Newsletter

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Application of unsaturated soil mechanics to open pit slope stability

by Professor Harianto Rahardjo, Alfrendo Satyanaga, Nanyang Technological University, Singapore; and Professor Ken Mercer, Australian Centre for Geomechanics

Introduction

Unsaturated soil mechanics is an important new branch of soil mechanics and has only very recently begun to be applied to the field of open pit slope stability. The main concern related to any open pit mining excavation is obviously the stability of slope during excavation. Seepage and stability analyses for the walls of open pits developed in sediments and highly to completely weathered in situ rock, have historically been performed using the principles of classical (saturated) soil mechanics. However, this method does not account for the additional benefit of unsaturated shear strength that can develop in the pit walls. The unsaturated state of these materials may develop as a result of them being situated either above a pre-mining water table or due to phreatic surface drawdown from dewatering activities during mining. In both cases, an unsaturated zone will be formed within the cut slope forming the excavation. When soils become unsaturated, the pore water pressure within the soil become negative with respect to the atmospheric pressure. The negative pore water pressure or matric suction contributes to the shear strength and overall stability of the cut slope around the excavation. The slope however, may become more susceptible to failure during the rainy season as the infiltration of rainwater will result in a partial reduction of the shear strength of the unsaturated soil. Nevertheless, it is reasonable and advantageous to incorporate unsaturated soil properties in the stability analyses of open pit excavation, particularly in rapidly mined pits, provided rainfall is accounted for.

This article illustrates the application of unsaturated soil mechanics theory for solving seepage and slope stability problems applied to open pit mining. The fundamentals and theories of unsaturated soil mechanics are summarised. Measurements of unsaturated soil properties are presented to describe the measurement technology that is available

for the laboratory testing of unsaturated soil. The numerical analyses of slopes with different matric suction profiles and different unsaturated shear strength properties are presented to illustrate the effectiveness of matric suction in maintaining slope stability.

Unsaturated soil mechanics principles

Two stress state variables are required to describe the behaviour of unsaturated soil: net normal stress (σ-ua), and matric suction (u_a-u_w), where σ is total normal stress, u_a is the pore air pressure and u_w is the pore water pressure. Relationships between shear strength or volume change with stress state variables are expressed as constitutive equations. All constitutive equations used to describe the mechanical behaviour of unsaturated soils can be presented as an extension of the equations used for saturated soils. Table 1 summarises several unsaturated soil mechanics equations related to seepage and slope stability problems. The constitutive equations for unsaturated soils show a smooth transition to the constitutive equations for saturated soils when the degree of saturation approaches 100% or matric suction goes to zero.

Soil water characteristic curve, permeability function and shear strength

The soil water characteristic curve (SWCC) defines the relationship between the water content and suction of the soil and is usually described by both a drying curve and a wetting curve. It was suggested that the SWCC be used to represent the relationship between volumetric water content (θ_w) and matric suction. The volumetric water content of the drying curve is higher than that of the wetting curve due to hysteresis (Figure 1). Hysteresis can be observed in the SWCC due to the non-uniform distribution of pore sizes and the presence of air in the soil. The drying SWCC test can be carried out using a Tempe pressure cell (Figure 2(a)) and a pressure

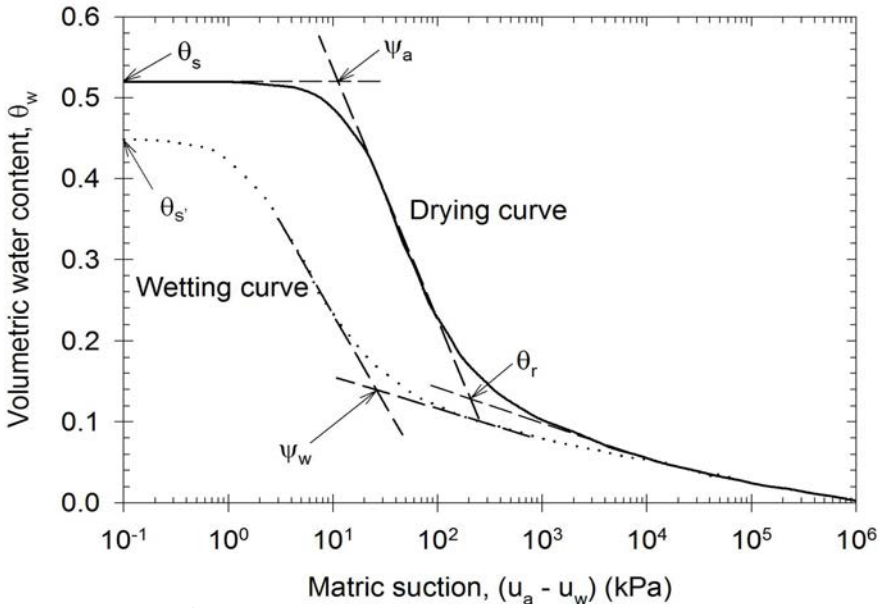


Figure 1 A typical SWCC

Table 1 Principles and equations for unsaturated soil mechanics

Principle	Unsaturated soil	Equation
Stress state variables	$(\sigma - u_a)$ and $(u_a - u_w)$	1
Shear strength	$\tau = c' + (u_a - u_w)\tan\phi^b + (\sigma - u_a)\tan\phi'$	2
	$c = c' + (u_a - u_w)\tan\phi^b$	3
Flow law for water (Darcy's law)	$v_w = -k_w(u_a - u_w)(\partial h_w / \partial y)$	4
	$h_w = y + (u_w / \rho_w g)$	5
Unsteady state seepage	$\frac{\partial}{\partial x}\left(k_w \frac{\partial h_w}{\partial x}\right) + \frac{\partial}{\partial y}\left(k_w \frac{\partial h_w}{\partial y}\right) = m_2^w \rho_w g \frac{\partial h_w}{\partial t}$	6
Slope stability based on limit equilibrium		
Moment equilibrium	$F_m = \frac{\Sigma\left[c'\beta R + \left\{N - u_w\beta\frac{\tan\phi^b}{\tan\phi'}\right\}R\tan\phi'\right]}{\Sigma W x - \Sigma N f}$	7
Force equilibrium	$F_f = \frac{\Sigma\left[c'\beta\cos\alpha + \left\{N - u_w\beta\frac{\tan\phi^b}{\tan\phi'}\right\}\tan\phi'\cos\alpha\right]}{\Sigma N \sin\alpha}$	8

Where:
τ = shear stress, c′ = effective cohesion, c = total cohesion as the sum of c′ and the increase in shear strength due to matric suction, ϕ′ = effective angle of internal friction, ϕ^b = angle indicating the rate of increase in shear strength due to increase in matric suction, k_w = unsaturated coefficient of permeability, v_w = low rate of water, $\partial h_w / \partial y$ = hydraulic head gradient in the y-direction, g = gravitational acceleration, y = elevation at certain point above the datum, ρ_w = density of water, h_w = hydraulic head, t = time, N = the total normal force on the base of the slice, W = the total weight of the slice of width 'b' and height 'h', β = the sloping distance across the base of a slice, α = the angle between the tangent to the centre of the base of each slice and the horizontal, f = the perpendicular offset of the normal force from the centre of rotation or from the centre of moments, x = the horizontal distance from the centreline of each slice to the centre of rotation of the centre of moments, R = the radius for a circular slip surface of the moment arm associated with the mobilised shear force, S_m for any shape of slip surface.

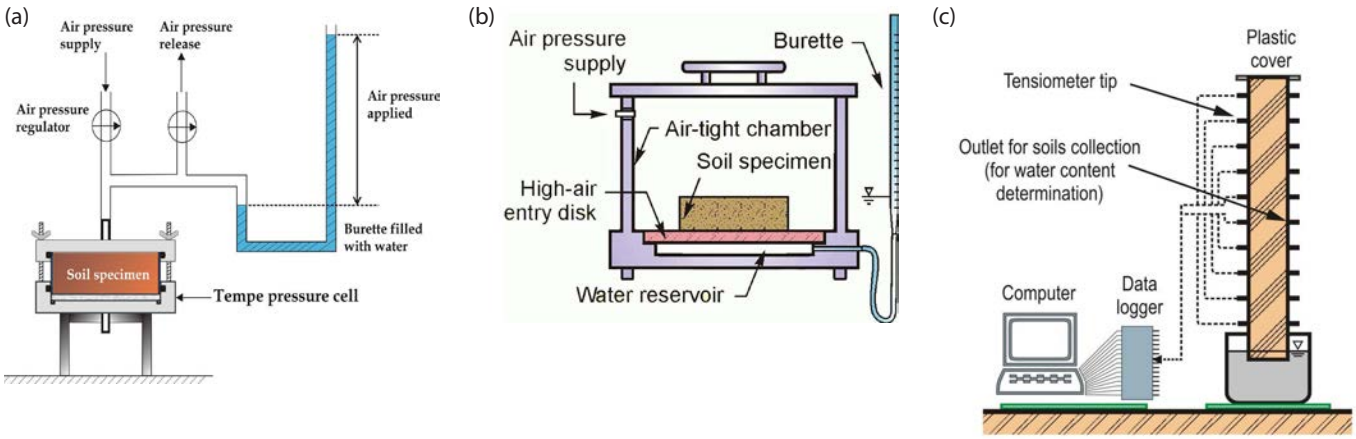


Figure 2 Schematic diagram of (a) Tempe cell; (b) pressure plate; and (c) capillary rise tube for SWCC test

plate extractor (Figure 2(b)), while the wetting SWCC test can be performed using a capillary rise open tube test Figure 2(c)).

The SWCC variables shown in Figure 1 are: saturated volumetric water content (θ_s), residual volumetric water content (θ_r), air-entry value (Ψa) and water-entry value (Ψw). The saturated volumetric water content is the water content in a saturated condition (at very low matric suction), while the residual water content is the water content at which a large suction change is required to remove additional water from the soil. The air-entry value of a soil is the matric suction value where air starts to enter the soil during a drying process. Various equations have been proposed to best fit the SWCC.

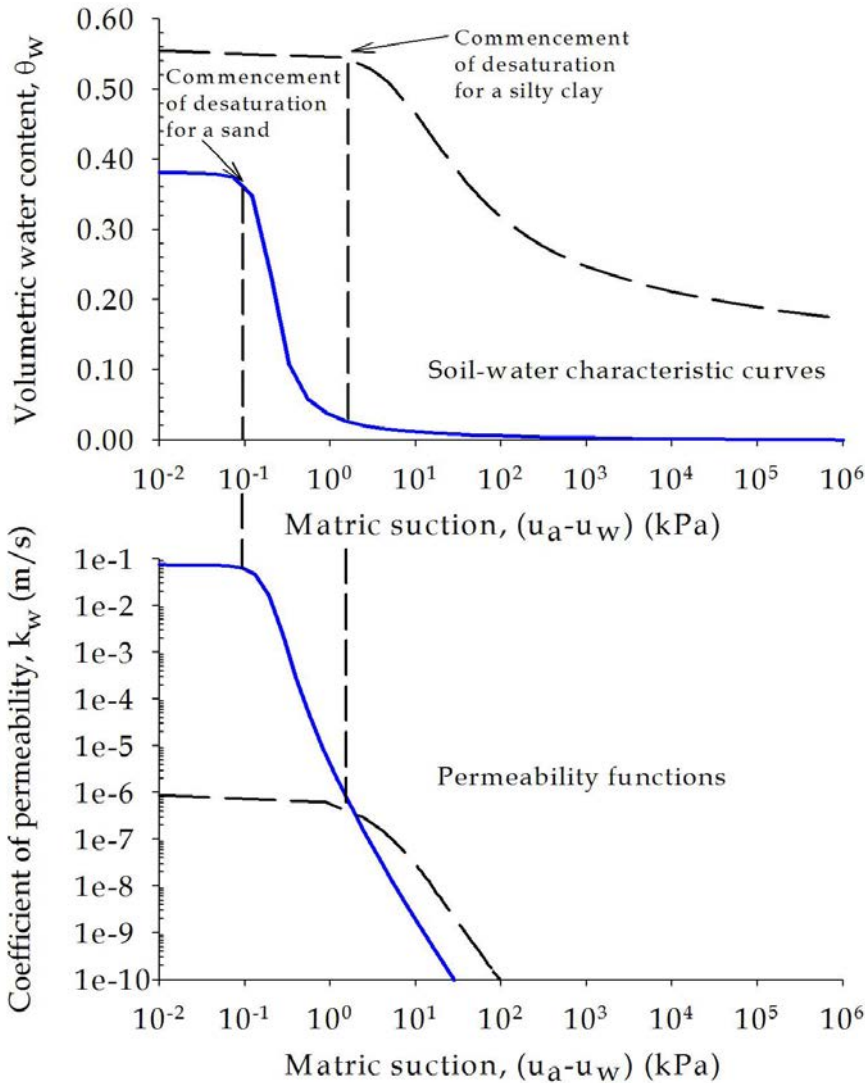


Figure 3 Relationship between SWCC and permeability functions for a sand and a silty clay

Those equations are required to model the SWCC as a continuous function and to estimate the permeability function of soil from the fitting parameters. The SWCC variables can also be determined using the parameters of the best fitting equations, as suggested by Zhai and Rahardjo.

Fredlund and Rahardjo defined the coefficient of permeability with respect to the water phase as a measure of the space available for water to flow through soil. Unlike the coefficient of permeability of a saturated soil, which is constant, the coefficient of permeability of an unsaturated soil is not constant, but it is a function of the volumetric water content of the soil (Figure 3). The determination of unsaturated coefficient of permeability by laboratory experiment is a tedious and time-consuming process. Therefore, an indirect method using a statistical model is commonly used to predict the permeability function from the saturated coefficient of permeability, k_s, and the SWCC. The statistical method is based on the assumption that the permeability function and the SWCC are primarily determined by the pore size distribution of the soil.

The measurement of the shear strength of soil under unsaturated conditions can be carried out using a modified triaxial cell, (Figure 4(a)). The modified triaxial apparatus is capable of controlling and measuring pore air and pore water pressures in the soil specimen independently using the axis-translation technique. Details and procedures pertaining to performing unsaturated triaxial tests can be seen in Fredlund et al. The results of the unsaturated triaxial tests are interpreted using an extended Mohr–Coulomb envelope. The Mohr circles at failure can be plotted in a three-dimensional graph (Figure 4(b)). The failure envelope intersects the shear strength versus matric suction plane at a total cohesion, c. The total cohesion obtained at various matric suctions are plotted to give the ϕ^b angle. Fredlund et al. assumed a constant value of ϕ^b for the entire matric suction of soil. However,

further studies have shown that ϕ^b is equal to ϕ' for matric suctions less than or equal to the air-entry value, and ϕ^b is less than ϕ' for matric suctions higher than air-entry value.

Slope stability analyses

Two types of unsaturated slope stability analyses are used to illustrate the effect of the matric suction on the stability of an open pit slope. The first type of analysis was conducted using total cohesion method while the second type of analysis was conducted using an extended shear strength method. The stability analyses using the total cohesion method were carried out to obtain the variation in Factor of Safety (FS) for slopes with 35 and 40° overall slope angles, under different matric suction profiles and with different ϕ^b angles. Stability analyses using the extended shear strength method were undertaken to obtain the variation of the FS both during and after rainfall, for a slope with a 35° overall slope angle.

