NEWSLETTER

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IN THIS EDITION

Northparkes Mine, New South Wales, Australia

High resolution microseismic monitoring is shedding new light into caving mechanisms

by Yves Potvin, Australian Centre for Geomechanics

Introduction

Block and panel caving are, without a doubt, the most productive underground mining methods. However, they are methods where mine operators have very little control once production has commenced. The main and practically only means of controlling the caving process is by using an appropriate undercutting strategy and adjusting the sequence and rate at which the ore is drawn.

Adding to the operating challenges of these ultimate mass mining methods is the lack of access to the caving front; where ore breakage occurs. Block and panel caving miners have virtually no visual access to the cave and limited opportunities to install instrumentation in the large volume of ore to be caved. Generally, only a handful of observation holes and simple instruments such as extensometers or Time Domain Reflectory (TDR) are installed to monitor the multi-million cubic metres of ore. Therefore, caving operators not only have limited control over the caving process, they are also literally blind to what is occurring at the caving front. This situation is far from ideal in terms of risk mitigation but can be improved with the introduction of high resolution seismic systems.

Continued page 2

Continued from page 1

High resolution seismic monitoring has been used in open stopes and cut-and-fill Canadian mines for a few decades and in Australian mines during the last decade. However, to date very few caving mines have used seismic monitoring in a high resolution mode. Some caving mines have been using seismic monitoring systems for a long time but their focus has traditionally been on monitoring and performing seismological analysis on large events. The wide sensor spacing of their systems did not allow for recording of smaller events or for the precise location and tracking of the seismogenic zone — where the stress front ahead of the cave is breaking the rock. Until recently, only a few block and panel caving mines have had this capability, including Northparkes Lift 2.

Aerial view of Northparkes Mine

Caving mechanics concepts

In his book "Block Caving Geomechanics", Brown (2003) has a section (1.2.2) on "Basic Caving Mechanics" where he describes some of the elementary caving mechanics concepts. For example, "… caving occurs as a result of two major influences — gravity and the stress induced in the crown or back of the undercut or cave." A number of distinct mechanisms are proposed: the first involves a low stress environment where the dominant mechanism is gravity fall resulting from the de-confinement of the rock mass in the cave back. In the second mechanism, also referred to as "stress caving", the induced tangential stresses which are high compared with the compressive and shear strengths, are responsible for breaking the rock mass. This is clearly the case for deeper caves like Northparkes Lift 2 and Palabora.

Brown (2003) mentioned a third mechanism where the tangential stress induced in the crown is high enough to clamp the discontinuities, yet in comparison to the rock mass strength, remains too low to induce failure. Some form of pre-conditioning may be required to weaken the rock mass and facilitate caving under these conditions.

A fourth mechanism is called subsidence caving "...in which a large mass of rock subsides rapidly as a result of shear failure on the vertical or near vertical boundaries of the block". This type of caving may lead to catastrophic consequences, especially if a large air gap is present between the cave back and the caved material.

Duplancic (2001) proposed that the stress driven "rock breaking" process of caving mines, which he called the seismogenic zone, can be identified using high resolution seismic systems. He defined a seismogenic zone as a band where the "stress front" is actively failing (or fracturing) the rock mass. As the state of rock mass within this band reaches a post failure state, the stress is shed outwards and the seismogenic zone (or stress front) migrates upwards. Microseismic systems can be a very powerful means of tracking "real time" the seismogenic zone where the vital rock breaking process of caving mines is happening. It can also allow for timely corrections and the implementation of control measures if the cave is not following its anticipated behaviour. For example, the rate of advance of the seismogenic zone will provide early warning that the cave is stalling (Brown's third mechanism) which may require some form of induction or a change in the undercutting sequence. Conversely, the seismogenic zone can accelerate indicating that the breakthrough and potential consequences will happen much earlier than anticipated. Risk mitigation measures can then be brought into action in a timely manner.

Cave initiation during undercutting

In this section, the well-known block caving concept of cave initiation will be discussed with parallels made to its associated seismic behaviour.

At the early stage of undercutting and prior to cave initiation, seismicity tends to be entirely localised near the blast, and this activity subsides within a few hours of the blast. Once a certain hydraulic radius is reached, the caving process will initiate. The seismic activity will then have migrated some distance above the undercut (at least 50 m in the case of Northparkes Lift 2), formed the seismogenic zone in which the rock fracturing process will continue even during prolonged periods when there is no mining activities, in between undercut blasting. In the case of Northparkes Lift 2, once cave initiation started, the seismogenic zone moved at a constant rate of approximately 0.15 m/day. Plotting the seismicity on time-magnitude charts is an effective method to observe and monitor cave behaviour during undercutting. Figure 1 (top graph) shows data during the early undercutting period. The blue line is the cumulative number of events and the stars at the top of the graph represent undercut blasts. The vertical alignment of seismic events corresponds to designated blasting days and the stepwise

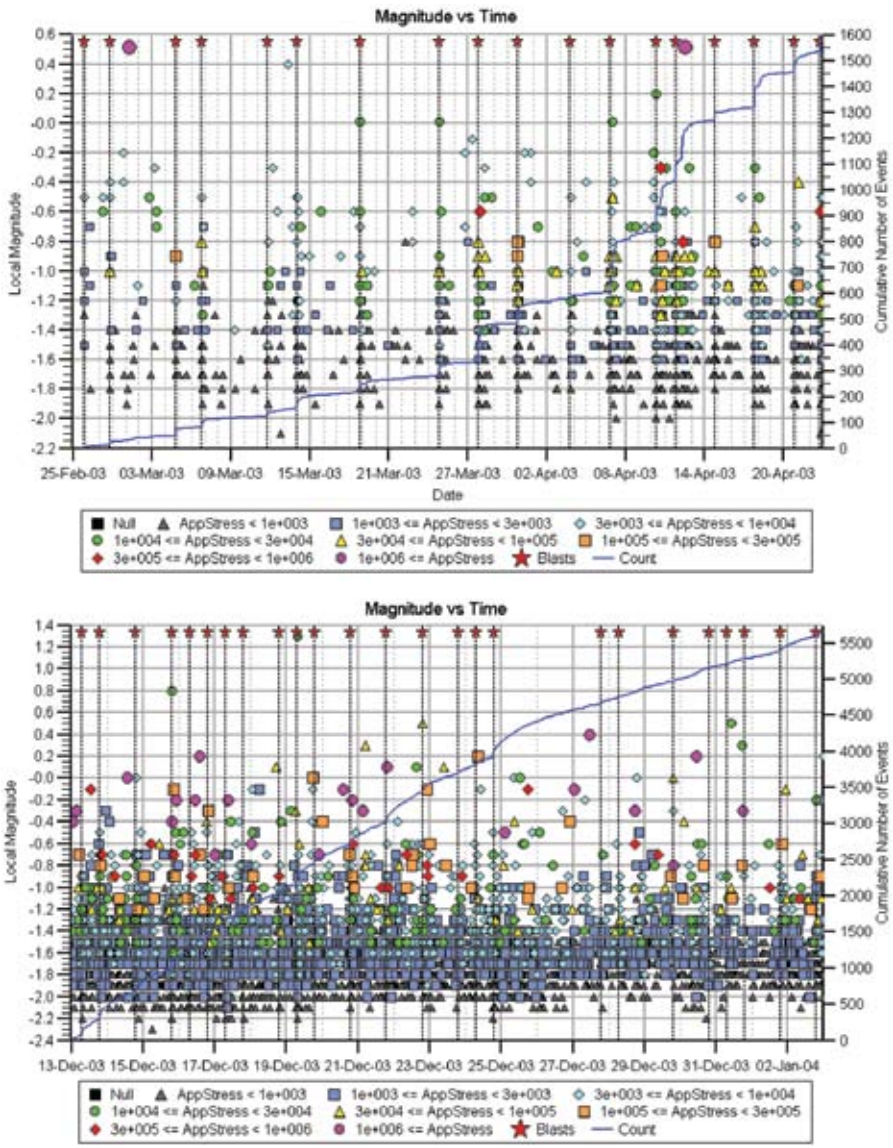

Figure 1 Magnitude-time history chart of seismic events that were recorded during the early stage of undercutting (top graph) and late stage of undercutting (lower graph) at Northparkes Lift 2

shape of the curve indicates that seismicity subsides rapidly after each blast and that cave initiation has not yet commenced. The magnitude-time history in the lower chart (Figure 1) contains data from the late undercutting period and the cumulative number of events (blue line) is no longer stepped but instead increases at a relatively constant rate, even in between blasts, indicating that the caving process has initiated.

Cave propagation and breakthrough during production

During undercutting only the swell from the undercut blast is removed keeping the broken ore tight against the cave back and the overlying rock mass in a certain confinement. As production starts, the ore is removed at a certain rate creating temporary voids or gaps underlying the cave back area, which stimulates the caving process. The de-confinement of the rock mass migrates upward from the cave back where rock blocks are falling off to the seismogenic zone; weakening the ore at that location and increasing the rate of the rock breaking process. For example, once the production started at Northparkes Lift 2, the seismogenic zone migrated at an increased rate of 0.5 m /day, compared to 0.15 m/day during late undercutting. It was also found that this rate of migration remained relatively constant, even if production was interrupted for at least 10 days in one occasion. From the Northparkes experience it appears that a relatively long period of production slowdown or shutdown is required to re-build the confinement within the seismogenic zone and slow down its migration rate. This observation challenges the conventional thinking that altering production rates is an effective way to control the caving process. It will certainly have an impact of the cave back migration but its effect on the breaking process (seismogenic zone migration) appears to be delayed significantly.

The migration rate of the seismogenic zone can, however, change rapidly and drastically when it reaches areas where ground conditions change. At Northparkes Lift 2 for example, when the seismogenic zone moved 150 m below Lift 1 (the area shaded in red in Figure 2), the migration rate suddenly increased from 0.5 m/day to 2.5 m/day and accelerated to almost 4 m/day in the last 100 m (green shaded zone, Figure 2), before breaking through Lift 1. Whilst the seismogenic zone was moving through the red area, four large seismic events of a local magnitude greater than one (ML>1) were recorded within the cave column. During the very rapid migration of the seismogenic zone through the green area, another 21 large events were recorded. However, this time the large events were located outside and to the west of the cave column, roughly where a diorite dike is present.

It is interpreted that the ground condition of the last 100 m of crown pillar under Lift 1 was already heavily broken due to pre-conditioning work done by the lower abutment stress from the mining of Lift 1. Consequently, the front of stress (lower abutment stress under Lift 1) was likely located further down in the red shaded area, when the mining of Lift 2 commenced.

Therefore, when the seismogenic zone moved through the high stress abutment zone (red area), the breakage process increased (2.5 m/day migration) as there was higher stress available to do the "work". This could also explain the four large seismic events that occurred in the crown, as the higher stress conditions resulted in higher seismic energy being released from the rock breaking process. In the last and rapid stage of seismogenic zone migration through the broken crown pillar (4 m/day migration), the stress front broke through to Lift 1 and suddenly redistributed from the crown to the

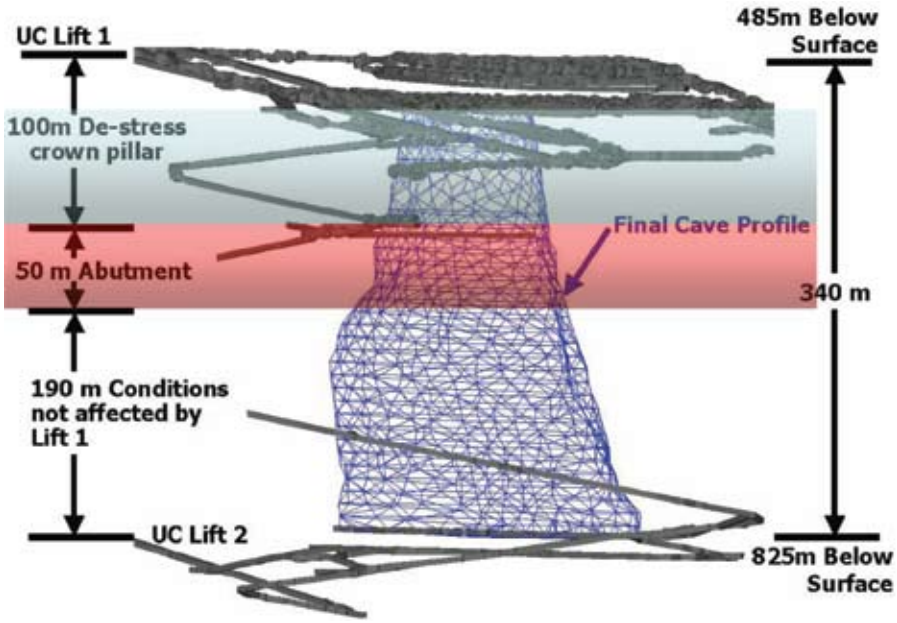

Figure 2 Diagram showing the geometry of Northparkes Lift 2 and an interpretation of the conditions prevailing under Lift 1, with a zone of 100 m de-stress crown pillar and (green shaded) and a zone of abutment stress (red shaded)

outside of the cave column, producing 21 large seismic events. Effectively, the crown pillar was rapidly removed and a regional stress readjustment occurred creating a period of high seismic hazard. After this stress readjustment was completed, the whole mine became seismically quiet and the seismic risk from that time became very low.

