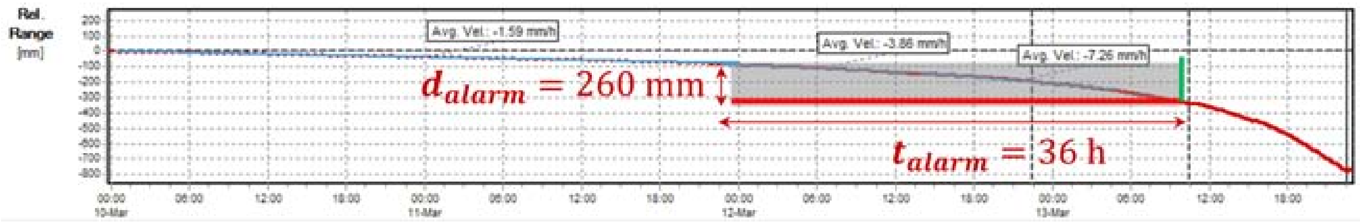


(b)

Figure 10 Comparison of average velocity graphs using different time windows showing the designed alarm thresholds

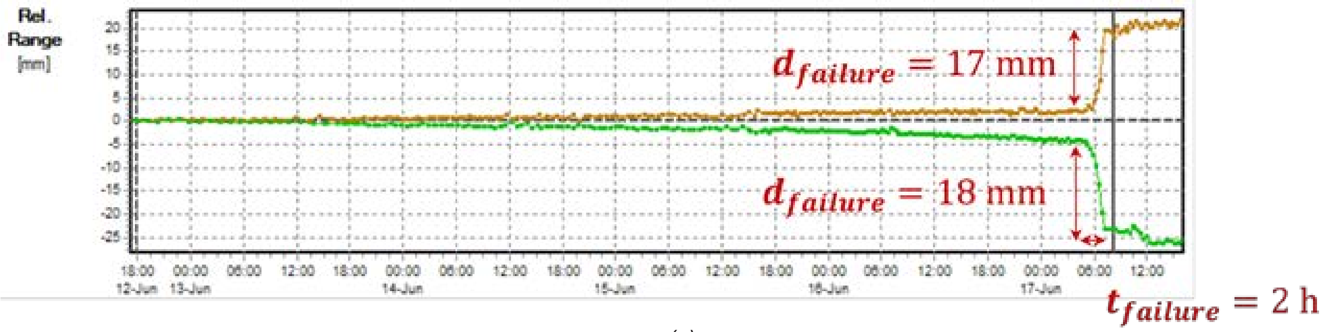


(a)

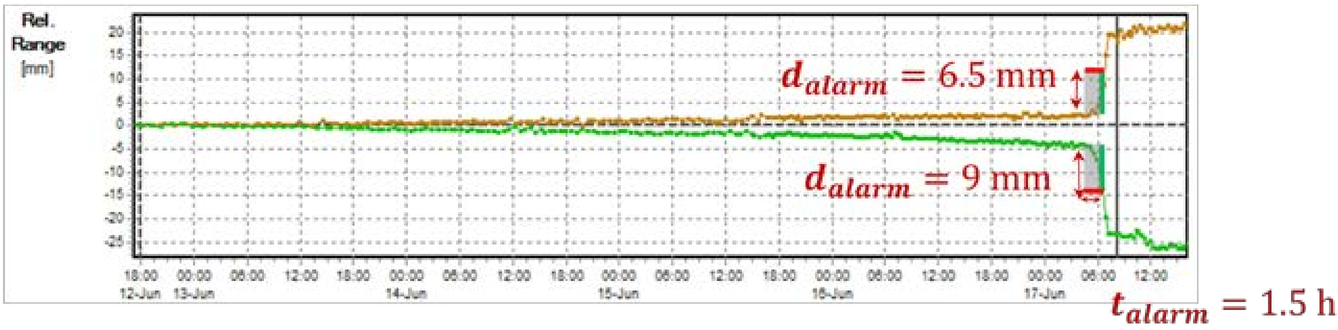


(b)

Figure 11 Scaled displacement benchmark profile of past failure, showing failure and alarm values (long time to collapse)



(a)



(b)

Figure 12 Scaled displacement benchmark profile of past failure, showing failure and alarm values (short time to collapse)

past failure captured by the radar provided geotechnical engineers onsite with the scaled slope movement profile (Figure 9).

During the progressive stage of the failure the displacement was roughly 33 mm and the time to failure roughly 9.5 hours. Based on the mine's operational requirements it was decided that a critical alarm of two hours prior to collapse was sufficient. This requirement resulted in an alarm time of 7.5 hours and a corresponding alarm displacement of 10 mm (Figure 9).

These values result in an average alarm velocity, v_{alarm} = 1.33 mm/h. Figure 10(a) shows the average velocity calculated using the designed time window, with two solid red lines indicating the ±1.33 mm/h limits. The velocity increases beyond the desired thresholds at approximately two hours prior to the collapse, as designed.

Figure 10(b) shows the result if a shorter time window is used for calculating the average velocity. The alarm time is earlier, but the chance of false alarms increases if there is any small disturbance to the signal.

Example 2: long time to collapse

An MSR300 system was deployed to monitor a wall where engineers were concerned about failures that develop over extended periods of time. An analysis of a past failure, captured by the radar, provided geotechnical engineers onsite with the scaled slope movement profile (Figure 11).

The displacement recorded during the progressive stage of the displacement was roughly 690 mm and the time to failure was roughly 48 hours. Based on the mine's operational requirements it was decided that a critical alarm of 12 hours prior to collapse was sufficient. With this requirement in mind, an alarm time of 36 hours was used, giving an alarm displacement of 260 mm (Figure 11). This resulted in an average alarm velocity, v_{alarm} = 7.2 mm/h.

Example 3: short time to collapse

In this example, a pit highwall was monitored by an MSR300 and a very rapid failure occurred. The displacement profile is shown in Figure 12(a). This failure is interesting in that there was some rotation, with the upper part of the failure moving away from the radar, while the lower part moved towards the radar. This is indicated by the two trend lines. Both exhibited a similar amount of displacement – approximately 18 mm in two hours.

With such a rapid failure, the warning cannot be triggered very long in advance. In this case, the design was done for just a 30 minute warning (Figure 12). The limit for the average velocity towards the radar is 6 mm/h and the limit for the average velocity away from the radar is 4.33 mm/h. The difference in the two is due to the slightly different displacement curves in the two directions.

In the earlier examples, the average velocity limit away from the radar was not discussed. Generally, it is set equal to the approach velocity, unless there is a benchmark profile that specifically suggests otherwise.

Conclusion

Continuous slope monitoring plays a major role in ensuring that high safety and productivity standards are achieved on surface mines around the globe. Advance warning mechanisms are an essential part of continuous slope monitoring programmes, but are only effective if the correct alarm thresholds are selected.

Understanding the concepts of slope failure profiles and pit geometry scaling as well as the associated impact on back analysis and warning times, will aid geotechnical engineers in selecting applicable alarming thresholds. The guidelines presented in this article therefore serve as a framework that can continuously be referred to.

As powerful a tool as the slope monitoring radar is, it is critical to understand its limitations and how to correctly interpret its data. The use of additional methods to monitor and analyse the slope movement is well advised to better mitigate the risks associated with slope failures.

Please click here for article references.

Alex Pienaar,
Reutech Mining, South Africa

Slope Stability 2013 Symposium report

by Maddie Adams, Australian Centre for Geomechanics

The International Symposium on Slope Stability in Open Pit Mining and Civil Engineering (Slope Stability 2013) attracted more than 380 mining professionals from over 100 companies. Throughout the three day symposium, open pit mining and civil engineering practitioners, consultants, researchers and suppliers worldwide were provided with an opportunity to exchange views on best practice and state-of-the-art slope technologies in relation to pit slope investigations, design, implementation and performance monitoring.

The Australian Centre for Geomechanics was delighted to host this symposium in September 2013, in Brisbane, following on from past symposia in the biennial symposium series held in Vancouver, Canada, 2011; Santiago, Chile, 2009; Perth, Australia, 2007; and Cape Town, South Africa, 2006.

Slope Stability 2013 attracted significant international interest, with attendees from countries including: Australia, Austria, Botswana, Brazil, Canada, Chile, Finland, France, Germany, Ghana, India, Indonesia, Italy, Laos, New Zealand, Pakistan, Papua New Guinea, Peru, Republic of Korea, Saudi Arabia, South Africa, Spain, Sweden, Switzerland, Thailand, United Kingdom, United States of America and Zambia.

The Slope Stablity symposium featured 68 presentations and two poster sessions, featuring 33 posters. The symposium began with opening speaker Mr John McGagh, Head of Rio Tinto's Innovation team, discussing the impact of slope failure, challenges of slope design and the future of slope stability.

In his address, John briefly discussed the impact of geotechnical engineering to Rio Tinto. The Bingham Canyon Mine failure in April 2013 demonstrated to Rio Tinto the benefits of excellent slope monitoring systems and a diligent technical and operations team.

He emphasised that for any slope design, the understanding of site structural geology, material properties of the materials forming the slopes and the impacts of surface and groundwater were critical. As the first two are not within engineering control, it is vitally important to understand their impact on slope performance. Without this knowledge it is not possible to understand the risks to the mining business.

Industry needs to work smarter but not choke itself on data without the ability to synthesise, analyse and understand what the data is saying. More data is insufficient to manage the risks.

John went on to explain how Rio Tinto was addressing such issues technically, and through the use of key nodes called centres of excellence. Such nodes can then operate 24/7, assisting operations around the world.

Following this presentation, keynote speaker Professor Derek Martin, University of Alberta, presented his paper on 'Pit slopes in weathered and weak rocks'. Professor Martin examined the application of sampling techniques and laboratory testing methods, prominent failure processes, soft iron ores and the Águas Claras failure and mudrocks, Cobre Las Cruces open pit.

Dr Loren Lorig, Itasca Consulting Group Inc., then presented a keynote paper written by Dr Jonny Sjöberg, Itasca Consultants AB, on 'Numerical analysis, slope design and in situ stress'. The presentation focused on the history of modelling, issues and examples of stresses and the vision for the future.

