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**9–11 November 2011,
Novotel Langley Perth,
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Since 2005, this series of international seminars on strategic versus tactical issues in mining has been well attended by senior mining professionals and managers in Australia, South Africa and Canada. The ACG is pleased to bring this innovative and highly topical seminar back to Perth.

**ABSTRACTS DUE
21 FEBRUARY 2011**

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Sheet flow of ultrafine tailings

The evolution of paste and thickened tailings

by Richard Jewell, Australian Centre for Geomechanics, Australia

Background

Under the guidance of Dr Eli Robinsky, the first attempt to produce thickened tailings for surface disposal was made in 1973 at the Kidd Creek Mine in Canada. Implementation of the first central thickened discharge (CTD) operation commenced using a conventional thickener, but it was not until 1995, after several iterations of thickener upgrade as the technology developed, that the original vision was achieved at the Kidd Creek Mine. In the mid 1980s, the alumina industry set a goal of converting existing “wet” storages to “dry” storages to achieve environmental benefits. This was accomplished with new generation thickeners and in a relatively short period the modus operandi for the alumina industry

globally involved thickening tailings to a high consistency prior to discharge. “Dry stacking”, in which tailings were first dewatered using filters into a “cake-like” consistency, was feasible and was indeed applied to some low throughput operations in regions such as the Atacama Desert in Chile where water retention was of paramount importance. However, it was only in the first decade of the 21st century, as thickener technology advanced, that the concept of surface deposition of high-density slurries bordering on the consistency of a paste became feasible for the general mining industry.

In parallel with developments for surface disposal, the technique for underground disposal of tailings as a cemented backfill, or

Continued page 2

component thereof, was being developed. In this application, backfill strengths in the MPa region are often required, orders of magnitude higher than associated with surface disposal. There are considerable differences between surface and mine backfill disposal in terms of transport and disposal systems, and yet the production of the thickened tailings itself is often identical.

The equipment now exists to make and to transport very high-density tailings and, in general, it will be the practicability and environmental and social advantages of any system that will determine whether it will be adopted by the industry. A practicable above ground disposal system will require that the tailings flow away from the point of discharge for a sufficient distance to avoid the need to locate the discharge points at very close intervals, or to install a discharge system that is excessively expensive to construct or to operate. The thickened tailings slurries used in most CTD projects will not segregate, but a limited amount of supernatant water may separate from the deposited tailings and flow down slope.

In general, the consistency (solids content or density) of these tailings will also be limited by the capability of a centrifugal pump to drive the material through a pipeline. Positive displacement (PD) pumps will pump much higher density materials at correspondingly higher discharge pressures.

The cost of installing and operating positive displacement pumps has to be evaluated over the lifetime of the project to make a meaningful comparison with a system using multiple centrifugal pumps (or pump stations) required to generate a comparable pump discharge pressure.

The implementation of paste and thickened tailings (P&TT) technology on any mining operation can only be justified if it is the most cost-effective solution available. The cost of the plant required to thicken tailings, transport and then discharge them is not insignificant and it is only when full life-cycle costs are taken into account that a true economic



Positive displacement pumps at Richards Bay (Hillendale)

comparison is possible. The implementation of P&TT technology, however, can provide a number of specific benefits to the extent that these become the reasons or “drivers” for adopting the technique.

The reasons for implementing the thickening of tailings vary between sites. When first introduced, the principal drivers in all operations were to save on the costs of storing the tailings safely while ensuring minimum potential for pollution of the environment. Saving on costs satisfied the need of the operator to optimise profit, while the provision of a safe and stable storage facility contributed to meeting the demands of the stakeholders and to the long-term “license to operate” of the mining company. In more recent times however, the perceived potential to retain process water at the plant from thickener overflow, rather than expose the water to evaporation and seepage losses at the TSF, has become a very important factor for considering the adoption of thickening technology in many mining provinces around the world. In the economic assessment of the process, the need for any additional treatment of the retained water

for it to be of value to the plant needs to be considered.

Savings are possible in water, energy and reagent conservation, reduced impoundment needs, improvements in impoundment and embankment stability, more rapid closure and reduced financial provisions. Additional capital and operating expenses for thickening equipment, pumps and piping may partially offset these savings.

To fully evaluate the benefits of P&TT technology, an economic study based on full life-cycle costs is needed. Capital and operating costs, timing and the time value of money need to be applied across the full spectrum of the mine plan, including closure. Often, complete mine and/or tailings plans need to be run to closure to truly understand the benefits of this technology. Non-monetary benefits like improved public perception should also be evaluated, using appropriate qualitative measures.

P&TT seminars

In 1999, the first of what has become an annual series of seminars on P&TT was held with the objective of providing a

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forum for the dissemination of information on and promotion of the technology.

One outcome over the past decade has been rapid advances in thickener design. This was an industry that had traditionally evolved and adopted new designs, increased the size and throughput of thickeners and introduced changes to improve feedwell designs etc. primarily on the basis of trial and error. However, industry is evolving and increasingly benefitting from research programmes such as the AMIRA funded P266F project entitled "Improving Thickened Technology", investigating feedwell mechanics. The number of submissions of papers from pump manufacturers and participation in the trade presentations at the seminars are increasing and it would appear that they also are benefiting from their involvement and adapting their products to the requirements of P&TT operations.

These P&TT seminars are now being held as a quadrangular series rotating between Australia, Southern Africa, South America and the Northern Hemisphere. It would appear that these seminars have become the first choice for industry in this particular field and the number of registrants is increasing year by year.

An important outcome arising from the initial seminars has been the production of

a guidance and advice handbook, published to provide sufficient information on the technology to enable miners to determine whether it may be of value to their operations. The handbook published by and available through the ACG is entitled, "Paste and Thickened Tailings – A Guide", and in excess of 1,000 copies in two editions have been sold.

Paste 2011

The 14th International Seminar on Paste and Thickened Tailings returns to Australia in April 2011 accompanied by workshops on beach slope prediction and rheology.

Papers have been especially commissioned to cover the significant changes that have been made to thickeners and transport systems. Beach slope prediction remains the greatest unknown for TSF design. The current status of beach slope prediction is that there are two "semi-rational" methods under development and investigation and both are complicated and very parameter dependent. A collaborative paper has been commissioned to draw together and compare the current systems. The hope is that this exercise will lead to a better understanding of the state-of-the-art and, with luck, will be a start towards the development of a rational method

for accurate beach slope prediction and at the very least will discredit the claims of those who maintain that beach slopes can be predicted from the results of small scale flume testing. The parties to this collaborative paper will also mount the workshop on beach slope prediction.

Acid mine drainage (AMD) has been and is a major issue for TSFs retaining conventional tailings. Although there are few, if any, P&TT storages at the stage of closure, the expectation has been that AMD potential will be reduced. A keynote paper has been commissioned to look at this aspect of closure and will hopefully provide a valuable contribution to an environmental issue that could very much influence the take up of P&TT technology into the future.

Many Paste 2011 papers will cover the whole range of the issues encountered in the application of the technology and the seminar promises to more than maintain the quality of the series and to justify the expectations of those attending.



Richard Jewell,
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14TH INTERNATIONAL SEMINAR ON PASTE AND THICKENED TAILINGS

5–7 April 2011, Esplanade Hotel Fremantle, Perth, Western Australia

Keynote speakers

Fred Schoenbrunn, FLSmidth, USA
Angus Paterson, Paterson & Cooke, South Africa
Gavin Mudd, Monash University, Australia

Associated workshops

Sunday 3 April 2011 | ACG Beach Slopes Workshop

Monday 4 April 2011 | ACG Rheology Workshop

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Consolidation, fragmentation and the structure of the mining industry

by Knud Sinding, University of Dundee, Scotland and University of Southern Denmark, Denmark

Introduction

Suppose that one large mining firm is attempting to take over another similar firm, in an unfriendly and contested way. The offer is, as such offers frequently are, based on the broad assumption that large cost savings can be achieved by merging operations. The target company has rejected the offer on the grounds that it is too low. Several important competitors and clients have joined the fray by acquiring significant shareholdings in the target company and possibly also in the potential acquirer. The latter has claimed that the merger will allow significant synergies to be realised from properties in which both companies are already partners or which are adjacent to each other. However, before the issue could be resolved all these matters were rendered irrelevant by the deepening global financial crisis.

That proposed merge, and others like it, might have contributed to a higher degree of concentration among producers of a number of key minerals. This could lead to concerns about the accumulation of market power in the hands of a few, very large mining companies. The proposed and abandoned merger would follow others in the mining industry that has created a series of top firms that are much larger and have greater combined market shares than they had before. Examples within the last decade include the mergers between Xstrata and Falconbridge, Freeport McMoRan and Phelps Dodge, CVRD and Inco, and, most recently, Rio Tinto and Alcan.

While the trend at the top seems to suggest that larger firms are being created, a corresponding trend among smaller firms can most precisely be characterised as fragmentation. This covers the existence of a long established trend (Humphreys, 2001) towards new deposits being explored for, discovered and moved towards a feasibility stage by companies that are sometimes labelled “junior” or “medium-sized”. Some

projects eventually become mines and somewhere along the way a major mining firm may acquire the junior company that had developed the project or the portfolio of projects, to a point where it became an interesting proposition to the large mining firm.

The apparent very strong desire of large mining companies to engage in mergers raises the question of motive. Research has shown that mergers are costly, in the sense that the acquiring firm has to pay a premium price for the shares of the target. Further, the mergers frequently (anywhere between 50 and 70%) fail to deliver the returns they were supposed to generate, the loss of shareholder value generally appearing within a month and persisting thereafter (King et al., 2006). Various explanations have been offered for the persistence of mergers despite the evidence that they destroy value, including managerial egotism and perverse incentives for bankers to press for such transactions (Pfeffer and Sutton, 2006).

One of the most frequently suspected explanations for mergers is the desire to attain market power. No respectable company would ever use that explanation, since market power, and by extension, monopoly, is almost universally frowned upon. The high failure rates would tend to refute this effect. Nevertheless, “consolidation”, “restructuring” or “top level” mergers are taking place in the mining industry. Assessing the degree of concentration, measured as the fraction of global production by top firms, is a way of saying something meaningful about the real extent of this consolidation. This article examines changes in concentration over the last 20 years in a number of key mineral segments: gold, copper, nickel, zinc, aluminium and iron ore.

While the level of concentration has increased some percentage points over the past two decades, the increase is not as large as might be expected from the rhetoric emanating from various

regulators and downstream interests. One reason that concentration may not have increased as quickly may be related to events at the other end of the supply chain. Even if mining firms do merge, the resulting entities are still being depleted by ongoing production, and reserves are only augmented to the extent that new deposits or prospects are acquired. These new deposits, however, are rarely discovered or even brought forward by major mining companies.

It seems as if these firms only enter the development and production stage by way of an acquisition, or at least some sort of partnership arrangement, when it is tolerably clear that the worst uncertainties have been overcome. This refers not just to the geological and technical uncertainties present everywhere, but also to institutional and regulatory uncertainties. Furthermore, while there has been some agreement that junior exploration firms are very active, little is known about these companies, about why they undertake the upstream section of the deposit discovery and development stream, or about whether this is an efficient way to organise the mineral supply process in the first place. To shed light on this part of the industry, this article uses cross-sectional data from the different stages of project generation in combination with selected longitudinal case studies of individual deposit development histories.

Some possible motives for mergers or consolidation were briefly alluded to above. However, market power is an unpalatable explanation, at least when proponents of mergers try to gain support for their ideas. An expression widely used for promoting any merger in any industry is that certain “synergies” will be realised. The synergies are sometimes specified as a precise monetary amount e.g. savings to be made over a fixed period of time. Apart from elimination of duplicate services, many of these expected synergies essentially refer to economies of scale and/or scope, but may also refer to improvements in efficiency that take place upon the installation of new management. However, it is not always clear that such gains will ensue or that their value will accrue to the acquiring firm, as opposed to the seller. Further, there are legions of examples, as well as systematic research, which indicate the high risks associated with mergers (King et al., 2004). There may, however, be additional reasons specific to the mining industry that can explain the ongoing fashion for big mergers.

Strategic versus tactical

Consolidation among major mining companies

The hostile merger offer sketched in the introduction was to be just one of a number of such proposals. Some have been realised, others not. Some proposals have been received with great hostility; others have been accepted after protracted haggling over price. The outcome of these mergers has been slight increases in concentration levels in the industry.

Concentration can be measured in a number of ways. The simplest is possibly the calculation of the fraction of market capitalisation of the largest mining firms. This, however, obscures the fact that many of the largest mining firms are composed of a variety of businesses. Some firms are strong in just one commodity, as exemplified by Codelco Chile, which dominates copper mining (and as a state company is not involved in the merger game). Therefore, measures of concentration based on the market value of firms are virtually meaningless.

A slightly more fine-grained approach is used here, with an emphasis placed on the combined market share of the 10 largest producers in copper, iron ore, aluminium, zinc, nickel and gold. These market shares are examined at the mining stage, regardless of any downstream processing.

The picture presented in Figure 1 is one of a gradual but not a very strong and sustained trend towards higher concentration ratios for the five metals examined. Even if the stages of production are not fully comparable, the trends are fairly clear: despite a series of quite significant mergers among large firms in the industry, concentration has not risen much – or not at all. One possible explanation is that most of these mergers have taken place during a long period of high and rising metal prices. Under these conditions, total production, not just that of the top firms, is likely to have risen. Indeed, assuming that the top firms apply conservative criteria when evaluating new projects, additional production may in many cases have come from sources where high production costs overshadow the effect of mergers.

The trend in Figure 1 suggests that concerns over concentration may not at this time be warranted. However, if the assumption about production cost levels holds, then a downturn in metal prices may bring a higher degree of concentration, as the marginal producers close mines and delay production. Conversely, the larger firms may be quicker to reduce output

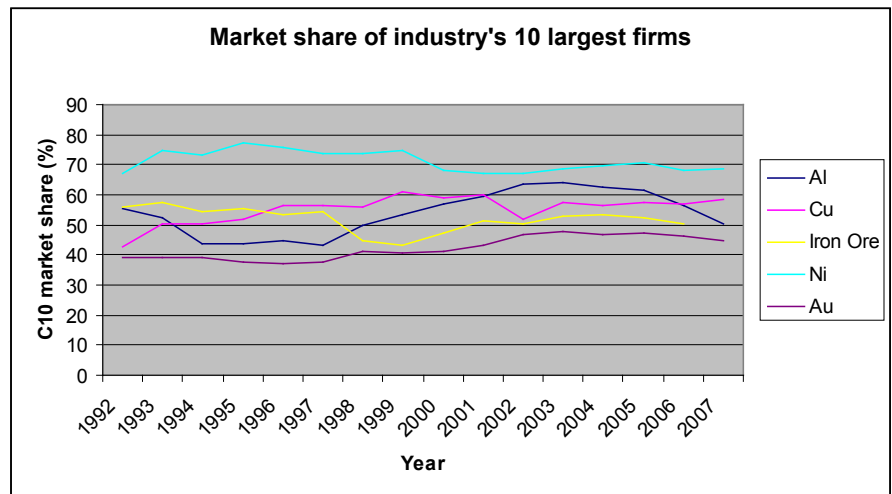


Figure 1 The market share for five important metals measured cumulatively for the top 10 producers in each metal. Based on data from Raw Materials Group

during a recession.

While Figure 1 paints a fairly rosy picture of competition, more careful consideration of individual market segments or niches can change the picture. A merger between two top iron ore producers would have great impact in that segment of the iron ore industry that happens to be seaborne. Since iron ore shipping facilities tend to be large, those firms engaged in seaborne exports may come to hold a very strong market position. Even if this results in higher prices, this effect will last only as long as it takes to find new ways of bringing minerals to market.