Experienced cave mining operators are aware that prior to and during the breakthrough period, the seismic hazard becomes elevated. In particular, when seismically active geological features are present in or around the cave, they will be particularly susceptible to produce large events when the regional stress change occurs. The risk can be particularly high when such seismically active features intersect the production area. In the case of Northparkes Lift 2, the large events luckily concentrated near the diorite dyke, relatively far away from production areas.

Discussion

Because instrumentation relies heavily on observation holes and extensometers, cave mines monitoring programs have traditionally focused on the position and migration of the cave back. However, when the cave back moves, it is simply the very last stage of the caving process, when the already broken rock mass is falling under gravity. This is an important piece of information but it does not provide much insight into what is really happening with the caving process itself. The installation of a high resolution seismic monitoring system enables the tracking of the seismogenic zone which is a strong indicator of the status and the position of the stress front. Only the rock mass going through the seismogenic zone is likely to cave in the future. Therefore, monitoring the seismogenic zone will facilitate the prediction of areas that may not cave and reduce the ore recovery. A slow down in the rate of migration of the seismogenic zone can also be indicative of an upcoming hang-up of the cave.

The period of stress breakthrough, when the seismogenic zone goes through the crown pillar, is a period of high seismic hazard. This results in a regional stress redistribution which may combine with existing seismically active geological features such as faults and dykes to produce flurries of large events. Only the monitoring of the seismogenic zone can provide early warning that this period of elevated seismic hazard is imminent. The physical breakthrough of the cave back will typically occur much later, depending on the rate of production.

High resolution seismic monitoring can provide great benefits but requires significant data processing and interpretation effort. To define and keep track daily of a seismogenic zone would require a dedicated, experienced seismic system operator working full time on the data. The ACG is currently developing seismogenic zone tracking tools within its MS-RAP software that could potentially automate this task, as part of its "Mine Seismicity and Rockburst Risk Management" research project.

Acknowledgement

The ACG acknowledges Northparkes Mines for granting us permission to use the seismic and other data generated from mining Lift 2.

Article references are available on request.

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The ideal shotcrete system

by Alby Loncaric, StrataCrete Pty Ltd
and Brendan Donnelly, Safe Mine Training

Introduction

What does the underground mining community require of shotcrete? The answer is usually a 32–40 MPa unconfined compressive strength (UCS) at a minimum thickness of 50 mm and a cost of $650–750/m^3. This article proposes that a nationally recognised and standardised training program would be an effective step in achieving this target.

A barrier to achieving this target is the over-design of a shotcrete mix to achieve strengths from 50–65 MPa UCS. Concrete suppliers are forced to use excessive quantities of cement to overcome poor procedures, poor adherence to good procedures, poor materials (sand and gravel), poor operator skills and uncontrolled use of water. The goal of developing best shotcrete practice should be to achieve extremely consistent and, therefore, lower cost shotcrete. Accepting a wide variation in quality, covered up by an excessive use of high cost materials, fails to address the underlying problem of a lack of training of shotcrete operators who can contribute to a shotcrete system that can produce consistent quality shotcrete. With a process as complex as applying high quality shotcrete underground in a producing mine, training of shotcrete operators produces measurably better shotcrete. This leads to safe and rapid development conditions underground. The aim of a nationally recognised and standardised training program must be to achieve a consistent quality of shotcrete to ensure rapid development whilst underpinning the existing safe working conditions underground.

Setting up the system

When setting up a shotcrete system the mine needs to take an overarching approach, i.e. the mine needs to know what is happening across the whole system starting with the raw materials, and continuing right through to the contractor doing the testing of the cylinders. In the end, the mine pays for all of these elements and should check that they are getting what they have paid for. The components of a shotcrete system are:

- The supply of materials.
- The batching of the shotcrete.
- The delivery of the shotcrete underground.
- The pumping and application of the shotcrete.
- The testing of the materials and shotcrete from delivery to application.

Each of these components require trained personnel who understand how what they do impacts on the quality of the shotcrete eventually placed underground. Each component requires its own set of safe work procedures that have been developed by the personnel involved. These procedures should be reviewed or audited on a regular basis to ensure the system is consistently producing quality shotcrete.

Existing system

Where there is an existing shotcrete system in place the mine should appoint a suitable person as the responsible officer for the shotcrete system. This person will receive copies of all of the reports and information associated with the shotcrete system. They will be responsible for taking action to correct any issues that arise out of the information received. They will also be responsible for chasing up any missing information or reports.

Audit

Where the mine has an existing shotcrete system that is not performing to the required standards, the responsible officer should establish what is happening in the system. By establishing a baseline the responsible officer can then measure the effects of the steps that they take to improve the shotcrete system. The responsible officer will establish a baseline by conducting an audit.

Because there are different kinds of contracts of varying quality, it will be better to look at what an ideal shotcrete system should look like rather than an audit against the contract, which may be flawed and may itself be the cause of the shotcrete system not working.

The audit should look at capturing any reporting that has been completed during the previous month of operation. The responsible officer should also observe the shotcrete system for at least one month to establish an accurate picture of the actual procedures and conditions that represent the current accepted standard of operation.

Usually there will be some improvement in the system simply because operators are aware that the system is under evaluation.

The audit can initially be split between batching and application. It can then be broken down into several large groups that can be applied throughout the process. These are:

- Test results (contract).
- Personnel (training).
- Procedures.
- Equipment.

Batching

The following list of questions can be used as a guide to establishing an audit process for the batch plant.

Testing

Are the batch plant operators (and truck drivers) taking a set of test cylinders and a slump test for every 50 m^3 of shotcrete produced from the plant?

Can the batch plant operator perform a moisture content test on the sand and gravel stockpiles?

Do they understand the specifications for the raw materials?

Personnel

What training has the batch plant operator received?

What training have the truck drivers received?

What additional training do they require?
When is this training planned to be completed?
How long do they take to batch a load?

Procedures

Do they adjust the water added to the mix to take moisture content into account?
Do they measure all water added to the mix prior to delivery of the mix to the shotcrete pump?
Do they know how to maintain a stockpile?
Do they follow the correct procedure for signing off on the acceptance of materials delivered to the batch plant?
Do they understand the correct use of chemicals in the mix design?
Do they follow the correct procedure for batching a load?
Do they use the correct mix design?
Are they recording the correct information associated with the batching of each load?
Is this information forwarded to the responsible person on-site?

Equipment

Is a pre-start check conducted and recorded for the batch plant at the beginning of every shift?
Is a pre-start check conducted and recorded for each truck whenever a new operator starts using it?
Are there any outstanding repairs required on the batch plant?
When are the repairs scheduled to be completed?
Are there any outstanding repairs required?

The goal of developing best shotcrete practice is consistent and low cost shotcrete application

Application

These questions can be used as a guide for establishing an audit process for the application of the shotcrete underground.

Testing

Do they apply and use depth indicators correctly?
Do they use depth stamps correctly?
Do they spray a Round Determinate Test Panel for every 50 m^3 of shotcrete through the shotcrete rig?
Do they prepare test cylinders and do they carry out a slump test for every 50 m^3 of shotcrete through the shotcrete rig?

Personnel

What training has the operator received?
What additional training do they require?
When is this training planned to be completed?
How many compliant test panels have they sprayed?

Procedures

Do they collect accurate information from the production meeting?
Can they conduct a shift change over meeting correctly?
Can they prepare a site for spraying, including:

- Hydroscaling
- Understanding ground conditions
- What needs to be reported to the shift supervisor
- Placement and set up of the rig
- Communication with the batch plant?

Can they tram the rig safely from site to site?
Can they operate the pump correctly?
Can they operate the nozzle correctly?
Do they clean up the rig correctly?
Do they record and report:

- Fallout
- Ground conditions
- Mix problems
- Testing
- Equipment failure
- Equipment damage?

Equipment

Is a pre-start check conducted and recorded for all plant used whenever a new operator uses it?
Are there any outstanding repairs required on the shotcrete rig or other equipment?
When are these repairs scheduled for completion?

Training

Once the current state of the shotcrete system has been established via the audit process, the responsible officer will then need to implement changes to improve the system. The most effective way to implement change is to set up a training program to ensure that all operators are aware of what is required of them. During the audit process the responsible officer can also be establishing a training needs analysis to identify what training is required for the operators currently working in the shotcrete system.

The following figures show the improvements that are possible with a minimal training program implemented over a three month period.

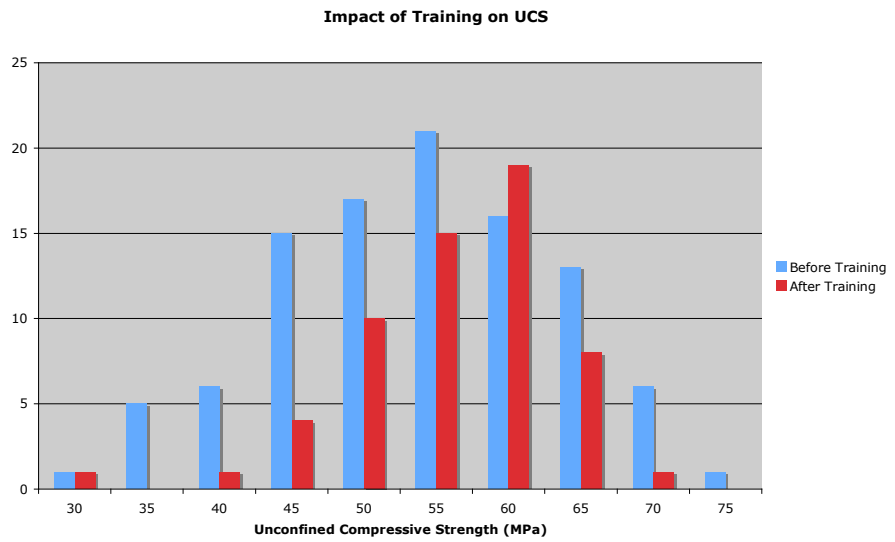


Figure 1 Unconfined compressive strength (UCS)

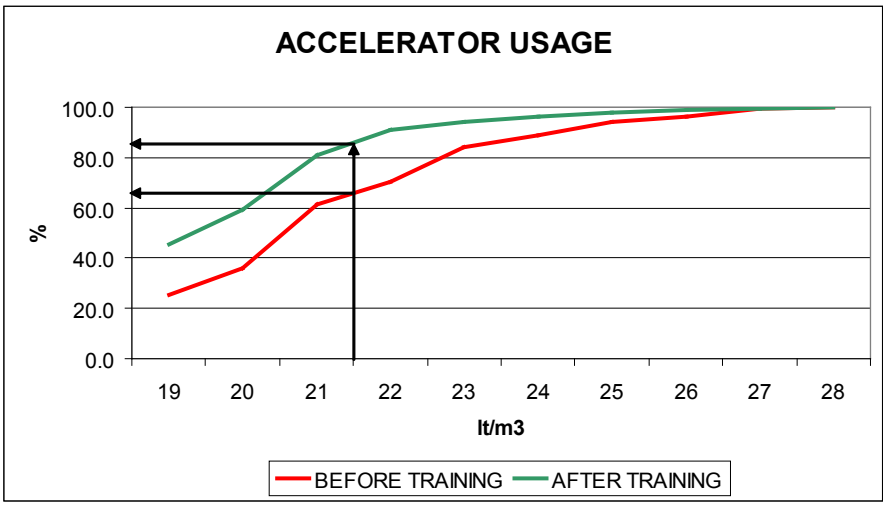


Figure 2 Accelerator usage

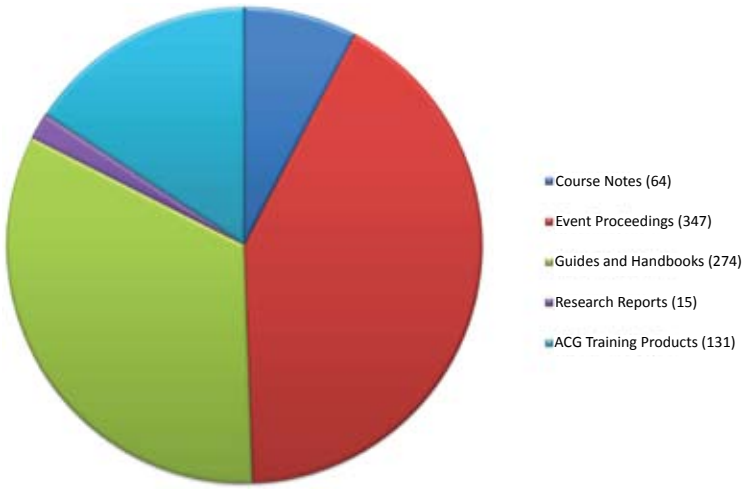
The main UCS improved from 55 MPa to 60 MPa. The standard deviation of UCS results was reduced from 7.1 before training to 6.8 after training.