A panel discussion was held at the end of day one, chaired by Martyn Robotham, Rio Tinto. The panel comprised John Read, Geoff Beale, Bob Sharon, Loren Lorig, Phil Carvill and Pete Stacey. Each panel member gave some ideas as to where they saw challenges for the geotechnical fraternity. This helped develop discussions with the audience. Unfortunately the meeting had to finish after 1.5 hours. Why unfortunate? The audience was just starting to warm up.

Day two of the symposium began with Geoff Beale, Schlumberger Water Services, presenting his keynote paper, 'Water and slope stability – the application of a new science' . Geoff discussed topics including the effect of water on slope stability, controls on pore pressure, application of depressurisation methods, implementation of programmes and considerations for eventual mine closure.

Professor Tim Sullivan, Pells Sullivan Meynink and The University of New South Wales, continued the session with his keynote address, 'Global slope performance index', which provided an overview of a simple empirical system,

Slope Stability 2013 opening speaker, John McGagh, Rio Tinto

Delegates had the opportunity to visit exhibitor booths throughout the symposium the global slope performance index, for assessing the expected performance and risks associated with excavated slopes.

Dr John Read, CSIRO Earth Sciences and Resource Engineering, was the first keynote presentation on the final day of the symposium, with his paper, 'Data gathering, interpretation, reliability and geotechnical models'. Dr Read discussed open pit slope design and the process involved in constructing the geotechnical model.

Following Dr Read, Dr John Simmons, Sherwood Geotechnical and Research Services, presented his keynote paper, 'Excavation control, management of blast damage, and quality control' with discussion on design and implementation, design documentation and ownership and excavation control hazards.

The ACG was grateful to have the support of Slope Stability 2013 sponsors, namely the industry co-host, Rio Tinto; principal sponsor, Reutech Mining; and major sponsors: 3D Laser Mapping Ltd, GroundProbe Pty Ltd, IDS Australasia Pty Ltd, and Specialised Geo Pty Ltd. Delegates had opportunities throughout the symposium to visit the trade exhibition areas, which included 30 booths, to network and acquire information from the event exhibitors.

The symposium was generally considered to be most rewarding and inclusive, and the atmosphere was friendly with many opportunities for people to engage.

There were a number of events associated with the symposium which took place before and after the event. The Instrumentation and Slope Monitoring Workshop took place prior to the symposium, on 23 September, with programme content including developing and understanding slope failure mechanisms from monitoring and inground monitoring, TDR, extensometers, inclinometers, geophones and piezometers.

This workshop was followed by the Slope Analysis and Design in Anisotropic Materials Workshop which attracted over 90 delegates. Topics included coal and iron ore mines case studies, impact of structural anisotropy on material parameters, and numerical and limit equilibrium methods.

The Business Case for Risk-based Slope Stability Design Workshop took place after the symposium. This workshop's programme included presentations from South Africa and Australia and was attended by over 50 delegates. The workshop included an introduction to statistics – different methods and the importance of being quantitative versus qualitative, and case studies. The collaborating organisation for this event was SRK Consulting and Dr Oskar Steffen, SRK Consulting, opened the workshop with a presentation on risk components, the management decision process, geotechnical risk, drill spacing and other areas relating to developments in open pit planning.

Following Slope Stability 2013, the ACG, in conjunction with Stanwell Corporation Ltd, gave the delegates the opportunity to visit Meandu Mine, which provides thermal coal to the adjacent Tarong Power Station, and the Blackbutt Range Slope Stabilisation, where a summary presentation on the site conditions, scope, and nature of the stabilisation works was provided. The ACG gratefully thanks Dr John Simmons and Stanwell Corporation Ltd for facilitating this field trip.

The 2015 International Symposium on Slope Stability in Open Pit Mining and Civil Engineering will be held in Cape Town, South Africa in 2015.

Introducing the Bowen Basin Open Cut Geotechnical Society

by Alison McQuillan, Anglo American, and Nicole Tucker, BHP Billiton Mitsubishi Alliance

The Bowen Basin Open Cut Geotechnical Society (BBOCGS) was founded in 2012 following the success of the Bowen Basin Underground Geotechnical Society (BBUGS). The two groups share the same ethos in which both societies were formed with the following aims and objectives:

- To advance the knowledge of all factors affecting the design, construction and operation of open cut coal mines in the Bowen Basin, including technical aspects, health and safety, and sustainable development.
- To promote and review research and development of relevant projects.
- To communicate and link with other local and international like associations concerned with the application, planning and practice of geotechnical aspects of open cut coal mining.
- To review the adequacy of education and training of technologists associated with the science, engineering and practice of the geotechnical aspects of open cut coal mining.

BBOCGS aims to advance the knowledge of all factors affecting the design, construction and operation of open cut coal mines in the Bowen Basin

To date, BBOCGS has over 80 members ranging from site-based geotechs, consultants, suppliers, and students to those with a general interest in coal geotechnics. Members are invited to attend and present at quarterly meetings held either at Bowen Basin mines or in Brisbane. The nature of site-based meetings generally entails three to four technical presentations followed by a site tour of the host mine.

The calibre of presentations to date has been outstanding, with presenters from operating mines, consultants, and suppliers, all generously giving up their time to share their knowledge. Highlights have included presentations from Dr John Simmons, Sherwood Geotechnical and Research Services, detailing the importance to back-analyse failures, using a case study of a Rolleston highwall failure; Clare Murray, Board of Professional Engineers Queensland, outlining the responsibilities of geotechnical engineers in Queensland under the Queensland Professional Engineers Act; and John Latilla, AMC Consultants Pty Ltd, providing an insight into coal mining in Mongolia where seams dip up to 17° and temperatures plummet to as low as -40°C.

Members were also provided comprehensive overviews of the geotechnical constraints of Peabody's Moorvale Mine, BMA's Blackwater Mine and Anglo American's Foxleigh Mine by site representatives prior to observing conditions for themselves during site tours. Technical meetings also provide a forum for industry suppliers to present their latest technological advances. To date, IDS and GroundProbe respectively presented the range and capability of their products to assist geotechnical engineers with

BBOCGS members at the Moorvale technical meeting, December 2012

Thirteen heads are better than one

monitoring ground movement. In line with the society's ethos to advance knowledge and research, Dr Marc Elmouttie, CSIRO Earth Science and Resource Engineering, and Dr David Williams, University of Queensland, presented their research into streamlining the Geotech Hazard Mapping Procedure and the Geotechnical Stability of High Coal Mine Spoil Piles at a joint AGS/BBOCGS meeting.

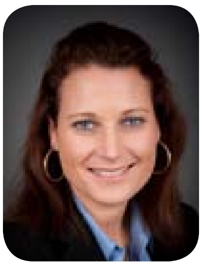
BBOCGS is a not-for-profit organisation. We thank our sponsors Peabody Energy, IDS Australasia, Xstract, Mining One, Trilab, and major sponsors: GroundProbe, Maptek and Specialised Geo, for supporting our society and ensuring its success well into the future. BBOCGS would also not have evolved from more than an idea without the support and input of founding committee members Dan Payne (BMA and co-founder of BBUGS), John Simmons (SGRS), Chris Strawson (Xstract), Chris Hanson (Adani), John Latilla (AMC Consultants), Nicole Tucker (BMA), Ian Kelso (GHD), Gavin Lowing (Peabody), Alex Hossack (RTCA) and Ismet Canbulat (Anglo American and co-founder of BBUGS).

Whatever the coal price, we believe there will always be pressure for mines to reduce costs by increasing batter angles, reducing bench widths, shortening dump circuits, etc. The risks associated with these design changes have to be proactively managed by geotechnical engineers to ensure that the safety of mine personnel is not compromised.

It is at forums like these BBOCGS meetings that like-minded geotechnical engineers can collaborate, sharing experiences and enthusiasm for the geotech field, trade war stories, and push the boundaries of knowledge and problem-solving whilst simultaneously moulding the younger engineers of tomorrow – all in the comfort of friends.

For more information, visit www.bbocgs.org.au.

Alison McQuillan
Anglo American



Nicole Tucker
BHP Billiton Mitsubishi Alliance

Mass mining projects come of age

observes Dr Raul Castro, University of Chile, Chile

CAVING 2014
3RD INTERNATIONAL SYMPOSIUM ON BLOCK AND SUBLEVEL CAVING

5–6 June 2014 | Santiago | Chile

Introduction

To be profitable, the extraction of large amounts of valuable minerals from the ground requires the use of efficient mine technologies. Equally important is the sustainability of the operations and high safety standards. Underground mining methods produce less impact on the environment than open pit practices. Caving methods are also the natural replacement of open pit operations as the ore reserves near the surface become depleted. Mine caving offers the lowest cost and highest production method, provided that it is correctly selected and implemented for the orebody's geotechnical and geological conditions.

Sublevel caving, block/panel caving and super caves

There are many caving method variants. The methods could be classified as either sublevel caving or block caving. In the first instance, the overlying waste is caved as the ore is extracted by the drill and blast technique. While in the second, the ore, and sometimes the overlying waste, are caved. Block caving could also be classified according to undercutting and extraction method, and the position of the production level into block, panel caving and inclined caving. The methods could also be classified on the use of explosives for preconditioning and other blasting methods, and by the distance of the cave front from the extraction level (pre-undercut and post-undercut).

Finally, caving operations could also be classified according to the scale of operations. Small scale caving operations may be defined as a production rate less than 30,000 t per day; medium size operations, from 30,000 to 60,000 t per day; and large scale operations from 60,000 t upwards. It must be emphasised that the ore could come from different mines in a complex, i.e. Codelco Chile's El Teniente and Freeport Indonesia. The classification according to the size of the operation is relevant not only in terms of production, but also by the amount of capital, development, equipment and personnel that are required to reach high productivity. Future mines are planned to extract more than 140,000 t per day. The question is, what is the production limit for the 'super caves'?

Research and development

Australia, Canada, Chile, China, Indonesia, Mongolia, South Africa, Sweden, and the USA all have cave mines. Currently, worldwide mine caving research is being pursued within mining companies, universities and research bodies. This research examines some of the technical challenges that the block caving industry faces, including the:

- Large amount of development required in a short period of time.
- Scarcity of highly qualified people.
- Need for high productivity material handling systems.
- Understanding and tracking of the cave and the flow.
- Mud rush, and rockburst prediction and control, especially when the mud has a high grade content.
- Mine costs and dilution control.
- High stress conditions.
- Ventilation and high temperature conditions.
- Stability of the mine infrastructure.