Total cohesion method

In the total cohesion method, the stability analysis was carried out by incorporating matric suction into the cohesion of the soil. In this method, the matric suction was considered as a percentage of the hydrostatic negative pore water pressures above the groundwater table (25, 50, 75 and 100%), (Figure 5(a)). Matric suction was multiplied by tan ϕ^b to give an increase in total cohesion due to matric suction (Equation 3

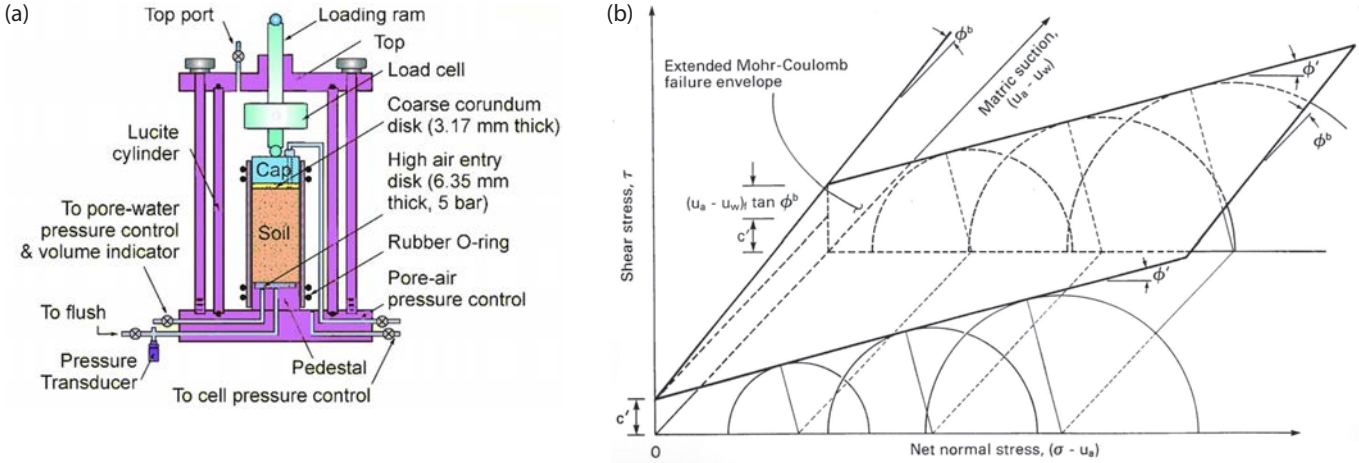


Figure 4 *(a) Schematic diagram of modified triaxial cell for unsaturated shear strength test; and (b) extended linear Mohr–Coulomb failure envelope for unsaturated soil*

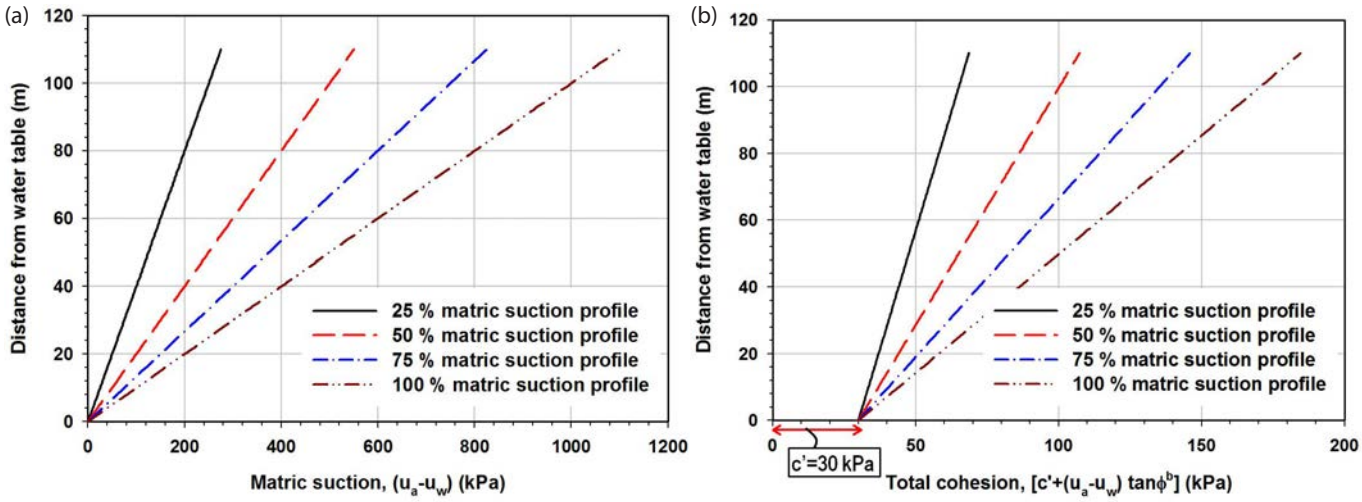


Figure 5 *(a) Different matric suction profiles as a percentage of the hydrostatic profile above the groundwater table; and (b) total cohesion profiles corresponding to the different matric suction profiles for ϕ^b = 25% ϕ'*

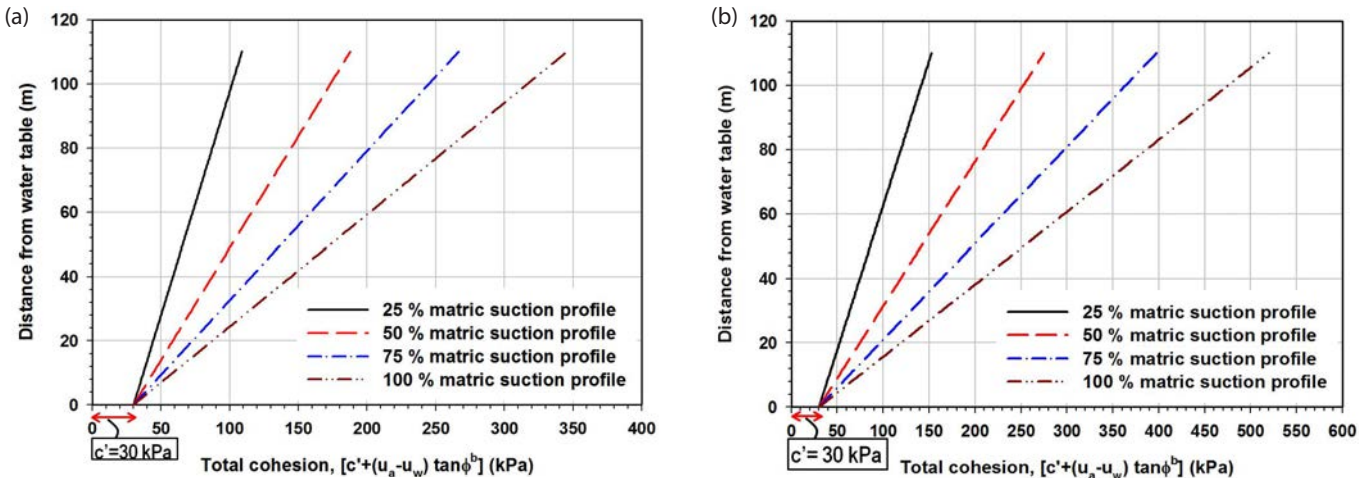


Figure 6 *Total cohesion profiles corresponding to the different matric suction profiles; (a) ϕ^b = 50% ϕ'; and (b) ϕ^b = 75% ϕ'*

in Table 1). 24 slope stability analyses were carried out using this method. Analyses 1 to 12 were carried out on a slope with a 40° slope angle. Analyses 1 to 4 were conducted using four different percentages of matric suction profiles with ϕ^b = 25% ϕ' (Figure 5(b)). Analyses 5 to 8 were conducted using ϕ^b = 50% ϕ' (Figure 6(a)). Analyses 9 to 12 were performed using ϕ^b = 75% ϕ' (Figure 6(b)). Analyses 13 to 24 utilised the same soil properties as those used in analyses 1 to 12, but they were conducted on a slope with 35° slope angle.

The slope stability analyses were performed using SLOPE/W in accordance with the Bishop's simplified method of slices. The height of the slope model was 100 m and the groundwater table was assumed at 10 m below the toe of the slope (Figure 7). The soil within the slope layer was considered to be silty clay with an effective cohesion of 30 kPa and an effective friction angle of 32°. The shear strength contribution from matric suction was incorporated into the total cohesion of soil (Equation 3). The unsaturated zone above the groundwater table was divided into seven sublayers (Figure 7). Each layer had a total cohesion corresponding to its distance from the water table as given in Figures 5(b), 6(a) and (b) for different values of ϕ^b angle.

The variations in the FS of slopes with 35 and 40° slope angles for different matric suction profiles and different ϕ^b angles are presented in Figure 8. The FS increased with the increase in the percentage of matric suction profile and the higher value of ϕ^b angle. The lowest FS was observed when matric suction was not considered in the analysis (0% matric suction profile). On the other hand, the highest FS was observed when 100% matric suction profile was considered in the analysis. Figure 8 shows the importance of matric suction in maintaining the stability of the slope. If matric suction is ignored in the stability analyses, the slope should have already failed since the FS of either slope was close to or less than 1. However, a matric suction profile as low as 25% would be sufficient to significantly improve the stability of slope in most of the cases analysed, illustrating the benefit of considering matric suction in the stability analyses of open pit excavation.

Extended shear strength method

In the extended shear strength method, a transient seepage analysis using the governing flow, Equation 6 (Table 1), was carried out to calculate the changes in pore water pressures (matric suction)

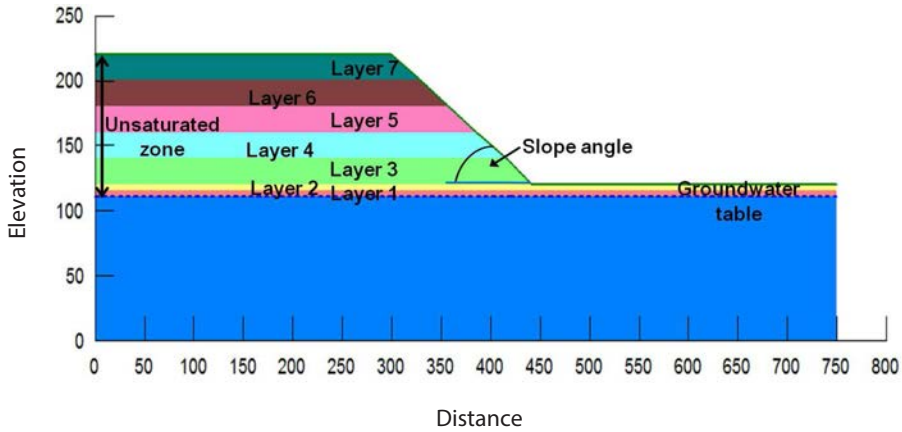


Figure 7 Numerical model for slope stability analyses using total cohesion method

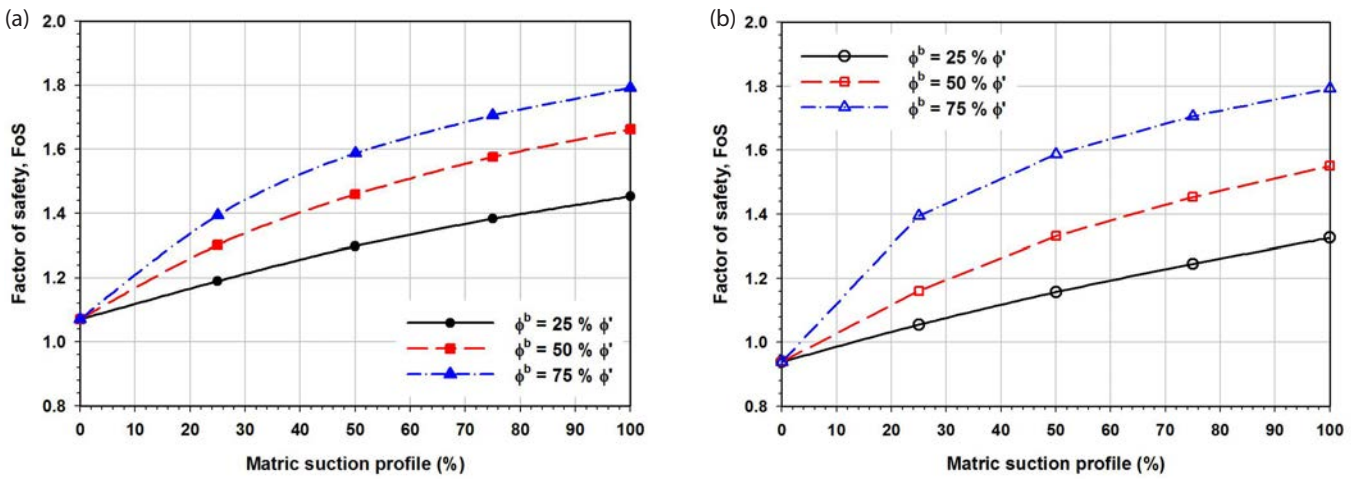


Figure 8 Variation of FS for slope with (a) 35° slope angle; and (b) 40° slope angle for different matric suction profiles

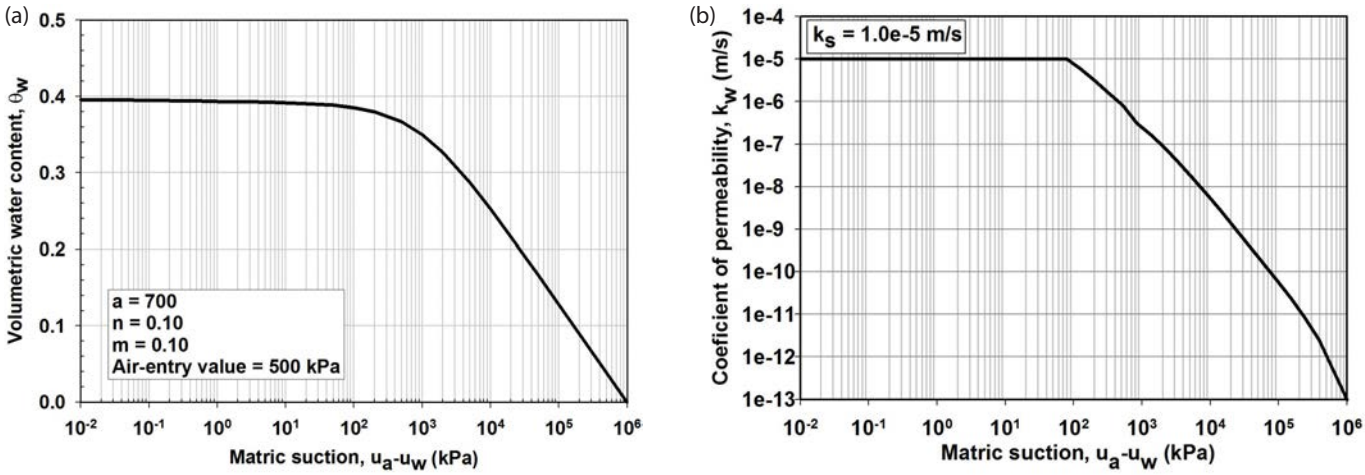


Figure 9 (a) SWCC; and (b) permeability function of soil used in this study

during and after rainfall. The changes in matric suction were then incorporated into SLOPE/W to calculate the variations of FS during and after rainfall using Bishop's simplified method of slices (Equation 7). The seepage analysis was performed using a finite element seepage modelling software, SEEP/W. The required primary data for seepage analysis were the SWCC and the unsaturated coefficient of permeability of soil (Figure 9). The geometry and the boundary conditions for the seepage analysis are illustrated in Figure 10. The left and right edges were located at a distance of 3 Hs (Hs = slope height) from the crest and the toe of the slope to avoid any effect of the boundary conditions on the seepage analysis within the slope area.