After training there was a 20% reduction in accelerator usage, i.e. before training 65% of the time the operators were using less than 21.5 litres per cubic metre, after training the operators were using less than 21.5 litres per cubic metre 85% of the time.

The key to setting up the ideal shotcrete system is to set up a training program that ensures that everyone involved in implementing the system has the skills and knowledge to fulfil their role competently and safely.

Brendan Donnelly
Safe Mine Training

ACG Publications and Training Products Sales 2008



More than 830 ACG publications and training products were purchased by local and international mining personnel during 2008. To find out why industry turns to the Centre as one of the main sources of geomechanical knowledge, contact the ACG for a detailed list of event proceedings, training products and research reports.

Application of earthquake ground motion simulation in the evaluation of tailings storage facility performance

by Jonathan Liang, David Elias, Gerrie Le Roux, GHD Pty Ltd

Introduction

Evaluation of stability conditions and crest displacements of tailings dams during earthquakes is an important consideration for the mining industry. Failure of tailings dams induced by seismic loading may result in uncontrolled spills of tailings and other materials, potentially leading to environmental catastrophes, losses in terms of property and human life, and negative company image. Several cases of damage to tailings dams during earthquakes have been reported. In Chile, failure of the new and old El Cobre tailings dams following the La Ligua earthquake in 1965 killed 200 people. In Amatista, Nazca, Peru, failure of an upstream-type dam during an earthquake released more than 300,000 m^3 of tailings in a run-out of about 600 m, spilling into a nearby river and contaminating crops.

The main seismic design factors for tailings dams are the governing site seismicity, the properties of the tailings and the design and structural integrity of the containment. Site-specific earthquake-induced motions can be determined by regional seismic hazard analysis. In recent years, a revision of attenuation models of peak ground acceleration (PGA), peak ground velocity (PGV) and ground motion spectral accelerations for southwest Western Australia (SWWA) have been developed by Liang et al. (2008a) using a combined stochastic and Green's function simulation method. In comparison to other models currently used in SWWA, it is shown that the new attenuation models provide a more reliable prediction for available SWWA records than other models considered. Therefore, it is expected that the new equations are likely to provide more reliable seismic risk results in SWWA. The revision of seismic hazard analysis in SWWA, based on updated regional seismicity and the new ground motion attenuation model, has led to the need to re-assess the seismic safety of tailings dams previously built or planned. In this article, a new seismic hazard analysis for Perth metropolitan area (PMA) is reported. A case study of the performance of a hypothetical tailings dam in SWWA under simulated earthquake ground motion is presented.

Seismic hazard analysis

By way of example, the following analysis describes a hypothetical tailings embankment situated in metropolitan Perth. Data is available to conduct similar analysis for most locations in Australia and overseas.

The latest seismic hazard analysis for the PMA has been performed by Liang et al. (2008b) based on the new ground motion attenuation model and updated seismic source zones and their recurrence relationships in SWWA. Some extracted highlights are presented below.

The definition of seismic source zones and their recurrence relationships in SWWA have been made by Gaull et al. in 1990. It was in the Gaull et al. (1990) study that the earthquake hazard maps in the previous earthquake loading standard (AS1170.4-1993) were derived. Modifications presented in Hao and Gaull (2004) were applied to the original zone boundaries and recurrence relationships to include the most recent activity in the Burakin area. This updated seismic source zone map and recurrence relationship model are largely adopted in the study. The selection of an appropriate ground motion attenuation in relation for use in probabilistic earthquake hazard evaluation, is almost always critical to the results. In the Liang et al. (2008b) study, the attenuation model presented by Liang et al. (2008a) is applied to derive design response spectra of rock site ground motions corresponding to the 475 year return period earthquake (10% chance of exceedance in 50 years) and the 2475 year return period earthquake (2% chance of exceedance in 50 years) in SWWA.

The seismic risk for PGA

PGA on rock in PMA with a 10% and a 2% probability of being exceeded in 50 years are calculated (Figures 1 and 2). As seen in Figure 1, PGA on bedrock estimate ranges from 0.14 g in the northeast through to 0.09 g in the southwest for a return period of 475 years. This compares with a value of 0.09 g (1:475 year) given in the current Australian earthquake loading code. In other words, the code might underestimate PGA in the northeastern part of PMA by some 50%. The PGA for a return period of 2475 years is estimated in the range of 0.24 to 0.36 g.

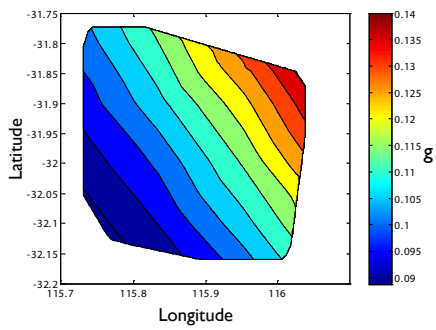


Figure 1 Rock PGA in PMA with a 10% chance of being exceeded in 50 years (equivalent to the return period of 475 years)

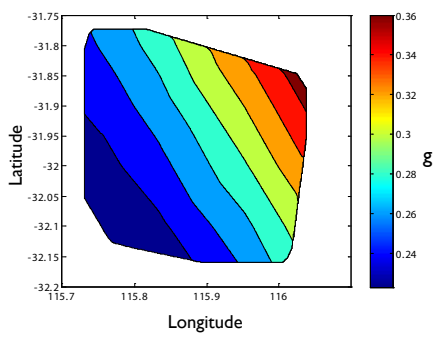


Figure 2 Rock PGA in PMA with a 2% chance of being exceeded in 50 years (equivalent to the return period of 2475 years)

Probabilistic seismic hazard spectra

The 5% damped spectral accelerations corresponding to the probabilistic seismic hazard levels for return periods of 475 years and 2475 years are estimated. The spectral accelerations observed at the central business district (CBD) of Perth (longitude 115.85° and latitude 32.00°) are used. The comparisons of the calculated spectral accelerations and those of the code spectrum are given in Figure 3. The comparisons show that the spectral accelerations on rock sites corresponding to the 475 year return period in general lie between the code spectrum of strong rock and rock sites within the range of 0 to 0.5 sec. However, the code spectrum might underestimate spectral acceleration in the period range of 0.5 to 2.0 sec.

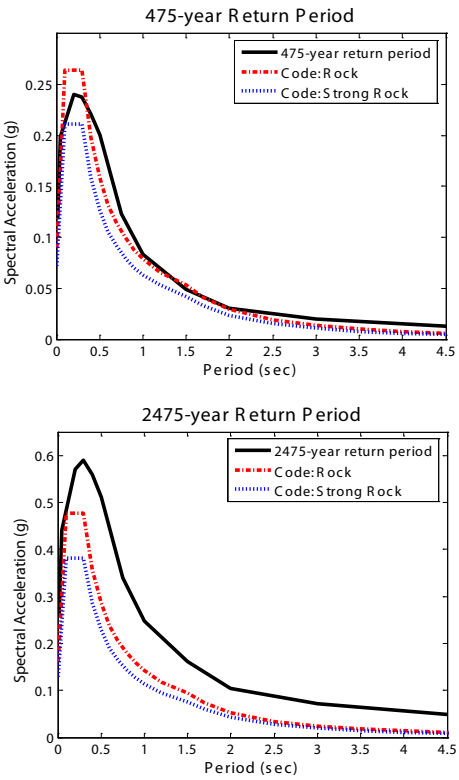


Figure 3 Comparisons of the calculated spectral acceleration and the code spectral acceleration

Earthquake ground motion simulation

To investigate performance of tailings dams under earthquake ground motion, seismic ground acceleration time histories corresponding to a 475 year return period are generated in this study. The PGA value of about 0.1 g and the proposed design response spectrum shown in Figure 3 are adopted in ground motion time history simulation as the tailings dam's site is assumed to be located adjacent to the PMA. The duration model presented by Liang et al. (2008a) is used to estimate the strong ground motion duration since it is based on a database of ground motions on rock sites located 5-200 km from the epicentre and magnitudes 3<ML<6.2 in SWWA. The time history duration value of about 12 sec for 475 year return period event is estimated. The simulated time history and the comparison of the response spectrum of the simulated motion and the target design spectrum derived from Liang et al. (2008b), are shown in Figures 4 and 5.

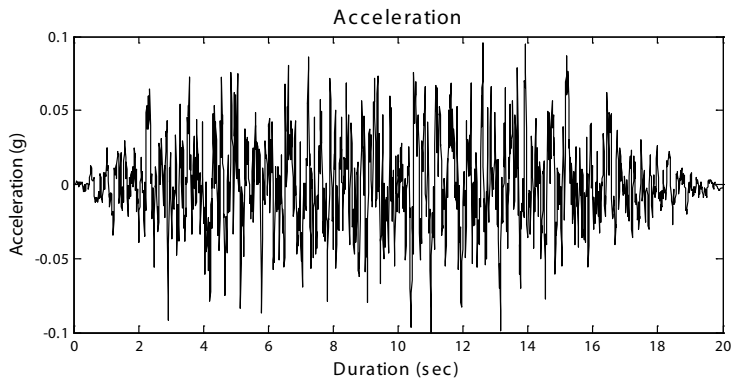


Figure 4 Simulated time history

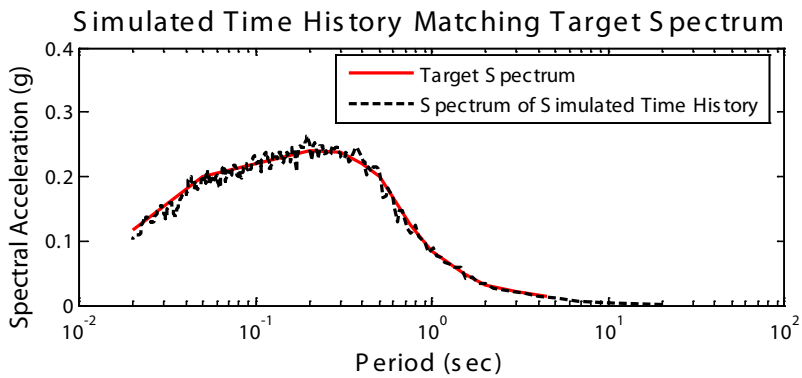


Figure 5 Response spectrum of the simulated time history and the proposed spectrum

"The main seismic design factors for tailings dams are the governing site seismicity, the properties of the tailings and the design and structural integrity of the containment."

Seismic response of a tailings dam embankment

Finite element model

A typical section of a tailings dam is modelled as shown in Figure 6. Two points (E and F) were selected to track the displacement histories at the crest of the dam during the earthquakes.

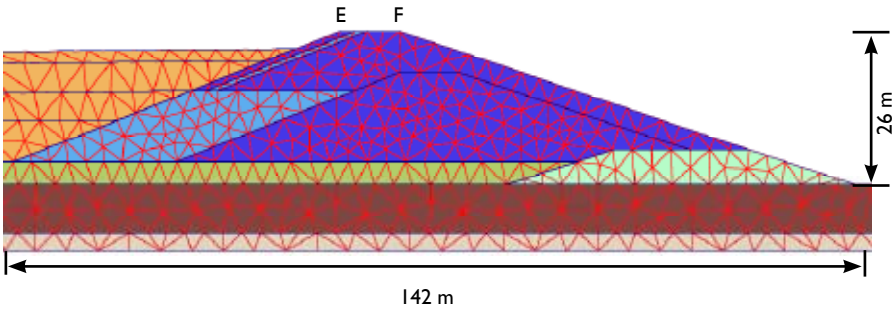


Figure 6 Typical section of tailings dam and mesh used in numerical analysis

The tailings dam embankment stability and deformation under the proposed earthquake ground motion are evaluated using finite element (FE) techniques. Numerical modelling was carried out using the Finite Element Package, Plaxis. The program allows for a two-dimensional analysis of elastic-perfectly plastic soils with a Mohr–Coulomb failure criterion utilising fourth order 15-node triangular elements. The calculation consisted of two phases. The initial stresses due to soil weight and pore pressures were generated and activated in the first calculation phase. In the second calculation phase, seismic loads were introduced at the model base by applying the simulated ground acceleration time history corresponding to the 475-year return period. The geotechnical properties adopted are listed in Table 1 and illustrated in Figure 6 for the section.

Table 1 Geotechnical material parameters adopted for seismic analysis

Description	Colour	Unit Weight (kN/m^3)	E	Cohesion (kN/m^2)	Friction Angle	Permeability (m/s)
Compacted tailings		20	12 MPa	0	30	1.16E-07
Embankment		20	12 MPa	1	28	4.05E-09
Existing tailings		18	2.5 MPa	0	26	1.16E-06
Foundation soil		19	8 MPa	10	28	2.32E-09
Fresh tailings		15	500 kPa	0	25	1.16E-05
Bedrock		24	30 GPa			1.16E-05
Rockfill		20	25 MPa	0	35	1.16E-04

Performance of a tailings dam embankment

Figures 7 and 8 show the horizontal and vertical displacements of selected points at the crest of dam, respectively. The numerical results indicated that the permanent horizontal displacements of the embankment are about 30 mm at point E and 12 mm at point F. The estimated permanent vertical displacements of the embankment are about 16 mm at point E and 2 mm at point F. Stability conditions are evaluated by progressively reducing the effective cohesion and the angle of shearing resistance by a factor of safety until large displacements of the dam are obtained. Using this procedure, the safety factor of post-earthquake stability of the dam is estimated to be about 1.48, indicating that the tailings dam embankment should not suffer significant damage from the 475 year return period ground motion.