Whether a very large merger justifies some form of intervention, either by regulators or by supply chain stakeholders, remains to be seen. However, the relentless impact of depletion means that even very large mining firms must either discover or acquire new deposits to keep up with the volumes being extracted and sold.

Fragmentation

One overall reason that concentration and consolidation should not be taken too seriously is that the industry is very dynamic, in its own way. The essential characteristic is that individual deposits are depleted over time. Concerns about depletion have a long history, one that usually overlooks the limited incentives to acquire definitive data on the existence of stocks to be mined more than 20 years into the future. However, even if the geological stocks are very large, various environmental concerns may be the biggest source of shortages, or, more precisely, of steeply rising costs. The time for such discussion is some time away and at the present time, the source of new deposits is

our focus.

The question is not if they exist but who finds them. The very small numbers in Table 1 indicate that it is certainly not the top 10 mining firms that undertake any of the four stages involved, with the possible exception of Xstrata and to a lesser extent, Freeport and Antafogasta. Indeed, some firms are curiously absent from the list or have very little going on. This suggests very different approaches to vertical integration and strategy in general. On the one hand, there are firms like Xstrata that seem to be involved from a fairly early stage. On the other, several of the largest industry firms (i.e. Rio Tinto and BHP Billiton) are not involved directly in any of the stages identified in Table 1.

To further explore the issue of fragmentation, consider the history of the Pebble project in Alaska. According to the Northern Dynasty website, the original discovery was made by Cominco in 1997 but the property had gradually been transferred to the Northern Dynasty between 2001 and 2005. Subsequently, a pair of very large low-grade orebodies was discovered. Rio Tinto acquired a 19.8% stake in the company, but Anglo American was identified as the partner for developing the project. The history of the project, briefly sketched here, illustrates major firms entering and exiting over a long period of time. Many similar paths can be identified for other projects.

To some degree this example of an acquisition path confirms the notion of fragmentation, but also indicates an ongoing effort by large mining companies to create networks of opportunities for themselves. Sometimes these efforts will lead to full control over a deposit, at other times the

	Conceptual	Pre-feasibility	Feasibility	Construction
Freeport	8	0	0	0
CODELCO	2	2	0	1
BHP	1	0	0	0
Xstrata	12	4	5	3
Rio Tinto	0	0	1	0
Grupo Mexico	0	1	3	0
Anglo	0	2	2	0
KGHM	0	0	0	0
Norilsk	3	1	0	0
Antofagasta	8	1	4	0
Total projects	288	86	93	37

Source: data from Raw Materials Group

Table 1 Greenfield copper projects with the top 10 copper producers as of 2008

web fails or the major firm chooses not to pursue the opportunity. While the details surrounding the agreement about the Pebble deposits are limited in number, the fact that two of the largest mining firms were involved indicates that some form of bidding war took place, with Anglo as the winner.

Making sense of mining mergers

Mergers and acquisitions in this industry do not just involve the very large mergers, such as those mentioned previously. Most M&A activity in fact goes on at a far less glamorous level, where the focus is on individual projects. The latter are clearly pursued by producing mining firms, seemingly with the sole objective of replenishing their ever depleting portfolio of operating projects.

Larger mergers are a different matter altogether. The literature on post-merger financial performance is not encouraging (King et al., 2004), in the sense that M&A activity does not create superior financial performance for the acquiring firms. The motives for engaging in M&A have been hinted at. Market power, synergies of an operational or financial kind, tax advantages, inefficient target management and managerial self-aggrandisement may all be motives that contribute to the decision about takeover offers (Capron and Pistre, 2002).

In cases where competing firms have adjacent operations and extensive infrastructure, the operational synergies may be obvious. Indeed this may be the principal driving force behind the offer BHP Billiton made for Rio Tinto. Both companies have an extensive presence in the iron ore industry in Western Australia. The synergies expected in such a case are likely to come from economies of scale, in this case transporting the ore to market.

Realising such economies of scale, however, does not depend on a merger

taking place. The alternative to a full-scale merger, which on average is a fairly risky undertaking for the acquiring firm, is to realise the specific economies of scale through contracting solely with respect to where the improvements are expected to come from. If it is a matter of operating a joint rail transport system rather than building competing systems that overall are less efficient than a joint system, then contracting is certainly an alternative.

Since no firm will voluntarily claim that attaining market power is their goal, we can dismiss that explanation for large-scale merger activity. What remains are other synergies, tax advantages, poor management in the target company, and self-aggrandisement. The last option is likely to always be involved and even if this is not the reason, top managers may commit themselves to such a degree that they cannot change their mind.

Financial synergies refer to the possibility that the merged firm may be able to obtain financing for its activities on better terms than they could before. However, since the firms involved are already among the largest in the industry, the improvement in cost of capital will probably be low.

Apart from scale benefits from adjacent operations, synergies are harder to identify. Economies of scale in mining are primarily dictated by deposit features and they are also somewhat locked in during the mine design stage. Against this background it would be difficult to point to specific merger benefits. Economies of scope are a different matter. Scope economies are available when carrying out several activities at the same time, leading to lower unit costs for both activities. However, while economies of scope may follow from the broader knowledge base in a merged company, realising such benefits requires more than just a merger. These benefits can only be obtained if the collected resources of acquirer and target firms are integrated.

While large mergers of companies of roughly equal size are not doomed to succeed, other mergers may still be a good idea. In an otherwise very acerbic commentary on poor management, Jeff Pfeffer and Bob Sutton (2006) highlight the success of a company like CISCO, which has grown much of the way by taking over much smaller firms. The same authors (Pfeffer and Sutton) also use an example from the mining industry, that of Cyprus Minerals, to illustrate how enormous gains in performance achieved by good management practices can be undone quickly by a merger.

Concluding remarks

Very little is known about mining company profitability. Of course, the bottom line is published, as are the profits and losses of subsidiaries. Investigating the sources of profitability is, however, particularly difficult as the question invariably involves the matter of rent. Rent is the abnormal profits that accrue to firms as a result of the quality of a deposit. This type of profit is very specific to the extractive industries. All industries, however, may employ other resources and skills that competitors do not have and, as a result, generate abnormal rents. Separating the two is difficult, to say the least, but also contentious for any politician looking for money to spend.

Mergers may in fact be a way of dissipating rent. Mergers are very costly to implement. They take place on the basis of accumulated earnings. To the extent that merger-related costs can be deducted from operational revenues, the net result is that shareholders must forego profits that managers deem better spent on mergers.

Many questions remain unanswered. How does the junior mining sector work? How good is it at finding new deposits and what can we do about it? The list is quite long for anyone wishing to get started.

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Article references are available on request.



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Tailings-based mine backfill and geotechnical considerations

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Introduction

Over the past decade there has been increasing use of “paste” for backfilling of mined-out voids (“stopes”) in underground mines. Paste backfill is generated from full stream tailings and is almost always blended with cement, or other additives, before being placed underground. Hydraulic backfill generally refers to tailings that are either coarser than paste backfill to start with, or where the fine fraction has been removed, e.g. via cycloning. In the design and management of cemented mine backfill, it is common to adopt specific “rules of thumb” for paste fill while other rules are adopted for hydraulic fill. Conventionally, the distinction between the fill types is rather crude, such as that provided by Potvin et al. (2005), which defines hydraulic fill as having a maximum fines content (finer than 10 μm) of 10%, and paste fill as having a minimum fines content (finer than 20 μm) of 15%. This criterion has been used to define appropriate operating criteria. However, it is widely recognised (Bawden 2005; Qiu and Segó 2001) that different tailings streams can possess very different characteristics. When combined with variations in geometric conditions (such as rate of vertical rise), fills made from different tailings (but which all classify as paste backfill according to the above definitions) can behave quite differently during deposition and, as a consequence, require different design and management systems.

The stresses within mine backfill and the stress acting on the containment barricade are a function of complex relationships involving the geometry of the stope and the barricade(s), and the timescales associated with consolidation, cement hydration, and filling. Because of these complex interactions, it is difficult to derive simple closed-form solutions for this problem. Until recently, no rational approach incorporating all these important aspects has existed for estimating stresses acting on barricade structures. Lack of certainty about this critical aspect of paste fill design has undoubtedly contributed to at least some of the 12 known paste

and hydraulic fill barricade failures that occurred between 2003 and 2006, as well as resulting in production delays.

Over the past few years, the authors have addressed this problem using a new approach based on the principles of effective stress in which the processes of hydration and consolidation are fully coupled. To couple these mechanisms the authors developed the Minefill-2D software (Helinski et al., 2010). This software was verified against a series of field experiments to demonstrate the applicability of the analytical approach for a range of different filling conditions in Helinski et al. (2011). Following on from the back analysis of the in situ monitoring results, the authors presented a brief sensitivity study to show how subtle differences in characteristics associated with the consolidation behaviour can influence suitable strategies for designing and managing stresses applied to barricade structures.

This article follows on from Helinski et al. (2011), presenting a summary of the previously presented field studies and back analyses. Using the two presented cases, an extended sensitivity study is presented, which investigates the significance of aspects such as binder addition, vertical rate of rise and drawpoint configuration on both barricade stresses and the rate of application of effective stress to cemented mine backfill during curing. This work demonstrates that, even though both fill types are classified as paste (according to the system presented in Potvin et al., 2004), the behaviour is vastly different, and different mechanisms control the behaviour.

To assist in understanding the behaviour of cemented paste backfill, a new framework for characterising the backfill deposition process is presented. This framework draws on an analytical solution originally proposed by Gibson (1958), which combines the coefficient of consolidation (c_v), time (t) and the vertical rate of rise (m) in a non-dimensional time factor (T) to provide an indication of the degree of consolidation that occurs during the initial stages of filling. Given

an appropriate characterisation of the consolidation behaviour immediately after deposition (using Gibson’s solution), a series of relevant strategies are recommended for the design and management of aspects such as barricade loads and appropriate curing strategies.

Influence of effective stress on backfilling

Tailings-based backfill (hydraulic fill or paste) is almost always transported with water. This provides an efficient form of transportation, but if the characteristics of the backfill are such that no consolidation can take place, no effective stress can develop in the backfill and therefore no arching can take place. As illustrated numerically by Helinski et al. (2006) and Fahey et al. (2009) and experimentally by Fourie et al. (2007), if no consolidation occurs during filling, no stress arching can take place and the stress applied to the barricade structure would be equal to the self weight of the overlying fill, even if cement hydration creates a considerable increase in strength and stiffness. Furthermore, these authors all illustrate that if there is no development of pore water pressure during filling, the stress imposed onto containment bulkhead structures would be very low. Helinski et al. (2006) showed that in a 50 m tall stope containing fully cured paste, if pore pressure was neglected, the barricade stress would be approximately 80 kPa, but if no consolidation took place during filling, the stress applied to a barricade would be in the order of 800 kPa.

During the progressive accretion of mine backfill, previously-deposited fill is exposed to an increase in total stress, which is dependent on the rate of vertical rise, bulk unit weight of the fill and extent of arching that develops. The rate at which this total stress is converted to an effective stress is dependent on the rate of consolidation.

To investigate the significance of applying effective stress to a cementing specimen during curing, the authors undertook a series of laboratory experiments. Each of these experiments was undertaken

using tailings from an Australian mine mixed with 2.7% binder to an initial solids concentration of 78% by weight. The testing programme involved experiments to investigate the significance of the rate of application and the duration of application of effective stress on the final strength.

To maintain lateral rigidity, split moulds, tightly secured using a hose clamp, were used to contain the specimens, as shown in Figure 1(a). Within these moulds, a latex membrane was sealed to the top-cap and base with O-rings. These top and bottom caps were fitted with a drainage tube, with each of the tubes set to maintain a constant water head of 0.1 m above the top of the specimen, thus ensuring that the effective stress in the specimen was not influenced by any pore water suctions (see Figure 1(b)).

Load was applied to the specimen top cap via a loading frame that straddled the specimen resting on a support frame. Figure 1(a) presents a photograph of a specimen being cured under loaded conditions while Figure 1(b) presents a schematic of the setup.

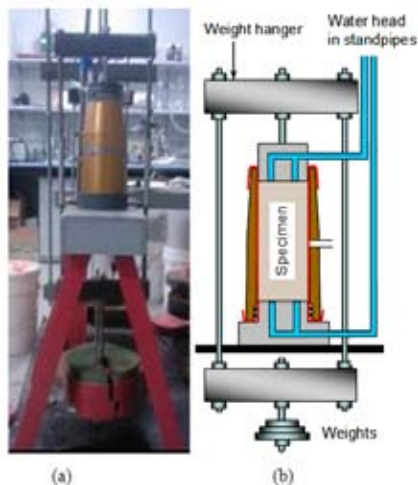


Figure 1 Sample preparation setup

To investigate the influence of loading rate on the resulting material strength, identical specimens were exposed to loading rates that corresponded to vertical effective stress (σ'_v) rates of increase of approximately 0, 1.2, 2.4 and 4.8 kPa/hr. Assuming a bulk unit weight (γ) of 22 kN/m³, these rates correspond to the rate of total stress application in a slope when filling at vertical rates of rise of 0, 0.05, 0.11 and 0.22 m/hr, respectively. For each of the loaded specimens, the loading rate was continued until σ'_v reached 60 kPa, which was maintained for the remainder of the hydration period.

After 28 days hydration the specimens

were removed from the moulds for testing. As illustrated by Walsh (1997), pore water suctions can influence the unconfined compressive strength (q_{UCS}) of weakly cemented soils. To eliminate the influence of suction, all UC testing was carried out under water using a specially-designed container. A photograph showing a specimen being testing is presented in Figure 2.



Figure 2 UCS test with specimen submerged in water bath

The results of the UCS testing on each of these specimens are presented in Figure 3, which shows q_{UCS} plotted against average rate of application of σ'_v . Figure 3 clearly shows that when σ'_v is applied at rates that correspond to rates expected in the field, q_{UCS} is highly sensitive to the rate at which σ'_v increases, with an increased loading rate resulting in significant increases in q_{UCS} .

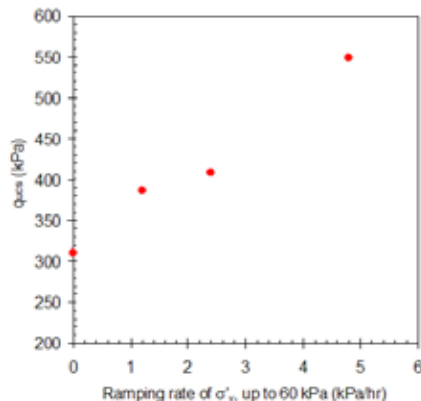


Figure 3 Effect of rate of application of vertical stress during curing on the unconfined compression strength

To investigate the significance of the duration of σ'_v ramping, σ'_v was applied to a second series of four specimens at a constant rate of approximately 2.4 kPa/hr, but with the duration of ramping being 6,

12, 24 and 48 hours for the four specimens. In addition to the base case mix, i.e. 2.7% binder, these tests were also undertaken on mixes containing 1.5 and 3.0% binder. The preparation, curing and testing strategy was the same as that adopted in the loading rate programme. The results from this series of experiments are presented in Figure 4, which shows q_{UCS} plotted against load ramping duration for each of the mixes.

Figure 4 shows that, even when ramping only occurs for the first six hours, the application of σ'_v created a considerable increase in strength. The results illustrate that the most significant increase in strength is achieved during the early stages of loading. This result is somewhat expected as it is volumetric compression (leading to a higher density) that is thought to be creating the higher strength, and the application of σ'_v at the beginning of curing would create the largest volumetric compression.