Rainfall of 18 mm/h over a 9 h period was applied on the ground surface as the net flux of water (q). No ponding condition was allowed in the seepage analyses. The side boundaries below the groundwater table were specified as constant total head boundaries and the side boundaries above the groundwater table were specified as no flow zone or nodal flux, Q, equal to zero. The applied total head corresponded to the hydrostatic condition. In this study, a hydrostatic pore water pressure distribution with a limiting negative pore water pressure of 100 kPa was set as an initial condition for the slope.

Figures 11(a) and (b) show the changes in the pore water pressure profiles during and after rainfall respectively, below the crest of the slope. During rainfall, pore water pressure in the slope moved gradually towards positive values. After rainfall stopped, the top part of the slopes experienced a drying process as indicated by the movement of pore water pressure towards negative values. However, the lower part of the slope still experienced a wetting process for up to 48 h, as indicated by the movement of pore water pressure towards positive values. The variations of pore water pressures with time from the seepage analysis were incorporated into slope stability analysis to obtain the variations of FS during and after rainfall. The effective cohesion and effective friction angle used in the previous analyses using the total cohesion method were also used in this extended shear strength method. The ϕ^b angle was assumed to be 50% of ϕ' in this analysis.

Figure 12 shows the changes in factor of safety during rainfall and after rain stopped. It can be seen that the presence of matric suction within the unsaturated zone contributed to the stability of the slope as indicated by a FS of 1.38 before rain started. However, matric suction in the upper soil zones (within 20 m from ground surface) decreased during rainfall due to water infiltrating the slope (Figure 11(a)). In addition, the groundwater table was found to rise during the rainfall. As a result, the

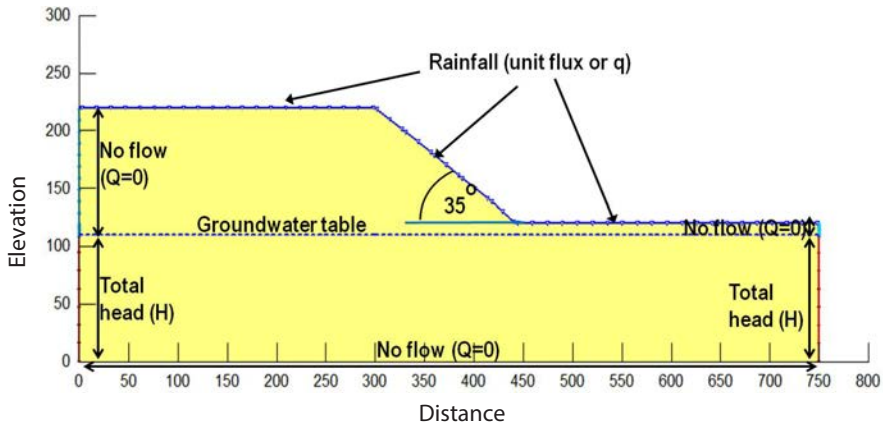


Figure 10 Numerical model for seepage analyses of slope with 35° inclination

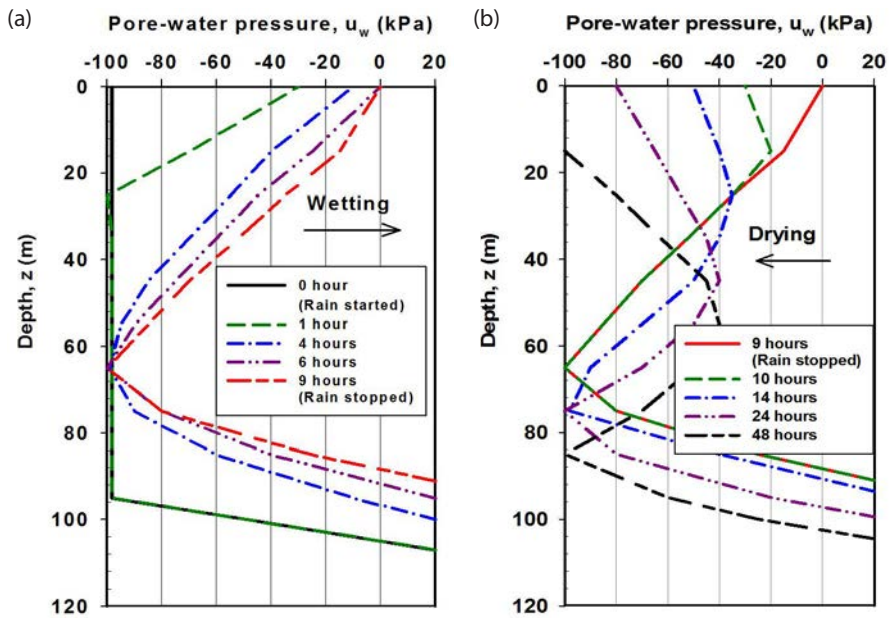


Figure 11 Pore water pressure profiles; (a) during 9 h of rainfall; and (b) after rain stopped

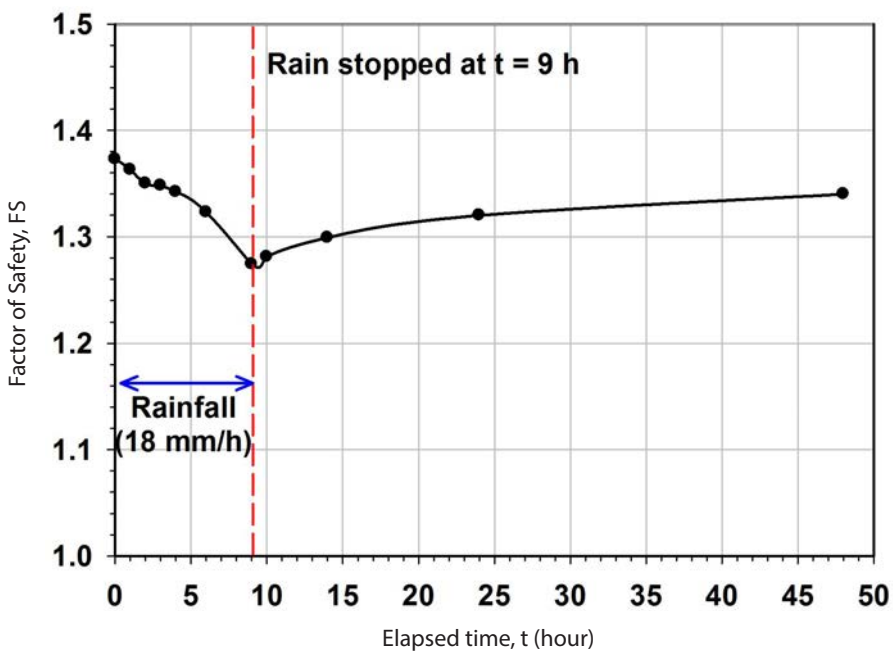


Figure 12 Variation of FS during rainfall and after rain stopped for 35° slope angle

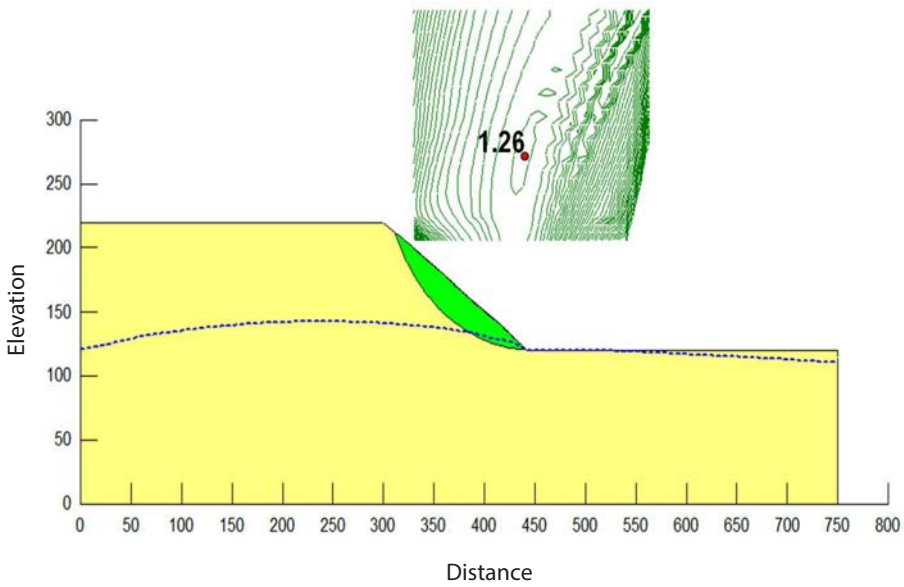


Figure 13 Critical slip surface obtained from slope stability analysis at the end of rainfall

shear strength of the unsaturated zone decreased and consequently the FS of the slope reduced during the rainfall. The minimum FS of 1.26 was observed at the end of rainfall (t = 9 h). The critical slip surface corresponding to the minimum FS was located near the ground surface as shown in Figure 13. After the rain stopped, the matric suction in the upper soil zones began to recover resulting in the shear strength of the soil increasing and consequently, the FS of the slope improved (Figure 12).

Conclusions

Unsaturated soil mechanics principles and theory are required to account for the behaviour and shear strength of soils that are commonly unsaturated in nature. Laboratory testing equipment, procedures and standards for unsaturated soil testing are now becoming available which makes the application of unsaturated soil mechanics to the design of pit slopes more readily achievable. By using the additional shear strength developed by matric suction in designs, more beneficial pit slope angle may be achieved with resulting savings in stripping costs. The application of unsaturated soil mechanics principles in the slope stability analyses during rainfall enables the variation in the FS during and after rainfall to be evaluated in order to determine the critical periods during which the FS of the slope is at its lowest. The unsaturated soil properties of individual pit slopes must nevertheless be properly characterised and design assumptions and simulations validated by field instrumentation that monitors the matric suction.

Article references are available on request.

Professor Harianto Rahardjo,
head of division,
Nanyang Technological University,
Singapore

Tracking the cave and rock mass damage in block caving mines

A new ACG physical modelling project may well be the first model of a block cave to be tested in a geotechnical centrifuge writes Daniel Cumming-Potvin, Australian Centre for Geomechanics

Introduction

Block caving is an underground mining method which yields a high rate of production. Due to the low cost of operation, block caving has become increasingly popular in recent years. Some of the key geotechnical risks in block caving, such as large uncontrolled caving events, poor fragmentation and undesirable cave propagation, stem from the fact that there is no access to the orebody and, as such, the state of the rock mass within the cave column is largely unknown.

Geotechnical monitoring is routinely conducted in modern block caving operations. Monitoring often includes depth measurement from open holes: time domain reflectometry (TDR) or microseismic monitoring methods. As microseismic monitoring methods can give a full three-dimensional picture of cave development in real time, and the other methods only pinpoint measurements at a given time, this article will focus on microseismic techniques.

Microseismic analysis methods

Duplancic divided the caving profile into five zones: the pseudo-continuous domain, seismogenic zone, zone of loosening, air gap and caved zone (Figure 1). This model for the caving front is widely accepted and has been adopted by the industry as the framework within which monitoring results are interpreted.

A detailed literature review by Cumming-Potvin (to be published in the Caving 2014 Symposium proceedings) shows that there have been a number of analysis techniques used to track the cave back and seismogenic zone, and to quantify rock mass damage. Each analysis technique has its own strengths and weaknesses, however, it is unclear which of these is most successful in describing cave development. It is possible that a combination of these techniques would yield better results than any individual technique. The literature review also reveals a systematic lack of quality validation through all studies. There is a need for an independent method of verification to determine which analysis method is most effective. The verification method will need to allow objective identification of the cave back and the damage ahead of it through space and time. This has prompted the ACG to begin a physical modelling project in conjunction with the University of Pretoria, South Africa.

Figure 2 Daniel Cumming-Potvin at the geotechnical centrifuge in Pretoria, South Africa

The physical modelling project involves creating a scale model of a block cave mine using a brittle cementitious material. The two-dimensional sample will be progressively undercut in the University of Pretoria's state-of-the-art geotechnical centrifuge facility (Figure 2), in order to create cave development in the sample. The sample will be instrumented with sensors to record acoustic emissions released as the sample fractures and the cave develops. A video of the test will also be taken so that direct visual observations can be used to quantify the rock mass damage in an objective manner. The acoustic emission monitoring results will then be analysed using a variety of analysis techniques so that the optimum technique, and application of this technique, can be determined. Another potential application of the physical modelling project is the potential for calibration of numerical models. The physical model has known input parameters and assumptions and the final result can be easily quantified, so the numerical model can be calibrated with a high level of confidence. The physical modelling project is, to the author's knowledge, the first model of a block cave to be tested in a geotechnical centrifuge.

Article references are available on request.

The author will present a paper on this topic at the Third International Symposium on Block and Sublevel Caving to be held in Chile, June 2014. See www.caving2014.com

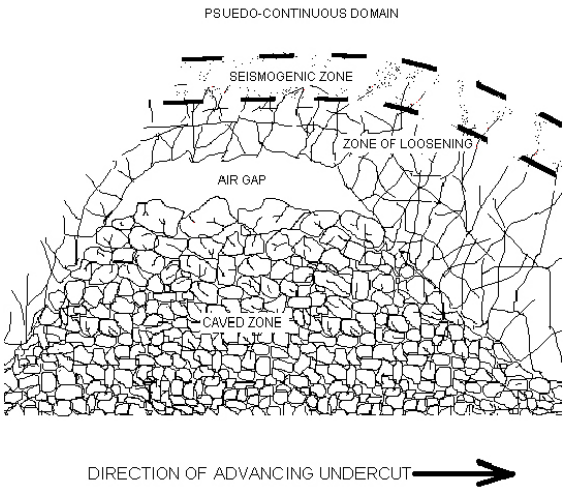


Figure 1 Zones of ground behaviour in block caving mines

Daniel Cumming-Potvin, PhD student,
Australian Centre for Geomechanics,
The University of Western Australia

Interpreting risk in geotechnical engineering

by Julian Venter, Rio Tinto Iron Ore

Introduction

Much engineering literature has been devoted to providing ways of calculating probability of failure but rarely is guidance provided on making decisions based on the results. This article presents guidance for making decisions based on the results of probabilistic analysis for economic outcomes. Background information is provided in the form of a definition for risk, followed by a discussion on risk assessment techniques and interpretation tools.

The article concludes with practical advice to improve risk assessments based on the discussion of risk assessment tools in the text.

Defining risk

Risk is defined as:

Risk = likelihood × consequence

This definition is the same as presented in most texts on risk such as Baecher and Christian. This definition implies that two variables are required before risk can be defined: likelihood and a consequence. The practical implication of the equation is that for risk to be understood, a consequence or scenario needs to be identified and a likelihood assigned to it. Most of the literature on risk focuses on estimating likelihood, as it is generally assumed that consequences require no further guidance.

This has resulted in many risk-based decisions being made by reducing likelihoods instead of managing consequences. While manipulating likelihoods may be unavoidable in some cases, it is often not the best approach. The methodology presented in this article focuses on understanding the shape of the risk distribution, not just the expected outcome, and addressing the consequences.

Assessing risk

Many risk assessment tools have been used in geotechnical engineering to produce both qualitative and quantitative risk assessments.

Qualitative risk assessment is where the consequences are documented for each scenario and a likelihood estimated, and is still the more common approach.

An example of a qualitative analysis would be to use one of the many risk matrices published in literature that categorises risk according to a predetermined table of consequences and outcomes.

Quantitative risk assessment is where consequences are documented and analysed, and the likelihood and severity of the outcome is quantified.