"Reliable regional seismic hazard analysis and ground motion simulation are key issues in the analysis."

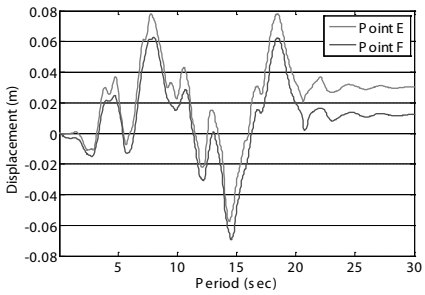


Figure 7 Horizontal displacement histories of selected points

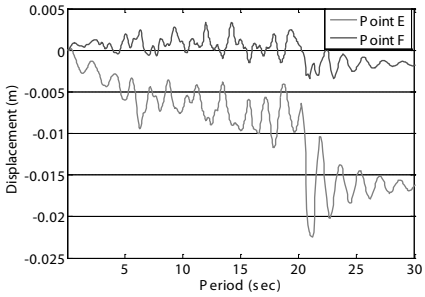


Figure 8 Vertical displacement histories of selected points

Conclusion

The seismic response analysis of the dam embankment is a complex problem that generally requires the use of dynamic methods of analysis. Reliable regional seismic hazard analysis and ground motion simulation are key issues in the analysis. The safety of tailings dams should be continually reviewed as more reliable regional seismic hazard results become available.

Article references are available from the ACG.

Jonathan Liang, GHD Pty Ltd

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Rehabilitation of bauxite residue — a soil science perspective

by Martin Fey and Talitha Santini, School of Earth and Environment, Faculty of Natural and Agricultural Sciences, The University of Western Australia

Introduction

Processing bauxite for alumina production carries a high environmental price tag because of the large amount of alkaline tailings that require disposal. There are some exciting challenges still waiting to be tackled despite the fact that the process involved is still much the same as a century ago.

As a first step in alumina production, bauxite is finely crushed and digested with caustic soda to dissolve the alumina. The residue of solid impurities, known as red mud, consists mainly of iron oxides and quartz together with accessory alumina and alkaline substances. Roughly 70 million t of red mud worldwide have to be disposed of annually (Energetics Inc., 2000), including about 20 million t in Western Australia. The most common method of disposal is 'cap and store', whereby mud is deposited in storage dams with containment walls being built using a sand fraction of the residue. Residue sand consists of particles, mostly of a diameter greater than 0.15 mm, that are separated during the washing process (Figure 1).

Figure 1 Fragments of bauxite residue sand, about 0.5 to 2 mm in diameter, from the Alcoa refinery at Kwinana, Western Australia. Photograph courtesy of Bill Wilson, UWA

The fine fraction (<0.15 mm and amounting to about two-thirds of the residue) is disposed of within the sand walls after dewatering to remove some of its solution phase (Cooling, 2007). Further consolidation is promoted by evaporative and gravitational dewatering, which takes considerable time (Nikraz et al., 2007). Leachate from the landfill is intercepted and returned to the alumina plant. While the residue disposal area grows, the outer sloping walls are given a cover of vegetation with a mixture of indigenous plant species.

The next step accompanying closure is to extend the vegetation cover across the top of the storage facility after covering it with a layer of either soil or residue sand, separated from the underlying mud by an impermeable barrier in order to isolate the residual alkalinity in the fine mud. Slow drainage of alkaline leachate from the mud will continue after the refinery closes and needs to be disposed of imaginatively. There are many strategies for reducing the environmental impact of red mud, with research and development currently focussed on neutralising the alkalinity by various chemical means such as carbonation (Cooling, 2007). In this article we present some of the principles of dealing with bauxite residue that emerge from a soil science perspective based on a review of current practices.

The pedological perspective

Soil scientists find bauxite interesting because: a) most bauxite deposits are, or once were, soils formed through the action of water and heat on parent rock, tempered by topography and biotic influences over time; and b) disposal of residue from the Bayer process involves trying to convert what is left of the bauxite back into soil again.

Turning solid waste into productive soil can be seen as more than simply creating a sustainable growth medium for a selection of ecologically desirable plant species. To maximise our confidence in creating a stable ecosystem, the new soil should be designed so that it becomes pedogenically consistent with its surroundings. Thus, theories of soil formation can inform the rehabilitation process. In thinking about how to turn residue disposal areas (RDAs) into subtle elements of landscape, we need to consider the factors (Jenny, 1941) and processes (Simonson, 1978) of soil formation, and the concepts of biomantle (Johnson, 1990), soil-landscape system (Huggett, 1975), and pedogenic threshold (Chadwick and Chorover, 2001). Treating the soil solution as though it were evolving brine in a saline lake (Hardie and Eugster, 1970) also has merit.

Bauxite residue becomes more interesting when ranked alongside other natural parent materials which may be even more alkaline in terms of total alkalinity (limestone and basalt, for example). Mineralogical and chemical subtleties then begin to take on special significance. Climate as a determinant of rehabilitation success, including its modification through irrigation in the early stages, becomes

especially interesting when we consider the effect it can have on soil formation. Vegetation assumes extra relevance when established not just for the sake of being there, but as an agent of soil development (Angers and Caron, 1998; Mendez and Meier, 2008). To do this successfully requires the application of soil fertility principles.

Soil fertility principles

Fast-tracking reclamation by planting and irrigating a perennial, fast growing grass such as kikuyu which is salt-tolerant (Mills et al., 2004), with a generous amount of water containing tailored nutrient formulations enriched with more nitrogen than the pasture can assimilate, may sound like a travesty of ecological principles. However, not if we can show how much quicker (and probably less expensively) a soil can be created this way as a means to an ecological end.

A workable neutralisation-amelioration strategy might consist of pre-treating with phosphoric acid and gypsum, then applying copious nitrogen fertiliser (ideally, ammonium sulfate) and other nutrients including trace elements, dissolved in the irrigation water. Part of the philosophy of this formulation is that acid inputs (apart from the phosphoric acid and that generated by bacterial oxidation of the nitrogen fertiliser) are, to a large extent, being achieved through photosynthesis of plants growing in situ rather than with expensively imported organic amendments (Fuller et al., 1982). We have termed this phytocarbonation, because atmospheric carbon dioxide is effectively transferred, via carbohydrate synthesis, to the soil through the respiration of plant roots.

A regular leaching fraction of water should be applied, carefully calculated to achieve a progressively deeper wetting front. Leaching of sodic alkalinity should be maximised under irrigation when combined with gypsum application. Such a combination is standard practice for converting naturally alkaline soils to productive farmland. Soil acidification under productive grass pastures can be as much as 3.4 kmol ha^{-1} $year^{-1}$ (Helyar and Porter, 1989).

Physical properties

Some current research is interested in manipulating particle size distribution to improve water (and nutrient) holding capacity (Courtney and Timpson, 2005; Anderson et al., 2007). Recombination of red mud with residue sand helps physically, but it has the disadvantage of increasing chemical amelioration costs because the fines are much richer in sodic alkalinity than the sand fraction. Addition of natural clay to residue sand has been considered as an alternative to red mud addition. The creation of a duplex profile with a B horizon of sodic red mud at a suitable depth in the residue sand has also been suggested (A. Fourie, personal communication). This would have the same water-retaining advantages as other fabricated layers in hyper-draining sands, such as asphalt barriers. How to get the mud down into sand which is already in place could be challenging, but simulating soil forming processes after establishing critical flocculation conditions (Frenkel et al., 1992) for dispersed mud might make it possible to create an argillic or luvic, B_t horizon by irrigating a dilute suspension of red mud into suitably pretreated sand. With the closure of RDAs, the thickness of the sand capping could be adjusted to achieve the best compromise between a duplex soil and a shallow sand aquifer.

The converse problem of intractable red mud could also be tackled imaginatively, one suggestion being to exploit the wide cracks which develop in drying mud by filling them with ameliorated residue sand, thus allowing roots to penetrate deeper down in sandy tongues than they would directly into the mud itself.

Treatments which primarily aim to neutralise or alleviate alkalinity, such as carbonation or gypsum addition, or improve nutrient availability, may also influence the physical properties of the residue through mineral dissolution or precipitation (Menzies et al., 2004; Nikraz et al., 2007), or tendency for flocculation or dispersion (Wong and Ho, 1993; Shainberg et al., 1988). This interaction can improve revegetation outcomes or impair them.

Mineralogical considerations

The mineralogical analysis of the residue sand shown in Figure 2 indicates that beside some secondary sodalite (alkaline aluminosilicate) the main constituents are remnant oxides and hydroxides from the bauxite: quartz, gibbsite, goethite and hematite. The iron- and aluminium-rich nature of the residue means that even the sand could have sufficient chemical reactivity to affect the rehabilitation process.

The sodalite represents the tip of the iceberg in terms of secondary minerals that can form in amended bauxite residue. Carbonation will yield a variety of carbonate minerals such as dawsonite, calcite and hydrotalcite. Ettringite, a well known mineral in cement chemistry, has the potential to form after gypsum addition along with tri-calcium aluminate. All of these minerals will affect the buffering of alkalinity to some extent and therefore the rate at which a more normal soil environment will be achieved.

Leachate management

The importance of topography is paramount. If there are low points in the landscape that cannot discharge solutes, then even if neutralisation is achieved to a large degree, the soluble salts generated will accumulate and result in barren patches devoid of vegetation. In addition,

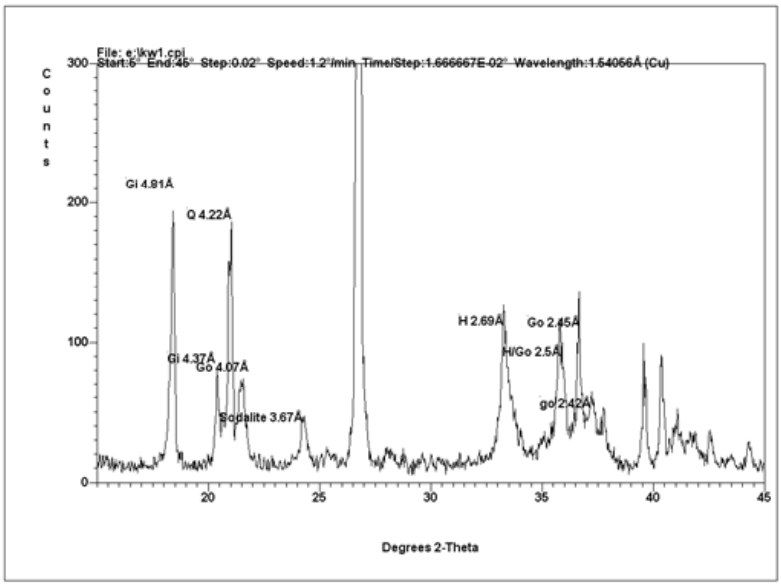

Figure 2 X-ray powder diffraction pattern of the residue sand from the Kwinana refinery shown in Figure 1, obtained using Cu Kα radiation (Peak labels indicate d spacing and mineral identification as follows: Gi – gibbsite, Go – goethite, Q – quartz, S – sodalite and H – hematite)

drains would need to be installed to ensure the discharge of leachate.

Alkaline leachate will continue being generated but will no longer be recycled after refinery closure and its potential value needs to be researched so that it does not become an environmental burden. Seawater neutralisation research in the past has included evaluating the synthesis of hydrotalcite-like reaction products (Palmer et al., 2009) in which Mg and bicarbonate from seawater combine with Na aluminate in red mud effluent. These compounds, also known as anionic clays because they hold exchangeable interlayer anions instead of cations, could be synthesised from red mud leachate and seawater bittern, but with added nitrate as the principal interlayer anion (e.g. Gillman and McCallum, 2004), thus producing a soil amendment that might not only add water-holding and wetting characteristics to coastal sandy soils but also serve as a slow release (and therefore environmentally considerate) nitrogen fertiliser.

Learning from experience

Strategies for achieving desired rehabilitation outcomes should be based upon prior knowledge of relationships between amendments and responses under similar conditions. Relationships between amendments and the response of bauxite residue properties can be identified by carrying out a detailed assessment of progress that has been made on sites that have been revegetated for a decade or more. Assessment of vegetation cover, and soil quality in relation to a range of soil properties and site history can be instructive. Techniques such as quantile regression help to identify, through boundary lines that encapsulate plots of biological response to environmental determinants (Cade and Noon, 2003; Schroder et al., 2005), which factors may still limit the progress of revegetation. Even the manner in which biodiversity is affected by soil properties can be unravelled (Mills et al., 2009).