Block caving in the future

Many operations are considering, or have decided, to use block caving as their preferred mining method. Presently, about 400,000 t per day are extracted by caving methods. It is estimated that this figure will increase to a rate of 1 M t per day by 2018. Production rates would also increase. This will present new and exciting challenges and opportunities for the mining industry.

Caving 2014

In June 2014, the Third International Symposium on Block and Sublevel Caving will be held in Santiago, Chile. Chile has three large block cave operations: El Teniente, Andina and Salvador, with an annual production of 74 M t. Codelco, the largest copper producer, is developing two new block caving mines at El Teniente and Chuquicamata, that will produce additional resources for Chile's future.

We look forward to welcoming delegates to Santiago de Chile, a city near the sea and the mountains and home to some of the world's largest mines.

Visit www.caving2014.com.

Block and sublevel caving have been applied in the successful extraction of large orebodies for more than a century

Dr Raul Castro
University of Chile, Chile

Mine backfill – a cost centre or an optimisation opportunity?

by Tony Grice, AMC Consultants Pty Ltd, Australia

Twenty years ago, geomechanics professionals were only found on the largest and more technically advanced mine sites. Nowadays, all mines have a ground management plan and a small but dedicated geomechanics team to assist the mine manager. By contrast, mine backfill is utilised on many underground mines and, although it accounts for up to 30% of the mine operating cost budget, it is still a rarity to find a single person directly responsible for backfill.

Now, as commodity prices ease off from the recent boom, costs and productivity are receiving laser-like attention. Backfill is clearly an area that warrants management focus. This article discusses bringing the backfill system up to standard, and how to identify cost savings and productivity improvements that will add value to the mine and increase production flexibility.

Backfill is typically made from waste rock or dewatered tailings residues and is often mixed with cement to achieve moderate strengths. It can be delivered to stopes in several ways: either by truck, by pumping and/or gravity or as dense slurry or paste through boreholes and pipelines.

Backfill serves a number of functions in underground mines. Filling of open stope voids maintains stability of the adjacent working areas and reduces risk of local or regional ground failure. If cementitious binders are added, the blasting of adjacent pillars enables higher recovery of ore reserves by exposing the cured fill.

In benching and open stoping mining methods, stable vertical fill exposures can be created as the pillars between stopes are removed, or as the mining front retreats back to the access point. In underhand mining methods such as drift and fill or up-hole retreat, the cured fill can form a homogenous stable roof that enables safe ore extraction. In overhand mining methods such as cut-and-fill, benching or open stoping, the fill can also provide a stable working platform for people and equipment. Each of those applications requires specifically designed strength and filling requirements.

Backfill offers many environmental benefits. Paste backfill, tailings dewatered to a yoghurt or toothpaste-like consistency, can enable up to 50% of the total tailings produced by an underground mine to be placed back underground. In some mines, acid-generating waste can be encapsulated in the backfill, sealing it into virtually impermeable cells. In most mines, some development waste rock is disposed of into stoping voids. Each of these activities reduces the environmental footprint of the mine and assists with final site rehabilitation.

The design, operation and management of backfill systems involves a number of technical disciplines that often cross several management boundaries on a mine site. The processing department on surface is generally in charge of production and delivery of the backfill, and is responsible for the quality, operating costs and process monitoring. The underground geotechnical department specifies the strength of the fill, the cement dosing and the fill recipes. They also review the quality control (QC) data and analyse fill performance to improve the fill recipes. The mine planning department develops the schedules and specifies where and how much fill is placed.

On most mines, up to three departments are responsible for the backfill systems. The processing department carries all the production costs but the geotechnical department is responsible for the cement dosing rates. The mine planning and operating departments plan and place the fill. The net result tends to be a lack of focus on meeting the mining requirements and maximising quality. Excessive costs and low productivity are often the results of these multiple responsibilities.

So what's to be done?

There are two actions that all mines can take to initiate optimisation, improve productivity and reduce costs in the backfill system in a rational manner.

The first action is to assign one person to have overall responsibility for the backfill system. The key performance indicators (KPIs) are both the quantity and quality of the fill placed underground. Most commonly, this is a superintendent level management position who is directly responsible for all underground backfill activities. Where management roles are split on a mine site by surface and underground, that person also has a leadership role to assist the other managers. The role is a coordinating function between processing, mine planning and operations,

Paste fill on a conveyor

and geomechanics. At most mine sites, however, backfill is a technically complex system and an engineer is required for this role.

The second action is to initiate a backfill system audit. Ideally this should be a review of the current backfill system against the backfill management plan. However, many mines do not have such a document and the audit then serves to provide a systematic review that will initiate this plan.

Cured cemented hydraulic fill exposed underground

The audit should start with a review of mining needs and work back through the system via placement, delivery and processing to the supply of the backfill components. Invariably, the backfill needs of the mine will have changed since the original design and commissioning of the backfill system. Increases in production rates, changes in mining area locations and mining methods will have occurred. This is now an opportunity to fine-tune each component to ensure that the backfill types and recipes are all relevant to the current operations.

Productivity improvements can be found in focusing on activities that will shorten stope fill cycle times. Longer fill runs, higher slurry density, reduced drainage and improved barricade construction will all contribute to faster stope turnaround. Installation or recommissioning of line pressure sensors and video links will enable operators to identify and avoid impending line blockages, and the long and costly delays that accompany them.

Cost reductions can be found by addressing the fill quality and the components, particularly binders. Is the mine using the correct binder type or grade for your strength curing targets? When was the last review of the tailings or rock characteristics and how do the QC test results compare with the design specifications? Have there been recent changes to recipes in response to unexpected or excessive fill dilution that have resulted in increases to binder or additive costs?

In an efficient mining system with backfill, the backfill costs being expended will actively contribute to improved underground safety, efficient ore recovery, fast stope cycle times and flexible production options. Focusing management attention on this large proportion of the mining costs will ensure that productivity opportunities are identified and implemented, and that costs are not being incurred unnecessarily through poor quality fill and misaligned KPIs. The improvements from the single point of responsibility and technical audit will pay for itself many times over and will have continuing long-term benefits to the mine.

By now, all mines with backfill should have a backfill management plan. There should be one person assigned to manage the backfill system and to ensure compliance with the plan. The plan will require updating to maintain currency and be reviewed on an annual basis. Backfill is an integral part of the mining operation and a well-managed backfill system will provide many opportunities to optimise production by increasing productivity and reducing costs.

Tight filling in progress in a stope

Tony Grice
AMC Consultants Pty Ltd
Mine Fill 2014 Symposium Chair

Paste 2014 – a new world of tailings technology

by Jack Caldwell, Robertson GeoConsultants Inc., Canada

A paste and thickened tailings seminar has been held every year since 1999. Paste 2013 was sold out, with many turned away. With Vancouver as the host city in June 2014, delegate numbers promise to be even higher.

The issues of thickened, paste, filter-pressed, and polymer-amended tailings are not yet solved. Paste 2014 will be an opportunity to advance our knowledge of these topics and work towards solutions.

Paste 2014 will focus on paste for mine backfill. It is responsible mining to put as much tailings as possible back underground. And I know this works, for I have been at three mines this year that use paste backfill. A mine in the far north of Canada is experimenting with centrifuges to get good material to fill underground stopes. A Vancouver Island mine has been successfully placing paste tailings underground for so long that it has become routine. A mine in Guatemala has just started production, planning for half their tailings to go underground, and half to go to the dry-stack.

Paste 2014 will also have sessions focusing on dry-stack tailings – another responsible way to mine. Andy Robertson's recommendation in the mid 1980s that Greens Creek employ filter-pressed tailings was initially met with skepticism. Eventually they took his advice and today the mine predicts at least another 40 years of dry-stack tailings disposal. Additionally, the latest dry-stack facility started in October 2013 at Escobal, in Guatemala.

The most exciting aspect of Paste 2014 is a series of papers on polymer-amended tailings from the oil sands industry. Paste 2013 included fine papers on the plans of the oil sands industry and their quest to better manage tailings. At Paste 2014 we are promised a series of papers on the theory and techniques being evaluated in the laboratory, and in the field.

In South Africa I saw how the best slimes dams are operated. One worker had total control of the slimes deposition; he decided which spigot to open next, which walls to raise, and what penstocks to lower to reduce the pool of supernatant water. At Paste 2014 we may get a paper from Fraser Alexander on the amazing success of this system – a system that the oil sands operations are replicating, as their polymer-amended operations advance to success. Currently, flume testing of tailings, to which polymers from the leading suppliers are added, are in progress. Beach angles are being measured and predicted. Dewatering and strength gain rates are being measured and predicted. Designs for full scale polymer-amended tailings facilities are being formulated.

There will be many more papers at Paste 2014 from the experts on beach formation. From my research I concluded that 0.3 to 1.0% is ideal. Only when years of deposition prove otherwise should you revise your designs. I consult to a mine where the measured slope varies from 0.3 to 3.0%. After much investigation I concluded that the slope variability is a function of distance from the mill. As the mill manager reported,

"When we are pumping to close spigots, we reduce the amount of water we add to the tailings. When we are pumping to distant spigots, we increase the water we add to the tailings. It is all about keeping the pumps happy. Don't overload them, otherwise they cut out. So maybe your beach slopes are simply a result of the water we add to make the pumps happy."

Dare we hope for a paper on this unexplored practice?

A question that I would like to see discussed at Paste 2014 is this: how do paste and thickened tailings perform in an earthquake? In 2010 there was a major earthquake in Chile. The report tells of boils in the tailings, significant settlement of tailings, and failure of dykes some days after the earthquake, when excess pore pressures had time to migrate to impermeable layers to form a layer of water along which failure occurred. I have seen no paper that honestly explores this. I believe that the challenge to the tailings industry in earthquake-prone places is this: predict how the thickened and paste tailings will respond to big earthquakes, and tell us how to design for such events.

An exciting new world of tailings technology is being explored. Paste 2014 promises to be a venue to explore these new ideas.

Over and above the invaluable technical content, Vancouver is a wonderful venue for a seminar. For a start, it is home to an incredible concentration of mining companies and consultants. There are over 700 juniors, more of them successful than cynical reporters would have us believe. Teck and Goldcorp are the majors, and their offices are a stone's throw from the seminar venue. Also here are AMEC and Tetra Tech. SRK has been here for 35 years; Knight Piésold for 38. This seminar is a great opportunity for the manufacturer, supplier, or installer of mining materials to meet these consultants.