Quantitative analysis can take many forms but the most common are:

- event trees,
- fault trees,
- butterfly diagrams,
- probability of failure; and
- decision tree analysis.

Often these methods are combined. For instance, the output of a fault tree can be combined with probability of failure and an event tree to estimate the probability of not meeting production targets as a result of slope instability.

In practice, there are three major sources of error stemming from the incorrect use of quantitative risk analysis methods. This has resulted in many decision-makers becoming sceptical about the value of quantitative risk analysis methods.

The first source of error is that quantitative risk analysis methods are still dependant on subjective inputs for at least some of the decision/fault/event tree nodes (Figure 1). Subjective input has resulted in widespread abuse or misinterpretation caused by a frequentist interpretation of risk by contributors instead of a degree of belief interpretation, as defined by Vick, rather than malice. The distinction between frequentist approaches and degree of belief approaches is that frequentist interpretation considers risk to be a property of the object (slope) being assessed, while degrees of belief interpretation considers risk to be a statement of confidence regarding the knowledge about the object (slope).

The second source of error is that the input for fault/event/decision trees is often evaluated based on single estimates of input variables instead of stochastic input variables. This prevents a reality check to be carried out and results in decisions made without understanding the shape of the risk distribution. If fault/event/decision trees are first evaluated using uniform distributions as inputs for each node with probabilities ranging from zero to one, it will be noticed that the outcome is nonuniform with a much reduced standard deviation. This output distribution can be used as a reality check as any subjective choice of inputs that fall outside the range of likely outcomes is probably biased and should be revised. Evaluating a fault/event/decision tree using stochastic inputs first (Figure 2) provides the added advantage of comparing the effectiveness of different trees representing, for instance, different slope monitoring configurations or different risk management plans. This attribute alone may provide

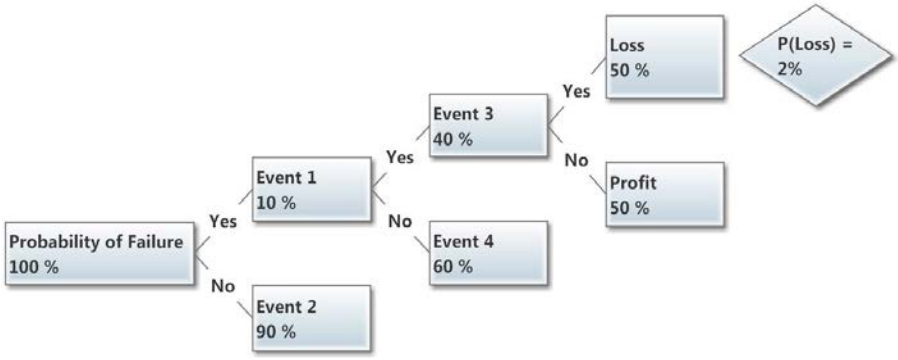


Figure 1 Event tree subjective inputs

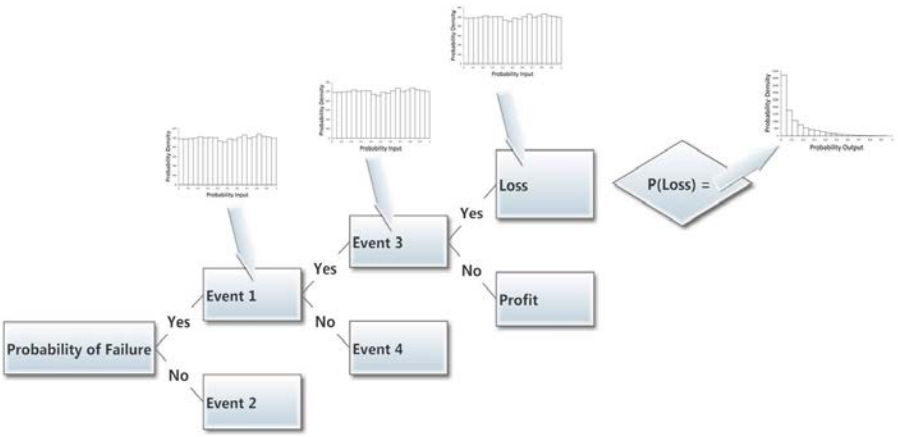


Figure 2 Event tree probabilistic inputs

sufficient risk management information without relying on subjective node inputs.

The third source of error is that quantitative risk assessment methods are often used to evaluate only the likelihood of a single scenario instead of multiple scenarios which often results in an underestimation of the risk. The underestimation occurs as the risk of a system is the sum of all the risks. In open pit terms, it is often incorrect to assume that the maximum risk is at maximum pit depth, as the economic consequence on the net present value (NPV) may be higher for a slope instability half way down than at the bottom – even if the probability of failure is higher at the pit bottom. To better understand the full risk profile it is advisable to assess various scenarios.

Evaluating likelihood for multiple risks in one system

Contreras and Steffen realised that combining all the risks for one system, such as a mine site, is unnecessary as the risk profile can be evaluated by only considering the risks on the risk frontier, as opposed to considering all the risks in a system, which is an impossible task.

The risk frontier is determined by plotting the likelihoods and financial outcomes for many risks in the system on a chart with the likelihood on the x-axis and outcome on the y-axis (Figure 3). The result is similar to the F-N charts available for benchmarking safety risk (Figure 4), presented by Wesseloo and Read, Baecher and Christian, and Steffen et al.

While the risk frontier does not contain all the information required to determine a risk profile, it contains all the necessary information for informed decision-making. The benefit of this approach is that not all risks need to be defined to evaluate a system, only the ones defining the risk frontier. This simplifies the analysis considerably.

Alternative ways to interpret risk

Before presenting guidance on interpreting risk using the risk frontier, it is first necessary to introduce a few new concepts that relate to the shape of the risk frontier.

In his book "Antifragile", Nicholas Taleb presented a system of classifying risk based on the shape of the probability density functions of the consequence.

Three categories are provided: robust, fragile and antifragile. Antifragile is a new word defined by Taleb that has the opposite meaning to the word fragile. Taleb considered that since fragile items break easily, the word robust cannot be the opposite of fragile, since it only implies that robust items do not break easily. Antifragile implies that items get better

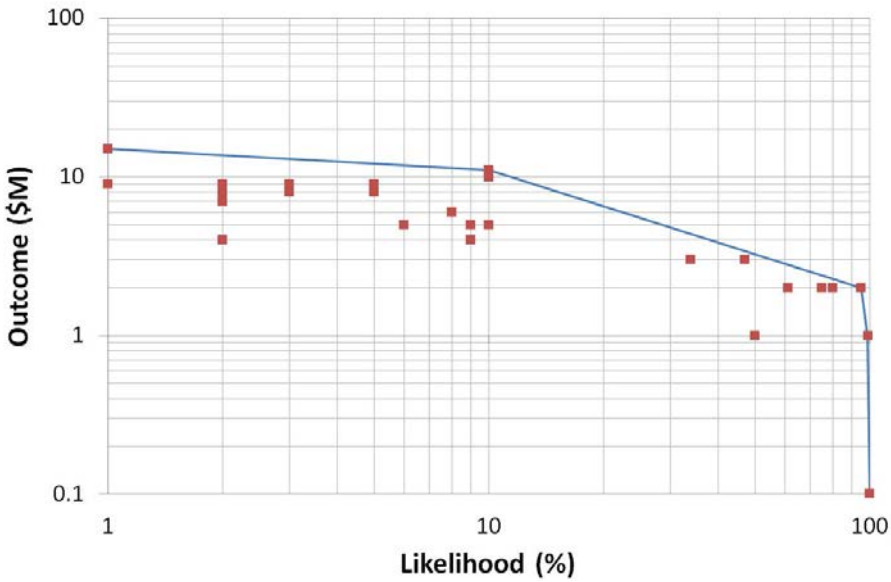
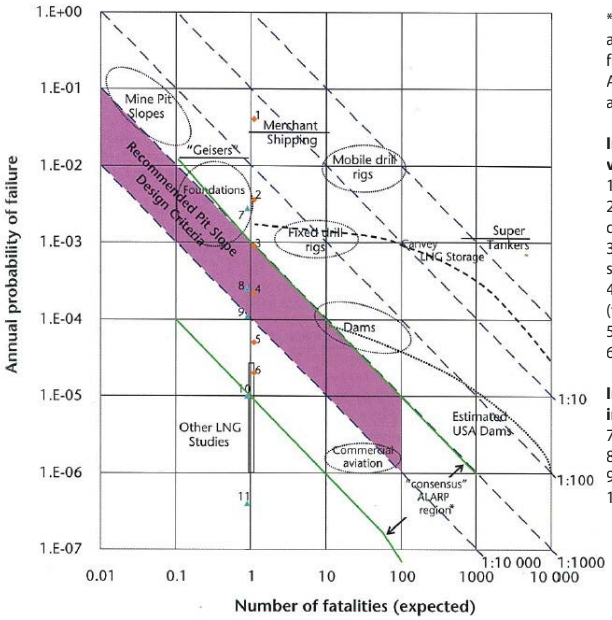


Figure 3 Risk profile and risk frontier



*Region defined by the upper-most and lower-most ALARP boundaries from Hong Kong Planning Department, ANCOLD, US Bureau of Reclamation and UK HS executive guidelines.

Individual fatality statistics USA – voluntary
1. Space shuttle program (per flight).
2. Cigarette smoking (one pack per day).
3. Average individual voluntary sporting risk.
4. USA police killed in line of duty (total).
5. Frequent flying profession.
6. Alcohol (light drinking).

Individual fatality statistics USA – involuntary
7. Cancer.
8. Motor vehicle.
9. Home accidents.
10. Huirricanes/lightning/tornadoes.

Figure 4 F-N chart for safety risk

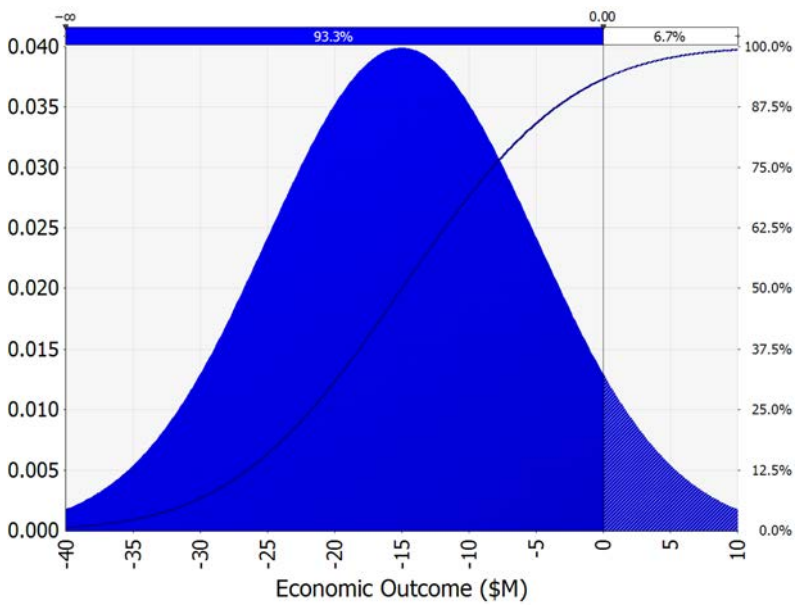


Figure 5 Fragile distribution shape

when stressed. Antifragile is not a property that can be attributed to inanimate objects; many biological systems exhibit antifragile properties when stressed. Antifragility in biological systems is caused by the system's response to the stressor as its defence mechanisms react to improve its defences against the threat caused by the stressor. This response is only observed if the biological systems survive the damage caused by the stressor, i.e. do not experience *force majeure*.

Taleb also presented the concepts of fragile, robust and antifragile in terms of statistical distributions related to economic outcomes, potentially NPV. The term fragile is represented by statistical distributions with unlimited downside potential (Figure 5). Robust describes distributions with limited upside and downside potential (Figure 6). Antifragile describes distributions with limited downside but unlimited upside potential (Figure 7).

These distribution shapes are ranked from most desirable to least desirable as follows:

- antifragile,
- robust; and
- fragile.

The reason why antifragile is superior to robust is that it is open to upside potential while upside potential is eliminated for purely robust solutions. The result of being open to upside potential is that the mean or expected value of the outcome is more favourable for distributions with unlimited upside potential.

In the open pit context, mining operations, similar to termite colonies, exhibit the behaviour of biological systems. Through various management strategies and behaviours, mining operations respond to challenges such as varying geology and slope instability. Through the choice of response to these situations, mine management can steer mining operations to become fragile, robust or antifragile.

An example of a fragile geotechnical response is to not carry out a good geotechnical investigation before starting an open pit. This results in various risks remaining hidden until they materialise. Due to the significant time involved in drilling, modelling and designing slopes, and the inconvenience and cost (direct and opportunity cost) of having to mine unplanned slope cutbacks to flatten slopes, the downside financial consequence of this behaviour is unlimited.

An example of a robust geotechnical response is to carry out a good geotechnical investigation prior to mining, followed by incorporating multiple access ramps into the open pit leading to various dig faces accompanied by a geotechnical monitoring plan without active geotechnical data collection and reconciliation. While this approach appears to be industry best practice, it is still only considered robust as the downside consequences are limited. However, nothing is done to proactively identify risks nor to identify possible opportunities based on more accurate data from newly exposed geology, as opposed to boreholes from the pre-mining analysis.

An antifragile geotechnical response is to carry out all items, as for the robust response, to improve survivability, but to follow up with active and ongoing geological and geotechnical data collection, modelling and analysis. The purpose of this work is to identify hidden geotechnical risks in time to provide solutions early on and to identify geological and geotechnical opportunities in time to realise the value. The upside potential of such a process is unlimited while the downside is limited to the cost of carrying out the work.

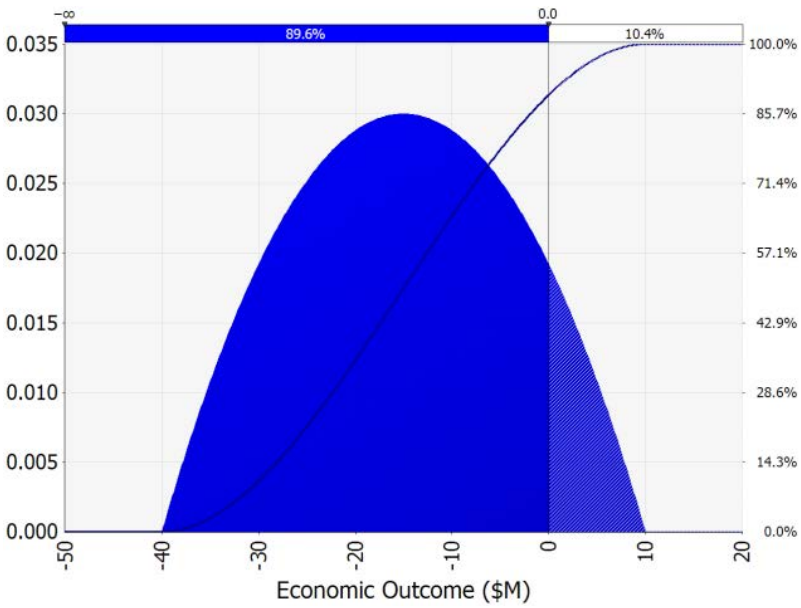


Figure 6 Robust distribution shape

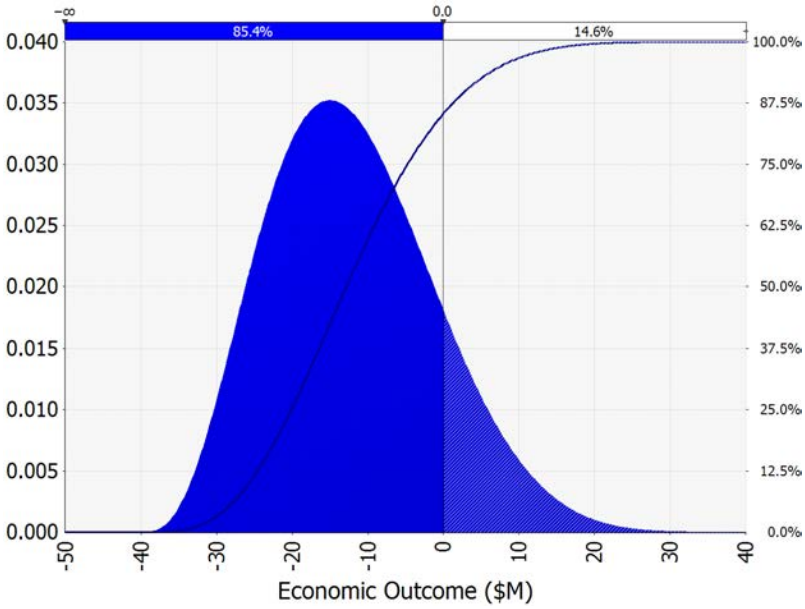


Figure 7 Antifragile distribution shape

Open pit risk mitigation strategies

Taleb proposed that the most effective way to improve the expected value of the risk profile is to address the risks with more severe consequences instead of trying to change the entire risk profile. The result of such an approach is that the expected loss due to risk will decrease at a lower unit cost than if all risks are addressed equally. In effect, the expected value of the risk profile is improved by clipping the downside and extending the upside instead of trying to move the entire distribution.