Another valuable activity on older sites is to examine profiles for signs of soil horizon development. Incipient pedogenesis is detectable in young regolith, especially with techniques such as scanning electron microscopy (SEM) and energy dispersive spectrometry (EDS). Who knows what fledgling films of what soil scientists call ortstein, calcrete, plinthite or duripan lurk deep in the oldest residue sands? If there are signs of such materials, then already we are much wiser about pedogenic trajectory. Silica is particularly interesting because by all accounts it should be highly mobile under the alkaline conditions that prevail but it seldom appears as a solute in routine lab reports. We speculate that manipulating its concentration could prove useful in controlling alkaline mineral buffers and therefore the pedogenic trajectory of the residue.

Acknowledgements

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Article references are available from the ACG.

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Tactical versus strategic monitoring of open pit mines

by Phil Dight, Australian Centre for Geomechanics

Monitoring in open pit mines is typically undertaken by measuring slope deformation. This takes the form of visual monitoring, crack monitoring, prism monitoring, surface and borehole extensometers, piezometers and, more recently, radar, photogrammetry and light detecting and ranging (LIDAR). The success with locating prism monitoring and surface/subsurface instrumentation relies on the experience of the planner to anticipate where instability may arise, and on access to instrumentation installation. Prism monitoring has become pervasive as reflectors are cheap, data can be recorded using automatic theodolites and the movement interpreted using standard software. The monitoring data is often used to calibrate numerical models where the ability to determine the rock mass modulus from any other method other than empiricism is extremely difficult. The implied assumption with this type of monitoring is that the rock behaves in a relatively homogeneous manner and does not respond to progressive movement on individual structures. Prism monitoring is generally a strategic form of monitoring when initially installed. It can then be used tactically should instability be identified and more intense arrays installed with the requisite increase in monitoring frequency. A limitation can be the distance between the survey station and the pit wall, the atmospherics and the spacing of prisms among other things. Radar, photogrammetry and LIDAR have the ability to provide more extensive surface mapping of the deformation. This removes some of the uncertainty of where to monitor and what to monitor inherent with the placement of prisms. Photogrammetry and LIDAR would also fit more into the strategic monitoring category because the turnaround in obtaining information for analysis and interpretation presently cannot be completed quickly without dedicated personnel. On the other hand, the radar has proven very successful in tactical monitoring of impending instability with very fast scan times and response to developing instability. The initial limitation of range has been largely overcome for large pits. Its downside has been the inability to determine the vectors of movement and, hence, the placement of the radar requires intelligent use of location and interpretation to ensure alarms are correctly assigned. As a tactical monitoring system, it would satisfy the mining requirement of an early warning system without the need to interpret the mechanism.

The interpretation of the mechanism/s of failure relies on understanding the vectors of movement, where the movement is occurring, and the structural controls. This aspect is of more importance to the geotechnical engineer than necessarily the mine operator: that is, until the question is asked; "Could it happen somewhere else and how quickly it is likely to occur?"

It is not possible from surface monitoring to determine the depth behind the surface unless there are obvious expressions of tensile cracking behind the face from which an estimate of the volume of failure can be made. Where movement is expected on individual structures or contacts, subsurface monitoring is required. Early movement can often be detected from changes in the pore pressure recorded by the piezometers or on extensometers/inclinometers, before any surface expression can be detected. However, it is hard with this approach to consider volume measurements of the rock mass as the systems are essentially point locations. The only way this can be done is to use microseismicity/acoustic emission to listen to the rock. Every movement or new/existing crack extension will emit noise. This will occur well before any subsurface or surface movement can be detected. The level of detail available from this monitoring is extensive; but it depends on the signal to be measured, the signal to noise ratio and the likely attenuation. Tactical seismic monitoring looks at noise counts and likely source location, provided the array is well designed. Such a system could comprise an acoustic emission network (often used in highway monitoring), or triaxial/uniaxial geophones located in drill holes and on the surface of the slope. Examples of where tactical seismic monitoring could be implemented would be along, or adjacent

to, pit entrances/haul roads, railways, or adjacent to expected failure zones. When looking at the implementation, consideration needs to be given to the possible sources of energy that could initiate movement - for example, local seismicity (typically events from Richter? -1 to 2) which can be picked up with say a regional seismic network that has now been implemented at several mines throughout Western Australia, or from blasting in the near field or intermediate field adjacent to the slope being monitored. The importance of the regional seismic network should not be underestimated. A failure that occurred a few years ago happened one hour after a seismic event of 2.4 was recorded 70 km away. What is not known is whether the mine was experiencing local events smaller than 2.4 because there was no system in place to measure such events. The recent seismic activity at Beacon, Western Australia, which attracted much attention in the press, appeared to be localised. However, seismic activity of events less than 2 were recorded on the regional seismic network at mines as far away as Lawlers, Darlot, Golden Grove, Leinster and Agnew.

Strategic monitoring would involve triaxial sensors located in the boreholes looking for the initiation or extension of existing cracks. In open pit mining the stress drop due to unloading the slope is low. As a consequence, the likely size of these new features is expected to be 1 to 100 m². This translates into seismic events of approximately -3 to -1, with many more events in the range -4 to -3. As such, the sensors need to be located at centres of approximately 100 m. It is recommended that the majority of the sensors are triaxial. This ensures that good source location and waveform can be achieved for subsequent analysis. It is also recommended that some sensors are not amplified so that larger seismic events (blasting and earthquake) can be recorded with the full waveform. The critical information being sought in this instance is the dominant frequency of the event leading to possible instability and the resonant frequency of the slope/structures. With information on the type of fracturing, and the location of the fracturing or movement, a much better understanding of the developing mechanism can be achieved.

The ACG has made good progress with developing such systems for the tactical and strategic monitoring required at mines. The first stage of the ACG's High Resolution Seismic Monitoring for Open Pit Mines project has now been completed. One of the major conclusions from this study was that most of the movement detected was the result of extensional cracking. This was unexpected as it had been thought beforehand that most of the movement would be associated with shear. Such cracking occurs below the base of the pit (often seen as low angle structures), and in the pit wall where the structures are more typically oriented at 15 to 30° from horizontal. These cracks are expected to develop in response to the change in stress resulting from the open pit mining. This can happen in a relatively low stress environment due to the unloading of the slope, while at the same time the stress acting normal means that the wall must deviate under the pit. Very little laboratory testing has been undertaken on the unloading response of rock subject to constant axial load. The research work is ongoing to develop predictive techniques to define when these features should occur. With the advent of new fracture systems which may not align with the structural model, this amounts to a change to the design assumptions. It also may assist explaining the observation of the fracturing of rock bridges. An essential input to the prediction is the understanding of the in situ stress field which is not commonly measured in open pits. The research work also suggests that the extensional type of cracking becomes more prevalent for pits deeper than 200 m below surface and nearer the pit wall. Certainly it is expected that the situation will be significantly more likely in the very deep pits currently in design (up to 1000 m below surface). The timeframes for these pits to be developed is a long way into the future, i.e. up to 20 to 40 years. It is unlikely that even a strategic monitoring system would be implemented at this stage for such long lives. Nevertheless, the only way that volume measurement of the rock mass can be undertaken, is using microseismicity. It is then up to us to understand the mechanisms that are leading to the noise and develop interpretations on the likely risks that mining operations will face in the future. The ACG would like to hear from any operation which would be interested in following up on these developments. Please contact Phil Dight via acg@acg.uwa.edu.au for further information.

Phil Dight,
Australian Centre for Geomechanics

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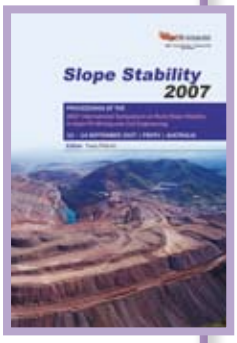

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Underground mining — geotechnical aspects of mining method selection

by Hakan Urcan, Coffey Mining Pty Ltd

Introduction

Mining projects, particularly in massive mining, require high capital investment to access and prepare the ore deposit for extraction. Hence, selection of an appropriate mining method prior to the commencement of access development is of paramount importance. If the mining method selection procedures are not applied properly, the viability of the project is at high risk, because the ability to change methods once extraction has been initiated is difficult and usually increases capital expenditure.

The selection of a suitable underground mining method for an ore deposit involves consideration of a diverse set of criteria. In this article, the geomechanical aspects of underground mining method selection are discussed.

Factors that influence mining method selection

In an underground mining environment, the geotechnical conditions are important as well as the stress regimes acting on the excavations. Under these circumstances, the discontinuities in the rock mass, specifically jointing, the excavation size, the induced stress and the rock mass strength are all critical aspects. Variations in the rock mass geotechnical properties will have significant effect on the excavation design and stability, highlighting the importance of identification of geotechnical regions, and local variations, on a routine basis. The ability of production staff to identify changes in geotechnical properties and to pre-empt implementation of inappropriate designs is important.

Within this environment several factors must be considered:

- Excavation size.
- Interaction of excavations.
- Variability of geotechnical regions.
- Pillar design procedures.
- Rock reinforcement techniques.
- Backfill.
- Blast vibration.
- Mining methods.
- Economic factors.

Excavation size and interaction

Increased excavation size in any jointed rock mass, increases the probability of a gravity induced failure occurring once the excavation intersects sufficient joint sets. Excavation size estimates are generally initially obtained through rock mass ratings and via empirically determined stability graphs, as well as through numerical modelling.

Numerical modelling is often required as an increased excavation size escalates induced stress changes and the probability of stress related failures. The approximate distribution of stresses around multiple openings may be obtained by the application of the principle of superposition. A general rule that arises is that openings can be considered to have no influence on each other if separated by a distance of three to four times their representative dimensions. As excavations become larger it is more difficult to avoid this interaction. Hence, mining induced stresses change (both increase and decrease) in the areas that are periphery of stopes and may include associated development such as ramps.

Variability of geotechnical regions

Owing to the sensitivity of excavations to the variation in geological discontinuities, it is important that the geotechnical regions be established across the mining reserves. Each region defines an area of similar geotechnical characteristics. This will then facilitate the correct support and pillar design aspects. Geotechnical mapping and rock mass classification should be continued on a routine basis when development is in place to establish trends, and to rapidly identify changes that may occur in the development's conditions.

Pillar design procedures

The stress-strain curve of a pillar is shown conceptually in Figure 1. The initial straight line portion up to the yield point corresponds to the elastic response of the pillar. The slope of this portion of the curve is the effective Young's modulus of the pillar. The yield point indicates the onset of localised inelastic behaviour, or failure of some material in the pillar. After the yield point, the pillar exhibits strain hardening until the peak strength is reached. Load shedding then occurs until a residual strength is reached.

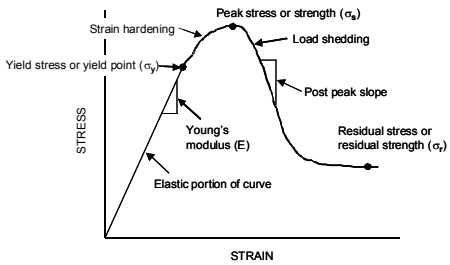

Figure 1 A qualitative diagram of the complete stress-strain curve of a pillar

It is generally accepted that the strength of pillars is a function of:

- The strength of the intact rock of which the pillar consists, suitably downrated to take into account the scale effect.
- The geometry of the pillar, including both its shape and its width to height relationship.

Several pillar strength formulae have been developed to take into account these factors. One such formula is:

$$\text{Pillar strength} = K.W_{eff}^{\alpha}/H^{\beta}$$

Where,
'K' is the 'large' scale strength of the rock mass, say of every 1 m^3;
'Weff' is the effective pillar width;
α and β are constants.

Since pillars are often irregular in shape, particularly in massive mining and in situations where orebody thickness is variable, pillar widths and shapes may vary considerably. The effective pillar width, which has been included in the above formula, can take these variations into account.

$$W_{eff} = 4 \times \text{pillar area/pillar perimeter}$$

The values of the constants in the above equation will vary for each mine, and will be best determined by back analysis of the observed behaviour of the mine's pillars. However, such back analysis is not possible at the project development stage, so that the design of pillars must be carried out without this information. In such cases it is recommended that the α and β exponents in the formula are those derived for hard

rock pillars by Hedley and Grant (1972). It is also recommended that the value used for K is Laubscher's design rock mass strength (DRMS). The pillar strength formula recommended for use in design and stability evaluation therefore becomes:

$$P_{strength} = DRMS.W_{eff}^{0.5}/H^{0.75}$$

It must be noted that this is a general formula, and its applicability to particular mining cases needs to be verified by observation of pillar behaviour and by correlation of this observed behaviour with the expected behaviour determined from the application of the above formula.

Experience has indicated that the DRMS usually falls in the range of 20 to 50% of the UCS of the intact rock and is commonly about 30% of the UCS.

The pillar safety factor (SF) is then determined by dividing the pillar strength by the load carried by the pillar:

$$SF = P_{strength} / P_{load}$$

A desired safety factor pillar design derives from the estimated pillar strength and a comparison of it to the estimated pillar load. Pillar load can be determined with some degree of certainty through elastic and inelastic modelling techniques, coupled with in situ measurements, back analysis and fine tuning of numerical models.