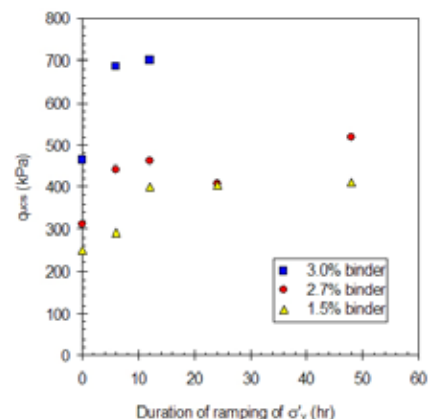


Figure 4 Effect of ramping duration on UCS strength

Overall, these results illustrate that the application of σ'_v during filling can potentially create considerable increases in the strength of cemented mine backfill. Importantly, the results show that in order to provide an accurate prediction of the magnitude of strength increase, it is critical to be able to accurately predict the rate that σ'_v is applied to the material during the early stages of hydration.

Development of effective stress during backfill deposition

Because the rate of consolidation during filling is critical to both the distribution of total stress (and specifically barricade loads) as well as the evolving material strength, the authors set about investigating the consolidation behaviour in a material that is undergoing all of the changes that typically occur during the consolidation

of mine tailings, but also changes that come about as a consequence of cement hydration and stress arching. Specifically, cement hydration during mine backfill deposition modifies the consolidation behaviour by:

- increasing the material strength and stiffness, often by over an order of magnitude;
- reducing the permeability as a consequence of hydrate growth; and
- inducing volumetric changes associated with the cement hydration process, termed “self-desiccation” (Helinski et al., 2007a).

In order to simulate the cemented mine backfill deposition process, taking account of the cement induced characteristics, the authors developed a new finite element (FE) program called Minefill-2D (Helinski et al., 2010); a standard FE formulation, based on conventional FE coding. The model is a fully coupled consolidation model based on the Biot consolidation theory. While the model is programmed using a Eulerian coordinate system, it takes account of the changes in stiffness and permeability that result from both soil compression and cement hydration, as well as the volumetric changes that occur during the hydration process (the self-desiccation mechanism). The elements used in the program are typically 8-noded quadrilateral elements for displacement calculations, and 4-noded quadrilateral elements for drainage analysis. Numerical integration is carried out using four integration (Gauss) points per element.

Minefill-2D is able to take account of the accretion of cemented mine backfill in a stope during filling as well as providing an opportunity to represent drawpoints and any stress redistribution (arching) onto the vertical (rock) boundaries. Being a two-dimensional model limits the precision by which three-dimensional characteristics, such as drawpoints and “squarish” type stopes, can be represented, but using appropriate adjustments such as those suggested by Fahy et al. (2009), a reasonable representation can be achieved.

In situ studies and back analysis

To investigate the field behaviour of cemented mine backfill during deposition, the authors undertook a program of in situ monitoring and back analysis using Minefill-2D. Details of this work are presented in Helinski et al. (2011). Both studies were undertaken with cemented paste backfill (CPB) and the cases investigated included a CPB from the Kanowna Belle (KB) mine,

which is obtained from fine gold tailings, and one from Savannah Nickel Mines (SNM), which is composed of coarser nickel tailings. The particle size distribution (PSD) of these two tailings are presented in Figure 5, which also shows the definitions given by Potvin et al. (2005). According to these definitions, both the KB and SNM materials can be classified as paste.

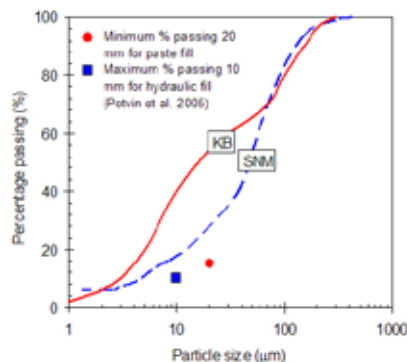


Figure 5 Grading curves for the two backfill materials

To define relevant geotechnical properties for these CPBs, Rowe cell, hydration cell and triaxial testwork was carried out on both materials, in accordance with the procedures described by Helinski (2008). For the KB case, a General Portland Cement was used, but for the SNM case, the cement used was Minecem, a proprietary cement used by a number of Australian mines. In the case studies discussed, the KB material was deposited at a slurry concentration of 75% solids with 3.1% cement content (C_c), while the SNM material was deposited at a density of 79% solids with C_c (Minecem) of 2.7%. Model parameters for the SNM material were determined from laboratory tests on samples mixed to these proportions, but the parameters adopted for the KB material are those from laboratory testing carried out prior to undertaking the field studies. This material was tested at the corresponding density, but with C_c of 3.0% rather than 3.1%. The results were identified as “Minefill B” in Helinski et al. (2007c).

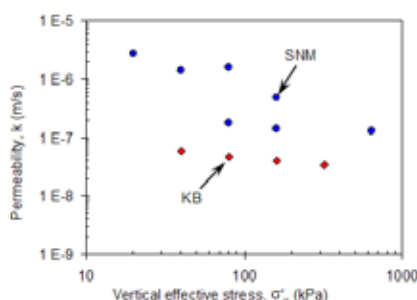


Figure 6 Variation in permeability with effective vertical stress for uncemented backfill materials

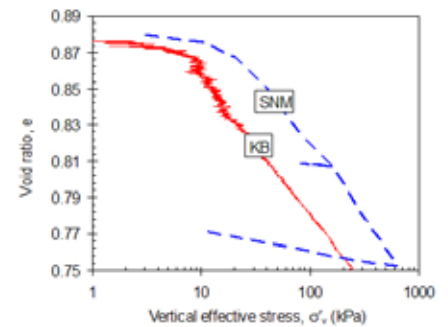


Figure 7 Compression curves for uncemented backfill materials

Figures 6 and 7 present the results from the Rowe cell tests on uncemented backfill, with Figure 6 showing the effect of applied stress on permeability while Figure 7 shows the one-dimensional compression curves. The results from the hydration cell test (Helinski, 2008) are presented in Figures 8 and 9, with Figure 8 showing the development of small strain shear stiffness (G_s , as measured using piezo-ceramic bender elements) with time, while Figure 9 shows the change in permeability with time due to cement hydration.

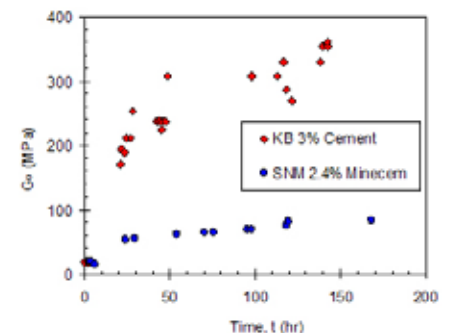


Figure 8 Variation in permeability with effective vertical stress for uncemented backfill materials

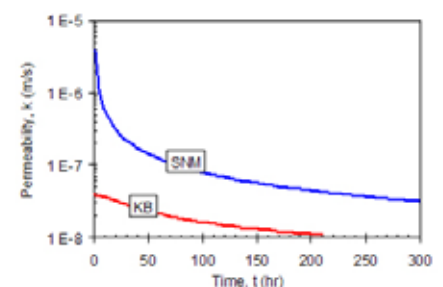


Figure 9 Compression curves for uncemented backfill materials

Overall, the testwork results indicate that the SNM material has a high uncemented permeability and stiffness, but a relatively low increase in stiffness as a consequence of cement hydration, while the KB paste has a low uncemented permeability and stiffness, but the onset of cement hydration

creates a significant increase in stiffness.

KB case study

Field testing at KB involved filling a 40 m tall slope, with plan dimensions of 15 m × 18 m, and with a single 6 m wide × 6 m tall drawpoint at the centre of one of the long sides. The containment barricade was constructed in the drawpoint 6 m from the slope brow. The slope was filled with CPB containing 3.1% cement at a density of 75% solids by weight. Figure 10 shows the slope layout and the position of the instruments.

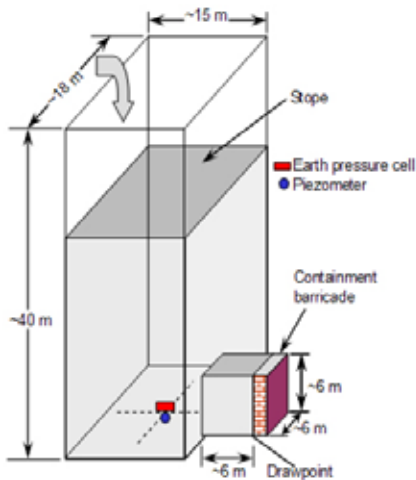


Figure 10 Schematic diagram of the KB slope

Pore pressure was monitored at the centre of the slope floor using a vibrating wire piezometer. Total vertical stress was measured at the same location using a Geokon 4850-01 total pressure cell. This cell is designed to measure total stresses in concrete, and hence is designed to be as stiff as possible and would therefore minimise the likelihood of under-registration due to the increasing stiffness of the backfill as it hydrates.

The filling sequence consisted of filling the first 10 m at a vertical rate of rise of 0.2 – 0.5 m/hr prior to a 24-hour rest period. After the rest period, filling continued at a vertical rate of rise of 0.3 – 0.6 m/hr until the slope was full (184 hours after the commencement of filling).

Figure 11 presents the measured vertical total stress (σ_v) and pore pressure (u) plotted against time during filling for a point on the floor in the centre of the slope.

Previous studies (Fahey et al., 2009; Helinski et al., 2006) demonstrated that arching can only occur if some effective stress develops. Examination of Figure 11 shows that σ_v and u initially increase at the same rate as the applied overburden stress,

indicating that there is no consolidation, and hence no arching early in the process.

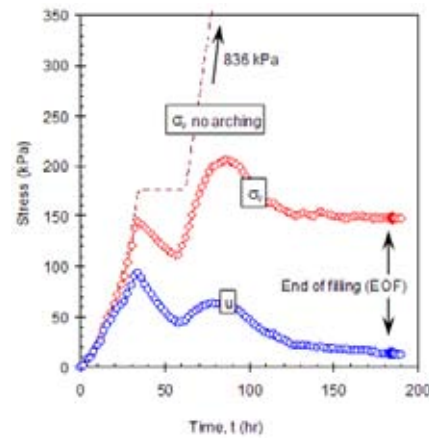


Figure 11 Measured results from the KB slope

However, after about 20 hours, some effective stress starts to develop (u starts to plot below σ_v), and also (though not very clear at the scale of this plot), σ_v is slightly less than “ σ_v no arching”, indicating that load shedding due to arching has started. During the period when filling is suspended (from 34–58 hours), σ_v reduces, but u reduces by a greater amount, indicating some further consolidation and increase in effective stress, which permits a greater degree of arching to develop. When filling recommences, both σ_v and u increase again, but at a lower rate than the “ σ_v no arching” rate, until, at about 90 hr, both σ_v and u start to decrease in spite of filling continuing. These results are discussed further in a later section.

SNM case study

The experimental work at SNM involved filling a 23 m tall slope, with plan dimensions of 10 × 12 m and with two 6 m wide × 6 m tall drawpoints that extended 5 m to previously filled stopes on either side. During filling, pore water pressure was monitored at elevations of 7 and 17 m above the floor using vibrating wire piezometers. Figure 12 presents a section view showing the stope layout and instrument locations. In this diagram, the adjacent stopes are not shown. The filling sequence consisted of filling the first 6 m at a vertical rate of rise of 0.04 m/hr, and then filling the remainder of the stope at a constant rate of rise of 0.1 m/hr, such that the stope was filled in just over 200 hours.

The results from this filling sequence indicate that, unlike the KB case, the rate of increase in pore water pressure is significantly less than the rate at which total stress is applied right from the start

of filling above the transducer depth. This indicates that considerable consolidation is taking place during placement. It is also interesting to note that, again unlike the KB situation, the pore pressure continues to increase while filling takes place, and it is not until filling terminates (at 205 hours) that the pore pressure reduces.

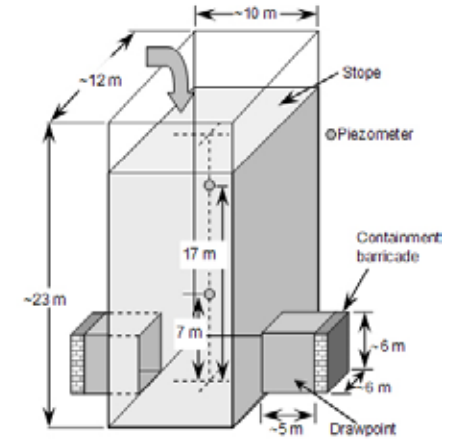


Figure 12 Schematic diagram of the SNM stope

Modelling with Minefill-2D

Input material properties used in the Minefill-2D model for the two backfill materials were obtained from the laboratory data presented previously in Figures 6, 7, 8 and 9, using the procedures outlined by Helinski et al. (2007c) and summarised in Table 1, where G_{o-i} is the initial small strain shear stiffness, G_{o-f} is the ultimate small strain shear stiffness, d is a constant controlling the rate of hydration, t_0 is the time until initial set, E_h is the efficiency of hydration self-desiccation term, c'_u is the ultimate cohesion, ϕ' is the friction angle, q_{u-f} is the ultimate unconfined compressive strength, and c_k and d_k are constants relating permeability and void ratio.

	KB	SNM
G_{o-i} (MPa)	18	19
G_{o-f} (MPa)	560	180
d (1/day)	1.4	2.2
t_0 (day)	0.3	0.5
E_h (cm ³ /g)	0.032	0.005
c'_u (kPa)	125	100
ϕ' (°)	30	38
q_{u-f} (kPa)	433	400
c_k (m/s)	5.0×10^{-6}	5.0×10^{-4}
d_k (-)	40	30

Table 1 KB and SNM material properties adopted for back analysis of in situ test results

These properties were used with Minefill-2D in the back analysis of the in situ measurements. Details of the boundary conditions and drawpoint representation

used for the back analysis are presented in Helinski et al. (2011), but the main assumptions are fully permeable barricades, fixed displacement impermeable flow boundary conditions around the stope boundary. The KB stope was idealised as an 8.2 m wide plane strain stope and the SNM stope as a 5.5 m plane strain stope, in accordance with the recommendations of Fahey et al. (2009).

Kanowna Belle (KB)

Figure 13 compares the measured results from the KB stope with the Minefill-2D output. The high pore pressure (u) during the early stages of filling is due to the low coefficient of consolidation of the paste immediately after placement, and the high vertical rate of rise during filling. However, when cement hydration commences, volumetric reductions occur due to the self-desiccation mechanism, which, combined with the high cement-induced stiffness of the solids matrix, results in a reduction in u and a consequent increase in σ'_v . Because the self-desiccation process occurs relatively quickly after "initial set", the associated reduction in u is rapid.

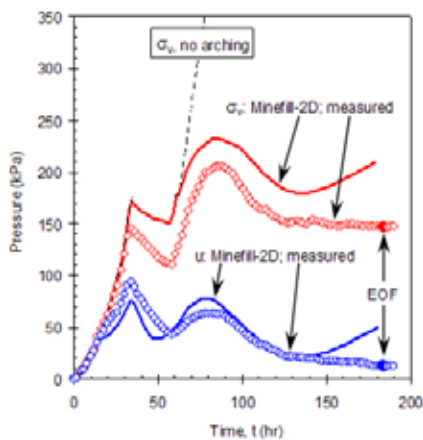


Figure 13 Measured behaviour in the KB stope compared with model results from Minefill-2D

Savannah Nickel Mine (SNM)

Figure 14 presents the results of the measured and calculated pore pressures at the two positions (u_{7m} and u_{17m}) indicated in Figure 12 in the SNM stope. This figure indicates that the calculated pore pressures are higher than the measured values, but the trends are consistent with the field measurements. Both measured and calculated u values are considerably less than the increase in σ'_v from the accretion of fill, and both calculated and measured u values progressively increase until filling is complete, at which point there is a

reduction.