A practical approach to incorporating the methods proposed by Taleb is to define the hazards and scenarios for a risk assessment and define the associated likelihoods and consequences with sufficient precision to determine which hazards and scenarios define the risk

Figure 8 *While no amount of risk assessment can guarantee achieving positive results, a lot of risk can be managed with sufficient planning and the pro-active anticipation of risk. Copyright © 2014 Rio Tinto*

frontier. The risk frontier can be defined using either quantitative or qualitative methods depending on the level of rigour required and resources available. As this initial definition of the risk frontier is before mitigation and control, it is the inherent risk frontier.

After the initial definition of the inherent risk frontier, the extreme downside and extreme upside of the inherent risk frontier is often left undefined. The next step is to fill in the gaps in the inherent risk frontier by identifying the hazards and scenarios, representing the extreme downside and also finding opportunities that define the upside. Once the extreme values are added, the inherent risk frontier is complete and the risk mitigation activities and controls can be identified and the associated changes in the risk frontier assessed. The risk frontier after implementing controls becomes the Residual risk frontier.

To follow the advice of Taleb, the risks with low likelihood but high consequence must be controlled first. The emphasis must be on decreasing the consequences for extreme value risks and not changing the likelihood.

Once the downside is limited, opportunities must be actively sought to improve the upside potential of the system. Two types of questions need to be asked to identify upside potential. The first question relates to identifying actions that may unlock more value such as gaining further orebody knowledge or further design improvement. The second question relates to identifying opportunities that are unlocked when downside risks materialise. A lot of value is gained if extreme downside risks are turned into opportunities. Examples of such opportunities are purchasing insurance or hedging activities on one of the financial markets.

Many risk management plans focus on preventing risks but significant value can be realised by also planning for the period after a risk has materialised. A good example is the motor vehicle seat belt. A seat belt does not prevent the initiating event (accident), but it makes a significant impact on the consequences. It is therefore prudent to also prepare contingency plans for severe risks should they materialise.

Concluding remarks

This article set out with a definition of risk followed by a discussion on carrying out risk assessments and defining a risk frontier. The importance of understanding the extreme values on the risk frontier was stressed and advice to improve risk assessments provided.

The methods presented here can be used for both quantitative or qualitative risk assessments depending on rigour required and resources available.

It must be noted that while no amount of risk assessment can guarantee achieving positive results, a lot of risk can be managed with sufficient planning and pro-active anticipation of risk.

Article references are available on request.

Julian Venter,
superintendent geotechnical engineer,
Rio Tinto Iron Ore

Communities, advocates and experts discuss Philippine mining issues and waste landform and closure technologies

by Dianne Marah E Sayaman, research and advocacy officer, Center for Environmental Concerns-Philippines, Philippines

Delegates from the Filipino people's organisations (POs), non-government organisations (NGOs), government agencies and The University of the Philippines staff and students attended a course delivered by academics and experts from the Australian Centre for Geomechanics (ACG). The three-day seminar introduced strategies to help both operationally manage and close mine waste landforms.

Around 70 individuals from more than 20 organisations attended the The International Mining for Development Centre (IM4DC)/ACG Mine Waste Landform Management and Closure Workshop, held in February in Quezon City, Metro Manila, Philippines. The seminar was organised by Manila-based environmental NGO, Center for Environmental Concerns-Philippines (CEC) and IM4DC. IM4DC is a partnership with the Australian Government through the Australian Aid initiative, The University of Western Australia and The University of Queensland, which was established to assist in lifting the quality of life in developing nations through a more sustainable use of mineral and energy resources.

"The training presents a great opportunity for ordinary people with environmental advocacies and struggles for their rights. Contrary to what mining firms claim, advocates like us do what we can to understand the problem and find solutions to it. This is the scientific approach to resolving issues in mining," said CEC executive director Frances Quimpo.

Frances Quimpo, Center for Environmental Concerns-Philippines

Since its inception in 1989, CEC has assisted the Filipino people to address environmental challenges in research, campaigns and advocacy, education and community services to respond to these issues and raise critical awareness for a healthy ecology.

To familiarise the participants with the Philippine context, Filipino professors and experts explained the laws covering mining practices, the implications of climate change on the vulnerability of mining areas, and the various issues currently confronting the mining industry. Rolando Peña of the Mines and Geosciences Bureau, Ricardo Saturay Jr of the Philippine Institute of Volcanology and Seismology, and AGHAM - Advocates of Science and Technology for the People chairperson Dr Giovanni Tapang led the talks on the Philippine mining situation.

Following the Philippine presentations, Professor Ken Mercer, Bill Biggs and Dr Augustine Doronila each discussed and shared their expertise with the training participants. They discussed the most up-to-date methods to design and manage mine waste landforms as well as risk management systems that need to be in place to prevent mining disasters. Very importantly, the presentations covered the key aspects of mine closure and rehabilitation of the waste landforms in order to minimise long-term environmental impacts.

As a practical example of waste mismanagement and irresponsible planning of mining systems and structures, CEC volunteer, Peter de San Miguel, shared mining practices gone wrong in the Philippines, such as the failure of the Philex mine tailings dam in Padcal, Benguet in late 2012, and the 1996 Marcopper mine tailings spill in Marinduque.

"Both cases show how some mining companies in the Philippines have abused the laws and ignored environmental and social losses. Sadly, many mining-affected communities are not aware of the causes of these disasters and, more importantly, of the appropriate measures to mitigate effects and effectively rehabilitate damaged land and water bodies," said CEC head researcher Rog Amon.

True to its cause of pursuing a pro-people and patriotic environmentalism, CEC has led and participated in several campaigns against environmentally irresponsible mining that has destroyed the lives and livelihood of local communities. Together with its network organisations, the CEC holds environmental investigative missions to assess impacts of mining disasters, explain the implications of such events to the communities, and aid ordinary people in asking for corporate and government accountability over these incidents.

Seventy delegates attended this highly interactive seminar

The three-day forum closed with the signing of a declaration that demanded the urgent rehabilitation of compromised and unstable mine tailings facilities to effectively prevent further mine disasters and for industry players and Philippine authorities to ensure that only the safest and most environmentally appropriate mining and waste management technologies are used in the country.

The ACG, in conjunction with the CEC, intends to repeat the Mine Waste Landform Management and Mine Closure Training Seminar in either the North or South provinces of the Philippines in February/March 2015, targeting primarily NGO and PO participants. In conjunction with this seminar the ACG/CEC are considering launching the First International Conference on Mining in Remote Communities in South-East Asia. The forum is intended to provide an opportunity for representatives from governments, NGOs, industry, professionals and mining affected communities throughout the South-East Asian region to share different aspects of their environmental, social and community challenges and provide an opportunity to hear about solutions and share learnings both positive and negative.

The CEC is also planning to host further local conferences on mining affected communities within the Philippines.

Can we generalise seismic hazard assessment to include the effect of mining?

"Due to the complexity of mining, finding the many unknowns involved with seismic hazard may be problematic. This needs to be addressed by the industry to ensure accurate seismic hazard forecasting" writes Gerhard Morkel, Australian Centre for Geomechanics

Introduction

In a seismically active mine, seismic hazard analysis plays an important role in determining an applicable ground support design. This article briefly outlines the different seismic hazard assessment methods commonly used, as well as discussing the shortcomings of such methods for mine design. In order to create a generalised seismic hazard assessment method, we need to consider additional parameters. I will postulate a model based on such parameters and show with a real world example its usefulness in describing seismic hazard. I will theorise the problem and develop some concepts that will need to be expanded with further work.

Current seismic hazard methods

Determining the seismic hazard is vital in defining an appropriate seismic management plan. This includes ground support design. There are several analysis techniques to determine an applicable seismic hazard. This section briefly discusses three of the most commonly used methods:

- Determining an upper peak particle velocity (ppv) value from observation, modelling and measurements.
- Calculating the highest possible magnitude event based on the frequency–magnitude (f–m) distribution of the site.
- Calculating the next highest magnitude event, based on the extreme value distribution from the f–m distribution of the site.

The first seismic hazard method determines a constant upper value ppv. Milev and Spottiswood, and Wagner suggest using 3 m/s for rockburst-prone South African gold mines. According to Jager and Ryder, this value is determined from "underground observations, seismic measurements, rockburst back analyses and numerical simulations". Although based on a South African gold mining context, the methodology can be used for other mining areas.

The techniques for hazard methods two and three rely on the statistical quantification of the f–m distribution. Figure 1 is a typical example of a f–m distribution. A truncated function is used to describe the f–m relation of the data. This is indicated by the red line in Figure 1 and is the best fit for the observed data, based on the current mining conditions. Three characteristic magnitude values are often used and shown in Figure 1. First, an x_{max} value, which is the highest recorded event in the database. Second, a m_{ext} value, which is the extrapolated magnitude based on the f–m fit: statistically, it signifies the most probable magnitude for the next largest event. Last, a m_{max}, value, which is the theoretical upper bound of the magnitude of the next largest event, visually represented as the asymptote of the f–m fit.

The second technique for finding seismic hazard aims to find the mmax value in Figure 1. Due to the complexity of finding an appropriate m_{max} value, numerous techniques exist, as Kijko and Singh state:

"… it is astonishing how little has been done in developing appropriate techniques for estimating this parameter. Presently, there is no universally accepted technique for estimating the value of m_{max}… ".

This statement is made with earthquake seismology in mind and, therefore, it is truer for the complex mining environments discussed in this article.

A good approach in finding a relevant m_{max} value is to consider several of these estimation techniques and choose the most represented value amongst them. From the m_{max} value and some distance attenuation relation, a ppv value can be calculated at any point in the mine.

Wesseloo and Potvin optimise the second method by determining an extreme value distribution for the largest expected event. From such a distribution, a design can be based on the probability of having a certain magnitude for the dataset. The probability of having events less than the mmax is 100%, and for events equal to or less than the m_{ext} it is 63% ($m_{max} >> m_{ext}$).

When considering this method for a ground support design, both short-term and long-term datasets should be considered. The short-term (six month) cloud highlights the current highest seismic hazard, typically on the present active workings. The long-term (three year) cloud highlights historical high seismic hazard, typically on geological structures or pillars. Figure 2 exemplifies the latter case. Long-term and short-term seismic hazards coincide in areas with a relative high hazard, and increased exposure to workers.

A further point to note from Figure 2 is the difference in hazard magnitude between the two hazards. The long-term hazard has a lower magnitude in some areas compared to the short-term hazard. One expects the magnitude of the short-term hazard to be equal to or less than the long-term hazard, because the long-term hazard encompasses the short-term hazard, and therefore already includes the effect described by it. This discrepancy is the result of using time to

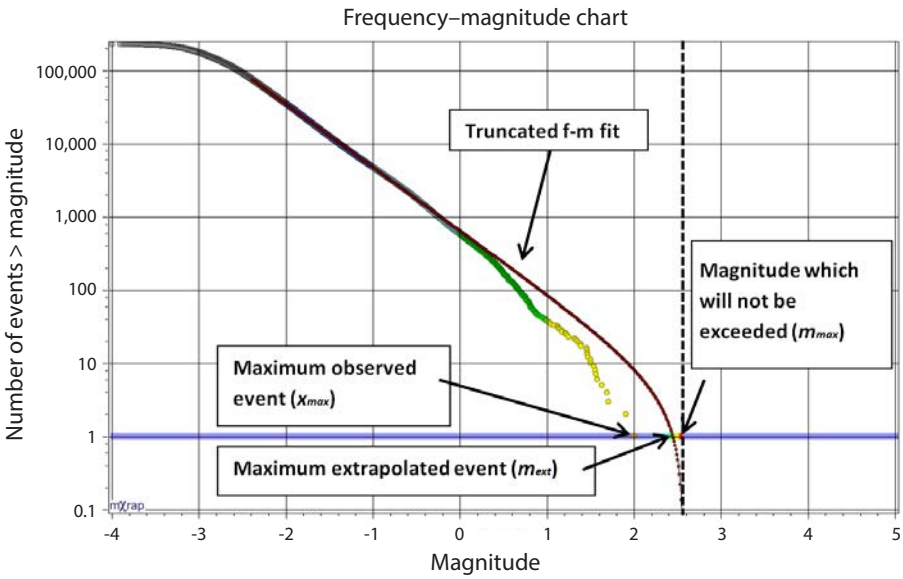


Figure 1 *The coloured dots represent the f-m plot of the dataset. The dark red line indicates the best f-m tapered fit, and the dotted line, its asymptote. Three relevant magnitude values are indicated by the arrows*

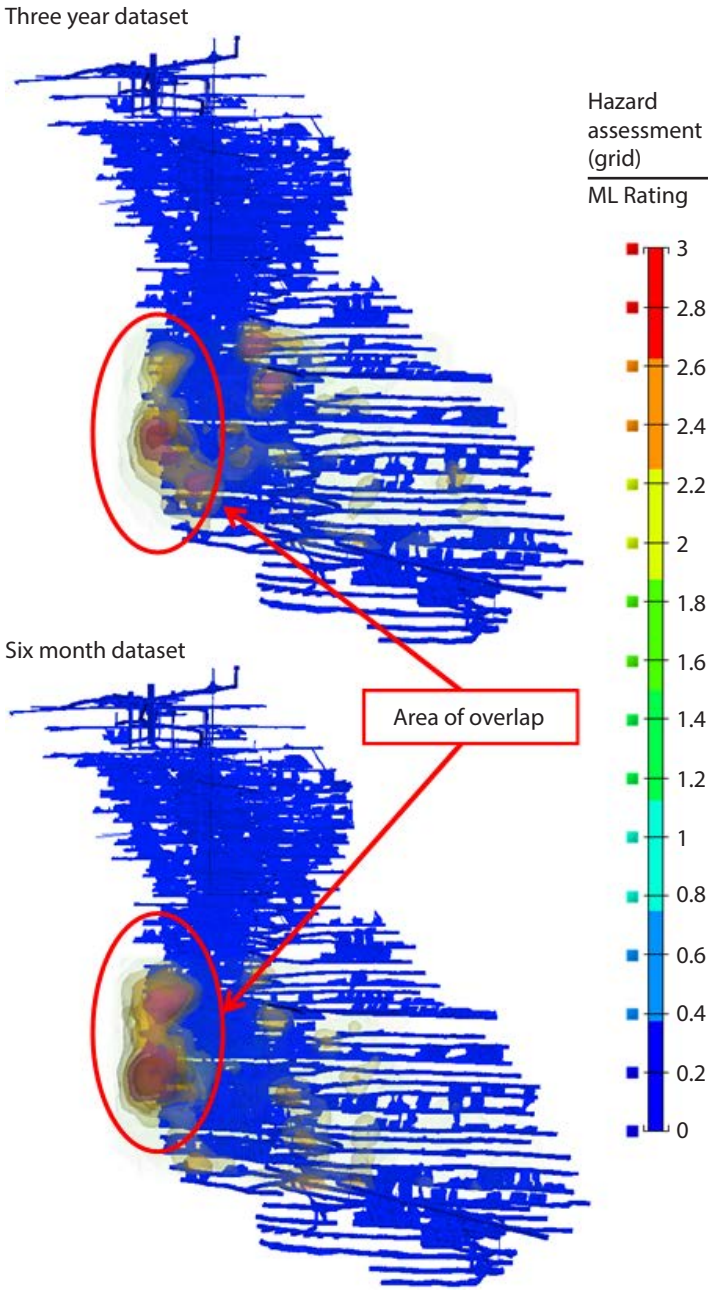


Figure 2 *Seismic hazard clouds. Both inserts are based on an 85% probability design. The top insert is for a three year historical period, and the bottom plot is for a six month data period*

normalise seismic hazard. Seismic hazard is not driven by time.