(a) Elastic pillars. If the average load of a pillar never exceeds the strength of the pillar (σs in Figure 1) such a pillar is defined as 'elastic', though the terms 'non-yield' or 'rigid' are sometimes used. If the load acting on the pillar is greater than the yield stress σ_y, then signs of localised fracturing of the pillar will be observe. However, this is not failure of the pillar as a whole because the strength of the pillar has not yet been exceeded. Strictly speaking, any fracturing is non-elastic behaviour, but because the strength of the pillar is not exceeded, the pillar as a whole can be deemed to be elastic.

(b) Crush pillars. These pillars are in a post-failure state, i.e. the peak strength has been exceeded and the residual strength of the pillar has been attained. Crush pillars are able to support much less load than elastic pillars. Strictly speaking, a normal safety factor applies only to non-yield (elastic) pillars, which are designed to support the hangingwall to surface, and in which the applied load on the pillar should always be less than the strength of the pillar. Crush pillars are designed with SF<1 such that the strength of the pillar is exceeded at an early stage.

(c) Yield pillars. These are a special case of crush pillars, designed such that the pillar is able to maintain an approximately constant stress after the yield point.

Rock reinforcement techniques

Rock reinforcement techniques generally need to cater for gravity and mining induced stress conditions, not for the dynamic loading associated with seismicity. Support systems can again be tailored to suit rock mass conditions and designs optimised to account for prevailing jointing. Support problems that arise are likely to result from poor ground conditions associated with alteration zones and weathering.

Stress problems that are encountered are usually associated with drawpoints, which by their nature are subject to stress concentration. The intensity of the support problem is often exacerbated by dynamic loading (due to blasting) and blast damage resulting from secondary blasting operations.

Backfill

Backfill systems need to be optimised within its specific approach and that may range from cemented rock fill through to classified tailings. When a key consideration is secondary extraction, the height of free standing fill face, the lateral closure during extraction, and the fill stability which then is required have to be taken into account.

Contrary to the application of backfill in tabular mining, in which significant stresses are generated in backfill, backfilling in massive mining operations usually takes the form of void filling. When bulk mining is considered, especially with extraction in both lateral directions, stress effects vary within the open stope fill.

The primary purpose of the fill is to stabilise large volume openings and prevent failure on a macro scale, and to facilitate higher extraction ratios. In blasthole stopes the backfill must be sufficiently strong to provide general mine stability as extraction proceeds. In cut-and-fill operations, backfill is required to provide a readily available working surface as well as bulk for overall stability.

Blast vibration

The impact of large blasts within a major stope can produce damage on critical excavations in close proximity to the stope as there may be an element of dynamic loading occurring. Similarly, the effect of blasting on freshly placed backfill should be assessed, as there is a possibility of liquefaction and of slump.

Mining methods

Mining methods need to be selected, not only to meet the targeted tonnage through-put requirements, but also for suitability to the rock mass conditions and the orebody geometry. Massive underground mining systems can be broadly classified as:

1. Naturally supported
   - Room and pillar (panel and pillar).
   - Sublevel and longhole stoping.
2. Artificially supported (backfill)
   - Cut-and-fill.
   - Vertical crater retreat (VCR) stoping.
   - Shrinkage.
3. Unsupported
   - Longwall mining (this is excluded from the further discussion because of its limited application in massive orebodies).
   - Sublevel caving.
   - Block caving.

Economic factors

Selective methods are usually designed for high ore recovery with minimum dilution whilst mass methods accept ore loss and dilution. Obviously, the higher the grade, the less desirable is the use of methods that result in high ore losses.

The methods indicated can be listed in order of increasing operating costs as:

1. Block caving.
2. Sublevel open stoping/VCR.
3. Sublevel caving.
4. Room and pillar.
5. Shrinkage stoping.
6. Cut-and-fill methods.

"Mining methods need to be selected, not only to meet the targeted tonnage through-put requirements, but also for suitability to the rock mass conditions and the orebody geometry."

Figure 2 summarises the influence of geomechanics in the selection of an applicable mining method for the varying rock mass quality using the Q index for the different zones of the orebody.

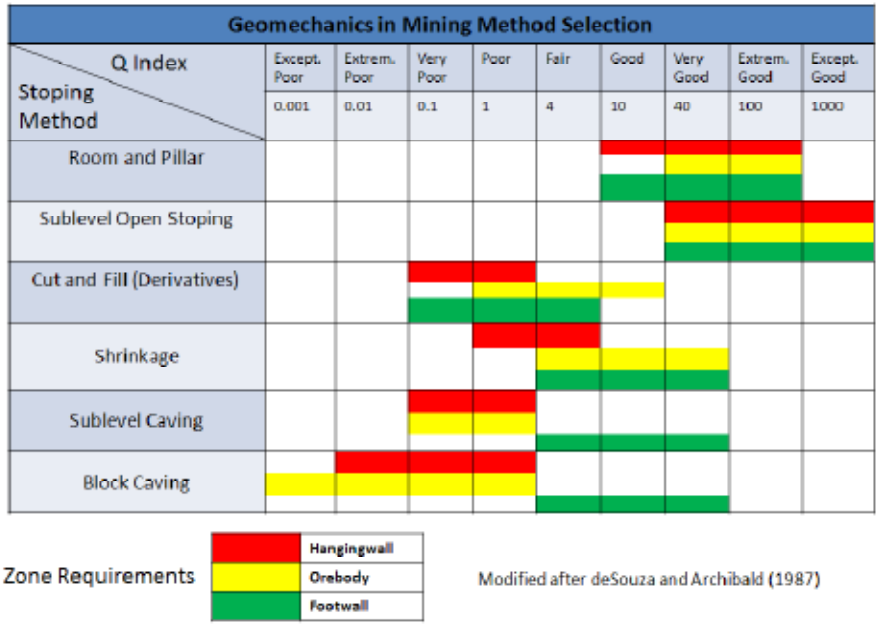

Figure 2 Geomechanics in mining method selection

Summary

In this article, an attempt was made to provide a basic outline of the needs for geotechnical considerations of various mining methods. Each deposit will have its unique features and a mining method should be selected accordingly. The geotechnical and design considerations for a mining method selection can be summarised as:

- Geotechnical investigations: core logging, outcrop mapping, detailed geological and geotechnical mapping, rock mass characterisation, rock types, location, boundaries, contacts, etc.
- Geometry of orebody.
- Depth.
- Structural features; faults, dykes, joints.
- Groundwater and surface hydrology (rivers etc.).
- Geomechanics rock mass classification of orebody and surrounding rock mass.
- Rock mass strength.
- Regional stress.
- Grade distribution in ore and dilution zones.
- General rock mechanics investigations: caveability of orebody and hangingwall, fragmentation analysis, rock mass stability estimates, etc.
- Mining sequences.
- Mining induced stress.
- Numerical modelling outputs.
- Production planning schedules.

Design considerations

- Identify potential failure mechanisms of system.
- Size of openings: stable open spans: empirical, rock mass classification systems, beam theory, keyblock theory, computer modelling, etc.
- Size of pillars.
- Mining sequence.
- Backfilling.
- Support.
- Equipment.
- Experience.

This is an edited article. For the full article and list of references please contact the ACG.

Hakan Urcan,
Coffey Mining Pty Ltd

First International Seminar on Safe and Rapid Development in Mining and International Forum on Development Productivity

The ACG's First International Seminar on Safe and Rapid Development in Mining was held in Perth, Western Australia, 6–7 May 2009 and attended by more the 75 local and international mining professionals. Prior to the seminar an International Forum on Development Productivity was held which included a workshop on how Australian mining can look to improve its productivity record while maintaining safety. For those who attended, the workshop proved to be highly successful. Speakers at the International Forum were Frode Nilsen, project director, Leonhard Nilsen and Sonner AS, Norway; Tim Gilbert, general manager, Australian Mine Assets, Norilsk Nickel Australia; Frank Greblo, principal mining engineer, AMC Consultants; Brett Ascott, senior mine planning and geotechnical engineer, Newmont Asia Pacific; Kobus Du Plooy, senior geotechnical engineer, SRK Consulting (Australasia); Chaim Sproles, mining engineer, Newcrest Mining – Cadia East; and Patrick Burke, group manager safety compliance, Macmahon Underground. The forum was facilitated by Tim Gilbert; Mark Adams, chief operating officer, Barminco; and Phil Dight, ACG.

Many issues were discussed during the forum and the associated workshop. Focus was placed on identifying and using appropriate equipment to maximise development productivity through maximising face utilisation. The key is to utilise the face to advance development. To achieve this, some Norwegian tunnelling operations use:

- Side tipping loaders.
- Trucks sized to the tunnel enabling drivers to perform a 3 point turn in the tunnel.
- Drilling equipment with navigation which is fully instrumented to implement a plan of the drill layout, including the survey of the hole collar and hole toe.
- Use of 5.8 m long drill steels (where ground is appropriate), logging of the drilling performance, blasting using emulsion, rapid setting shotcrete/fibrecrete (0.5 hr in freezing conditions).
- Most importantly, a dedicated, highly skilled workforce regularly achieving over 100 m of development per week in openings up to 38.5 m². The best week achieved 150.1 m of advance.

An important aspect of the tunnelling experience in Norway is the risk sharing principle (Figure 1).

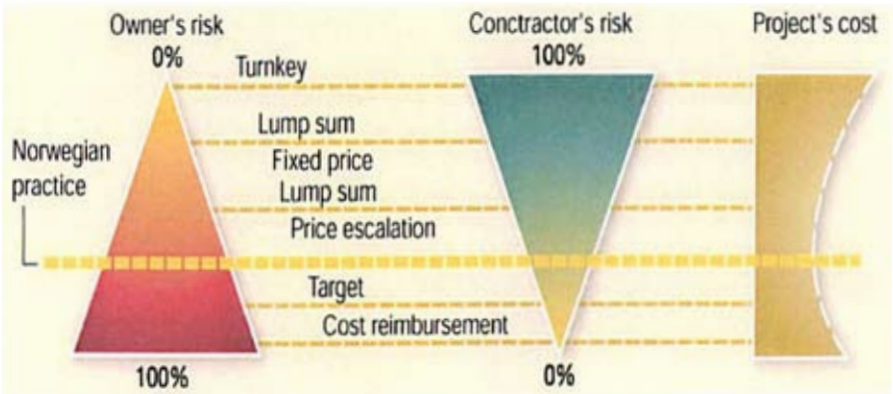

Figure 1 Leonhard Nilsen and Sonner AS risk sharing principle (Kleivan, 1987; Blindheim and Grøv, 2003)

Tim Gilbert noted that tunnelling can often be considered in a different vein to mining. The big difference is often attributed to the role of the client (in tunnelling it is often government and in mining, free enterprise). So, notwithstanding that the development rate in tunnelling is often higher than in mining, the cost per metre is not considered as important as the overall total cost. The objective of the development needs to be well understood. In some cases the development of a decline to achieve early access to ore and satisfy shareholder expectations can often result in a development which is compromised because production cannot be achieved without further implementation of stockpile bays etc. The five major reasons inhibiting development productivity are:

- Knowledge.
- Planning.
- Face size.
- Resources.
- Organisation.

A common point made by all attendees was the amount of time spent on ground support (27 to 35%). In one project, overbreak constituted 15%. Hence, reduction of overbreak by using appropriate drilling and blasting practices can save an enormous amount in development time. Much discussion revolved around using the Six Sigma process and the LEAN processes to improve productivity.

Frank Greblo presented the AMC benchmarking experience and the reasons behind why benchmarking has proved valuable to many operations trying to identify where they set within best practice. The development rate of 5.25 m per day was widely quoted by other presenters.

Brett Ascott provided a presentation on the MineGem implementation at Jundee. The success of the system was demonstrated through improved productivity, better working conditions, and less maintenance issues. Performance improvements have typically been 27 to 35%. It has also lead to a reduction in the amount of equipment required to achieve

the same productivity. Hot seating can be undertaken and the machines operated immediately following blasting which significantly improves utilisation. The approach is sensitive to good dust control and well maintained roads.

Chaim Sproles presented the Cadia East experience where they have adopted many of the techniques and equipment utilised by the Norwegian Tunnelling Society. Newcrest has achieved daily production rates of 9 to 11 m. While this is not as high as the 18 to 20 m per day achieved by the tunnelling people, it is up to double the rate identified in the benchmarking studies noted previously. Measurement of face utilisation at 15 minute intervals was one of the keys to understanding duration and variability of each portion of the development cycle with the aim of reducing the variation of each then minimising the duration. In many ways, the presentation by Chaim encapsulated the key points raised by the first four speakers.

Kobus Du Plooy from SRK made a significant contribution to the discussion by identifying what was necessary to reduce the 27% of time spent in ground support. This can be achieved by a properly developed ground control management plan. Picking up on the Scandinavian experience from Hakan Schunnesson where they have fully instrumented drilling equipment to record penetration rate, feed force, percussive pressure, rotation pressure, rotation speed, damp pressure, water pressure and waterflow, the ground conditions ahead of mining can be captured in real time allowing for changes to ground control requirements.