Jack Caldwell
Robertson GeoConsultants Inc., Canada

Inclinometers – the good, the bad and the future

by Mark Fowler, Pells Sullivan Meynink

Introduction

Inclinometers have been used in geotechnical monitoring since the late 1950s. They remain a vital instrument for measuring subsurface deformation, which is critical for a three-dimensional appreciation of failure path and an accurate assessment of the failure mechanism. Surprisingly, the technology has remained largely unchanged since the 1960s, with improvements confined to components and data acquisition equipment.

This slow growth is not reflected across the sector. Rapid changes are occurring in surficial measurements based on remote sensing techniques. Surficial monitoring has almost reached a 'data nirvana', where the quality and quantity of surficial data is so high that almost the only complaint that a practitioner can make is 'what do I do with all this data?'.

However, below the surface it is a different situation, as inclinometer development plateaued in the 1970s. This is not because it has reached perfection, far from it. The most widely employed technology of the traversing probe has significant drawbacks, including the high cost of the instrument, inclinometer surveys are time consuming, laborious, prone to errors and they are only capable of tolerating small movements.

This article provides a brief description of the history of inclinometer development, the author's experience in using the technology, overviews some promising developments, and poses the question 'what's next?'.

History

Inclinometers first appeared in the 1950s when researchers at Harvard University developed a cumbersome precursor to the traversing inclinometer. The first grooved inclinometer casing was installed in California in 1954. By 1959 a brass waterproof biaxial probe was being produced and installations had reached 150 m.

The Slope Indicator Company, named after its inclinometer, has been central to the commercialisation of the technology. By 1962 the Slope Indicator Company had developed a ruggedised traversing probe which remained in production until the early 1980s. In 1969 they developed a small diameter probe called Digitilt® using biaxial force balanced accelerometers borrowed from missile guidance systems.

Figure 1 shows a typical traversing inclinometer. The accompanying graph illustrates how displacement is measured by comparing the distance between successive readings over the true vertical or initial profile.

In-place inclinometers (IPI) were developed in the 1980s. They comprise sensors permanently located within the casing. However, due to cost considerations, wide spacing of sensors were usually employed resulting in lower resolution data.

Improvements to the traversing probe have largely been confined to refinements of the existing technology rather than any significant step changes. Improvements in sensors have increased the accuracy and reduced some errors. The latest sensors are solid state micro electro mechanical sensors (MEMS). Digital signalling enables thinner and more robust cables. Wireless connections to tablets or PDAs have replaced the more cumbersome and less user-friendly analogue read out units. Whilst automated winches have been around since the 1990s, traversing surveys are generally still undertaken manually.

Common inclinometer errors

The Sixth International Symposium on Field Measurements in Geomechanics (FMGM 2003) was held in Norway in 2003. At FMGM 2003, Erik Mikkelsen presented a paper titled, 'Advances in inclinometer data analysis'. This provided a comprehensive assessment of inclinometer errors and corrections, and suggested that 40 years since the invention of inclinometers industry still needed education about the errors and the potential for misleading results.

Today it is still common to view inclinometer data where errors are not corrected, resulting in data that is at best, noisy, and at worst, indicates deformation that is false.

Bias shift

Bias shift is the most common systematic error experienced in inclinometer measurements. When the data is presented on a cumulative displacement plot, the error appears as a rotation of the entire installation about its base. The magnitude and direction of the rotation is random. This means that in some cases the sense of displacement between surveys appears nonsensical.

The error is attributed to a small uncontrolled adjustment of the tilt sensor in the probe, between or during traverses. It can be corrected, providing the installation extends into stable ground which can be reliably assumed not to have moved. The data needs to be manually adjusted until the displacements in the stable zone are reduced to zero.

Such corrections are more difficult to apply where the installation does not extend into stable ground, for example, if measurements are taken adjacent to large scale caving.

Depth error

Depth errors occur when the probe is lowered to a different position in the casing. The recommended positional accuracy is in the order of 5 mm. Symptoms of depth errors are characterised by displacements that are exaggerated and offset vertically. Straight casing is less susceptible to this error than wavy or undulose casing. The causes of depth error include:

- A change in the cable reference position.
- Changes in cable length caused by substitution of cables or stretched/ repaired cables.
- Settlement of casing.

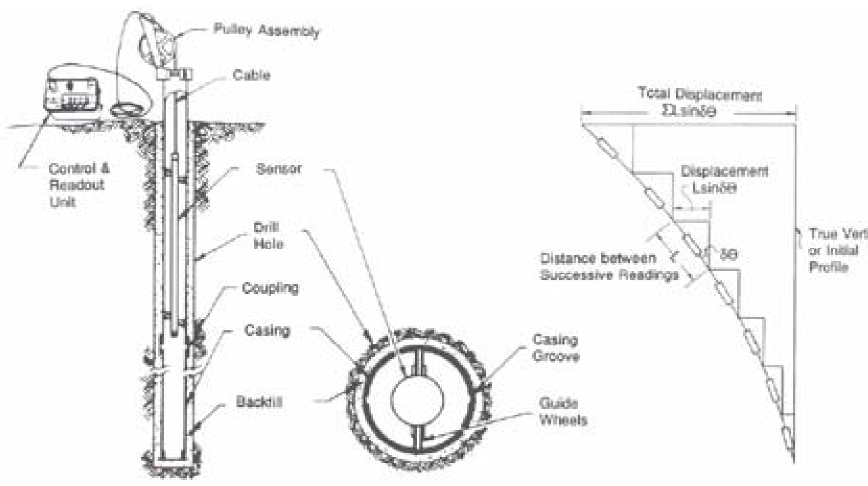


Figure 1 Traversing probe inclinometer

In most cases, the depth error can be remediated by interpolating the value which would have been read if the probe was at the correct depth. This is a one off exercise if the offset is known and, if not, a lengthy process of trial and error. Again, the correction is a manual process.

Bias and depth errors are illustrated in Figure 2. The cumulative frequency plot on the left is the raw data and presents both bias shift and depth errors. The plot on the right is corrected for both effects.

Discussion

Mikkelsen's identification of the errors and the solutions developed are very important to the continuing use of inclinometers. Further, it was understandable that the market accepted these major weaknesses in the technology, given that there was no viable, widely available alternative. Mikkelsen makes a striking statement that "the potential for error is greater than the potential for real movement".

Ten years down the track and with huge advances in other monitoring areas it begs the question, why hasn't this deficiency been addressed?

Experience

Corporate inclinometer experience at Pells Sullivan Meynink (PSM) is wide and varied. As is the nature of some projects, unusual and unorthodox adaptions have been required to meet the technical objectives of the project. The following sections demonstrate both the flexibility and the limitations of traversing probe surveys and describe some conventional and unconventional applications.

Civil tunnelling installations

The Epping to Chatswood rail tunnel, Sydney, Australia, involved the excavation of shallow cover wide span tunnel station caverns followed by the sinking of adjacent shafts. Inclinometers were successfully used to monitor stress relief and measure the magnitude of horizontal shearing on bedding planes to assess its impact on ground support. Monitoring was undertaken using a MEMS traversing probe. Bias shift correction, supported by surface survey of the inclinometer collar, was essential to provide useful results and correct the errors mentioned previously.

Figure 3 presents inclinometer survey results at one point in the tunnel. It shows the displacement occurring toward the cavern on the right as it was excavated first and then toward the shaft on the left. Displacements occur over 0.5 to 1 m reflecting deformations along horizontal bedding discontinuities.

Detailed survey

At some stages during the Epping–Chatswood project, quantification of the amount of bedding shearing was required to assess the structural impact on rockbolt capacity. Conventional inclinometer surveys identified significant movements. However, given the half metre spacing of probe readings, it was not possible to assess whether the displacement occurred at one discontinuity or, alternatively, across numerous discontinuity, the latter being more favourable in terms of impacts on bolts.

One of the versatilities of the traversing probe is that more detailed surveys can be carried out if required. In this instance, survey increments were decreased to 50 mm intervals (Figure 4). This significantly increased resolution of the survey and improved quantification of the magnitude of the shear.

In-place vertical

Experience with in-place vertical inclinometers has been positive from the perspective that they have performed exactly as promised. The key drawbacks are the inflexibility and general coarseness of results, although this has improved over time. The issues are:

- Sensors are expensive.
- The number of sensors that can be installed in a single hole may be limited.
- The spacing of sensors is often selected to be wider than readings obtained by traversing probes, or only concentrated around known movement zones with more widely spaced sensors outside of this. This often results in reliable but locally lower fidelity results.

In-place horizontal

A recently completed project in Sydney involved the use of a horizontal in-place inclinometer. The project comprised 40 tilt sensors connected together. Significant problems were experienced with the installation, such that the results were of little use. The cause of the problems has not been fully resolved, though may have been caused by vibration from construction plant, or more likely, due to thermal effects as the installation was shallow and affected by surface temperatures.

Large shear adaptations

One of the limitations of inclinometer casing is magnitude of shearing. The tolerance is controlled by the dimension between the internal casing diameter and the passing profile of the probe.