The applicability of current methods

These methods are useful, because they provide the current seismic hazard at a mine. The hazard is, at best, current, since it is based on historical data. To forecast the future seismic hazard, three main assumptions must be made:

- The mining rate is constant.
- The geological setting is consistently the same.
- The effect of mining geometry on seismicity is negligible.

When these three conditions hold, the historical seismic data can be used to forecast the seismic hazard at some future point. For most operations these assumptions cannot be made, consequently, direct forecasting is not possible.

A generalised seismic hazard method

For a generalised method, the seismic hazard should be explicitly expressed in terms of a measurable parameter. This parameter must describe the effect of mining rate, geological setting and mine geometry on the seismic response.

Mine seismicity is driven by mining; Gibowicz et al.; Mendecki; Holub; and du Toit show that inter-event mining volume outperforms recurrence time as a hazard indicator. Mining volume is therefore, the parameter which most likely describes seismic hazard. Kijko postulates that for a homogenous rock mass, with no geological structures, the relation between seismicity and mining volume can be expressed as:

$\Delta M_0 = c \cdot \theta \cdot \Delta V$ (Equation 1)

Where ΔM_0 is the change in seismic moment, ΔV is the change in volume of a mined layer, c a constant close to 1, and θ is a parameter dependent on depth and mining technique, mechanical properties of rock and rocks forming the opening, and the type of support of the opening.

Equation 1 can be expanded by assuming each mining variable of θ can be expressed by a function:

$\Sigma M_0 = c \cdot \{f(\theta_1,V)+f(\theta_2,V)+f(\theta_3,V)+ \cdots\} \cdot \Sigma V$ (Equation 2)

These functions should be numerically or empirically determined from site to site.

Equation 2 is further expanded by introducing the effect of mining excavations on geological structures as they approach them. This interaction is geometry driven and therefore, an additional term is introduced in Equation 2, independent of volume. It is easily represented by modelling parameters similar to excess shear stress (ESS) or dissipated plastic energy (DPE). These parameters do not take into account the effect of stored tectonic energy, and therefore a ctec value needs to be added to the ESS/DPE term. Equation 2 expands:

$\Sigma M0 = c \cdot \{f(\theta 1,V)+f(\theta 2,V)+f(\theta 3,V)+\cdots\} \cdot \Sigma V + (ESS/DPE+ctec)$ (Equation 3)

This equation is the expanded general form of Equation 1 and aims to describe mining-induced seismicity.

A practical example

Equation 3 must be solved for each site, or mining area. Scheepers et al. showed that for an ultra-deep gold mine in South Africa, the seismic potency can be described by modelled volumetric closure. Their study was on an ultra-deep narrow-reef tabular mine implementing a grid-based extraction layout. In their study, the mine has a 1 m stoping width, which effectively reduces the 3D geometry to a 2D geometry. Equation 1 can be reduced to:

$\Delta P_0 = c \cdot \theta \cdot \Delta A$ (Equation 4)

Where P_0 is seismic potency and A is the mined-out area. The mining parameter θ is then expressed in length and c is a unit-less constant of which the value is unknown. Equation 3 reduces to:

$\Sigma P_0 = c \cdot \{f(\theta_1,A)+f(\theta_2,A)+f(\theta_3,A)+ \cdots\} \cdot \Sigma A + (ESS/DPE+c_{tec})$ (Equation 5)

For a grid layout, as is employed in the Scheepers et al. case, each stope is mined in almost exactly the same conditions and therefore, stopes can be analysed as a group, rather than individually.

Figure 3(a) is an example of such a fit for six stopes. The best fit is a power law of the form $f(A)=aA^b$, where a and b are fitting parameters and A is the mined-out area.

Figure 3(b) is the modelled volumetric closure results for the same stopes, which

also follows a power law fit. The only difference from the potency fit is that it becomes linear after a reaching a certain mined-out area. This is due to the effect of backfill on closure in the model.

Therefore, Equation 5 can be rewritten as:

$$\Sigma P_0 = c_1 \cdot (f(closure,A) - c_2 \cdot f(backfill,A) + c \cdot \{f(\theta_2,A) + \cdots\}) \cdot \Sigma A + (P_{(ESS/DPE)} + c_{tec}) \quad \text{(Equation 6)}$$

When subtracting the backfill and closure functions from seismic potency, the resultant function has the form $f(A)=c_2 \cdot A^2+c_3 \cdot A$. The next step would be to find the mining variables which are described by this function. From recent papers by Hofmann; and Arndt et al., correlation between elastic energy release (EER) and energy release rate (ERR) could be seen. These two parameters might be able to describe the $f(A)=c_2 \cdot A^2+c_3 \cdot A$ function, and will be investigated in the future.

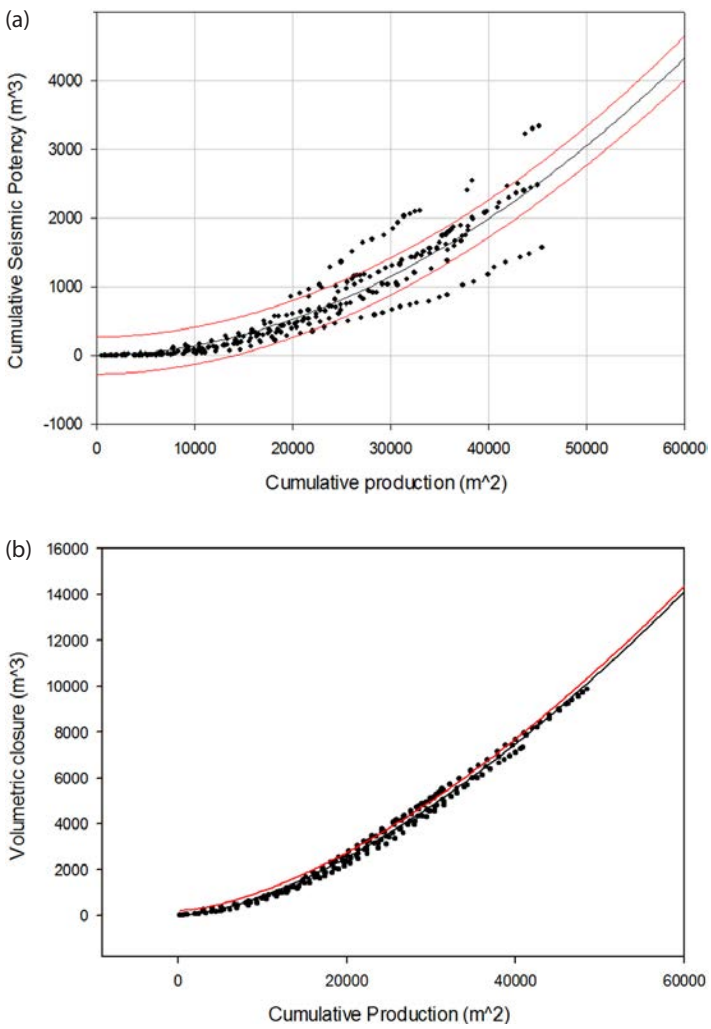


Figure 3 *(a) The best fit for seismic potency; (b) the best fit for the modelled volumetric closure. Both fits are sufficiently described by power laws*

Photo from Dave Ortlepp's library. Courtesy of IMS

Conclusion

The discussed seismic hazard assessment methods are useful in describing current hazard. When used for forecasting hazard, their usefulness is limited to a specific setting. To find a generalised method, the introduction of a production-dependent equation is necessary, where the production function is sufficiently described by geological setting and geometry of mining. With our current understanding of the applicability of modelling in certain scenarios, as well as a better understanding of the influence of rock properties on seismic response, this should be possible, as was shown for the Scheepers et al. case. Conceptually it appears that such a generalised seismic hazard assessment is possible. Due to the complexity of mining, finding the many unknowns involved with seismic hazard may be problematic. This needs to be addressed by the industry to ensure accurate seismic hazard forecasting.

Article references are available on request.

About the author

Gerhard joined the ACG in March 2014 as a research engineer for the ACG's Mine Seismicity and Rockburst Risk Management Project. See www.acg.uwa.edu.au/research/underground_mining/mine_seismicity_and_rockburst_risk_management.

Gerhard graduated from the North-West University with an MSc in Physics in 2008. After finishing his studies, Gerhard joined IMS, working in the operational aspects of mine seismology. In 2009 he joined Anglogold Ashanti, where he was involved with the seismological aspects of their Western Deep level mines. During this time, Gerhard completed his Chamber of Mines (South Africa) certificates in Stata Control and Rock Mechanics (metalliferous). In 2012 Gerhard relocated to Australia where he worked as a seismologist/geotechnical engineer for Kanowna Belle Barrick. Gerhard has five years of experience in the seismological and geotechnical aspects of highly seismically active underground mines.

Gerhard Morkel, research engineer,
Australian Centre for Geomechanics

11th International Symposium on Mining with Backfill

by Maddie Adams, Australian Centre for Geomechanics

The 11th International Symposium on Mining with Backfill, hosted by the Australian Centre for Geomechanics, took place on the 20–22 May 2014 in Perth, Western Australia.

Mine Fill 2014 was highly successful and attended by more than 130 attendees from countries including Australia, Bulgaria, Canada, China, Finland, Germany, Netherlands, Poland, PR China, Russia, South Africa, Switzerland, UK, and USA.

Mine Fill 2014 focused on the theoretical and practical aspects of the application of mine fill, with the three day technical programme consisting of sessions on new technology, geomechanics of mine fill – numerical modelling and field measurements, barricades, lab testing, fill system design, binder admixture, legal/environmental/safety/risk, case studies and a poster session.

The three-day programme comprised of 37 paper presentations, including three keynote presentations from diverse environments around the world. Additionally, a poster session took place on the first day of the symposium.

The ACG is appreciative of the support received from the Mine Fill 2014 sponsors, namely the principal sponsor MRC Global, major sponsors Beasley's Hydraulic Services, Imatech (ArmorPIPE) and Outotec, and industry sponsor Paterson & Cooke.

The Mine Fill 2014 symposium chair was Tony Grice, global leader backfill, AMC Consultants (Canada and Australia). The symposium opened with a presentation by Dr Ed Thomas, mine fill specialist Consultant, on 'The Mine Fill Symposium series – conception through to middle age', which focused on the progress of the Mine Fill Symposium series since its conception in Mount Isa, 1973.

Jacques Nantel, president, Nantar Engineering Ltd., Canada, presented the Mine Fill 2014 opening keynote address, 'Innovation and beyond', which explored what the mining industry needs to do to stay competitive and remain viable.

Dr Martyn Bloss, director Mining Studies and Technology, Olympic Dam, BHP Billiton, Australia, gave a keynote address on day two of the symposium, 'An operational perspective of mine backfill', which presented over 24 years of experience associated with underground operations that use mine backfill to discuss key operational issues, challenges and opportunities associated with mine backfilling.

Dr David Stone, president, Minefill Services, Inc., USA, presented his keynote on, 'The evolution of paste for backfill', examining the advances in the science and improved equipment associated with paste thickening of tailings since their first applications in the early 1990s.

Prior to the symposium, the Australian Centre for Geomechanics and Paterson & Cooke collaborated to host the Best Practices in Mine Backfill Technologies Workshop, held on Monday 19 May and attended by more than 50 delegates. The workshop facilitator was Stephen Wilson, director, Paterson & Cooke, UK, and focussed on best practices on mine design, plant design and operation, distribution systems, geomechanics, geochemical stability, instrumentation and safety.

A field trip to Frog's Leg Gold Mine, 20 km west of the city of Kalgoorlie, Western Australia, was held alongside the symposium, on Friday 23 May. Frog's Leg Gold Mine is wholly owned and operated by La Mancha Resources Australia Pty Ltd. Delegates learnt about the history, the present and the future plans for La Mancha Resources operations and went on a site tour and saw the mine workings.

Proceedings of the 11th International Symposium on Mining with Backfill, edited by Winthrop Professor Yves Potvin and Tony Grice, provide an excellent reference point for those people who are looking at the state-of-the-art and where the likely improvements are going to develop. These proceedings feature 44 technical papers, and are accompanied by a CD of colour figures from the publication.

Mine Fill 2014 proceedings and individual papers are available to purchase; visit www.acg.uwa.edu.au/shop.

Slope instability monitoring at MMG Century Mine, Queensland

by Michele Salvoni, Australian Centre for Geomechanics

Introduction

For an open pit mine, the evaluation and prediction of the mechanisms of instability, potential volume and timing of any wall failure are the main challenges at every stage of planning and operation.

Generally, the assessment of any possible instability requires a complete understanding of the geology, structural geology and rock mass proprieties of the area. Following this, the performance of the slope during and after the excavation is normally evaluated by monitoring surface deformations (survey, radar and/or laser scanner).

However, all this information may be insufficient to get a full comprehension of the rock mass behaviour and fracturing processes behind the surface.

Several studies have been carried out to get a better understanding of slope behaviour by the analysis of microseismic data. While this technique has been commonly used in underground mining operations, little has been researched in open pit slopes. The analysis of the seismic energy released in the form of acoustic waves during the fracturing processes or shearing shows the location of seismic events in the volume of rock behind the slope, providing a useful source of information to assist in the failure mechanism/s.

This ACG project will model rock slope instability through the combined use of geotechnical information, laboratory tests, surface and microseismic data. For this study, data recorded at MMG Century Mine, Queensland, will be used following the successful installation of a microseismic system in November 2013.

Background

MMG Limited's (MMG) Century Mine is located near Lawn Hill in North Queensland; approximately 250 km northwest of Mount Isa, close to the Northern Territory border. It is the largest open pit mine in Australia and one of the largest in the world, with production forecast of 465,000-480,000 t of zinc in 2014. The current ore deposit is estimated to continue production until 2015.