The last presentation was by Patrick Burke who challenged those present with the idea of alternative power supply (electric assist etc). The ideas were not necessarily new and indeed many have been adopted in coal mining. The main issue is the supply of diesel going into the future. The carbon footprint of open pit mines will become a significant issue if government succeed in implementing emission trading schemes. Underground mines have a significantly lower carbon footprint, however, it was suggested that mining companies with a long future should certainly be looking at alternative power supplies.

A workshop examined, "What does Australia need to do to improve productivity?". Key concerns were:

Planning

A strategic rather than a tactical mindset needs to be adopted. This includes retention of skilled personnel, looking for highly educated workers with a broad skill set, and moving from batch systems to continuous production systems.

People and systems

- Development appropriate key performance indicators to motivate operators — measurement analysis, interpretation and feedback.
- Development of robust cost benefit model to aid education and communication.
- Real-time analysis of information and communication.

Maximising face utilisation

- Use of equipment built for purpose.
- Utilisation of longer rounds.
- Development of longer range geotechnical mapping.
- Developing proactive maintenance.
- Design of ventilation to include smoke clearing and hence increased face utilisation.
- Identification of superfast acceleration for shotcrete.
- Monitoring road maintenance.
- Maintenance of design profile and accuracy.
- Embracing computer technology to improve quality and eliminate rework.

Measurement

- Undertaking benchmarking studies between the civil and mining industries to identify areas that could improve productivity.
- Develop appropriate key performance indicators to motivate operators — measurement, analysis, interpretation and feedback.

The overwhelming response from the participants was that the workshop was very valuable. Unfortunately, the global financial crisis meant some mining operation personnel were unable to attend the forum and seminar. Both the seminar and forum facilitated attendee interaction at a much closer level and provided an excellent opportunity to exchange experiences.

The ACG acknowledges the generous support of the forum and seminar sponsors: StrataCrete Pty Ltd, Dyno Nobel Pty Ltd, Geobrugg Australia Pty Ltd, SimMine, Geofabrics (Australasia) Pty Ltd, Maccaferri and Australia's Mining Monthly.

Article references are available on request from the ACG.

The ACG's SRDM seminar attracted more than 75 local and international mining professionals

Geocomposite faced rockfill — an innovative approach to tailings dam water-proofing in the absence of clay

by Craig Noske, ATC Williams Pty Ltd (previously Australian Tailings Consultants)

Introduction

Designers of dams, and particularly tailings dams, generally agree that the most cost effective solution is to maximise the use of natural materials available on-site. But what happens when a design simply cannot be made to work with the available materials? This is the situation recently faced by ATC Williams at the Sar Cheshmeh mine tailings storage in central Iran.

This article outlines the issues surrounding the design and construction of a 1000 m long, 40 m high downstream raise to the existing 75 m high tailings storage embankment, using an exposed geomembrane sealing system (GSS) on the upstream face as the water proofing element. The innovative means of anchoring the GSS to the embankment is the first time such a system has been employed anywhere in the world.

Background

Sar Cheshmeh is a large existing copper mine, originally commissioned in 1980. ATC Williams were engaged in 2001 to design tailings management and water saving options for a proposed production escalation involving almost 1 billion t of tailings over 31 years. The conclusion of this study was a scheme comprising paste thickeners and down valley tailings discharge into the existing raised storage, where decant water will collect in the vicinity of the retaining embankments.

The tailings storage prior to raising is shown in Figure 1. The existing tailings dam has an area of approximately 500 hectares, with a 125 hectare decant pond (to the left in the figure). The primary retaining structure is the 75 m high Main Embankment (shown in the foreground).

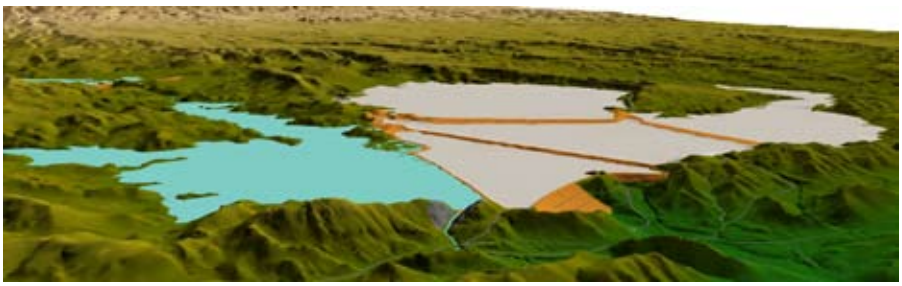

Figure 1 Sar Cheshmeh tailings storage prior to raising

Figure 2 Main embankment prior to raising

The Main Embankment prior to raising is shown in Figure 2. It was designed and constructed in the late 1970s, with provision for subsequent downstream raising by a maximum height of 20 m. Due to the changing political environment in Iran at the time, the documented design, construction and performance history of the embankment are incomplete. It was, however, evident from surveillance audits that various repair works were carried out during the 1980s and early 1990s to correct seepage, settlement and cracking issues relating to high pond levels.

An extensive geotechnical investigation program, comprising geological mapping, boreholes, test pits, seismic traverses and laboratory testing was undertaken by ATC Williams. It was concluded that the internal zoning and foundations were as shown in the inferred cross-section in Figure 3. Key features include an inclined clay core and an outer colluvial gravel shell. A number of clayey bands were encountered in the downstream shell, indicating poor construction control.

Raise design overview

Iran is one of the more seismically active regions in the world, evident by the peak ground acceleration for the Maximum Design Earthquake of 0.8 g. The critical design case for the 40 m raise included an allowance for up to 20 m of water ponding against the upstream face. The conventional downstream raising approach is to extend the existing clay core; however, the location of the raised core is constrained by the existing inclined core. The raised core effectively becomes an upstream diaphragm with little rockfill cover to act as a surcharge during seismic loading. Stability analyses showed that the resultant seismic factor of safety was unacceptable. Compounding the issue is a shortage of suitable clay-based soils in the semi-arid Sar Cheshmeh area, and that the borrow sources used in the original embankment construction are now covered by tailings.

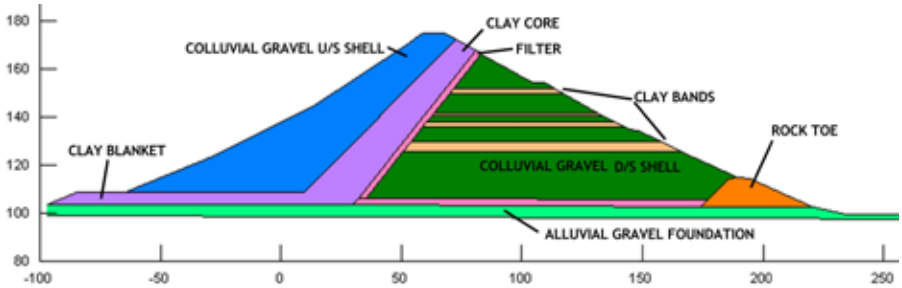

Figure 3 Main embankment inferred cross-section prior to raising

It was concluded that clay materials could not be used for the raise, and a detailed study of alternative water retaining elements was conducted. Options investigated included asphaltic cores, and bituminous and polymeric membranes. A cross-section consisting of a thin geomembrane sealing system on a rockfill embankment was subsequently adopted, on the basis of it being a more stable, efficient and buildable arrangement.

Geomembrane sealing system

A number of GSS options were identified, and the properties of each were analysed with respect to the following detailed design and performance criteria:

- Hydraulic properties.
- Mechanical properties.
- Durability properties.
- Practical requirements (cost, ease of installation, cover requirements).
- Precedence (of the GSS itself, and the installation method and contractor).

It became clear that from a construction, performance and cost perspective, an exposed cover would be preferred. It was then concluded that a PVC geocomposite represented the only GSS solution which would meet the necessary requirements for Sar Cheshmeh. Exposed HDPE geomembranes were found to have no application for this type of installation, whilst a covered HDPE system was found to be unviable from a construction and economic point of view.

A competitive tender process was initiated for the detailed design, manufacture, supply and installation of an exposed PVC geocomposite GSS. On the basis of demonstrable experience with respect to both materials and installation expertise, the contract was awarded to Carpi Tech, a Swiss/Italian specialist waterproofing company.

The selected PVC geocomposite was SIBELON CNT 4400 (manufactured exclusively for Carpi Tech by Flag SPA), consisting of a 3 mm thick PVC geomembrane, heat-coupled during manufacturing to a 500 g/m^2 non-woven polypropylene geotextile. This material has been developed specifically for dam applications in exposed environments. Its manufacture and materials are specifically formulated to impart to the geomembrane the tensile and weathering properties necessary to survive such conditions in the long term.

Embankment raise configuration

The embankment raise configuration, as shown in Figure 4, consists predominantly of good quality, quarried and blasted igneous rockfill, from a nearby source located as part of the geotechnical investigation. The side slopes for the raise are 1.5 : 1 (horizontal:vertical) upstream, and 2 : 1 (overall) downstream, made up of 1.75 : 1 slopes and 5 m wide benches every 20 m of elevation.

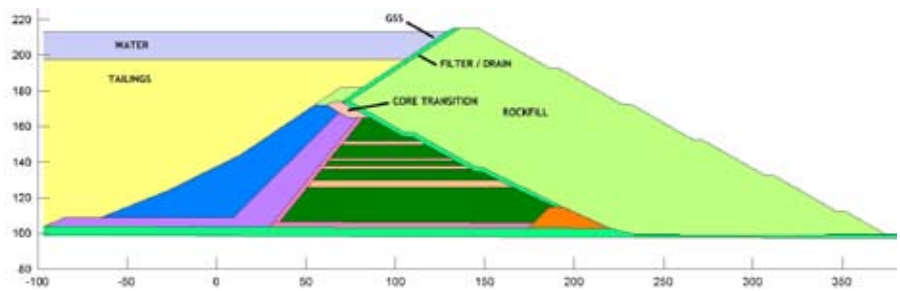

Figure 4 Main embankment final raise cross-section

To transition from an inclined clay core to an upstream exposed geocomposite, the arrangement shown in Figure 5 was devised. The core transition comprises an upstream sloping, 10 m wide compacted clay zone which is seated in the upper part of the existing core. This required the excavation of 60,000 m^3 of material from the downstream half of the existing embankment crest, the majority of which was reconditioned and reused in the core transition construction. The crest of the core transition forms the anchorage for the exposed PVC geocomposite.

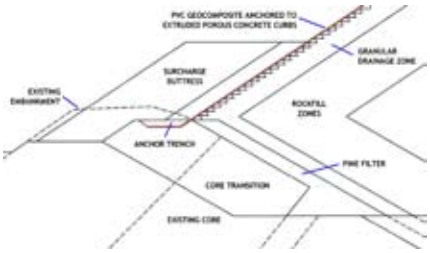

Figure 5 Core transition and GSS anchorage detail

GSS installation

A key factor in the raise design is the method of anchoring, supporting and providing drainage for the GSS. An innovative method was devised by ATC Williams and Carpi Tech, making use of the so-called Ita method of support layer construction. This involves porous concrete extruded curbs of low cement content installed on the upstream face, as shown in Figures 5 and 6. The curbs provide containment for the embankment fill, whilst providing a solid, uniform supporting layer for the placement, anchoring and drainage of the GSS. The extruded curbs were found to provide a very quick and effective means of installing the GSS when used in conjunction with a face anchorage system developed and patented by Carpi Tech. This involved embedding 0.5 m wide strips of the same geomembrane material in the curb support layer (Figure 7). The strips were positioned at 6 m centres to form continuous vertical lines, to which the sealing geocomposite was thermally welded (Figure 8).

Figure 6 Curb construction

Figure 7 Anchor strip installation

Figure 8 Welding geocomposite to anchor strip

Perimeter anchorage of the PVC geocomposite (Figure 9) was of the mechanical tie-down type on the abutments and crests of the raise, and via burial in trenches on existing embankments. At the abutments, tie-down was via stainless steel batten strips bolted to reinforced concrete plinths.

"From the perspective of both the designer and the owner, the selected GSS resulted in an efficient and economic waterproofing solution for the tailings storage raise."

Figure 9 Completing first stage GSS installation

Construction of the first two stages, incorporating a total raise of 20 m and approximately 40,000 m^2 of geocomposite, has recently been successfully completed (Figure 10). A joint venture of ATC Williams and local engineers provided full time site supervision of the works.

Figure 10 Commencing second stage GSS installation

From the perspective of both the designer and the owner, the selected GSS resulted in an efficient and economic waterproofing solution for the tailings storage raise. The extruded curbs have proven to be an effective construction method, whilst the anchor strip installation was a simple, routine process. Combined, these features greatly facilitated the installation of the PVC geocomposite. The installation was fast, with each stage installed within a four week mobilisation, commencing immediately upon completion of the earthworks. The overall construction period was also significantly reduced, given that the earthworks contractor was able to produce and place rockfill and concrete curbs at a greater rate than would have been achieved if the design had specified a moisture-conditioned and compacted clay core.