The low u values are thought to be due to a combination of the high permeability and stiffness of the material immediately after placement, and the low rate of filling. The result is that σ'_v from the accretion of fill is almost immediately transferred to the uncemented matrix as effective stress. This type of behaviour is typically associated with hydraulic fills, i.e. tailings fills where the fines have been removed. In these cases, the level of pore pressure is governed by the steady state drainage conditions in the fill, which depend on the restriction to flow created at the base of a stope due to a reduced flow area through the stope drawpoint (Cowling et al., 1988; Traves and Isaacs, 1991).

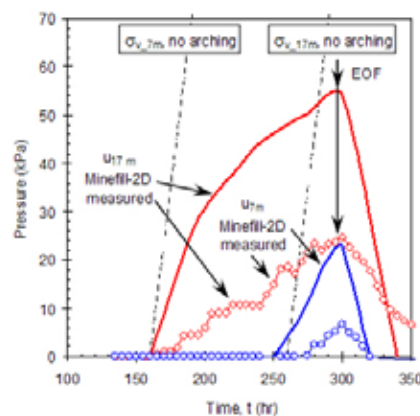


Figure 14 Measured pore pressure behaviour for the SNM stope, and predictions from Minefill-2D

Helinski and Grice (2007) presented monitoring results and back analysis of pore pressures in a number of cemented hydraulic fill stopes. Using in situ pore pressure monitoring, they deduced that the permeability was anisotropic – i.e. that the equivalent horizontal permeability through the drawpoint is often an order of magnitude greater than the vertical permeability within the stope.

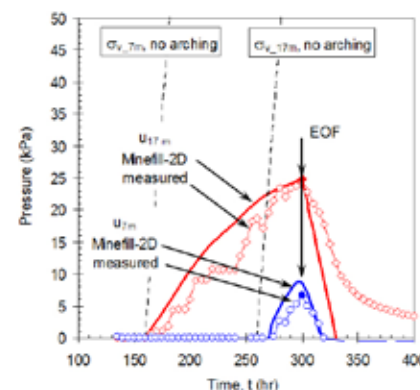


Figure 15 SNM modelling repeated with drawpoint horizontal permeability increased by factor of 10

On the basis of this conclusion, SNM modelling was repeated, but in this case the horizontal permeability through the drawpoint area was increased by an order of magnitude. The results of this modelling are compared with the in situ measurements in Figure 15, which shows much better agreement than in the previous modelling (Figure 14).

Considering the importance of the drawpoint permeability to the model results in the SNM case, KB modelling was also repeated, with the drawpoint permeability likewise increased 10-fold. The calculated pore pressure is compared with that from the previous analysis (Figure 13) and in situ measurements in Figure 16. This comparison indicates that in the KB case, drawpoint permeability has relatively little influence on the model results for the following reasons presented, though the u values from the modified analysis are slightly closer to the measured values than are those from the initial analysis.

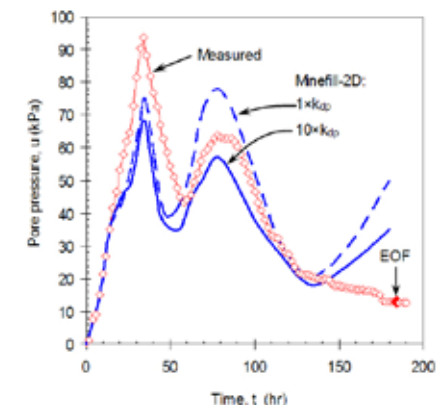


Figure 16 KB modelling repeated with drawpoint horizontal permeability increased by factor of 10

Monitoring discussion

Comparison between in situ measurements in these two different backfills indicates that the mechanism controlling the behaviour during filling varies significantly, depending on material properties and geometric conditions. The most significant factor is that the low permeability of the KB material, combined with the fast filling rates, makes the initial stages of deposition largely undrained. It is not until cement hydration commences, which increases the material stiffness and initiates the self-desiccation mechanism, that pore pressure reduction takes place. As the reduction in pore pressure is primarily dictated by self-desiccation, water drainage through the drawpoint has limited influence on consolidation.

The SNM material has much higher initial permeability than the KB material.

Combined with a relatively low rate of rise, these properties ensure that this material consolidates almost immediately after placement. During placement, pore pressures in the material correspond to the steady state flow condition, which is dictated by the flow restriction through the drawpoint, as explained by Traves and Isaacs (1991) and Helinski and Grice (2007).

Sensitivity study

The work described previously has demonstrated that the consolidation behaviour immediately after placement can vary depending on the material properties and filling rate. In order to demonstrate the consequence of different consolidation conditions, a sensitivity study is presented to assess the influence of cement content, filling rate, and base drainage characteristics on the development of effective stress (σ'_v) and barricade stress (σ_b) for the KB and SNM stopes.

Base case parameters

Figure 17 presents σ_b and σ'_v at the base of the stope plotted against time for both the KB and SNM cases, calculated using the Minefill-2D program with the base case set of parameters used in the previously presented back analysis. This modelling shows maximum σ_b of approximately 66 and 61 kPa for the KB and SNM cases. While the magnitudes of the maximum σ_b are similar, the way in which the stress varies with time is very different. In the KB case, the maximum σ_b develops soon after the commencement of filling, and then it reduces prior to the completion of filling, while in the SNM case, σ_b continues to increase throughout the filling cycle until reaching a maximum at the completion of filling.

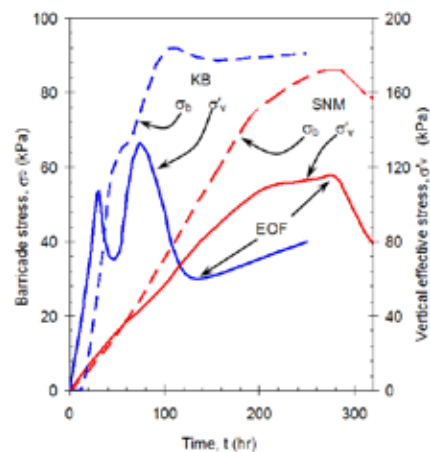


Figure 17 Barricade stresses and vertical effective stresses, KB and SNM modelling with base-case parameters

It is important to note that differences in barricade stresses are the direct consequences of different consolidation behaviour within the stope, which is shown by the rate of development of σ'_v in Figure 17. In the SNM case, σ'_v begins to develop immediately after the material is placed and continues to develop at a rate proportional to the vertical rate of rise in the stope, while for the KB case, σ'_v does not start to develop until about 15 hours after deposition. The lack of σ'_v in the KB case is a consequence of the low coefficient of consolidation (c_v) and high rate of vertical rise. However, as shown in Figure 8, after approximately 15 hours the KB paste begins to develop significant stiffness, and, when combined with the volume changes induced by self-desiccation, creates a significant reduction in u and a corresponding increase in σ'_v . Due to the favourable cementation characteristics associated with KB paste, the rate of σ'_v rapidly passes that for the SNM case and the increase in σ'_v promotes stress arching reducing σ_b .

Effect of changing cement content

To investigate the impact of cement content (C_c) on σ_b and σ'_v , the two case studies were re-analysed, but in this case with C_c reduced to 1% in both cases. The σ_b and σ'_v values thereby obtained for the KB and SNM stopes are compared with the base-case results in Figures 18 and 20 respectively.

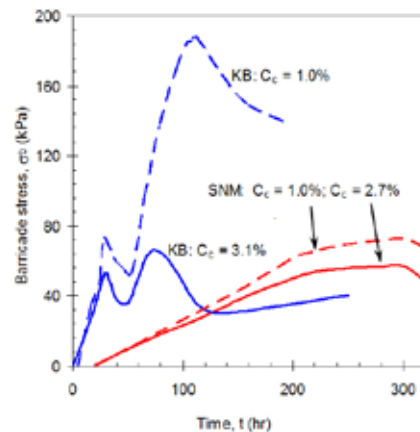


Figure 18 Effect on σ_b of changing cement content C_c

Figure 18 indicates that the SNM case is relatively insensitive to C_c , with the reduced C_c leading to an increase of only 18% in the barricade stress. In contrast, the KB case is highly sensitive to the change in C_c , with the reduced C_c leading to an increase of 170% in the barricade stress.

The reason for the greater sensitivity in the KB case is that for this material, pore pressure reduction is driven by the self-desiccation mechanism rather than by conventional consolidation. Reducing the cement content reduces both the material stiffness and volume reduction due to self-desiccation, resulting in less consolidation and greatly increased barricade stresses.

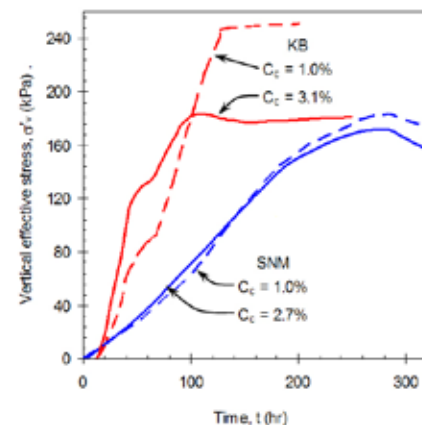


Figure 19 Effect on σ'_v of changing cement content C_c

The influence of C_c on consolidation is further illustrated in Figure 20, which shows little change in σ'_v for the SNM case, but in the KB case the reduction in C_c almost halves the rate of σ'_v development.

Effect of changing the vertical rate of rise

To investigate the sensitivity of σ_b and σ'_v to changes in the vertical rate of rise, the KB and SNM modelling was repeated, but with the filling rates increased by 50%. The calculated σ_b and σ'_v are compared with the base-case results in Figures 20 and 21 respectively. The plots corresponding to the base-case filling rates are denoted "rate×1", while those corresponding to the increased filling rates are denoted "rate×1.5".

Figure 20 indicates that increasing the filling rate by 50% increases σ_b by only 9% for the SNM case, but results in an increase σ_b of over 50% for the KB case. The greater sensitivity in the KB case is because increasing the vertical rate of rise changes the rate at which self-weight stress is applied, but as shown in Figure 21, σ'_v development remains almost constant, meaning that no further arching can take place. Figure 21 also shows that in the SNM case, the increase in total stress, from the increased vertical rate of rise, creates a corresponding increase in σ'_v , which is the reason the maximum bulkhead stress remains unchanged.

To investigate this aspect further,

another SNM simulation was undertaken, but this time the vertical rate of rise was increased by four times the base-case rate.

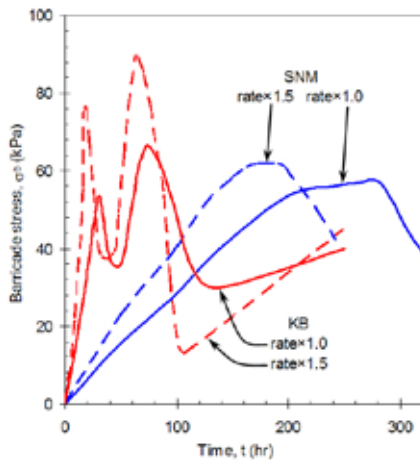


Figure 20 Barricade stresses and vertical effective stresses, KB and SNM modelling with base-case parameters

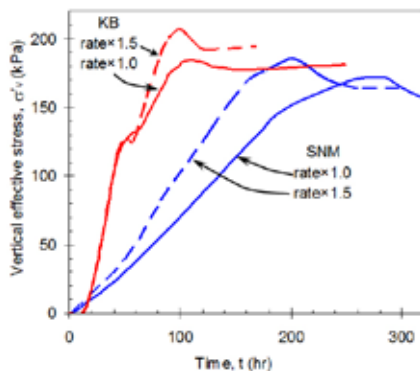


Figure 21 Barricade stresses and vertical effective stresses, KB and SNM modelling with base-case parameters

The barricade stress σ_b from this case is presented in Figure 22 (“rate×4”) along with σ_b calculated for the other filling rates for SNM. While increasing the vertical rate of rise by 50% was shown to create no tangible increase in barricade stress, Figure 22 shows that increasing the filling rate by 4 times creates a significant increase in the maximum σ_b . This result is probably because the combination of the initial paste properties and the increased rate of rise lead to excess pore pressure in the SNM case, which reduce arching, thus increasing σ_b . Figure 22 shows that soon after the completion of filling, the barricade stress falls to the value calculated for the base case. At this stage, the excess pore pressure is expected to have dissipated such that only steady-state seepage-induced pore pressure exists. As these pore pressures are similar to the final stages when filling at the lower rates, the calculated barricade stress is similar.

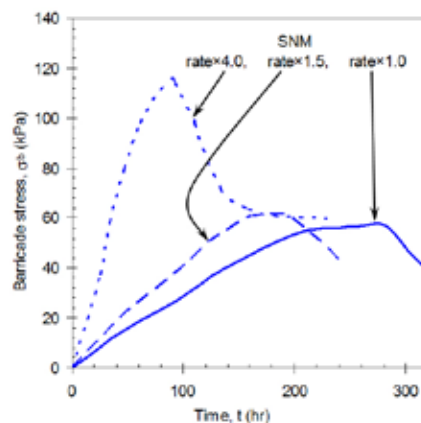


Figure 22 Barricade stresses and vertical effective stresses, KB and SNM modelling with “base-case” parameters.

Effect of changing number of drawpoints

The KB stope investigated had a single drawpoint, while the SNM stope had two drawpoints. To investigate the influence of number of drawpoints on the resulting barricade stress, KB modelling was repeated with two drawpoints, while the SNM modelling was repeated with a single drawpoint. The calculated σ_b and σ'_v are compared with the “base case” results in Figures 23 and 24, respectively, where the number of drawpoints is indicated on the plots (“1-DP” or “2-DP”). Figure 23 indicates that in the KB case, the barricade stress is relatively insensitive to the number of drawpoints, with only a 9% difference in barricade stress, while for the SNM case, the barricade stress is far more dependent on the number of drawpoints, with a 40% difference in barricade stress.

This sensitivity in the SNM case is because the pore pressure regime in the stope is controlled by the stope drainage conditions. A reduction in drawpoint flow area in the SNM stope creates an increase in steady state pore pressure, and a corresponding increase in barricade stress.

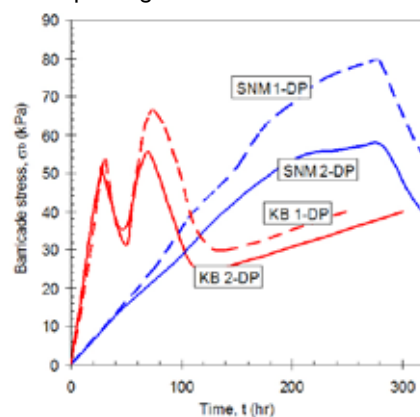


Figure 23 Effect of number of drawpoints on σ_b

The higher pore pressures are reflected in a lower rate of σ'_v development, as shown in Figure 24. In the KB case, the increase in drawpoint area has little influence on the pore pressure regime as pore pressure reduction in this material is dominated by the self-desiccation process, and not by water flow through the drawpoint(s). This is also demonstrated in Figure 24, which shows the development of σ'_v remaining constant, for the KB case, regardless of the drawpoint configuration.