For a longwall mining project, there was a need to monitor deformation across a known fault beyond the tolerance of conventional casing. The solution was to install the casing in a large diameter hole with an interval of very soft grout at the location of the expected movement. The idea was that the soft grout would allow the casing to bend rather than shear and thus achieve a greater range of

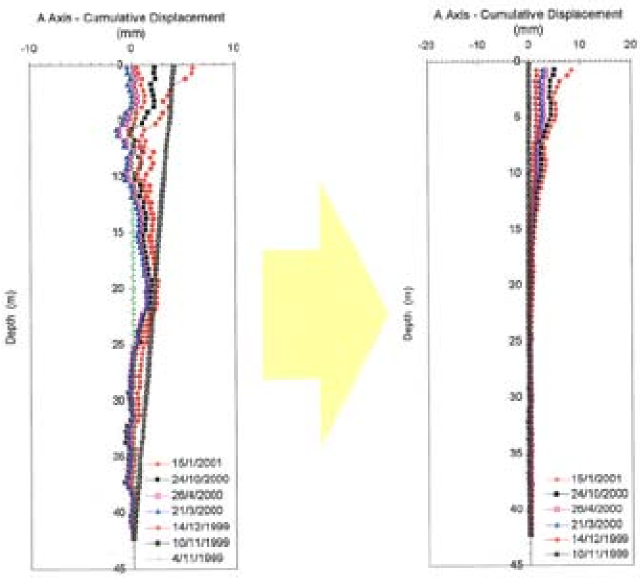


Figure 2 Combined bias shift and depth correction

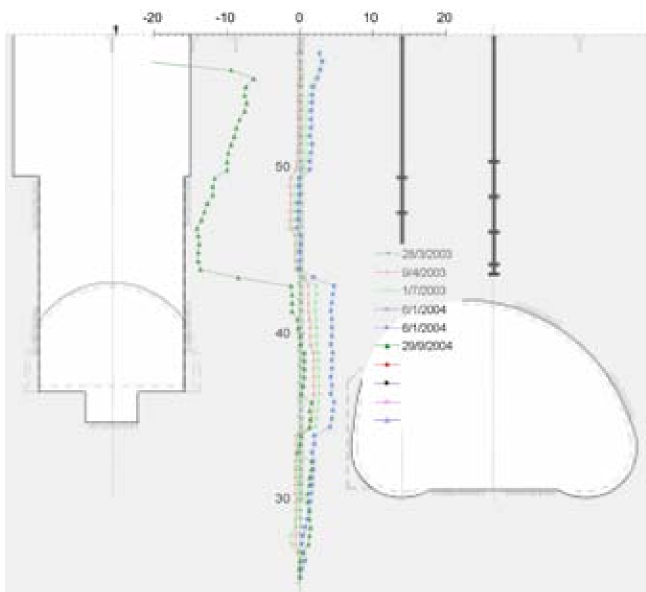


Figure 3 Inclinometer monitoring of the station cavern and shaft at the Epping to Chatswood rail tunnel

150 mm movement. This approach was a partial success in that an improved range was achieved, but it was not as much as anticipated. The installation is shown in Figure 5.

Very high temperature installations – Lihir

Monitoring of slopes with inclinometers at Lihir Gold Mine, PNG, is a challenge that stretches the use of the technology. The site is characterised by altered weak rocks in an active geothermal environment. In some instances the weak rocks result in partially stable slopes that creep to significant depths (Figure 6).

Notwithstanding the geotechnical challenges, high temperatures, often 150°C and sometimes in excess of 200°C, present difficult installation and survey challenges. Plastic casing melts, portland cement flash sets, and in some instances, steam pressures result in geysers. Site staff have developed procedures to accommodate the conditions including the use of aluminium casing, high temperature cements and quenching techniques where cool water is circulated in the hole. It is acknowledged that temperature change may affect surveys and so surficial monitoring is used in conjunction to confirm readings. The resulting inclinometer data is of surprisingly good quality, given the challenging conditions for installation and monitoring.

Recovery of lost probes

One of the downsides of traversing probes is that occasionally they become lodged in the casing. Probes can become trapped for many reasons, including discrete shears, casing separations and the unintended intersection of drill holes from adjacent underground workings. Primarily probes snag on the spring loaded wheel assembly.

Recovery is successfully executed using a metal sheath designed to locate the wheel assembly, fold it away and envelope the probe for retrieval to the surface. Such a device is shown in Figure 7, along with a cutting tool used to cut away a flap of casing that was blocking the hole.

Discussion

The author's experience has been that the traversing probe and grooved casing system is a useful tool for subsurface monitoring. Experience is that the errors described by Mikkelsen occur universally and need to be accounted for.

The traversing probe concept has unexpected benefits and weaknesses, for example, detailed surveys and probes becoming stuck downhole.

Alternative technologies

Alternatives are working their way into common use which address some of the down sides of the incumbent technologies. There are two in-place devices that are on the market; the Shape Accel Arrays (SAA), and the differential multi-parametric system (DMS).

Shape Accel Arrays

SAA comprises a series of 0.3 or 0.5 m long segments joined with flexible 'knuckles'. Each segment contains a triaxial, micro electro mechanical system accelerometers (MEMS) that measures tilt relative to the gravitational field. This permits the bend angles between each segment to be calculated and thus the shape of the SAA can also be determined.

DMS differential multi-parametric system

This is the big brother of the SAA. The differential multi-parametric system has the ability to house several monitoring sensors within a flexible and heavily robust jointed casing that is protected from the outside environment (Figure 8). Due to the overall strength of the jointed casing, installation is a more substantial exercise than standard casing.

The system maintains a preferred alignment within a borehole without the need for grooved inclinometer cases. This is achieved by monitoring the installation with a reference system while reporting the absolute orientation of movement based on an azimuth.