ATC Williams have concluded that the geocomposite faced rockfill approach is a viable, effective means of tailings dam construction where water retention in the vicinity of the embankment is necessary, and where natural materials are either not available, or are unable to be used from a technical perspective.

Craig Noske,
ATC Williams Pty Ltd

ACG Mine Tailings

The ACG actively assists mine personnel involved in the day-to-day management and operation of tailings storage facilities and the requirement to comply with the relevant operating standards and environmental and closure requirements through comprehensive and state-of-the-art tailings research, continuing education, and training products. Please contact the ACG for more details regarding our mine tailings activities.

ACG 2009 Mine Tailings Continuing Education Courses

Preparing and Implementing a Tailings Storage Facility Operations Manual Workshop
8 September 2009, Sheraton Perth Hotel, Western Australia

Practical Soil Mechanics in Mining Short Course
1 December 2009, Sheraton Perth Hotel, Western Australia

Tailings Management of Operators Seminar
2 – 3 December 2009, Sheraton Perth Hotel, Western Australia

ACG AUSTRALIAN CENTRE FOR GEOMECHANICS
CSIRO | Curtin University | University of WA
Joint Venture

The ACG newsletter aims to deliver to its readership of almost 5,000 local and international mining professionals topical and relevant geomechanical articles. The newsletter provides a forum for you to raise and explore the challenges facing industry. It also presents you with the opportunity to report on geomechanics initiatives occurring in underground, open pit and tailings environments, as well as state-of-the-art technological developments and mine safety advancements.

Do you have something to say? The ACG is listening. Please contact Josephine Ruddle, editor via acg@acg.uwa.edu.au

MINE WASTE 2010

First International Seminar on the Reduction of Risk in the Management of Tailings and Mine Waste

6–10 September 2010, Perth, Western Australia

This seminar will tackle the full range of issues that constitute risks in the management of mining wastes, particularly tailings and waste rock. It will provide a forum where practitioners, researchers and regulators can debate key shortcomings in our current understanding of the performance of mining waste storage facilities and associated risks faced by owners and operators of these facilities. Aside from the presentation of papers by selected authors, there will be a series of workshop sessions tackling specific issues of relevance as well as a number of keynote lectures from international speakers to ensure that the state-of-the-art is presented at this seminar.

Abstracts due 22 February 2010

www.minewaste2010.com

Tailings — From Concept to Closure

This ACG training DVD will provide guidance to personnel involved in the management and operation of tailings storage facilities that will facilitate the adoption of accepted best practices for the management of mine tailings.

To purchase your copy go to www.acg.uwa.edu.au/shop

Paste technology — improving our world

During the past three decades, the application of paste technology has progressed from a research based concept to a widely accepted and extensively practiced engineering solution for managing mineral waste. It is now readily recognised that significant engineering, scheduling, production and environmental benefits have been gained from the use of paste technology in the mining industry.
Paste 2010 will focus on discussing how the scientific advances made in the application of paste technology can improve the ability of various industries, in addition to mining, to manage their mineral waste streams.
Please join us in beautiful Toronto, Canada, to participate in this thought-provoking forum. Paste 2010 will present an excellent opportunity to meet other professionals from diverse industries to discuss and understand the application of best available paste technology practices, learn how these applications can enhance operations and, ultimately, appreciate how paste technology can improve our world.

www.paste2010.com

Paste 2009
Viña del Mar, Chile

For the third time, the International Seminar on Paste and Thickened Tailings returned to Chile. Since 2002, a lot of research has been done which makes P&TT technology more attractive and accessible. However, mining activities in Chile usually involve very large-scale operations. Studies have concluded that the paste tailings option still presents two main limitations: high cost of material handling and the absence of large capacity thickeners.

Paste 2009 addressed these issues and others considered to be of major interest to industry. The Paste 2009 proceedings comprises of 40 technical papers from industry experts from around the world.

These proceedings are a hard-bound, black and white publication featuring 40 papers comprising 400 pages. To order your copy go to www.acg.uwa.edu.au/shop

ACG On-Site Geomechanics Training Courses

The ACG's on-site training courses seek to deliver the latest technologies and information to the mining workforce with the mine site specific application. Our training and continuing education platform provides a solid base for the transfer of technological developments and practices based on knowledge gathered from local and international sources.

Management and Operation of Tailings Storage Facilities

Many high-profile failures of tailings storage facilities (TSFs) have occurred internationally during the last decade. Some have released large volumes of tailings resulting in environmental pollution, fatalities, huge clean-up costs and financial loss. These can be very damaging to the company concerned. In some cases, these failures have been attributable to lack of knowledge of the mine tailings' engineering characteristics and the possible implications for the design and operation of disposal facilities. While to a large extent favourable conditions have protected Australia from such a large failure to date, this could change and have a severe impact on the industry and must be avoided.

This on-site training course provides training in the management and operation of tailings and mine waste disposal facilities. The course seeks to improve the standard of mine waste management, in order to improve the safety and reduce the environmental and other liabilities associated with these facilities.

Ground Control at the Mine Face

Over the last 10 years, falls of ground have been the cause of one quarter of all lost-time injuries and 40% of all fatal accidents in Western Australian underground mines.

Ground Control at the Mine Face provides mine workers and front line supervisors with practical tools to protect themselves and their work mates against the 'unexpected'.

- Be aware of ground control hazards.
- Grasp the concept of ground pressure and its effect on your working environment.
- Understand how the rock will react.
- Identify the potential for ground falls.
- Make the right call with ground support.
- Fulfill your Duty of Care.

Ground Control at the Mine Face is simply about making the mine a safer place to work.

Over 650 mining personnel across Australia have attended this course.

Practical Rock Engineering Skills Development

The ACG offers the following courses:

- Geomechanics Data Training course.
- Ground Support Systems Training course.
- Stope and Pillar Design Training course.

This series of training courses seeks to raise the standard of geomechanics outcomes at mine sites through practical rock engineering skills development of site personnel.

Industry's newest rock doctor

Following many site visits and long evenings invested in data collection, analysis, results and writing, the ACG congratulates Dr Marty Hudyma on achieving his PhD.

Marty spent six years with the ACG as project leader of our internationally acclaimed "Mine Seismicity and Rockburst Risk Management" project. Marty left the ACG in early 2006 to return to Canada working with Itasca Consulting Canada Inc. Earlier this year, Marty was appointed assistant professor, Laurentian University, Canada.

Dr Hudyma's PhD thesis title is 'Analysis and Interpretation of Clusters of Seismic Events in Mines'.

Spatial analysis of seismic events in mines usually reveals that spatial clustering is a common phenomenon. This thesis proposes a clustering methodology that uses spatial and temporal data, as well as a number of seismic source parameters, as a data filtering technique. The clustering methodology can be used to identify individual seismic sources in mines. A set of seismic data analysis techniques are proposed, which can be used to make qualitative and quantitative conclusions about the seismic hazard and seismic source mechanism of populations of seismic events. Characteristic behaviours of typical seismic sources found in most underground, hardrock mines are identified and related to common rockmass failure mechanisms in mines. In addition, a methodology is proposed to create comprehensive seismic hazard maps and to forecast seismic hazard for individual seismic sources in mines.

Examples are presented using data from several mines in Australia and Canada. Topics such as: seismic system design, seismic data integrity, and bounds of self-similarity in seismic data are also discussed. The techniques and methodologies proposed in this thesis are applicable to understand the seismic response to mining for seismic data recorded in most mines.

Marty Hudyma, Laurentian University

ACG Board of Management Update

The ACG's Board of Management comprises of an independent chair, Ian Suckling, Newmont Asia Pacific; Yves Potvin, ACG director; and industry and Joint Venture Partner representatives: Steve Harvey, CSIRO Exploration and Mining; Paul Dunn, director, WA School of Mines, Curtin University; David Smith, UWA; Mark Adams, Barminco Ltd; Richard Butcher, Barrick Gold of Australia; and Chris Stone, BHP Billiton Nickel West. The Board meets up to four times a year to present strategic direction for the Centre, review and approve activities and operations and to provide counsel.

During recent times the Board was delighted to welcome onboard new members: Richard Butcher, David Smith and Chris Stone.

David Smith

Professor David Smith is the dean of the Faculty of Engineering, Computing and Mathematics at The University of Western Australia. Prior to his role at UWA he was associate dean (academic) at The University of Melbourne and played a crucial role in the introduction of The Melbourne Model and was associate dean of research at the University of Newcastle.

He was awarded a PhD from the University of Sydney for numerical analysis of a proposal to dispose of radioactive waste in the ocean floor. David undertook engineering at the University of Newcastle (Australia), and researched geotechnical, geoenvironmental and biomedical problems. He was appointed professor of Biomedical and Environmental Engineering at the University of Melbourne in 2004.

David is generally interested in all problems in computational biology, including cellular signal transduction, cell mechanics, physiology of the kidney, problems in developmental biology, and others.

When asked about his thoughts on strategic direction of the Centre, David said, "I am looking forward to joining the Centre's Board. I hope that my knowledge and experience will contribute to the challenging future of ACG. The collaboration between industry and academia is extremely important regardless of the economic climate. As the mining sector grows, it needs our support for each step of its progress. Through our research we improve safety in the working environment, we bring more effective technology solutions and we develop the skills and knowledge of industry personnel.

David Smith

During the financial crisis the mining industry still needs our expertise in deciding the best sustainable development practices. For this reason, our continued collaboration will be a crucial factor through tough and challenging times. As a new Board member I am relishing the challenge which lies ahead."

Richard Butcher

Richard Butcher has 29 years of experience in the mining industry with over 20 years involvement in geotechnical engineering.

Richard Butcher

He has wide experience in mining and geotechnical projects in 11 different countries and has held positions ranging from geotechnical engineer to general manager. He is well known as an international expert in caving and has been involved with mining research for over a decade. At the present time, Richard is the group mining engineer for Barrick's Australia Pacific and, as such, is functional head for both mining and geotechnical engineering within the region.

In terms of strategic direction, Richard believes that the ACG should continue to focus on research that has a practical benefit to the mining industry in terms of safety and cost. He further thinks that the mining industry is going to be challenged within the next two years to reduce costs and still maintain an acceptable risk profile. Therefore, the key challenge for the geotechnical engineer is to be able to define the acceptable risk/reward relationship.

Chris Stone

Chris Stone is a mining engineer and has 20 years of experience in the industry. He has worked in gold, nickel and coal across Australia. Chris' roles have been varied, including production, technical and business development activities, whilst ranging from operator to general manager levels. Chris' current role is as a project director with BHP Billiton Nickel West.

Chris Stone

On the ACG Board, Chris acts as an industry representative and seeks to ensure that the ACG continues to support the industry as it has to date. This means that a strategic focus is maintained on adding value to the industry through the application of geotechnical research and technology; directly improving safety, efficiencies and/or productivity at the mine sites. Further, in order to deliver the greatest value, the ACG should continue to seek bringing together the best geotechnical knowledge and resources of the Western Australia academia.

ACG Board of Management: Paul Dunn, Western Australian School of Mines, Curtin University of Technology; Yves Potvin, ACG; Steve Harvey, CSIRO Exploration and Mining; Ian Suckling, Newmont Asia Pacific; Mark Adams, Barminco Ltd; David Smith, The University of Western Australia; and Chris Stone, BHP Billiton Nickel West

ACG Event Schedule*

Geotechnical Engineering in Open Pit Mines Seminar	Brisbane, 9–11 June 2009
CSIRO Open Pit Mining Geomechanics Research Applications Seminar	Brisbane, 12 June 2009
Blasting for Stable Slopes Short Course	Perth, 14–15 July 2009
CLR Refresher Course in Soil Science for Environmental Managers	Perth, 2–4 September 2009
CLR Mining in Ecologically Sensitive Landscapes Seminar	Perth, 7 September 2009
Preparing and Implementing a Tailings Storage Facility Operations Manual Workshop	Perth, 8 September 2009
Fourth International Conference on Mine Closure	Perth, 9–11 September 2009
Applying Numerical Models to Mining Problems: Theory and Case Studies	Perth, 14–15 October 2009
Mine Backfill Seminar	Perth, 10 November 2009
Advanced Ground Support in Underground Mining Seminar	Perth, 11–13 November 2009
Practical Rock Mechanics in Mining Short Course	Brisbane, 25–26 November 2009
Practical Soil Mechanics in Mining Short Course	Perth, 1 December 2009
Tailings Management for Operators Seminar	Perth, 2–3 December 2009
Preconditioning Workshop	Perth, 19 April 2010
Second International Symposium on Block and Sublevel Caving	Perth, 20–22 April 2010
International Seminar on the Reduction of Risk in the Management of Tailings and Mine Waste	Perth, 6–10 September 2010
14th International Seminar on Paste and Thickened Tailings	Perth, 4–8 April 2011

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Application of earthquake ground motion simulation in the evaluation of tailings storage facility performance

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Rehabilitation of bauxite residue — a soil science perspective

by Martin Fey and Talitha Santini, School of Earth and Environment, Faculty of Natural and Agricultural Sciences, The University of Western Australia

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Geocomposite faced rockfill — an innovative approach to tailings dam water-proofing in the absence of clay

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