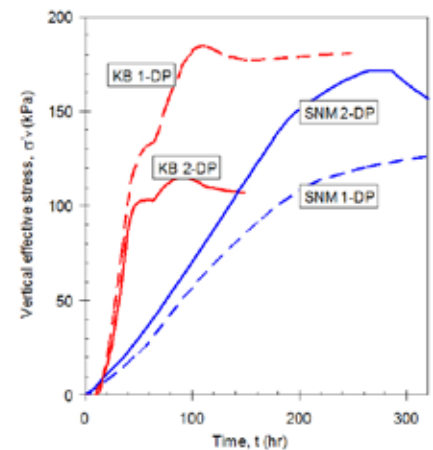


Figure 24 Effect of number of drawpoints on σ'_v

To further illustrate the significance of drawpoint drainage at SNM, a photograph showing the accumulation of paste drainage water in front of a bulkhead is presented in Figure 25. It is important to note that the paste in this case was typical of the paste prepared at this site, having representative fill rheological characteristics, i.e. 180–200 mm slump.



Figure 25 Water draining through the barricade, SNM stope

Sensitivity study discussion

The results of the sensitivity study illustrate that depending on the combination of initial fill material properties (specifically the initial coefficient of consolidation) and the vertical rate of rise, the mechanism controlling consolidation can vary considerably and, as a result, should be addressed differently in design and management.

Specifically, it was shown that if the material consolidates immediately after placement, σ'_v develops as a consequence of the rate of vertical rise and slope drainage. Therefore, bulkhead stress σ'_b is largely independent of cement hydration characteristics, but sensitive to the accumulation of steady-state seepage-induced pore pressures, which are influenced by slope geometry aspects.

If the material is unable to consolidate immediately after placement, then consolidation would be dominated by the cement hydration process. As a result, development of σ'_v is largely independent of geometry influences such as the vertical rate of rise and drawpoint configuration, but highly sensitive to aspects influencing cement hydration. As a consequence, the analysis must consider the rate of cement hydration and the vertical rate of rise when assessing and managing bulkhead stresses.

Gibson framework

Classification system

This article has demonstrated that taking account of consolidation of the backfill is critical for estimating and managing bulkhead stresses and for understanding the in situ rate of development of effective stress during curing. Furthermore, the presented sensitivity study showed that depending on the combination of initial fill material properties (specifically c_v) and the vertical rate of rise (m), the mechanism controlling consolidation can vary considerably, and as a result should be addressed differently in design and management.

To determine the expected behaviour, a fully coupled numerical analysis approach (such as Minefill-2D) could be adopted, but an analytical solution presented by Gibson (1958) provides a very useful initial classification tool that could be used to guide a suitable approach. This solution has been examined in detail in the context of slope backfilling by Fahey et al. (2010).

Gibson's equations express the degree of consolidation that occurs at the surface of a layer of saturated soil undergoing

continuous increase in thickness with time. This solution is based on one-dimensional consolidation conditions, for both drained and undrained base conditions, and assumes that the coefficient of consolidation, c_v , remains constant during consolidation. For the proposed application, this idealisation is considered reasonable.

The solution was presented by Gibson as plots of normalised surface isochrone gradient $(du/dz)/\gamma$ plotted against a non-dimensional time factor T , where T is given as:

$$T = \frac{m^2 \cdot t}{c_v} = \frac{m \cdot H}{c_v}$$

where m is the rate of rise (m/hr), and H is the total filled depth. The plot for the drained-base case is shown in Figure 26. On this plot, a surface isochrone gradient of -1 indicates zero consolidation, while a gradient of 0 indicates full consolidation.

Fahey et al. (2010) also used Gibson's equations to derive the average degree of consolidation throughout the whole thickness each time, and this plot is also shown in Figure 26 for the drained-base case.

On this diagram, regions of behaviour have been labelled consolidating (to the left of these curves), and non-consolidating (to the right of these curves).

Lines are also shown in Figure 26 to represent the values of T for the KB and SNM cases, calculated using the c_v value for these materials in their initial as-deposited state and the actual filling rates used. Also shown for the SNM slope is a line ("SNM×4") representing this case if the filling rate had been four times greater than the actual rate used.

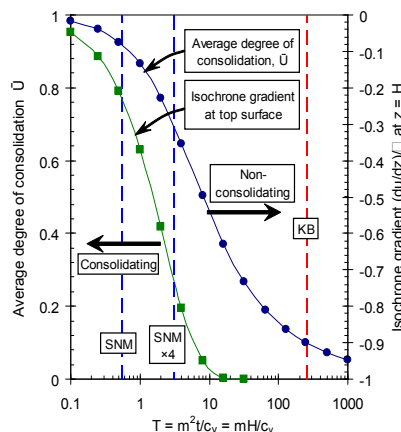


Figure 26 Gibson (1958) solution, as re-interpreted by Fahey et al. (2010)

Figure 26 shows that the SNM material,

at the filling rate used in practice, plots close to the consolidating region, whereas the KB material plots in the non-consolidating region. Fahey et al. (2010) showed that if some consolidation occurred during filling, c_v would change from this initial value: for the SNM material, c_v would increase during filling, so that the degree of consolidation during filling for this case is even more than suggested by Figure 26. However, the KB material is indicated as undergoing no consolidation during filling, and hence c_v would not change during filling. It should be noted here that this solution is only valid up to the start of cement hydration, since from that point onwards, the stiffness changes due to cement bonding, and the self-desiccation mechanism can also start to produce consolidation.

Results presented in the sensitivity study dealing with varying the vertical rate of rise showed that increasing the vertical rate of rise at SNM by a factor of four resulted in the development of excess pore water pressure. Increasing the vertical rate of rise fourfold has the consequence of increasing T to the location shown in Figure 26 ("SNM×4"). At this filling rate, Gibson's chart indicates that the increase in vertical rate of rise shifts the material from a consolidating fill to a partly-consolidating fill.

It follows from this that if a filling scenario plots in the consolidating region, i.e. the SNM case, or any case with $T < \sim 1$, consolidation typically occurs concurrently with filling, and any pore pressures within the fill result only from the restriction to flow imposed by the drawpoint. If the filling scenario is in the non-consolidating region of Figure 26, i.e. the KB case, or any case with $T > \sim 100$, the material properties remain largely constant until the onset of cement hydration, which makes the consolidation behaviour highly dependent on the cement hydration properties. Thus, this plot provides a useful means of initial classification. In this case, for both of these paste fills, according to the classification of Potvin et al., 2005, Figure 26 shows that different behaviours are to be expected, and this has been confirmed by the field measurements and the FE modelling.

Bulkhead stress management systems

When fill is classified as a consolidating fill, the stress imposed on barricade structures is largely independent of cement hydration, and the imposed stress progressively increases throughout the

filling process as the height of fill and steady-state seepage-induced pore water pressures increase. As discussed by Kuganathan (2002), for a given fill height, the barricade stress for a consolidating fill can be linearly related to the development of pore water pressure at the intersection of the stope and drawpoint. As a consequence, the conventional paste filling sequence (of filling to slightly above the stope brow before halting to allow curing) is not particularly appropriate. Rather, a more suitable approach would be to fill such that excessive seepage-induced pore pressure did not occur. This may involve a series of discrete filling and resting periods, much like is adopted for hydraulic fill. However, when applied to a paste fill condition (SNM), the high density of the paste would likely result in extended periods of filling. A real-time system for management risks associated with excessive seepage-induced pore pressures, and therefore excessive bulkhead stress, would be to install a piezometer at the intersection between the drawpoint and stope and ensure that a given trigger level is not exceeded.

In a non-consolidating fill situation, cement hydration dictates the consolidation behaviour. Therefore, the conventional paste fill approach of filling to slightly above the barricade before suspending filling, while hydration and therefore consolidation takes place in the material immediately behind the barricade, appears rational. However, as shown by Helinski et al. (2008), the barricade stress and pore pressure within the fill mass are closely related, such that should the desired cement hydration not take place, through under-dosage of binder, reduced paste density or adverse tailings characteristics, the resulting pore pressure would remain high during the "cure" period and higher-than-expected barricade stresses could be expected. Based on this theory, Helinski et al. (2008) recommended that in situ pore pressure monitoring be used to manage appropriate cure periods. With the installation of appropriate pore pressure monitoring equipment, pore pressure trigger values could be used to indicate an appropriate time to commence the second filling stage.

Curing stress systems

Laboratory experiments show that the application of effective stress during the early stages of curing can make a significant difference to the resulting q_{ucs} . Therefore, in order to more appropriately determine

the strength of cemented backfill from laboratory tests, an understanding of the rate at which effective stress is applied during the initial stages of hydration in the stope is necessary.

In a consolidating fill, effective stress develops immediately after placement. The rate at which effective stress develops is independent of cement hydration characteristics, but dependent on boundary conditions such as the vertical rate of rise (which controls the rate of application of total stress) and the restriction to flow through the base drawpoints (which controls the pore pressure).

A suitably conservative approach to curing consolidating fill specimens under stress may be to adopt a loading rate that corresponds to a lower estimate for the vertical rate of rise and to assume hydrostatic pore pressure conditions, i.e. the worst-case stope drainage situation. Given the same boundary conditions, all mixes would be cured under the same rate of application of effective stress.

If filling involves a non-consolidating fill, the material is not expected to develop any effective stress immediately after placement, but effective stress would be developed as the cement hydration process proceeded. The mechanism for applying effective stress is a reduction in pore water pressure as a consequence of cement hydration. Therefore, the rate of application of effective stress is largely independent of boundary conditions. As a result, a unique rate of effective application exists for each paste mix.

A suitable method for preparing specimens to take advantage of the expected application of effective stress during curing would be to place the specimen in a sealed mould, such as a latex membrane with O-ring seals on the end caps, confine the sides to ensure a K_0 stress path, and apply and maintain a single increment of vertical total stress. If fully saturated, the initial application of total stress would create an equal increase in pore pressure, but as cement hydration took place the pore pressure would reduce, creating an equivalent increase in effective stress.

Cemented fill management guidelines

Based on the understanding of the behaviour developed from the previously mentioned in situ monitoring and back analysis, a series of recommendations have been developed for the design/management of cemented mine backfills, where the

fills are divided into consolidating and non-consolidating fills, according to the proposed Gibson classification system. The proposed design/management system is summarised in Table 2.

Conclusion

This article presents an investigation into the behaviour of cemented mine backfill giving specific focus to the evolution of total and effective stresses during the filling process. Using both field and numerical studies it was demonstrated that the mechanism controlling the deposition behaviour, and therefore critical design and management systems for barricade loads and suitable quality control testing methods, is heavily dictated by the rate of consolidation of the fill immediately after deposition.

Gibson (1958) presented an analytical solution for the development of excess pore water pressure at the surface of a clay layer that is increasing in thickness with time. This solution was proposed in the article as a suitable strategy for understanding the mechanisms controlling the minefill deposition process and selecting appropriate design/management strategies using the fills coefficient of consolidation and vertical rate of rise. This strategy also provides limits for when changes in boundary conditions, such as the vertical rate of rise, can change the behaviour.



"Over the past three decades since paste fill was first developed, the preparation and distribution methods have quickly evolved ..."

Henderson et al., 2005, Handbook on Mine Fill.

Continued

Finally, a table is provided which, based on the classification using Gibson's chart, provides recommendations relating to appropriate mine backfill design and management strategies. Although these

proposed strategies still need to be tested in a range of applications, the framework provided here gives a useful initial indication of the aspects likely to govern the behaviour of a particular fill.

Article references are available on request.

Consolidating Fill ($T \leq 1$)	Non-consolidating Fill ($T \geq 100$)
Analysis with mechanical analysis coupled with steady state seepage induced pore pressures to calculated barricade stresses	Analysis requires full coupling of filling, consolidation and cement hydration
Little point in fill rest/cure period after covering barricade because of low stresses during initial stages of filling	Rest/cure period after covering barricades useful to ensure that hydration and therefore consolidation commences
Manage accumulation of steady state seepage induced pore pressure to ensure calculated barricade stress is not exceeded	Pore pressure good indicator of the onset of hydration – manage duration of rest/cure period with pore pressure monitoring
Barricade stress sensitive to: <ul style="list-style-type: none"> • Number of drawpoints • Flows through drawpoint 	Barricade stress sensitive to: <ul style="list-style-type: none"> • Binder type and content • Mix density • Vertical rate of rise
Significant pore water flows through drawpoint expected. Engineered barricade drainage system useful. Potential to erode waste rock barricades	Little water expected to report to barricades, specific wall drainage typically not necessary
Effective stress applied to fill material immediately after placement	Effective stress not applied to fill until "initial set" of binder
Application of effective stress during curing independent of mix design dependent on boundary conditions, i.e. vertical rate of rise and drawpoint configuration	Application of effective stress unique for each mix design and largely independent of boundary conditions, i.e. vertical rate of rise and drawpoint configuration
Considerable volumetric compression during filling	Little volumetric compression during filling

Table 2 Cemented mine backfill characterisation using the Gibson system



The 10th International Conference on Mining with Backfill will be held from 21–25 March 2011, in Cape Town, South Africa. Since its inception in 1973, the conference has recorded significant advances in backfill technology and has become the forum for international exchange of know-how about cost-effective backfill operations.

For further information, contact:

Julie Dixon, SAIMM, Conference Co-ordinator
Tel: +27 11 834 1273 · Fax: +27 11 838 5923 · E-mail: julie@saimm.co.za

www.saimm.co.za/minefill2011



**Sun International's
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**21–25
March 2011**



MS-RAP: the next generation

by Johan Wesseloo, Australian Centre for Geomechanics, Australia

MS-RAP, MSRRM and other acronyms

For those of you who wondered about it, yes, MS-RAP is the name of a software product and, no, the MS does not stand for Microsoft®.

MS-RAP is an acronym for Mine Seismicity Risk Analysis Program and this software has been part of many geotechnical engineers' toolboxes at some seismically active mines for nearly a decade. The software was initiated and developed in the second phase of the ACG's Mine Seismicity and Rockburst Risk Management Project (MSRRM) lead by Associate Professor Marty Hudyma, currently at Laurentian University, Canada.

Since those early days, MS-RAP has been a deliverable to the MSRRM project sponsors and is used as a way to turn research results into practical and usable tools for the sponsor sites.

Many things have changed during the subsequent phases of the MSRRM project, including the project leaders: Marty was followed by Daniel Heal, currently a technical services superintendent, BHP Billiton Nickel West, who was followed by the author who commenced the project in 2008. The one thing that did remain constant throughout this time was the MS-RAP software engineer Paul Harris, who developed version 1.0.14 and took it to its current state. All indications are that Paul will grow old with MS-RAP.

MS-RAP and the story of the caterpillar

During the current phase of the MSRRM project, MS-RAP v3 is maintained while v4 is being developed. The obvious thing to do at this stage is to provide the reader with a table comparing MS-RAP v3 and MS-RAP v4. After considering this, the author decided to abandon that idea for the sole reason that such a table will be meaningless; a bit like comparing the Serengeti with the Perth Zoo. The change from v3 to v4 can best be described as a metamorphosis. All components that formed the basis for v3 are still present in v4, although in a refined and matured manner, while added capabilities and changes to the internal structure and interface have allowed new opportunities.