The system can be operated remotely

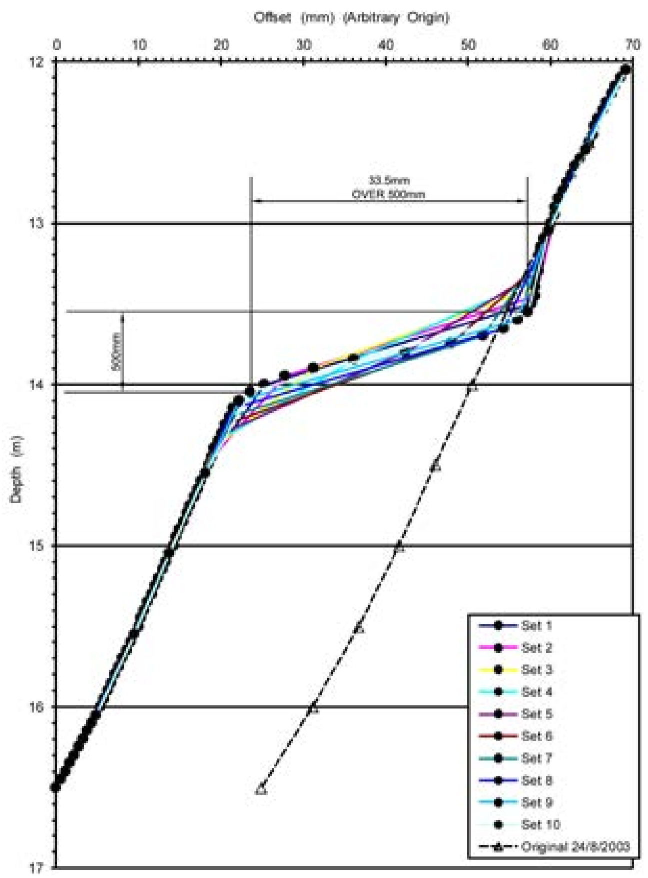


Figure 4 Detailed survey at 50 mm spacings

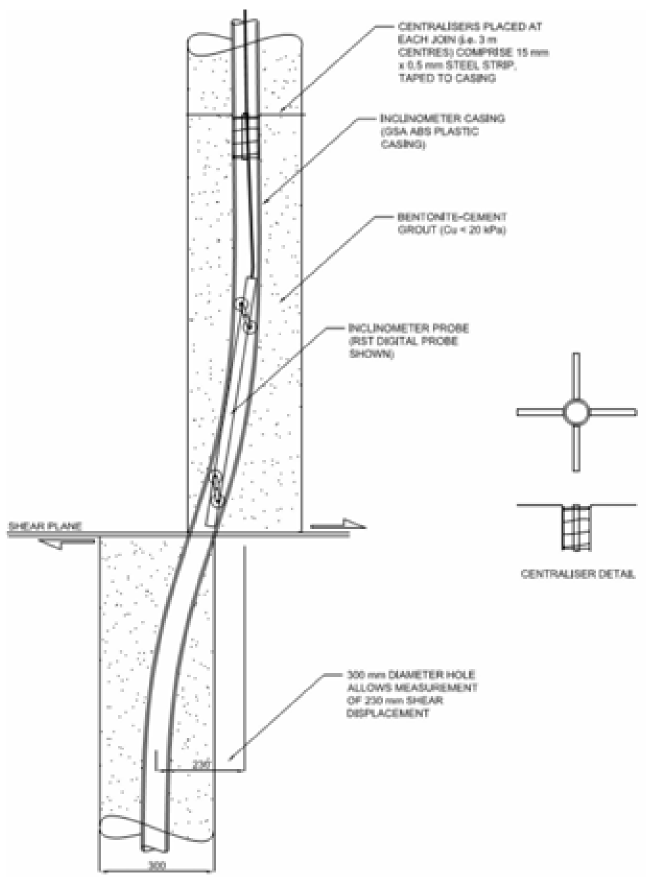


Figure 5 Softgrout backfill to increase shear tolerance at known fault location

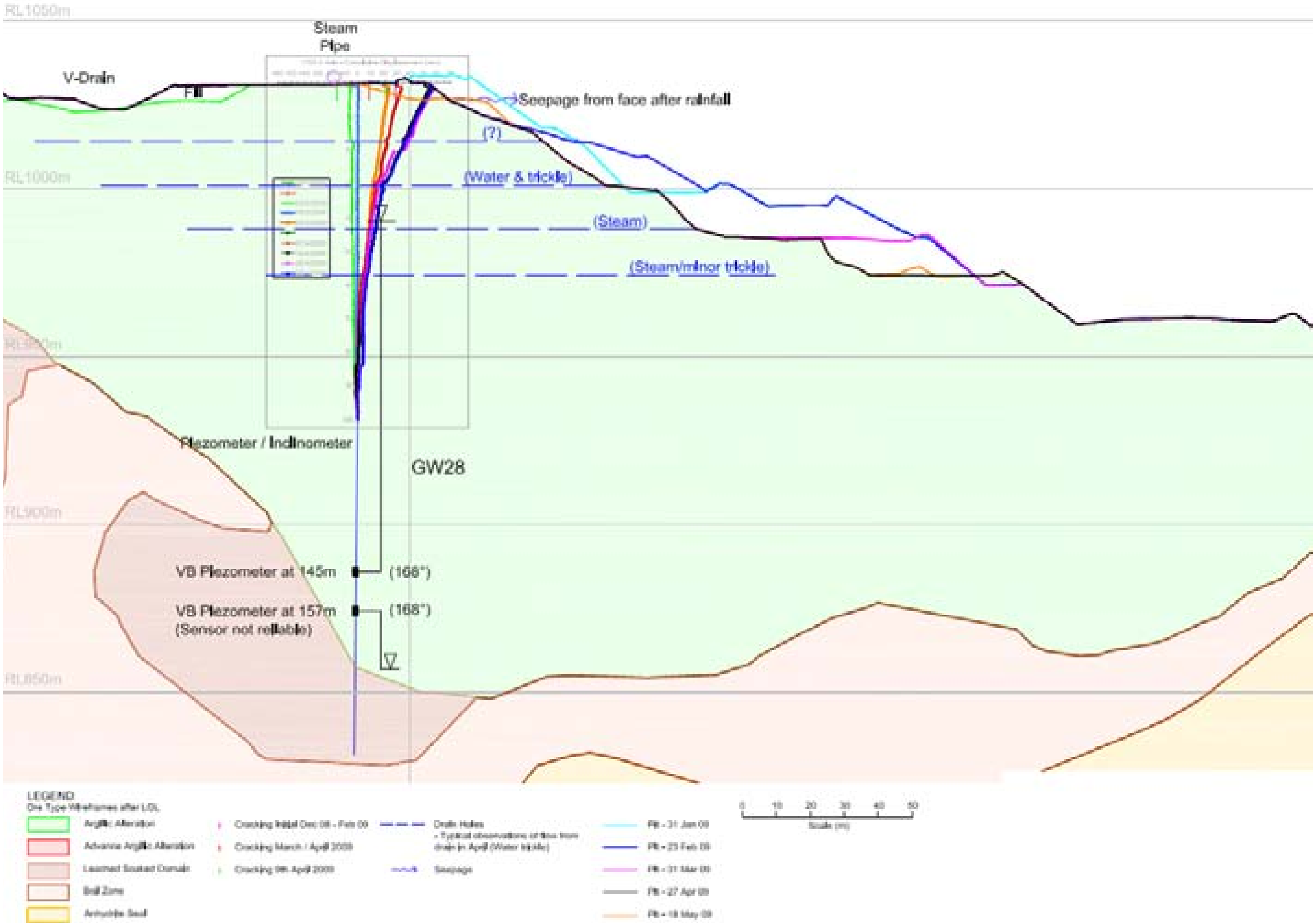


Figure 6 Inclinometer in argillic rocks at Lihir Gold Mine at 100 m depth

with the capability for remote real-time access and DMS is primarily designed for safety critical applications.

Emerging technologies

There are a number of technologies being investigated that may lead to improved inclinometers. Fibre optics has been applied to monitor the location and extent of strain, although direction of movement is still not resolved.

One exciting technology is the smart marker manufactured by Elexon Electronics that is currently utilised to monitor underground ore flow in sublevel and block cave mines.

Research and development of networked smart markers is progressing well. Networking enables communication between isolated sensors underground, creating a communication chain that reports back to the surface. A field scale trial is underway on a mine slope, where the sensors will report proximity of the neighbouring sensor only. Subsequent trials are intended to host additional sensors, hopefully including tilt.

Conclusions

Subsurface monitoring is critical for understanding failure mechanisms. Inclinometers, despite their limitations, are the primary subsurface tool available to geotechnical engineers.

Whilst for the most part we know and understand the sources of error in inclinometer surveys, there is plenty of scope for improvement. The ideal inclinometer would be:

- Automatic.
- Have systematic errors that are much smaller than the real movements being detected.
- Robust.
- Easy to use.
- Low cost.

The latest inclinometers tick some of these boxes, however adoption appears slow, presumably because of cost. There is a great reward for the enterprise that introduces the next big thing in subsurface monitoring.

Mark Fowler is the co-chair of the Ninth International Symposium on Field Measurement in Geomechanics to be held in Sydney, Australia in 2015.

Please click here for article references.

Figure 7 Inclinometer recovery tools

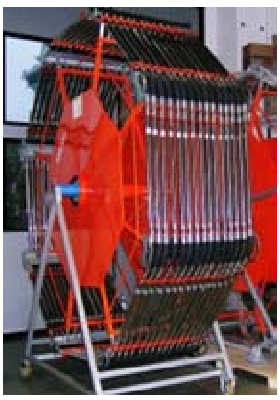
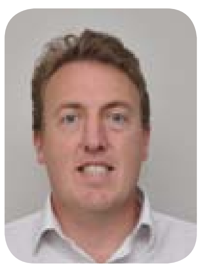


Figure 8 DMS technology



Mark Fowler
Pells Sullivan Meynink
FMGM 2015 Symposium Chair

Are we really living and mining sustainably or are we just heading for an epic fail?

an industry comment by Professor Ken Mercer, Australian Centre for Geomechanics

In the global mining industry one increasingly hears of sustainability, sustainability science and sustainable mining. The focus on sustainability has arisen as a result of increasing concerns regarding the path of human socioeconomic development and environmental impacts. These have emerged in a context of the ever increasing rate of extraction of natural resources in order to produce the goods and services demanded by an increasingly affluent population. To their credit, many companies have made it a corporate goal and appointed leaders to actively promote sustainability within their organisations.

But perhaps we should take a brief moment to consider what sustainability really means and whether it can ever be realistically achieved in the context of mining as we know it, and perhaps, even our society in its current state. Sustainability has been defined as the capacity to prevail. In ecology the word describes how biological systems remain diverse, self-sustaining and productive over time. The Brundtland Commission report defined sustainable development as,

"*development that meets the needs of the present without compromising the ability of future generations to meet their own needs*".

The key issues here are clearly living within the constraints of our environment and maintaining the long-term continuity of human society.

The first thing we need to consider is how does our performance rate in terms of sustainability on a macro scale? The Global Footprint Network (GFN) tracks humanity's demand on the planet's ecological resources, such as food provisions, raw materials and carbon dioxide absorption, which it calls its ecological footprint. The GFN measures this footprint against nature's ability to replenish those resources and absorb and biologically degrade waste products. Their data shows that currently, in less than eight months of each calendar year, human activities consume as much natural capital and ecosystem services as our planet can regenerate in that year.

The GFN 2012 national footprint accounts show humanity is now using ecological resources and services at a rate where it would take just over 1.5 Earths to renew. We are on track to require the resources of two Earths by mid century. Their data also shows that in terms of country footprint, China currently uses the ecological resources of 2.5 Chinas to self-sustain, the USA 1.9, Qatar 5.7 and Japan 7.1. Australia is still less than one in this assessment. Today, more than 80% of the world's population reside in countries that use more than their own ecosystems can renew within their boundaries. These ecological debtor countries either deplete their own ecological resources or deplete them from elsewhere.

In addition to the GFN, there is a growing body of literature which confirms the increasing degradation of the Earth's biosphere. Given the recorded loss of species over the past millennia and current rates of depletion of remaining natural environments, biologists now suggest that a sixth mass extinction may already be underway. Many global fisheries are in free fall or have collapsed altogether, as industrial fishing fleets with staggering overcapacity continue to scour the oceans for the remaining fish. This is not to mention accelerated climate change where continuously increasing levels of carbon in the atmosphere mirror the exponential economic growth and has now overwhelmed the Earth's capacity to absorb it. Rapidly receding and disappearing glaciers and exponentially collapsing ice sheets all point to an ice free artic summer, possibly as early as 2016. All these warnings of course go unheeded while industries and politicians endlessly debate the cost of carbon and its detrimental effect on economic growth.

> **"The fallacy of exponential growth on a finite planet has long been recognised but continually ignored"**

Despite what some might ignore, we live in a world that is finite in every respect. These finite systems impose absolute limits on the ecological processes and human activities embedded in it. The ever decreasing natural and ecological resources have in themselves a finite capacity to replenish themselves. Modern economies are based on ever increasing expansion and consumption of goods and services which is clearly unsustainable within the finite resources of planet Earth. The fallacy of exponential growth on a finite planet has long been recognised but continually ignored. The actual availability of energy and materials must ultimately limit human population and the level of socioeconomic development. Consumption by the current generation is constantly increasing, compromising future generations' ability to provide for their own needs. If civilisation in anything like its present form is to persist, it must start to take into account the finite nature of the biosphere.

The modern mining industry has made great strides in promoting sustainability, however is itself embedded in an increasingly unsustainable society. It has also made great improvements in efficiency, however, it is still dependent on

large quantities of finite natural resources such as fossil fuels and fresh water supplies as inputs. Over centuries, industry has logically extracted the easiest to obtain and highest quality resources first, but continuously declining grades means it is more and more costly in financial and environmental terms to extract additional quantities of these items. Orebodies have poorer grades, are deeper and require ever larger mining fleets, as well as energy requirements to mine cost effectively and to meet continually increasing demand. Consequentially, magnitudes of land disturbance and pollution continue to increase in scale. This will result in mining in its current form becoming compoundingly unsustainable.

The bottom line is that the growing human population and economies are being fed by both unsustainable use of finite non-renewable resources and by unsustainable harvests of renewable resources. We are producing ever larger quantities of waste and carbon emissions. Furthermore, attaining sustainability is additionally complicated by inevitable, yet unpredictable, changes in both human socioeconomic conditions and the extrinsic global environment.

We therefore need to be mindful of the bigger picture. The current unsustainable trajectory of the resource industry mirrors the course of modern civilisation. Society needs to transition to a truly sustainable steady state to ensure a future for the next generations. One that is not reliant on ever increasing economic growth and resource usage and one that does not result in the continuing decline in biodiversity. The mining industry has shown enormous ingenuity in finding ways to tap more and more inaccessible resources. Now is the time for the mining industry to put its tremendous resourcefulness to the test in finding and promoting the truly sustainable use of the finite resources it takes from the Earth. This will require nothing short of a complete re-invention of the resource industry to become global custodians of the ecosystems and resources it mines and manages. I do not know exactly what form this transformation can take but several trends are clear. The capacity for endless economic growth in a finite world is limited at best. There will be less linear resource use, a dramatic reduction in the removal of virgin resources from the Earth, and a lot more closing of the loop involving full recycling or reuse of these assets and resources. All of this will need to be done with very little fossil fuel usage, high energy efficiency, little or no net carbon emissions, minimal waste, a high degree of recycling and no loss in species diversity.

The hour is late. We cannot expect or wait for elected politicians to chart the difficult course of change needed ahead. As individuals we need to bolster change by aligning together to embrace these goals and force the rapid emergence of a truly sustainable society and supporting industries. Failing to do so will impair the viability of Earth's finite systems for future generations who will discover, possibly too late, that we have stolen everything from them, in order for us to live a very brief, selfish, and highly unsustainable life. For everyone, it's a case of rapidly adapt our society to our finite ecological limits or face the catastrophic consequences.