Since 2009, the southwest corner wall (sw wall) of the pit has been affected by the potential for instability. Further investigations are needed to be carried out to gain a better understanding of the possible instability mechanisms. This is of utmost importance for safety of personnel and to maximise ore extraction as Century progresses to the end of the mine plan.

Geological setting

Century is a shale hosted zinc, lead and silver deposit formed within the Proterozoic Lawn Hill Formation, representing the youngest unit of the Upper McNamara Group. It is mainly overlain by Cambrian limestone and younger Proterozoic sedimentary rocks. Century lies adjacent to the Termite Range Fault, a major northwest trending fault zone, occurring immediately northeast of the deposit.

The deposit itself is subdivided into two major blocks, separated by the Pandora Fault, a north dipping normal fault. Typical stratigraphy of the deposit consists of Cambrian limestone, Widdallion Sandstone, hangingwall siltstone-shale, mineralised sequence, footwall shale-siltstone, footwall black carbonaceous shale, Bulmung Sandstone and black laminated shale. Fault zones are characterised by the presence of intrusive carbonate breccia.

The area around the sw wall is characterised by the presence of three main units: black carbonaceous shales

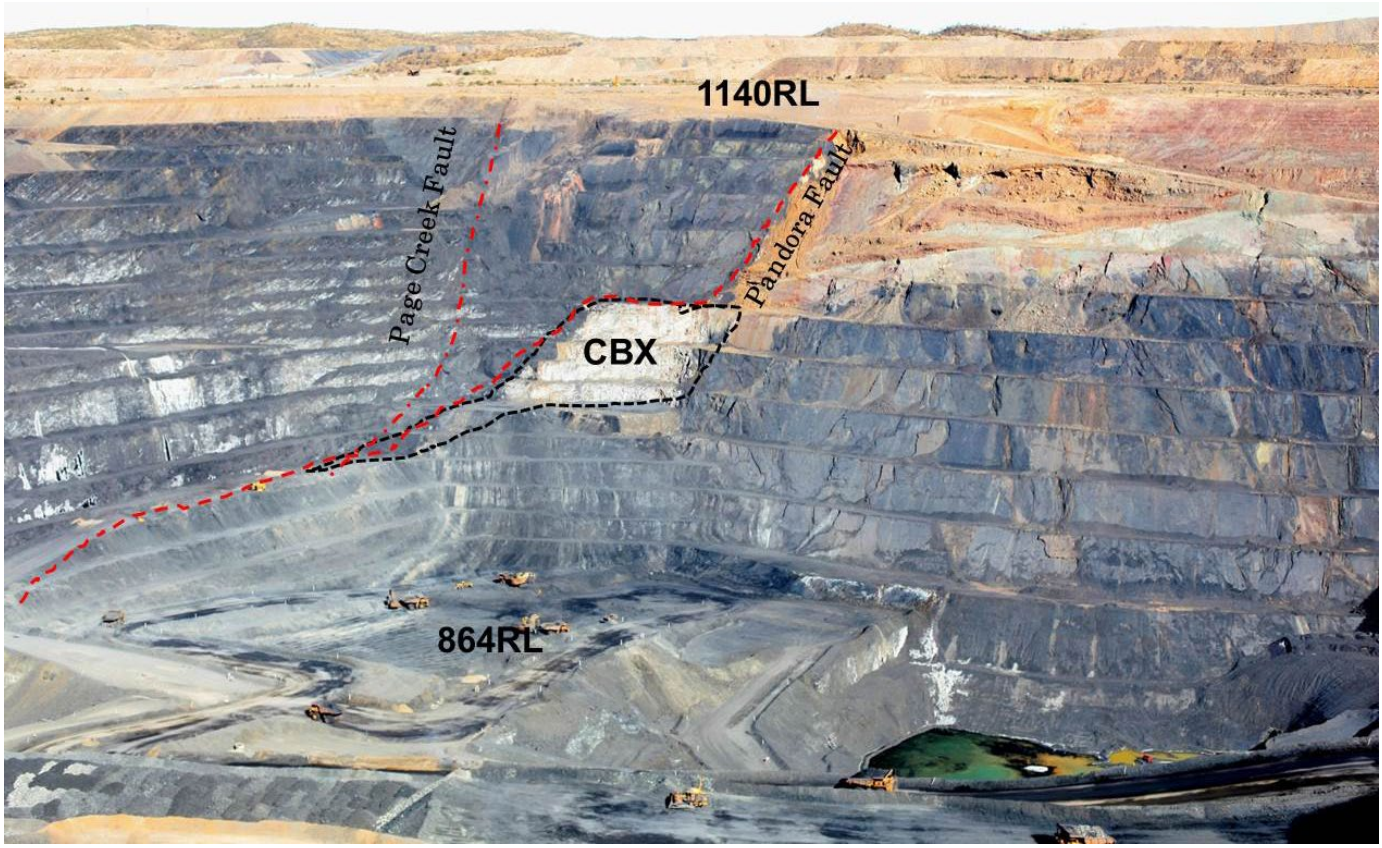


Figure 1 General overview of the sw wall

(BCS), black laminated shales (BLS) and carbonate breccia (CBX).

In the sw wall, both shales are represented by low strength rocks (~30 MPa) intersected by continuous fracturing and occasional bedding shears. A carbonate breccia known as the CBX outcrops as an intrusive block along the Pandora Fault. Another important feature is the Page Creek Fault, a north dipping subvertical fault (Figure 1).

Significant variation of the bedding is observed, with rotation in the proximity of the Page Creek Fault. The general trend is about 30-90°, dipping out of the slope.

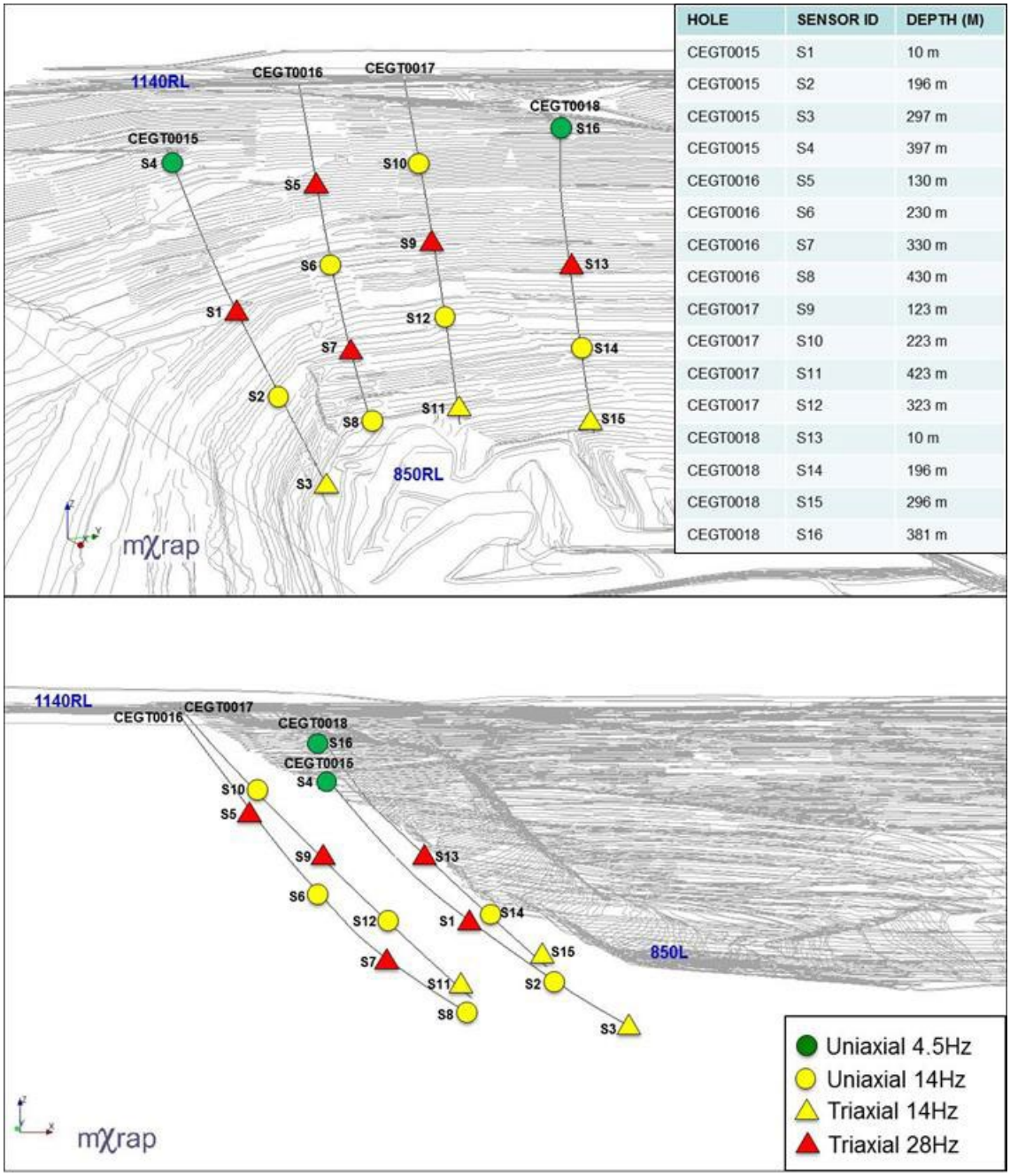
Monitoring system at Century

Surface deformations are measured by the combined use of prisms and radar. This permits a full cover of the surface of the wall, avoiding inaccuracies due to the direction of the movements. Radar can only measure variations of the distance to it. Without any idea of the movement vectors, there will be the risk of erroneous measurements and interpretations. If the actual position of the radar is pointing in the same direction as the movements, as recorded by the prisms, this is the best option. Few exceptions are presented at the top of wall and in the CBX, where the components measured by the radar may be slightly underestimated.

The microseismic system comprises sensors both near the surface and at the bottom of the monitored volume to maximise the location accuracy of the seismic events. It consists of uniaxial and triaxial geophones with frequencies of 4.5, 14 and 28 Hz (Figure 2). This range of frequencies permits the recording of a wider spectrum of seismic events in terms of local magnitude (ML~-5 to ML~2).

The system comprises 32 sensors, installed into a combination of four long inclined (~400 m) and two short vertical boreholes (~10 m). Each sensor is connected to one of the four data acquisition units, powered by solar panel. The seismograms are continuously transmitted to a central computer via digital radio, ready to be analysed.

Since 4.5 Hz sensors cannot be installed in holes more than 2° from the vertical, they were positioned near the surface in two vertical boreholes. Sensors were installed approximately every 100 m in each hole, with two sensors at each depth for the purpose of redundancy. Only 16 sensors are actually recording.



HOLE	SENSOR ID	DEPTH (M)
CEGT0015	S1	10 m
CEGT0015	S2	196 m
CEGT0015	S3	297 m
CEGT0015	S4	397 m
CEGT0016	S5	130 m
CEGT0016	S6	230 m
CEGT0016	S7	330 m
CEGT0016	S8	430 m
CEGT0017	S9	123 m
CEGT0017	S10	223 m
CEGT0017	S11	423 m
CEGT0017	S12	323 m
CEGT0018	S13	10 m
CEGT0018	S14	196 m
CEGT0018	S15	296 m
CEGT0018	S16	381 m

Figure 2 Overview of the microseismic monitoring system at Century Mine

Geophones were preferred to accelerometers because they are more sensitive in the typical frequency recorded from slope seismic events (10-400 Hz). Moreover, accelerometers are less reliable.

Project goals

The first part of the study involves the analysis of microseismicity. The current interpretation of the data is based on a homogeneous isotropic velocity model. It was obtained by a series of calibration blast tests, consisting of nine blasts shot in 15 m holes with approximately one second delay. The result has shown that Vp is about 3,774 m/s, while Vs is 2,046 m/s.

However, preliminary laboratory tests conducted on Century shales showed that the velocity of acoustic emission normal to bedding is about 3,533 m/s, while parallel to bedding 4,200-4,464 m/s. Therefore, one of the main challenges is represented by the strong anisotropic behaviour of the shales.

The rocks have been exposed to physical/chemical degradation processes. Consequently, the shallower portion of the wall is affected by a weathering surface. Even considering the seismic velocities, the assumption of homogeneous behaviour appears to not be adequate.

In February 2014, a ballistic test was performed in order to obtain a seismic tomography of the wall. The test consisted of 15 rifle shots, using a calibre of .308 to shoot the wall. The distance between the marksman and the impact was between 350 and 420 m at trajectories of 15-20°.

Unfortunately, the seismic energy generated by the impacts was insufficient to generate seismic signals with signal-to-noise ratios required to accurately predict the arrivals of seismic waves. The high fracturing level of the rock mass seems to have attenuated the seismic wave propagation.

Different hypotheses are currently under investigation. A solution may come considering the polarity of the seismic waves obtained by an accurate sensor orientation. Moreover, the use of an active source to generate a seismic tomography has also been proposed.

Another part of this research will focus on the modelling of the slope and rock mass properties. This is of upmost importance to identify any possible surface failures and the consequent volumes involved. A numerical model has already been made using Itasca software FLAC3D. In particular, to take into account anisotropy, the ubiquitous joint rock mass method (UJRM) was used, calibrated though synthetic rock mass strength (SRM). This assumes that SRM testing is an accurate representation of the rock mass strength and deformation behaviour in the tested loading directions and sample scales.

To verify those assumptions, a series of laboratory tests will be carried out to investigate the rock mass properties. Core subsampled in different directions will be tested axially, without confinement to investigate the anisotropic behaviour. This technique allows the magnitude and direction of in situ stress to be established.

A new numerical model will be built taking into account both the rock strength and stress parameters. A comparison with the previous model will be implemented. Finally, the accuracy of the model will be evaluated comparing the results to the actual surface and microseismic data.

Article references are available on request.

Michele Salvoni, PhD student,
Australian Centre for Geomechanics,
The University of Western Australia

Investigation into the lifecycle costing of tailings

by Professor Ken Mercer, Australian Centre for Geomechanics; and Professor Melinda Hodkiewicz, The University of Western Australia

The global tailings industry continues to investigate methods to increase the density of tailings produced from mining operations. This trend has been driven by practical needs, constraints such as increased fresh water scarcity and stricter environmental regulations. However, there are many other apparent advantages for adopting higher density tailings such as a smaller footprint, lower overall water usage as well as ease of management, and reduced costs for landform closure and underground support. Unfortunately these higher densities cannot be achieved without higher costs both in terms of higher capital and operating costs, and these costs can rise dramatically when considering very high density paste, filtered tailings and inline polymer addition operations. The questions that are typically raised are:

- What are the indicative capital and operating lifecycle costs typically associated with achieving different degrees of thickened tailings and filter cake?
- To what extent can these costs be offset by increased savings during operation and in closure?

In order to address these questions, the ACG is investigating the lifecycle costs of alternative thickened tailings management strategies ranging from conventional (unthickened tailings) through to the production of filter cake and mechanised transport. The primary objective being to assist tailings practitioners to more accurately undertake lifecycle costing for their projects.

From preliminary literature surveys, it has been immediately apparent that there is limited publically available data on this subject. There may be multiple reasons for this ranging from company confidentiality, site and country specific nature of the operations, vendor secrecy, the difficulty in extracting specific tailings related costing data, to a lack of interest by engineers or even the fact that no one has bothered to research it yet. Irrespective of the reasons, the ACG has set out to research the costs using a full lifecycle costing (LCC) modelling approach. The models which will be developed, like any model, are a simplified representation of the real world which captures the salient features of a real system and enables reasonable predictions to be made and sensitivity studies to be undertaken. The models will enable the costs to be broken down in a systematic way so that key cost drivers can be identified and evaluated across the different tailings disposal strategies.