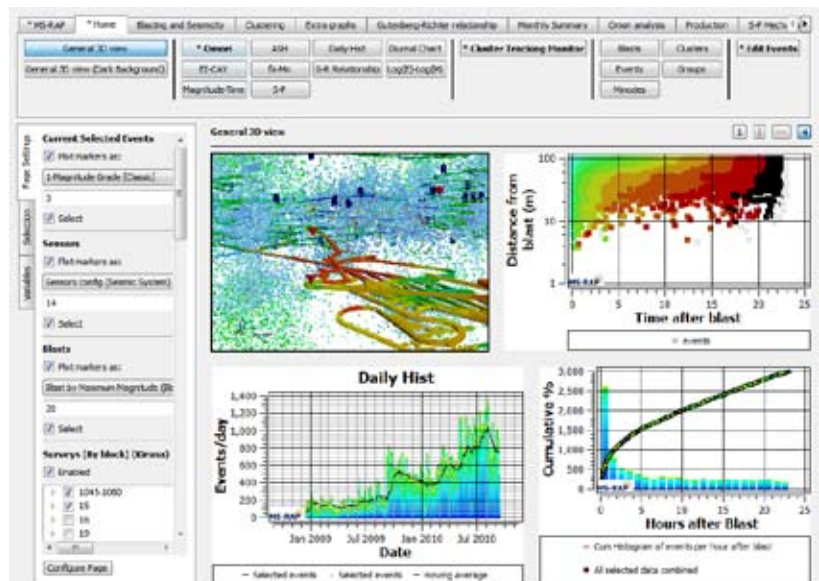
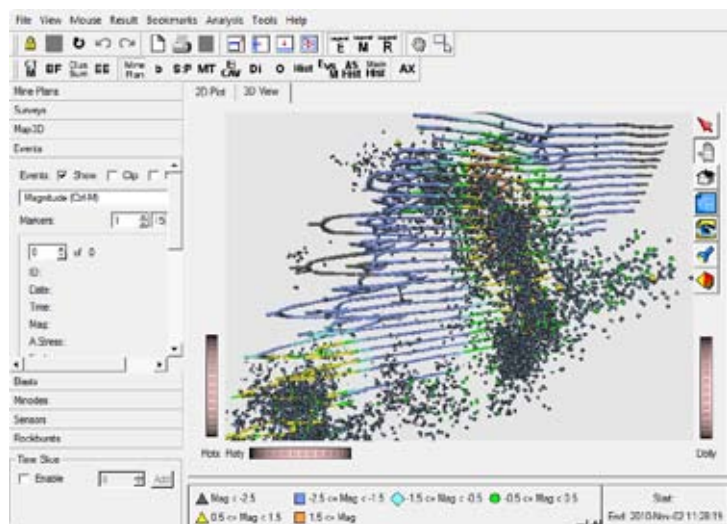
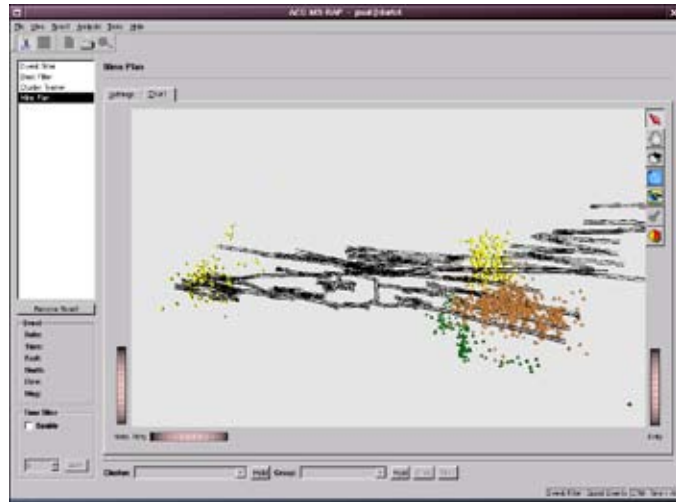


Figure 1 Screenshots of MS-RAP v2, v3 and v4

Several improvements, refinements and enhancements to MS-RAP v3 are being introduced into MS-RAP v4 and several features are added to make the user's life easier. What is more important, however, is the real, fundamental difference between v3 and v4 – that is design philosophy. The change in the design philosophy can be described as the aim to provide flexibility and context.

Flexibility

Mine induced seismicity and geotechnical engineering in general are too complex to be catered for satisfactorily with a “one size fits all” approach. It is important to design the software in such a way that it can be customised and configured to cater for the specific conditions at each site. Where v3 hardcoded the tools and provided a few settings controllable by the user, v4 provides more generalised tools that can be configured and combined to perform very specific calculations. One example is that it allows the mapping of one data set onto another. This enables, for example, performing re-entry analysis and mapping the result onto the blasts, which allows the plotting of blasts by re-entry time.

Phase Four of this ACG research project was financially supported and assisted by the following sponsors:

Minerals and Energy Research Institute of Western Australia (MERIWA)

Major

Barrick Gold of Australia
BHP Billiton Nickel West
BHP Billiton Olympic Dam
Lightning Nickel Pty Ltd
LKAB

Minor

Agnico-Eagle
AngloGold Ashanti Ltd
BCD Resources (Operations) NL
Codelco
Gold Fields Australia
Kirkland Lake Gold Inc.
MMG Australia Ltd
Newcrest Mining Ltd
Newmont Asia Pacific
Perilya
Xstrata Kidd Mine
Xstrata Nickel

Context

The second very important change is a stronger focus on the incorporation of non-seismic data. We are of the conviction that mining induced seismicity, is well ... mining induced, and cannot properly be interpreted or managed if it is not viewed within the context of the constantly changing mining and geotechnical environment. In order to achieve this, non-seismic production and geotechnical data needs to be incorporated. MS-RAP v4 is therefore able to import data from different sources as shown in the following examples.

As mentioned previously, it is not possible to make a simple comparison between v3 and v4. This will be illustrated by showing two of the v4 features that were not possible in v3.

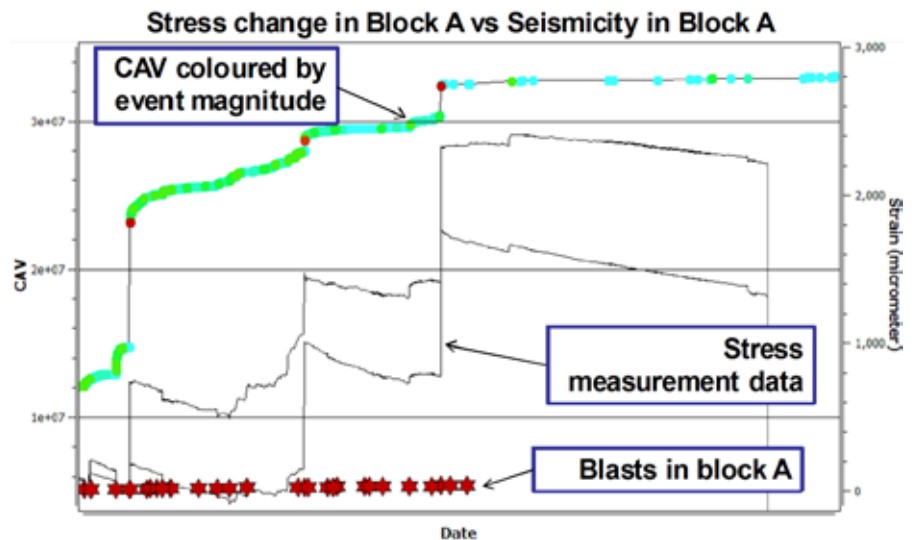


Figure 2 Cumulative Apparent Volume seismicity changes in the same area compared to measured stress changes MS-RAP version 4 – new features

Fence selection

One of the features that has proven valuable to the few pre-release users is the ability to perform selections on graphs with the use of a fence selection, and in 3D view with a three-dimensional version of a fence selection. Several fences can be created on different graphs and 3D views, and can be combined with AND and OR into a “Selection”, as illustrated in Figure 3.

For further illustration, visit www.acg.uwa.edu.au/msrapv4.

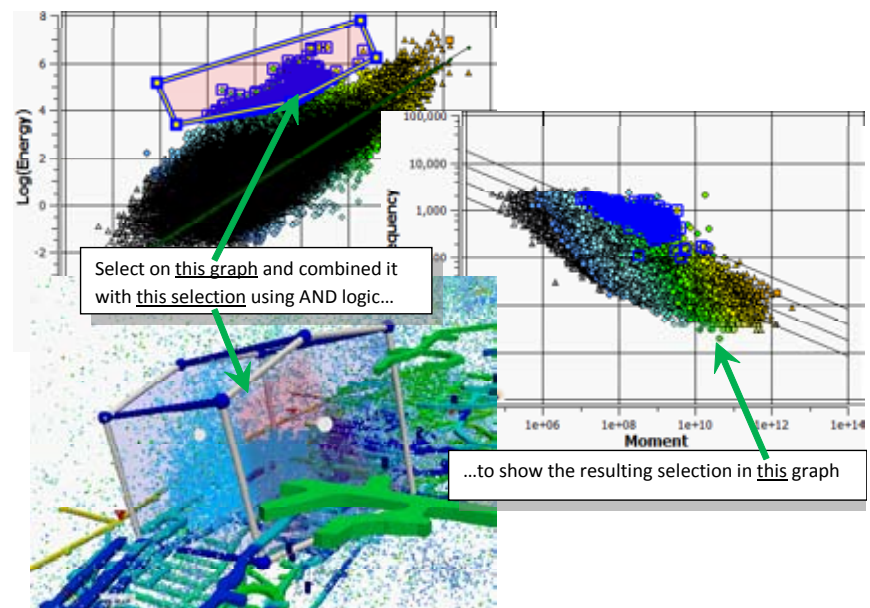


Figure 3 Two and three-dimensional selections in MS-RAP v4

Variables

With the generalisation of the functions built into the software, the ACG introduced a "variables" component. This enables one to configure a tool (calculation, graph 3D view, table view etc.), referring to a variable. Having configured the structure of the tool, one can then scroll the mouse over the variable to see the influence of those variable changes on the graphs (Figure 4).

For further illustration, go to www.acg.uwa.edu.au/msrapv4.

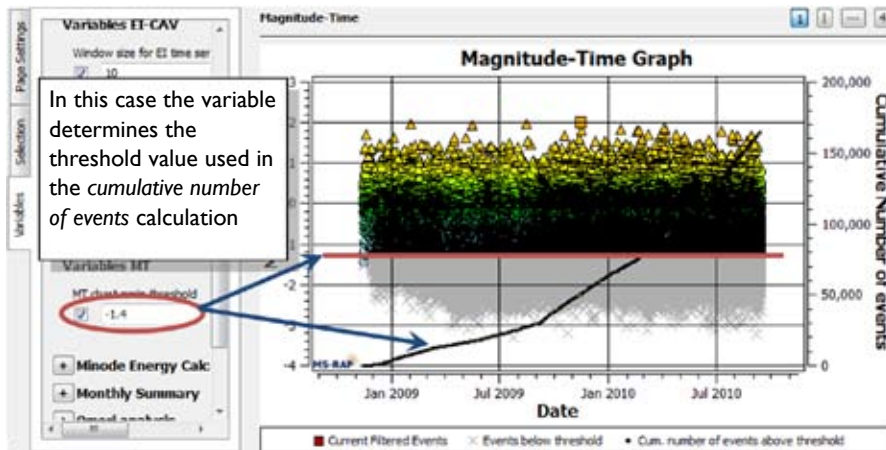


Figure 4 Magnitude-Time graph with a cumulative number of events graph for events above the specified threshold value



Applied research and the development of mine seismicity tools at the ACG are helping mine operators to proactively respond to mine seismicity and rockbursting hazards. For more information contact the ACG.

Deep Mining 2012

Sixth International Seminar on Deep and High Stress Mining

27-29 March 2012, Novotel Langley Hotel, Perth, Western Australia

The ACG is delighted to bring Deep Mining 2012 back to Perth, Western Australia. This follows previous International Deep and High Stress Mining Seminars held in Santiago in 2010; Perth in 2007; Quebec City in 2006; Johannesburg in 2004; and Perth in 2002.

The main objective of this seminar series is to document and disseminate the latest experiences and state-of-the-art technologies in the challenging and evolving area of deep and high stress mining.

Seminar Themes

- Geomechanics risks
- Financial risks
- Case studies
- Numerical modelling
- Rock behaviour under high stress
- Rockburst and seismicity monitoring
- Ground support
- Risk assessment and management
- Ventilation
- Blasting

www.deepmining2012.com

On the horizon ...

Underground Drilling and Blasting

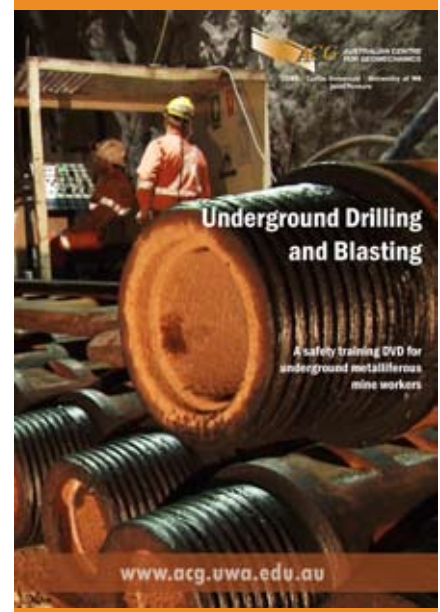
A Safety Training DVD for Underground Metalliferous Mine Workers

All underground mine workers will be exposed to drilling and blasting processes. The aim of this new DVD is to provide workers with the critical knowledge on drilling and blasting to aid appreciation of the importance of these mining processes and their related hazards.

DVD Project Sponsors

Barrick Gold of Australia; BHP Billiton Olympic Dam; Dyno Nobel Asia Pacific; Gold Fields Australasia; Newmont Asia Pacific; Orica Mining Services; and Xstrata Zinc.

This DVD is currently in development. Contact the ACG for further details.



Post-closure liabilities

by Ken Bocking, Golder Associates Ltd., Canada



Introduction

Over the past 20 years, closure planning has become an integral part of mining projects. Many mine closure plans are formulated on the premise that once mining ceases, the implementation of mine closure will be relatively straightforward, quick and final. It is supposed that a “walk-away” scenario will be achieved, with all hazards and liabilities resolved and the land returned to beneficial use. This premise is proving to be simplistic, and there is a growing inventory of mines that have ceased operations, but which have been unable to achieve final closure and to return the mine properties to useful post-closure land use. This article examines some of the issues related to post-closure liabilities.

Post-closure liabilities

Definition of post-closure liabilities

Mine closure plans typically provide a list of physical actions, e.g. demolition of buildings, site cleanup and revegetation, that are to be undertaken once mining ceases. For the purpose of this article, these actions are referred to as “agreed closure measures”. Agreed closure measures would normally also include planned monitoring activities used to verify the effectiveness of the closure measures. The planned monitoring activities that are allowed for in closure plans are often relatively short term in nature, typically in the range of five to 10 years.

At the majority of mine sites, it will not be possible to achieve a “walk-away” closure. In other words, execution of the agreed closure measures is unlikely to resolve all of the future liabilities associated

Potential Liability	Disruptive Agent	Post-closure Management
Erosion by runoff from site	Erosion, channel blockages	Regular maintenance of hydraulic structures
Unacceptable discharge water quality	Mine waste geochemistry	Ongoing treatment and monitoring
Unacceptable groundwater quality	Mine waste geochemistry	Ongoing interception and monitoring
Crown pillar collapse	Gravity, groundwater, collapse of supports	Establish safe setbacks and barriers or fencing, monitoring
Instability of open pit slopes	Gravity, weathering, erosion	Establish safe setbacks and barriers or fencing
Failure of shaft caps	Structural failure, corrosion	Inspection and repair
Dam failure - loss of containment	Extreme earthquakes / liquefaction	Repair dam and clean up spilled tailings
Dam failure - overtopping	Extreme rainfall, plugging of spillways	Repair dam and cleanup spilled tailings
Dam failure - overtopping	Plugging of decants or spillways	Ongoing inspection and maintenance
Public safety hazards	Public access	Ongoing security, fence maintenance
Loss of vegetative cover	Drought, fire	Monitor and revegetate as required
Dry cover failure	Vegetation, ground movement, frost, erosion, intrusion by burrowing animals or humans	Surveillance and maintenance
Water cover depletion	Drought, seepage	Surveillance, seepage mitigation, water diversion
Dusting	Erosion, lack of vegetation	Maintenance of vegetation cover
Excessive seepage	Failure of liner, dry cover or dam	Surveillance and repair

Table 1 Examples of post-closure liabilities

with the closed mine site. There may be long-term costs or liabilities that are typically not funded by financial assurance, nor allowed for in the calculation of the asset retirement obligation. This article defines “post-closure liabilities” as long-term residual liabilities associated with a former mine site that remain after the

agreed closure measures were completed.