Professor Ken Mercer
Australian Centre for Geomechanics

Managing organisational risk for mine closure

writes Bill Biggs, Biggs and Associates, and Professor Ken Mercer, Australian Centre for Geomechanics

Introduction

Managing the organisational risk of closure is a major issue facing operators in Australia. There are around 50,000 mine sites in Australia that are not currently operating, are in the process of closure, or have been abandoned.

During operations, mining activities generate significant safety and environmental risks that reach their maximum at the commencement of closure. These risks, which may include the stability and erosion of landforms such as open pits, tailings dams and waste dumps, as well as ongoing surface and groundwater contamination from contamination or ongoing geochemical reactions, can persist for a considerable time after mining ceases. Responsible mining organisations and regulatory authorities must ensure that adequate and appropriate organisational controls and defences are in place to enable future land owners to manage these risks after closure has taken place.

It may take decades, or even centuries, for the stability of landforms and contamination to decline to acceptable levels and for reconstructed ecosystems to reach a self-sustaining and stable state.

In addition to the persistence of these long-term risks there are a number of closure management aspects that have not historically been dealt with effectively in traditional approaches to closure. These are:

1. The transition from an actively managed mining operation to a more passively managed land use at closure.
2. The transfer of appropriate knowledge and skills required to manage these risks to the new land managers when control of the site is handed over.

The transfer of ownership between mining companies also presents a significant risk to the effectiveness of organisational controls where active mining assets are bought and sold. This is due to differing organisational structure, culture and technical capacity between the old and the new owners.

This article discusses a holistic risk-based approach to managing the risks associated with closing a mining operation. The article assumes the mining organisation is committed to leading practice mine closure and that closure will take place in an orderly and planned process. The article does not address issues associated with unplanned closure or abandonment. The approach is based on organisational risk management principals which require the provision of adequate controls and defences to prevent adverse incidents or impacts occurring after the mining operation has finished. This model is often referred to as the Swiss Cheese Model (Reason 1997) or the Layers of Protection Model.

The Swiss Cheese Model is shown in Figure 1 where the red line indicates whether there are sufficient controls or defences in place to prevent a series of incidents or actions leading to an adverse impact.

Overview of organisational risk management

Organisational risk management is a systematic approach to risk management based on the premise that humans are fallible and that errors or mistakes are to be expected. These are seen as consequences of the system rather than causes, having their origins not so much in human failure as in 'upstream' systemic factors. Central to organisational risk management is the provision of controls and defences, i.e. barriers and safeguards that prevent mistakes producing an adverse event or impact (Figure 1). Should an adverse event occur, the important issue is not who made a mistake, but how and why the controls or defences failed (Reason 2000). The most effective measures implemented at closure will recognise this and be designed from a fail-safe perspective.

Accidents happen when these barriers or safeguards are inadequate, they are defined by Reason as:

"Comparatively rare, often catastrophic, events that occur within complex modern technologies such as nuclear power plants, commercial aviation, the petrochemical industry, chemical process plants, marine and rail transport, banks and stadiums." (Reason 1997)

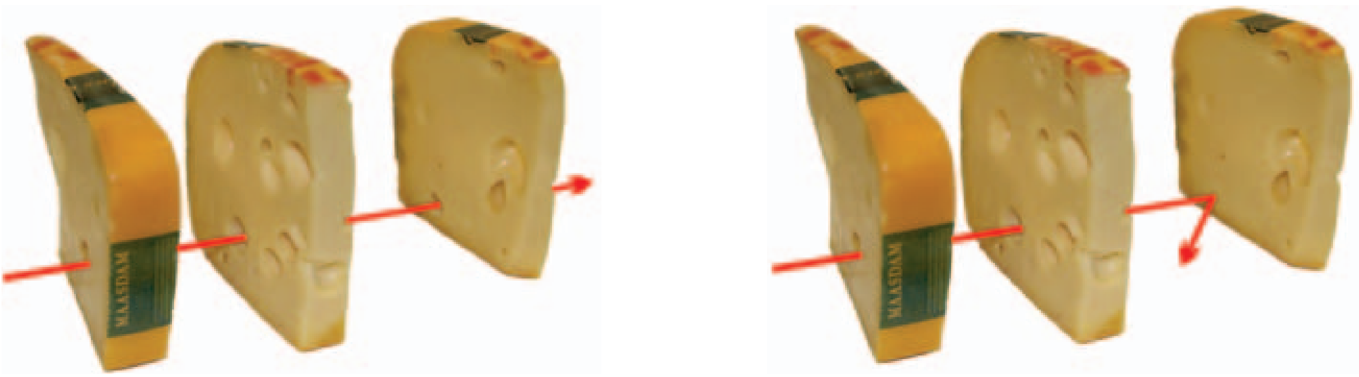
Organisational accidents can affect the physical safety of internal staff, as well as external communities, and can have devastating effects on environment and physical assets. The liabilities generated from these types of accidents can affect the profitability and commercial viability of the organisation involved, such as the impact on Union Carbide from the Bhopal incident in 1984; on Exxon from the Valdez incident in 1989; and on BP from the Deepwater Horizon incident in 2010.

Organisational accidents in the mining industry are numerous and include waste structure and tailings dam failures at Aberfan, Wales; Merriespruit, South Africa; Stava, Italy; Omai, Guyana; and Los Frailes, Spain. The management of organisational risk is therefore highly relevant both during active operations and at closure of the mine.

Managing organisational risk for closure

Safety and environmental risks are likely to persist for a considerable time after mining has ceased. These will need to be managed by future landholders or land managers after the miner has gone. At closure, the layers of protection and defences must deliver the agreed closure outcomes. The objective is to adapt the layers of protection and defences in three key areas to achieve this:

- Physical changes to landforms. Changing and developing new layers



(a) (b)

Figure 1 The Swiss Cheese Model (Reason, 1997); (a) without adequate defences; (b) with adequate defences of protection and defences to manage the physical change in landforms, water and material flows when active management no longer exists.

- Changes in ownership structure. Once the physical changes in the system have been accounted for, the layers of protection and defences must account for the change in organisational ownership type and structure.
- Change in ownership culture. As an organisation is essentially driven by culture, the layers of protection and defences must account for the culture of the new land managers.

Managing the transition of physical structures to non–mining land use

The first level of defence should always be to design and construct structures to withstand normal events and extreme events. Without stable landforms there is no possibility of achieving acceptable closure outcomes and for the site to reach a self-sustaining functional state.

To achieve stable structures, those designing and constructing mining landforms must understand the environment they are working within and the characteristics of the materials that will form these structures (Mercer & Biggs 2013).

Where extreme events cannot be managed within the constraints of the mine site, consideration should be given to implementing secondary – and tertiary – level defences, i.e. flood containment or diversion structures, as well as providing post-event controls.

Water management is a good example where active management controls and defences are removed when operations cease. Closure measures need to account for all the drainage and water movement from the site in a passive system requiring minimal management input. All recovery of water ceases at closure, therefore, closure measures must be adequate to manage flows and quality to acceptable standards (Figure 2). Identifying suitable measures to meet closure outcomes is challenging. The earlier the process is commenced the greater the opportunity to test these measures before closure and removal of active control measures.

Managing organisational defences for change of ownership

Each change during operations brings a new organisational cultural and operational skillset to the operation and will affect how the project is operated and closed. Poor communication and transmission of critical information can undermine organisational defences.

At closure, the transfer of ownership from miner to new land manager is an even more significant transition. Complex organisational risk management measures are more than likely beyond the skills and competency of the new owners to understand or implement effectively. The challenge is to identify appropriate measures that are within the capability of the new land managers. Effective defence measures should have considered these risks during design and be sufficiently robust to remain resilient to this change.

Defences and controls that are closely linked to an owner's organisational structure, hierarchy and internal reporting channels are particularly vulnerable to change. These can be rendered ineffective with a change in the type, scale and capacity of the new organisational structure.

At closure, mining organisations generally have a more complex structure to undertake mining operations, where the new land owners will have a simple structure suitable to undertake rural or conservation land uses. The new managers are unlikely to have the knowledge, experience or competence to manage the specialist consultants and contractors necessary for complex management defences put in place during operations.

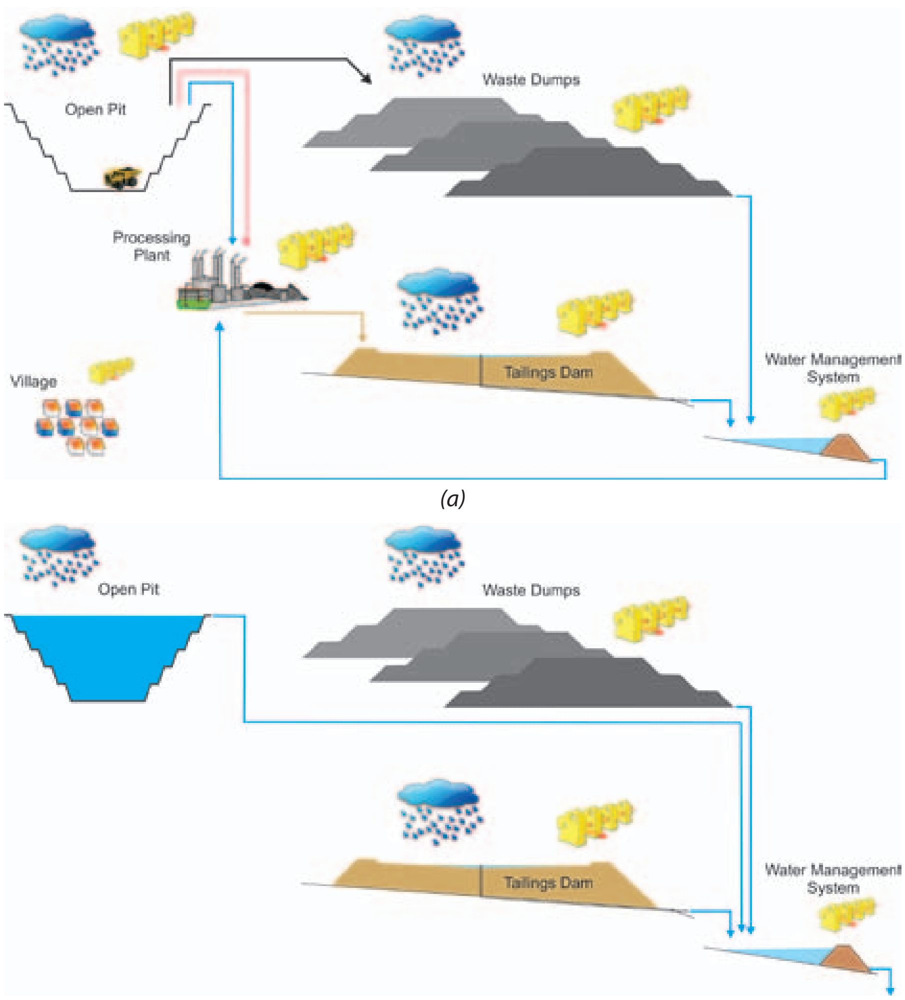


Figure 2 Surface water management during operations and at closure; (a) during operations; (b) at closure

Managing organisational defences for a change in culture

Culture plays an important role in driving and enforcing the integrity and effectiveness of system defences. The speed and effectiveness at which existing organisational defences are transitioned and implemented will reflect the culture of the new owners. How effective this is will depend on three factors at the core of a safe and effective culture (Reason 2008).