The initial objectives of the study are to develop an inventory of all the expenses that occur during the production of tailings for four different tailings disposal strategies which are:

- Conventional unthickened tailings.
- Thickened tailings and transport using centrifugal pumping.
- Paste tailings and transport using positive displacement pumps.
- Filtered tailings and mechanical transport.

The expenses typically include the full lifecycle range from acquisition, installation and construction, operation, energy, maintenance, refurbishment, through to decommissioning and disposal, and will be used to develop the LCC models for these scenarios. In order to evaluate the financial implications of the alternatives, these costs need to be offset against any benefits for each scenario which would include, but not limited to, a reduction in water usage and closure costs. Pumping, tailings storage facility (TSF) construction and operation costs are influenced by the degree of severity of the local topography. Consequently each management scenario has been subdivided into three generalised terrain types ranging from flat, moderate and steep. The ACG plans to expand the disposal strategies later to include underground and inline polymer addition.

The study will be to:

- Undertake literature surveys and develop preliminary cost models.
- Development and circulation of a costing questionnaire throughout the tailings industry and collection of costing data (±6 months).
- Input, reduction and analysis of the data (±3 months).
- Provide feedback to the industry in the form of papers and further articles.
- A workshop at Paste 2015, Queensland is planned in which the result of the study will be presented and discussed.

Depending on the findings and results of this research, the ACG is considering possibly producing a set of simple guidelines that can be used by current and future tailings practitioners that details the LCC models that have been developed. The development of these guidelines will, however, be subject to industry consensus that there is a requirement for them.

On behalf of the ACG we would like to encourage as many tailings related practitioners as possible to participate in this study so we can continue to grow the industry and promote the environmental benefits of thickened tailings.

Contact the ACG to register your interest via marketing-acg@uwa.edu.au.

References are available on request.

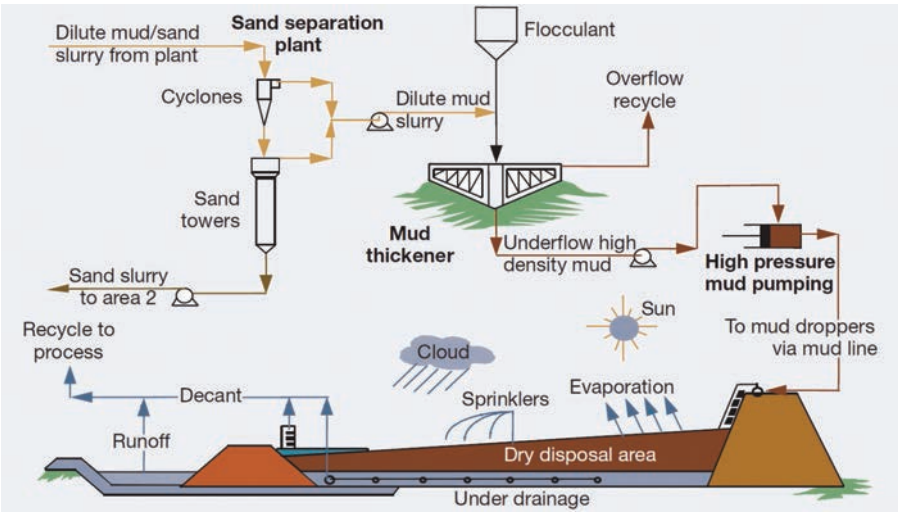


Figure 1 *The LCC model will encompass the components of tailings production, as illustrated in this schematic of an Australian dry-stacking process*

Professor Ken Mercer,
Australian Centre for Geomechanics

Professor Melinda Hodkiewicz,
The University of Western Australia

Paste 2014

An unprecedented number of papers have been submitted for the upcoming 17th International Seminar on Paste and Thickened Tailings

by Jack Caldwell, Robertson GeoConsultants Inc., Canada

It should be noted that a number of papers received for Paste 2014 relate to thickening oil sands tailings and mature fine tailings. This article details interesting aspects of some of these papers which are of topical interest to Western Canada.

One of them is entitled, 'Experimental study of the consolidation of mature fine tailings dewatered by using super absorbent polymer'. The authors write of laboratory tests to determine if it is possible to dewater oil sands mature fine tailings (MFT) using a super absorbent polymer. They placed super absorbent polymer in sachets made of cloth (textile) into containers of MFT. The water in the tailings passes through the textile and is absorbed by the polymer. The tailings moisture content decreases. The sachets are removed after a few days and the polymer is regenerated.

This interesting paper begs all kinds of questions, such as how one could achieve this on a commercial scale, or what could be done with the water squeezed out of the polymer? I was particularly pleased to read that they were able to further decrease the void ratio of the tailings by freezing and thawing it. Cold is so cheap in Northern Alberta that any rational scheme for tailings management must make use of the resource.

The other paper is entitled, 'Thickened tailings deposition – operational challenges and remedies'. The paper begins with a comprehensive evaluation of an aspect of tailing management. Yet I found this paper somewhat depressing: a long list of things that go wrong with thickened tailings and comparatively few ideas on how to deal with the problems. The authors conclude:

"The introduction of conventional tailings thickening in the Alberta oil sands is relatively new, and the challenges of thickened tailings deposition are only just beginning. It would be a pity if another promising tailings technology is discarded or passed by, or the true benefit of conventional tailings thickening is missed, simply because the remedies to these challenges are not readily or immediately apparent".

I have a paper entitled, 'Optimised Seasonal Deposition for successful management of treated mature fine tailings', written with co-authors A Revington, Suncor Energy Inc.; G McPhail, SLR Consulting Australia Ltd, Australia; and L Charlebois, Robertson GeoConsultants. We have been batting the ideas in this paper around for many years. Confidentiality agreements preclude us from writing about our test results. Now there are sufficient papers from previous seminars and from this seminar, that we can quote the literature to prove our concepts.

Here is what is involved in optimised seasonal deposition. Place polymer-flocculated MFT as follows:

- Spring deposition of MFT to a depth that consolidation by subsequent lifts induces consolidation and strength gain.
- Summer deposition of MFT in thin lifts which dewater and gain strength by way of evaporative drying.
- Fall deposition of MFT to a thickness where freezing and thawing induces a gain in strength.

Following are some of the papers to be presented at Paste 2014 that support the viability of our optimised seasonal deposition thesis. An impressive list of authors writing on topics germane to new ways of managing tailings, including:

- IDC Gidley and SS Jeeravipoolvarn, 'Key dewatering mechanisms of the environmental drying scheme for oil sands fine tailings'.
- T Moore, C Zhang, R Moffett and J Odle, 'Self-weight consolidation of Particlear™ treated fine tailings in a 10 m column'.

Our paper and many others at the seminar result from work directed to the deposition of flocculated MFT into dedicated disposal areas (DDAs), as mandated by Directive 74, a regulation issued by the Alberta Energy Regulators in order to promote the deposition of higher strength oil sands tailings.

Of course, the oil sands industry has already amply demonstrated that the 5 kPa, so central to Directive 74, is easy to achieve, and that 10 kPa does not produce a trafficable surface, despite published literature to the contrary. I know from personal experience that even 50 kPa is difficult to traffic.

The thesis of our paper is that, according to our understanding of the physical and financial requirements of such an operation, thin-lift drying alone will never be a stand-alone approach suitable for the dewatering of significant quantities of flocculated MFT, but that the management practice may be significantly improved by exploiting additional natural dewatering mechanisms and improving facility design. Flocculated tailings do not readily flow from spigots in thin, uniform lifts to form sheets over large areas as idealised in the thin-lift concept. This fact significantly reduces the reliability of thin-lift drying on a commercial scale. Additionally, the period of warm, dry weather available for drying is simply too short in northern Alberta, even if thin lifts could be consistently realised. Accordingly deposition will have to include profound consideration and the use of thin-lift drying (recognising its fallibilities), consolidation, and freeze-thaw as a means to increase the quantity of treated MFT that may be placed in a DDA of limited and fixed area.

This idea and the information in the papers of the Paste 2014 Seminar promise to generate deep discussion, new ideas, and industry advances.

Paste 2014 is accompanied by the following courses:

- The ACG Quantifying the Lifecycle Costs of Alternative Thickened Tailings Production and Management Methods Workshop.
- Rheology of Paste and Thickened Tailings.
- Tailings Preparation, Transport and Deposition.
- Paste Fill System Design, Operation and Management.

The 17th International Seminar on Paste and Thickened Tailings, to be held in Vancouver from June 9th to 12th, 2014, is shaping up to be a key industry event, following in the august tradition of the P&TT seminar series.

Jack Caldwell, civil engineer, Robertson GeoConsultants Inc., Canada

paste2014
17th International Seminar on Paste and Thickened Tailings
June 9–12, 2014 | Vancouver, Canada
www.paste2014.com

The ACG team at The University of Western Australia continues to grow!

Ariel Hsieh

Ariel joined the ACG in late 2013 as a research associate for the rock mechanics and in situ stress measurement project. Ariel completed Bachelor and Master degrees in geology and worked for Central Geological Survey in Taiwan. She relocated to Australia and completed a postgraduate diploma in mining engineering and PhD in geomechanics at The University of Western Australia. During her PhD study, she applied alternative stress measurement methods to more than 20 locations in Australia, Canada, Philippines, Peru, and Finland. The methods were evaluated and published in peer-reviewed journals. Since 2013, Ariel has undertaken a fundamental study of rock behaviour based on laboratory results. Ariel has experience and special interests in rock testing, which includes uniaxial/triaxial compression for physical properties, seismic velocity for anisotropy, CT scan for structure and porosity, and acoustic emission.

Thesis title: In situ stress reconstruction using rock memory

Ariel recently completed her PhD degree in geomechanics. Her PhD focused on two alternative in situ stress measurement methods based on using existing oriented rock core, namely the acoustic emission (AE) method and the deformation rate analysis (DRA) method. She has tested more than 500 rock samples from locations in several countries and artificial gypsum samples during her PhD. The mechanism, uncertainties and test procedure of the AE method were studied and the results give a fair view on the test conditions which affect the reliability/accuracy. The result was published in the Journal of Rock Mechanics and Mining Sciences. Another part of her thesis discussed the mechanism of DRA and the bending effect which occurs due to imperfections in the loading frame and/or sample preparation. The frictional sliding over the pre-existing crack, interfaces and/or grain boundaries was proposed, and the change in the rock stiffness caused by inelastic deformation was studied. In her postdoctoral research, Ariel will continue the investigation on the DRA method and start a fundamental study on rock behaviour under uniaxial/triaxial compression. AE, seismic velocity, deformation response and CT scan will be used in her research to understand the source of inelastic and elastic deformation.

Congratulations Dr Hsieh!

Ariel Hsieh, research associate,
Australian Centre for Geomechanics

Gordon Sweby

Gordon joined the ACG in early 2014 as a senior research engineer for the ACG's premier research project, "Ground Support Systems Optimisation" and will focus on the application of numerical models for ground support design.

Gordon is a rock engineering professional with 27 years' experience. He graduated from the University of the Witwatersrand in 1984 with a BSc in Geological Engineering, and then obtained a Masters in Rock Mechanics, also from Wits, in 1994. Gordon has the Chamber of Mines (South Africa) Certificates in Rock Mechanics (coal and metalliferous) and is a registered professional engineer with the Engineering Council of South Africa. He has practised as an operations engineer in both open pit and underground environments and in more recent times has held a corporate role as group geotechnical engineer for a multinational mining house. Additionally, Gordon has experience in a consulting environment, both as an independent and for various consultancy companies. Practical and technical expertise includes ground support design, mine seismicity, mine design and sequencing, ground control and safety, open pit design, technical studies and geotechnical monitoring.

Gordon Sweby,
senior research engineer,
Australian Centre for Geomechanics

ACG December 2013 newsletter correction

The ACG has been made aware of an error in the article entitled, "Performance of dynamic support system in highly burst-prone ground conditions at Vale's Copper Cliff Mine – a case study", by D Reddy Chinnasane et al., published in our newsletter volume 41, of December 2013. Mr Gary Davison of Mining Consultants Pty Ltd has pointed out that the data in Mr Chinnasane's Table 1 is incorrectly referenced. Specifically that the data, e.g. energy absorption of 46 mm split sets attributed as being sourced from Kaiser et al. (1996) is incorrect. Mr Chinnasane subsequently informed us that this data was sourced from Vale's own work. Kaiser et al. did not specify the size of the split set in their work and in subsequent personal communications with Mr Davison, Professor Kaiser stated that the data was not for 46 mm split sets but a smaller diameter version.

Mr Davison also points out that the peak loads for split sets and cone bolts appears to be transposed, and that Kaiser et al. refer to cone bolts, not modified cone bolts.

ACG event schedule*

Australian Centre for Geomechanics | Volume No. 42 | May 2014

ACG AUSTRALIAN CENTRE FOR GEOMECHANICS

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

2014

Practical Rock Mechanics (Introduction) Short Course	28–29 July 2014 | Perth, WA
Ground Support in Open Pit and Underground Mines (Introduction) Short Course	30 July–1 August 2014 | Perth, WA
Open Pit Geotechnical Analysis and Design Training Course	26–28 August 2014 | Perth, WA
Mine Dewatering and Mine Water Management Short Course	29 August 2014 | Perth, WA
Applications of Seismic Monitoring in Mines Course	13 September 2014 | Ontario, Canada
Practical Rock Mechanics in Underground Mines Course	13–14 September 2014 | Ontario, Canada
Getting the Most from a Seismic System Workshop	14 September 2014 | Ontario, Canada
Ground Support Subjected to Dynamic Loading Workshop	15 September 2014 | Ontario, Canada
Seventh International Conference on Deep and High Stress Mining www.deepmining2014.com	**16–18 September 2014 | Ontario, Canada**
Practical Calibration of Numerical Models for Meaningful Predictions of Ground Behaviour Course	19 September 2014 | Ontario, Canada
Blasting for Stable Slopes Short Course	3–5 November 2014 | Perth, WA
Mine Waste Landform Management and Closure Training Workshop	6–7 November 2014 | Accra, Ghana
Open Pit Slope Stability Training Seminar	10–11 November 2014 | Accra, Ghana
Radar and Monitoring Workshop	12 November 2014 | Accra, Ghana
Mine Dewatering and Mine Water Management Short Course	13 November 2014 | Accra, Ghana
Practical Geotechnical Logging Workshop	14 November 2014 | Accra, Ghana
Unsaturated Soil Mechanics for Mining Seminar and Unsaturated Soils Laboratory Testing Workshop	1–3 December 2014 | Perth, WA
Applications of Unsaturated Soil Mechanics to Tailings Workshop	4 December 2014 | Perth, WA

2015

Mine Closure Structured Training Seminar	**17–19 March 2015 | Perth, Australia**
Quantifying the Lifecycle Costs of Alternative Thickened Tailings Production and Management Methods Workshop	**4 May 2015 | Cairns, Australia**
Paste 2015 www.paste2015.com	**5–7 May 2015 | Cairns, Australia**
Ninth International Symposium on Field Measurements in Geomechanics www.fmgm2015.com	**8–10 September 2015 | Sydney, Australia**

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

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

PASTE 2015

Paste 2015

5–7 May 2015 | Cairns, Queensland

The ACG looks forward to welcoming many previous seminar attendees and those interested to learn more about mine waste management to Paste 2015. The last time that the event was held in Australia was Paste 2011, Perth; with more than 260 delegates attending.

www.paste2015.com

Abstracts due 22 July 2014