Examples of post-closure liabilities

Table 1 provides a list of common post-closure liabilities. It also indicates the disruptive agents, e.g. earthquake, flood, fire, etc., that cause each liability. Can these liabilities be eliminated by implementing

Mine closure

appropriate agreed closure measures, and indeed by designing the mining operation for closure? Unfortunately, the answer is “no, not usually”.

Typical examples of unresolved liabilities include the following:

- **Site runoff:** There will always be some risk of runoff at all closed mine sites. In wet climates, runoff may be continuous. In dry climates, runoff may be a very infrequent phenomenon associated with flash flooding. For this reason, agreed closure measures usually include provisions, i.e. hydraulic structures, channels etc., to direct runoff safely off the site. Unfortunately, hydraulic structures and channels are never maintenance free. They can be seriously eroded when flows occur that exceed the design basis flow for the channel. Serious flooding and damage can also result if hydraulic structures become blocked by debris, vegetation or by the actions of animals. For example, in parts of Canada, the requirement for the frequent clearing of beaver dams often presents the single largest demand for maintenance on closed mine sites.
- **Crown pillar collapse:** Around the world there are many historic mining areas that have been seriously affected by the post-closure collapse of crown pillars where underground mine workings approach the ground surface. For example, Carter and Miller (1995) list and assess unstable crown pillars at multiple sites in Ontario, Canada. Aside from property damage and the direct risk to public safety, unstable crown pillars have effectively sterilised large areas against future development. There is no question that rock mechanics and the application of modern mining

methods have greatly reduced the risk of crown pillar collapses above modern mines. Also, agreed closure measures sometimes include stabilisation of crown pillars by structural spanning, backfilling, or blasting. Nonetheless, it is not usually possible to completely eliminate the risk of collapse or subsidence.

- **Dam failure:** At many mine sites, one or more dams will be left onsite after closure to contain water and/or tailings in perpetuity. Currently, these dams are designed to remain stable under seismic loading, and this design normally also takes into account the possibility of seismically induced liquefaction of the tailings, of the dam itself and also of the foundation of the dam. The intensity of the seismic loading used for design purposes is selected by taking into account the seismicity of the area, the failure consequence category of the dam and the design life of the facility. As an example, it is not unusual to design a closed tailings dam to resist seismic loadings associated with an earthquake with a return period of 1,000 years. While this is a reasonable engineering approach, it does leave some residual risk. Firstly, there is uncertainty in predicting the seismic loading and also in predicting how the dam and its foundation will respond to the loading. Secondly, there is a chance, albeit small, that a seismic event will occur that exceeds the selected return period.
- **Cover failure:** Dry covers are now commonly used to rehabilitate tailings and rock dumps, especially in arid and semi-arid environments. Where acid rock drainage is an issue, the dry covers are designed to limit the ingress of water and/or air to inhibit oxidation

of the sulphide minerals. Covers on coal wastes aim to reduce oxygen flux in order to prevent spontaneous combustion and the resulting danger to the public. The integrity of the dry cover can be affected by climatic conditions (such as frost or drought), by erosion, and also by damage from vegetation roots. Ground movement has been especially problematic for dry covers on rock dumps. If the cover makes use of geosynthetics such as geomembranes or geotextiles, the finite longevity of geosynthetics will be a concern because the design life of closure works is effectively in perpetuity. It may be necessary to allow for the periodic replacement of geosynthetics over the long term.

Post-closure management of liabilities

Table 1 lists possible management approaches that can be taken in dealing with particular residual liabilities and disruptive events that can occur after closure. These approaches generally fall into three categories:

- **Routine inspection and maintenance:** Routine inspection and maintenance is necessary for water control and conveyance facilities, as well as for maintenance of vegetation. Where active effluent treatment is provided after closure, planned maintenance can include equipment replacement/upgrades, dredging and disposal of water treatment sludge, and similar activities. In many closed mining facilities, some degree of environmental monitoring may be required in the medium and long term. Because the actions in this category are by definition “routine”, it is relatively straightforward to predict the ongoing effort, cost and schedule associated with them.
- **Keeping the public away from unresolved hazards:** Where it is not practicable to remove public safety hazards, the next best thing is to keep the public away from those hazards. For example, rock mechanics studies can be completed to establish “safe setbacks” around open pit slopes or potentially unstable crown pillar areas. These areas can then be securely fenced off. Fences and other barriers need to be inspected, maintained and periodically replaced in perpetuity. It is relatively straightforward to predict the costs and schedule associated with such maintenance. Hazardous lands are likely to be unavailable for future productive use after closure. At the



Tailings area before remediation



Tailings area after remediation by flooding

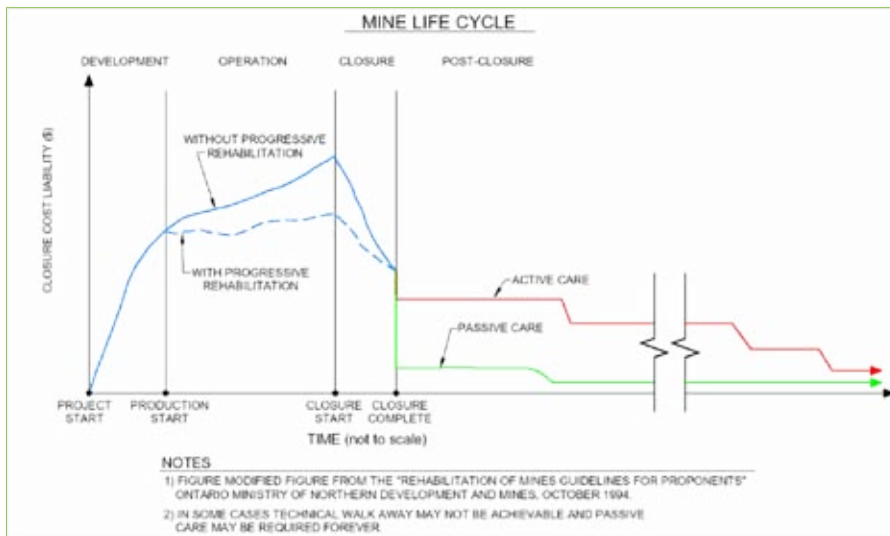


Figure 1 Schematic of mine closure liability variation through the mine life cycle

very least, their future land use may be restricted, and these restrictions may be registered on the land title.

- **Repairs as necessary:** The disruptive events that can bring about the need for repairs to closed facilities are stochastic in nature. Engineering facilities such as dams and spillways are necessarily designed for certain deterministic limits, such as maximum earthquake accelerations or flood flows of a particular return period. There will always be a residual risk that a disruptive event will occur which will exceed the design limit and such long return period events may cause serious damage. The most realistic way of dealing with this residual risk of unplanned disruptive events is to have contingency plans in place to repair the damage. For example, if a tailings dam fails because a long return period seismic event or flood occurs, then the dam needs to be repaired and any tailings that have escaped need to be cleaned up and put back into containment. It is not possible to predict in advance the magnitude and timing of future disruptive events; they can only be dealt with as a probability function. As a result, the cost and timing of repairs must be dealt with as a probabilistic risk.

Liabilities associated with final disposition of mining properties

By default, most closed out mining properties remain in the hands of mining companies. This is not a desirable outcome for companies because it leaves them with unresolved liabilities with respect

to properties that may have little or no remaining commercial value. They will likely incur costs on an ongoing basis for the inspection and routine maintenance of the property. They are also at some risk of facing large expenditures should low probability events occur, such as a tailings dam failure due to a large seismic event.

For this reason, it is desirable to dispose of mining properties after the closure measures have been completed. Bocking et al. (2009) discussed the following options for disposing of mining properties:

- Continuing to retain the properties and using the company finances to cover post-closure liabilities.
- Returning mining leases to the government (with a lump sum payment to cover unresolved liabilities).
- Transferring ownership to a special purpose company and using insurance to manage post-closure liabilities.
- Transferring viable assets to local communities.

Unfortunately, there will likely be costs associated with each of these land disposition options. These costs are yet another component of the total post-closure liability on mining properties.

Financial aspects of post-closure liabilities

Variation of liabilities through the mine life cycle

Figure 1 provides a schematic representation of how the financial liability associated with the closure of a mine will vary throughout the mine life cycle. The liability will increase while the project is

first developed. It will continue to increase throughout the operation of the mine, corresponding to increases in the size of facilities such as open pits, underground mine workings, tailings basins and waste rock piles. Normally, the financial assurance amount is calculated on the total liability that will occur at the end of the mine life.

The amount of the liability at closure can be partially offset by carrying out progressive reclamation during the operating period, for example by covering and revegetating parts of waste piles as they become inactive. In the first few years after mining operations cease, the agreed closure measures will be implemented, and this will substantially reduce the amount of the liability. The measures will not however reduce the liability to zero, i.e. a walk-away scenario was not achieved in the scenario shown in Figure 1. Rather, there will be some residual post-closure liability. The amount of post-closure liability will be greater if "active care" such as ongoing water treatment is necessary at the particular mine site.

Estimation of the post-closure liabilities

Procedures for estimating costs associated with mine closure, as distinct from the post-closure liabilities that are the topic of this article, are now fairly well established. Parshley et al. (2009), provide an explanation of how different types of estimates, each with slightly different estimating rules, are required for different purposes. These closure costs are usually expressed as a single lump sum cost, which is calculated deterministically. This deterministic approach can be justified for closure costing for several reasons:

- The agreed closure measures are well defined and are agreed upon between the mining company and the regulator.
- The majority of closure costs usually comprise capital costs incurred within one or two years of the date of mine closure, so the timing of these costs is predictable.
- Closure measures are normally engineered using design standards that incorporate defined return periods for disruptive events. The probabilistic liabilities associated with longer return period events are normally ignored.
- Closure plans often include a commitment to carry out certain actions after closure, i.e. inspections, periodic water quality sampling and even water treatment. However, the commitment

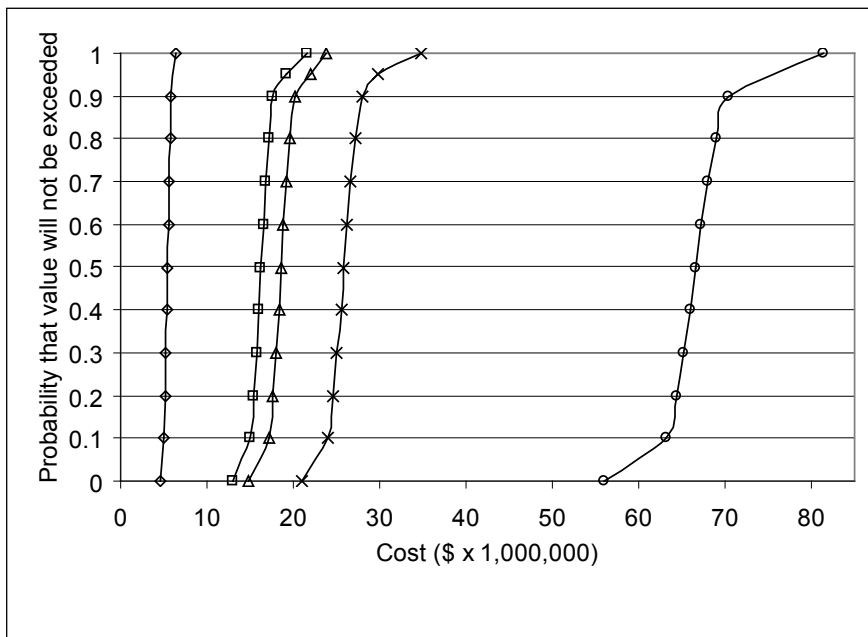


Figure 2 Typical stochastic predictions of care and maintenance costs for five tailings areas

event from 50 years of precipitation records. For this reason, risk assessment is usually conducted by an expert panel and includes three steps: hazard identification (failure modes), risk analysis (severity and likelihood of failure), and risk evaluation (risk acceptance and mitigation). Expert judgement is commonly used to provide input into the model with respect to the consequences of extreme events, the threshold at which damage first occurs, and the likelihood of its occurrence.

Post-closure liabilities are ultimately integrated in a computer model and they are expressed in terms of cost. While it is still commonplace to calculate a single value of expected post-closure cost, there is growing recognition that the cost estimate itself is inherently variable. A probabilistic based cost estimate can provide a more robust answer. The results are expressed as an “S-shaped” curve of cost versus the probability of exceedance. Figure 2 shows typical probabilistic cost predictions.

Based on a stochastic simulation with cost parameters that have predetermined probabilistic distribution functions, the cost model can answer questions such as, “how much money does a mining company need to satisfy the outstanding liabilities 50% of the time?” The process of computing the likely range of overall costs and also the distribution of each individual cost component can be very instructive. It can indicate cost-effective actions that can reduce the liability. For example, the construction of a back-up “dry spillway” in an impoundment can reduce the consequences of plugging the main service spillway. In short, risk modelling can serve as an invaluable tool to help manage the outstanding liabilities.

Conclusion

Mine closure planning has advanced significantly over the past two decades. As mining properties complete their agreed closure measures, the realisation is growing that most properties will continue to have residual liabilities post-closure. It is necessary to use risk modelling techniques to realistically evaluate the value of these post-closure liabilities.

Article references are available on request.

is often for a relatively short period such as five or 10 years. These costs are usually reduced to a net present cost (NPC) using an agreed discount rate.

The use of deterministic estimates is generally not adequate for the calculation of post-closure liabilities however, for several reasons:

- The “design period” for post-closure is effectively in perpetuity, so it is difficult to justify ignoring the residual liabilities associated with disruptive events that have a return period greater than the return period that was used for the deterministic design of the facilities.
- The timing of “repair as necessary” actions is completely unknown and thus the NPC of these actions can only be assessed using risk modelling.
- It is not realistic to represent as a single deterministic cost figure the sum of the liabilities from low probability events of uncertain timing over a design life of forever. The output from risk modelling is a cost versus probability curve which represents the liabilities in a more realistic manner.

Mine closure planning has advanced significantly over the past two decades.

Application of risk modelling approaches

Within the Canadian regulatory framework, the costs to mitigate unplanned, long return period events are typically excluded from the financial assurance for closure. However, in special circumstances, the proponent has been required to adapt a probabilistic approach to estimate the post-closure liabilities. During the closure of the federally regulated Quirke uranium tailings management area in Ontario, Canada, a probabilistic performance assessment was carried out for the decommissioned facility (Welch et al., 1997). The assessment considered long term performance and made provisions for extreme disruptive events that may occur over a long period of time. A Monte Carlo stochastic model was run repeatedly for time periods ranging from 200 to 1,000 years. The results were expressed in cumulative probability curves for damages (or lack of performance) and cost. Interestingly, the results indicated that the overall liabilities for the repair of damage from extreme events, such as seismically induced dam failure, are generally small compared to the costs for routine surveillance and maintenance.

Risk assessment is increasingly being used as a tool to evaluate residual liabilities for closed mining facilities. There are usually difficulties related to the lack of precedent data. For example, it is generally unreliable to extrapolate a 10,000 year return rainfall



Ken Bocking,
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ACG First International Seminar on the Reduction of Risk in the Management of Tailings and Mine Waste



Organised by the Australian Centre for Geomechanics, The University of Western Australia, the First International Seminar on the Reduction of Risk in the Management of Tailings and Mine Waste held in Perth, Western Australia was a great success.