Commitment

The long-term success of closure will depend on the investment the operator puts into understanding the requirements to transition from operations to closure. For best results, the commitment to closure needs to begin early in the life of the mine to enable the identification and integration of long-term risk mitigation measures and structures throughout its operational life. Too often this aspect is left to the end of the operation and requires retrofitting during the closure phase, adding a sizable cost to the closure process.

A fully integrated organisational risk approach goes beyond the physical activities required to make a site safe, stable and self-sustaining; it requires the operator to:

- Actively engage in measuring and

monitoring to gather adequate data to demonstrate that closure outcomes have been met.
- Understand the needs of the future landowners or managers and prepare them for the transfer of the organisational risk and to take control of the site after closure.
- Inform future landowners and managers of the layers of protection that are in place and any residual liabilities they may be taking on.
- Gain sign-off and release by government regulators from their conditions of operation.

If there are ongoing management requirements, the operator may need to establish funding arrangements to enable future landowners or managers to continue the activities. It is unreasonable to expect private or public funding of these activities.

Competence

Mining is a complex and challenging activity. At closure it is unlikely the new owners will have the necessary management skills or experience; this has to be recognised in the handover of responsibility for organisation risk. Provision should be made either to develop the skills of the new owners or, more appropriately, to ensure there is no requirement for high-level technical management for the long-term risk management measures.

Cognisance

Cognisance relates to the awareness and understanding of the nature and magnitude of the organisational risks that need to be managed both during and after closure.

Transitioning organisational defences for closure

The transition from operations to closure requires a structured and systematic approach and falls into three stages:
- Preparation. When all aspects of the operational design, conditions and constraints are reviewed to ensure they are achievable and will meet the closure outcomes. It may be necessary to modify or renegotiate some or all of these requirements based on experience gained during operations.
- Implementation of closure measures. Implementation of the controls and defences developed to meet the closure outcomes. A key aspect will be measuring how these measures perform and, where necessary, undertake remediation works.
- Actual closure and handover of ownership. This can take place when the closure outcomes have been met and ideally the new land managers have the appropriate skills and knowledge to manage the residual organisational risks.

Important observations

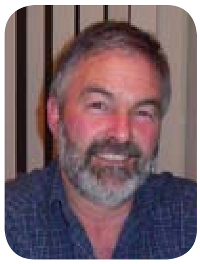
Managing organisational (system) risk is a complex and difficult task for any team of well-qualified managers. When operations cease, if the management system is too complex, future land managers are unlikely to have the same level of technical competence and knowledge necessary to manage these system risks successfully.

The system controls and defences required for closure must be developed before mining operations end. For the transition to closure, existing controls need to be adapted and modified to accommodate the changes that take place during closure activities. These changes must reflect the type and nature of the organisational structure of the new land managers and reflect their cultural differences.

The focus has to be on the transition from an actively managed operational mine into a more passively managed closed mine. The challenge is to implement measures during the closure process that will achieve the closure outcomes, are appropriate to the new land use, and are sufficiently robust to protect the outcomes in the long term – potentially into perpetuity.

To be successful, all stakeholders must be committed to undertaking the work necessary to achieve the closure outcomes. If this commitment is missing, the process will fail, and it is unlikely the desired long-term outcomes will be achieved. The mining industry is littered with examples of poor performance and dissatisfaction amongst landowners and stakeholders, with the responsibility for costly remediation ultimately falling to the community and taxpayers. This reputational damage to the industry affects its ability to develop new projects in the future.

Please click here for article references.



Bill Biggs
Biggs and Associates

To be successful, all stakeholders must be committed to undertaking the work necessary to achieve good closure outcomes

ACG Environmental Guidelines for Mine Waste Landforms, Design, Closure and Rehabilitation

Editors Professor Ken Mercer, Australian Centre for Geomechanics, and David Tongway, CSIRO Ecosystem Services

On the horizon

Industry's increasing emphasis on best practice mine waste management and sustainable mine closure has provided impetus for the improved management of rock dumps and heap leach pads during operations to facilitate economical and robust final closure. It is the ACG's intent to document the latest industry advances, practices and methodologies and provide owners, operators and designers with a valuable reference for mine landforms and their management through their 'life cycle'.

- Designing for closure
- Investigations and baseline studies
- Stakeholder engagement
- Environmental geochemistry
- Mine water management
- Landforms (design, construction and operation)
- Design of financial provision for closure
- Establishment of controls to mitigate risks during operation, care and maintenance and preparation for closure

Call for industry sponsorship

The guidelines' sponsorship packages has been designed for companies to provide their endorsement of excellent waste rock, heap leach pads and stockpile management, and commitment to reducing the impact of the mining industry on the environment. We believe that industry's adoption of best practice principles outlined in these new guidelines will be enhanced by your support. Contact the ACG for details.

www.acg.uwa.edu.au/publications/future_publications

On the horizon

Unsaturated Soils Guidelines (Volume 1)

Soil Water Characteristic Curves for Materials Classified according to International Unified Soil Classification Systems

These new ACG guidelines will detail the key aspects and behaviour of unsaturated soils. This is the first volume in potentially a series of volumes on this highly topical and relevant subject matter. The editors are Professors Ken Mercer, Professor Environmental and Mining Geomechanics, Australian Centre for Geomechanics, Australia, Professor Harianto Rahardjo, Head of Division of Infrastructure Systems and Maritime Studies, School of Civil & Environmental Engineering, Nanyang Technological University, Singapore and Alfrendo Satyanga, Research Associate, Nanyang Technological University, Singapore.

The ACG intends to launch this exciting new publication in early 2014.

Call for industry sponsorship

Companies may benefit from association with these highly topical and relevant guidelines by sponsoring this exciting new publication, and acknowledge the significance of unsaturated soil mechanics to industry and its critical application to the fields of mine waste, open pit design and rehabilitation.

www.acg.uwa.edu.au/publications/future_publications

Non-ACG industry events of interest

Coal Operators' Conference	12–14 February 2014 | New South Wales, Australia
Fifth International Mining and Industrial Waste Management Conference	10–12 March 2014 | Rustenburg, South Africa
12th AusIMM Underground Operators' Conference 2014	24–26 March 2014 | Adelaide, Australia
Fifth International Seminar on Strategic versus Tactical Approaches in Mining	7–8 May 2014 | Muldersdrift, South Africa
Sixth South African Rock Engineering Symposium	12–14 May 2014 | Muldersdrift, South Africa
Rock Mechanics and Rock Engineering: Structures on and in Rock Masses (EUROCK 2014)	27–29 May 2014 | Vigo, Spain
Third International Symposium on Block and Sublevel Caving	5–6 June 2014 | Santiago, Chile
First International Conference on Applied Empirical Design Methods in Mining	9–11 June 2014 | Vancouver, Canada
17th International Seminar on Paste and Thickened Tailings	9–12 June 2014 | Vancouver, Canada
Ninth International Conference on Mine Closure	1–3 October 2014 | Johannesburg, South Africa

www.acg.uwa.edu.au/events/nonacg_events

ACG event schedule*

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AUSTRALIAN CENTRE FOR GEOMECHANICS
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Joint Venture

2014

Are Your Ground Support Costs Too High Workshop	23 March 2014 | Adelaide, SA
Introduction to Environmental Geochemistry of Mine Site Pollution Short Course	26–27 March 2014 | Perth, WA
Best Practices in Mine Backfill Technologies Workshop	19 May 2014 | Perth, WA
11th International Symposium on Mining with Backfill – www.minefill2014.com	**20–22 May 2014 | Perth, WA**
Stress Measurement and Stress Modelling – Implications for Extraction Workshop	4 June 2014 | Santiago, Chile
Practical Rock Mechanics (Introduction) Short Course	28–29 July 2014 | Perth, WA
Ground Support in Mining (Introduction) Short Course	30 July–1 August 2014 | Perth, WA
Open Pit Geotechnical Analysis and Design Training Courses	26–28 August 2014 | Perth, WA
Practical Rock Mechanics in Underground Mines Course	13–14 September 2014 | Ontario, Canada
Ground Support Subjected to Dynamic Loading Workshop	15 September 2014 | Ontario, Canada
Seventh International Conference on Deep and High Stress Mining www.deepmining2014.com	**16–18 September 2014 | Ontario, Canada**
Practical Calibration of Numerical Models for Meaningful Predications of Ground Behaviour Course	19 September 2014 | Ontario, Canada
Design of Cover Systems for Rehabilitation and Closure Workshop	29 September 2014 | Johannesburg, South Africa
Mine Waste Management for Regulators Workshop	30 September 2014 | Johannesburg, South Africa
Blasting for Stable Slopes Short Course	3–5 November 2014 | Perth, WA
Unsaturated Soil Mechanics for Mining Seminar	1–2 December 2014 | Perth, WA
Tailings Management Seminar	3–4 December 2014 | Perth, WA

2015

Ninth International Symposium on Field Measurements in Geomechanics www.fmgm2015.com	**8–10 September 2015 | NSW**

www.acg.uwa.edu.au/events/current

*The ACG event schedule is subject to change. For event updates visit www.acg.uwa.edu.au/events

The ACG team wishes you and your family a

MERRY CHRISTMAS AND A HAPPY NEW YEAR

Our office will be closed from Monday, 23 December 2013, reopening on Monday, 6 January 2014.

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Performance of dynamic support system in highly burst-prone ground conditions at Vale's Copper Cliff Mine – a case study

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