Mine Waste 2010 was established as a forum to present and review the current state of practice in the field of mine waste. At the seminar, a diverse range of topics was presented and explored, including new and emerging technologies for managing tailings, appropriate design parameters and methods of analysis, utilisation of geosynthetics, long-term design and management issues, and achieving acceptable and sustainable closure. Although the majority of the papers presented at the seminar were by Australian authors, the international nature of the topic addressed by the seminar was evidenced by contributions from Austria, Canada, India, Poland, South Africa, Sweden, the United Kingdom and the United States of America.



Photo courtesy of Ashley Prince

As Perth turned on its spring charm, almost 120 mine waste and tailings practitioners, consultants, researchers and suppliers were offered an introduction from Winthrop Professor Andy Fourie, The University of Western Australia - seminar chair. This was followed by an

excellent keynote address presented by Michael Shelbourn, manager, geotechnical engineering, Barrick Gold of North America, entitled, "Geotechnical design verification and performance assessment of tailings storage facilities". Shelbourn noted that "while the increasingly stringent demands of fiscal performance, environmental stewardship and social accountability are likely to drive future mineral processing to the generation of drier waste products, most mine tailings are still discharged as conventional slurry of mineral solids and process fluids into above-ground storage facilities".

Tailings storage facilities can represent one of the greatest risk sources for a mine site, and the geotechnical design of any such facility should be conducted at a level appropriate for the satisfactory management of that risk. During operation, work is required to verify the geotechnical design parameters and conditions, which, at least for the deposited tailings, must often be assumed in the design stage, and to assess the performance of the facility, including consideration of eventual closure and reclamation.

The first day focussed on the related issues of managing waste rock, covering topics ranging from the design of appropriate landforms to visualisation tools and technologies.

Gary Bentel opened day two of the seminar with a keynote address titled, "The real value to the mining industry of leading-practice waste management". Bentel commented that the "real value to the mining industry of leading-practice waste management lies in the adequacy and sufficiency of the industry's investment in its future license to operate. If mining wastes and their associated risks are not

managed correctly, or if we do not provide adequate financial assurances to close waste facilities in such a way that they do not have enduring long-term impacts, we will almost certainly leave future generations with major environmental, financial and social burdens, and hence seriously jeopardise the sustainability of the mining industry".



Photo courtesy of KCGM and Ledestone TV

Mine Waste 2010 explored the current procedures and techniques for minimising the risks that are inherent to the management of mine tailings

Day two continued with an excellent presentation on the proposed Australian National Committee on Large Dams (ANCOLD) Guidelines – risk management aspects by Gary Bentel and Bruce Brown, Rio Tinto Technology and Innovation. Brown noted that, "ANCOLD sees the need to assist the mining industry and the mining community by providing technical advice on appropriate standards for tailings management and to provide a forum for the support of technical development of these structures. As part of this strategy, ANCOLD has established a sub-committee including leading practitioners from consulting, mining academia, and regulator backgrounds to review their 1999 Guideline on Tailings Dam Design, Construction and Operation. The scope

of the new guidelines had been extended to provide particular guidance on the use of risk assessment techniques to assist decision making in various aspects of tailings dam management and to provide guidance on design issues related to dam closure and post-closure performance. These include issues such as consequence assessment, freeboard requirements, seepage control, earthquake design methods and recommended factors of safety, with particular attention to the potential significant difference between tailings dams and conventional water dams”.

The topical issues of management and operations, geosynthetics in mining, and planning, legal and environment were also examined.

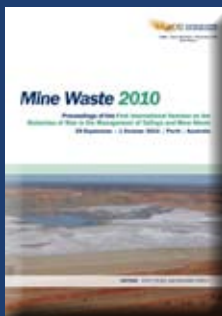
The presentations of the final day covered geochemistry, material characterisation, co-disposal, and thickened tailings. Those attending then departed to the four corners of the world having gained a shared knowledge and insight into mine tailings and waste management of their industry and research peers.



Photo courtesy of Ashley Prince

The ACG was enabled to host this event with the generous support of its sponsors, namely Golder Associates Ltd, Coffey Mining Pty Ltd and ATC Williams.

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



www.acg.uwa.edu.au/shop

Sixth International Conference on Mine Closure



*September 18-21, 2011
Lake Louise, Alberta, Canada*

Mine Closure 2011 will bring together multidisciplinary industry specialists to share information about innovative mine closure techniques, strategies, processes and products with the goal of minimising environmental and social impacts.

Delegates from North America and all over the world will be attending Mine Closure 2011.

Abstracts are due 1 February 2011 and must be submitted online to www.mineclosure2011.com

Connecting Mine Closure Practitioners Around the World

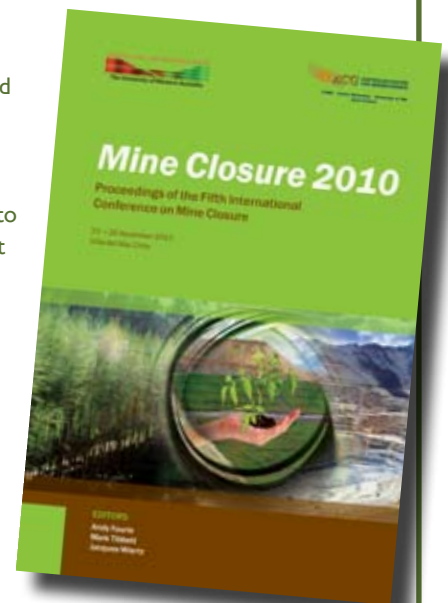
Proceedings of the Fifth International Conference on Mine Closure

The Mine Closure 2010 conference held 23–26 November in Viña del Mar, Chile, is the fifth in a series of international conferences initiated in 2006 by the Australian Centre for Geomechanics and the Centre for Land Rehabilitation of The University of Western Australia. The Mine Closure Conference series is a well-recognised international forum that puts technical excellence first. This event provides industry professionals committed to responsible and sustainable mining with a unique opportunity to interact with their counterparts from different countries, and share ideas and experiences on innovations related to mine closure design, planning and operation.

The Mine Closure 2010 proceedings reflect the shift in the business culture, changes in closure operations and practices as well as the increasing maturity of the mining community involved in the closure design and planning.

Contact the ACG to purchase the proceedings.

www.acg.uwa.edu.au/shop



Mining in saprolites

by Phil Dight, Australian Centre for Geomechanics, Australia

Mining in the regolith-MIRE; mining in saprolites; mining in unconsolidated sediments

The regolith is defined as the altered, unconsolidated or recemented cover that overlies coherent bedrock. Saprolite (from the Greek for putrid rock) can be described as the soft, typically clay rich decomposed rock formed in place by chemical weathering of igneous sedimentary and metamorphic rocks, while the unconsolidated sediments/tertiary deposits can be found in valleys overlying significant iron ore deposits. What do they have in common?

All of these units present challenges for slope stability assessment and design around mining projects. This includes coal mines, iron ore mines, nickel laterites, volcanogenic massive sulphides (VMS) systems, and gold projects. The major problem has been the ability to characterise the materials, and to obtain realistic strength and deformation parameters that adequately reflect the behaviour of the materials.

A proposal has been distributed to interested mining companies seeking to develop the geotechnical parameters and a classification scheme that could encompass mining in these units. Support for the project has been received from gold companies working in saprolites and lake basin sediments; iron ore mines with tertiary deposits; and coal mines with overburden comprising weak and weathered sediments and saprolitic granites.

It is clear that there is a need to better understand these materials. Some saprolite

slopes exceeding 160 m have been mined and some tertiary deposits could have slopes of more than 250 m. Often designers are resorting to an inappropriate use of the published rock mass failure criteria, e.g. the GSI system, and coupled with simplistic limit equilibrium analysis ignore issues of transient or coupled groundwater flow. Within the units there is a wide distribution of clay mineralogy which reflects the weathering process of the underlying protolith.

A review of literature highlights the lack of reporting on slope stability and geotechnical parameters obtained from testwork in the regolith/saprolites for mines. A review of a large database and some civil applications (for example, Hong Kong) gives an overall impression that poor sampling and testing undertaken during mining results in strength parameters which do not reflect performance.

Exploratory work has been undertaken by Monash University to investigate the intact properties of weathered mudstones. This has been reported by Donald, Johnston, Chiu and others, and has shown that there is a strong correlation between the saturated moisture content of the rock and its strength and deformation properties. There is a common link in the weathered materials surrounding the mineralised systems where the rock mass would be expected to have leached and has a high porosity. The mechanical properties are very sensitive to the strain rate used in the laboratory which, by necessity, needs

to be at least two orders of magnitude less than the recommendations of the International Society of Rock Mechanics. However, a review of the available published data would suggest that this very important aspect has been overlooked in some mining investigations.

Johnston and, more recently, Mostyn and Douglas, have shown that for weak rock the generalised Hoek-Brown approach, where "a" approaches one, could be applicable. To date, nobody has taken the intact material and looked at the influence of relic structure on the weak rock mass properties.

A concern has always been that there is very little data that could be used to build a comprehensive geological model of the weathered rock. As detailed by Deere and Patton (1966) and, more recently, by Santi (2006), without a comprehensive geological model any analyses are at best meaningless.

To this end, the work undertaken in characterising the geochemistry of the weathered materials by CSIRO (Butt, Anand, and Munday) could significantly improve the geotechnical model. When combined with geophysics to obtain in situ density, a much more detailed lithological profile could be developed to assist slope stability analyses. Until recently, most of this information has not been readily available to geotechnical engineers.

The ACG would welcome enquiries from industry who would like to participate in this project. Please contact Winthrop Professor Phil Dight for further details.

Article references are available on request.



Phil Dight,
Australian Centre for
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Australia

Slope Stability 2011

18–21 September 2011, Vancouver, Canada

International Symposium on Rock Slope Stability in Open Pit Mining and Civil Engineering

CALL FOR ABSTRACTS

Abstracts are invited for Slope Stability 2011. This specialty meeting will provide a forum for mine operators, consultants, engineers, suppliers and researchers to exchange views on how to promote best practices in slope stability investigations, design, monitoring and management. The technical programme will focus on the transformation our profession is undergoing as unprecedented amounts of quantitative data become available through new measurement and analysis techniques. Authors are invited to share their experiences through case studies together with papers featuring innovative solutions.

KEY DATES

Deadline for abstract submission	31 December 2010
Notification of abstract acceptance	31 January 2011
Submission of full papers	15 May 2011

www.slopestability2011.ca

New ACG events for 2011!

Advanced Application of Seismology in Mines Short Course

Tuesday 10 – Friday 13 May 2011, Perth, Western Australia

Seismic monitoring systems have become integral tools for monitoring in underground, hard rock mines. Most mines use only a fraction of the data recorded by their systems, with most data used reactively following large seismic events, rather than proactively to identify potential rock mass failures. This four day short course will provide a solid technical background about seismicity in mines. The course will utilise the ACG's MS-RAP software to investigate seismic data and evaluate seismic hazard and seismic source mechanisms in mines. Due to the significant tutorial nature of this course, the number of participants is limited to twenty.

MS-RAP Version 4 User Training Course

Monday 16 – Wednesday 18 May 2011, Perth, Western Australia

This three day user training course will focus on the advanced features of the software and ways of adapting the built-in tools to develop specialised site focused tools. This training is essential for users of MS-RAP version 4 that would like to utilise the full power of the software. During this course each of the basic components of MS-RAP v4 will be explained and tutorials and exercises performed. Topics will include, amongst others, understanding the data structures of MS-RAP v4, the setting up of marker styles, performing basic and complex calculations, building basic and complex filters, using variables, and data mapping. This training course will be of particular interest to the sponsor representatives of the ACG's Mine Seismicity and Rockburst Risk Management Project, attendees of the ACG's Advanced Application of Seismology in Mines Short Course, and companies interested to sponsor this leading ACG research project.

Dynamic Support Workshop

Thursday 19 – Friday 20 May 2011, Perth, Western Australia

With the worldwide trend of mining resources becoming increasingly deeper, many mines manage rockburst risks using, amongst other means, dynamic capable ground support. Given the current state of technology in this area and the lack of accepted design methods, one of the challenges is to base the selection of the dynamic support systems on a rational engineering design process. The ACG aims to facilitate a workshop that will collectively advance the dynamic support design processes presently used by industry. The workshop will concentrate on the back-analysis of rockburst damage case studies from the participants' sites. Attendance at this workshop will be restricted to the sponsor representatives of the ACG's Mine Seismicity and Rockburst Risk Management Project, and invited guests.

Please contact the ACG for more details about these exciting new events.

EAGCG Eastern Australia Ground Control Group

The Eastern Australia Ground Control Group held a meeting in October 2010 in Adelaide. Four interesting keynotes were presented:

- Stephen Priest presented a "personal journey" applying probabilistic analysis to rock slopes.
- Max Lee summarised the contents of Ground Control Management Plans.
- Tim Sullivan discussed the lessons learnt from the Yallourn coal mine batter failure.
- John Rodney presented the legal framework for OHS and Mining and the harmonisation of OHS laws commencing 1st January 2012.

Several case studies were also presented in which sites discussed the implementation of their Ground Control Management Plans. Presentations were also given on aspects of risk associated with caving and tunnelling, the role of the geological model, the perspective of the regulator and on chartered professional status. The next meeting will be held in April/May 2011.

www.eagcg.org

Proceedings of the Fifth International Seminar on Deep and High Stress Mining

Deep Mining 2010 was held in Santiago, Chile from the 6-8 October 2010 and attracted excellent international participation. These black and white proceedings feature 42 technical papers from around the world.



www.acg.uwa.edu.au/shop

ACG event schedule*

Environmental Geochemistry of Mine Site Pollution - An Introduction Short Course	Brisbane, 10–11 March 2011
Shotcrete Quality Assurance and Quality Control Workshop	Canberra, 20 March 2011
International Forum on Safe and Rapid Mining Productivity Development	Canberra, 24 March 2011
Prediction of Beach Slopes Workshop	Perth, 3 April 2011
Rheology Workshop	Perth, 4 April 2011
14th International Seminar on Paste and Thickened Tailings	Perth, 5–7 April 2011
Advanced Application of Seismology in Mines Short Course	Perth, 10–13 May 2011
MS-RAP Version 4 User Training	Perth, 16–18 May 2011
Ground Support in Open Pit Mining Seminar	Perth, 17 May 2011
Geotechnical Engineering for Open Pit Mines Seminar	Perth, 18–19 May 2011
Data Driven Slope Design Workshop	Perth, 20 May 2011
Advanced Ground Support in Underground Mining Seminar	Perth, 26–28 September 2011
Mine Backfill Seminar	Perth, 29–30 September 2011
Open Pit – Underground Mining Interaction Workshop	Perth, 7–8 November 2011
Fourth International Seminar on Strategic versus Tactical Approaches in Mining	Perth, 9–11 November 2011
Blasting for Stable Slopes Short Course (TBC)	Perth, 16–18 November 2011
Total Tailings Management Seminar	Perth, 7–8 December 2011
Stress Measurement Workshop	Perth, 26 March 2012
Sixth International Seminar on Deep and High Stress Mining	Perth, 27–29 March 2012

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

*Perth's
Kings Park tree
lights
at Christmas*

*The ACG team wishes you and your family a very safe and peaceful festive season.
We thank you for your support and encouragement during 2010
and look forward to an exciting 2011.
Our office will be closed from Friday 24 December 2010,
reopening on Monday 10 January 2011.*

ACG December 2010 Newsletter

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Consolidation, fragmentation and the structure of the mining industry

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Post-closure liabilities

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