Fundamental Techniques for Reducing Risk Associated with Instabilities in Mining Slopes

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Abstract

This paper discusses some of the fundamental considerations when managing mining slopes. The goal of management is to reduce all components that contribute to the geotechnical risk and by doing so reduce the risk to as low as reasonably achievable. The techniques and procedures suggested are not exhaustive; they represent a snapshot of some of the practical techniques the author has found useful for a range of scenarios.

Keywords: Mining Slopes, Instability, Risk Management

Introduction

Trends in mineral production rates are linked closely to population and economic growth. International population growth has been steady since the mid 1300s. The highest rate of growth occurred during the 1960s and 1970s, with a peak rate of 2.2% in 1963. Current projections show a continued increase in the international population, with high growth rates expected in India, Central and Northern Africa and parts of the Middle East, although there will be steady decrease in the rates in many other areas. The international population is expected to reach between 7.5 and 10.5 billion in 2050. International economic growth increased rapidly following the Great Depression from the late 1920s to the early 1940s aided in part by an increasing demand for goods. Although growth in western nations slowed after 1973, growth since then began in Japan and then spread to Korea, China, India and other Asian countries where it remains strong. This growth is being fuelled by rising incomes in some socio-economic sectors of these countries. These factors have contributed to the rapidly increasing annual growth rate in the demand for mineral products since the 1930s. The mining industry has had to expand evermore rapidly to satisfy this demand.

Since the 1900, the grades of most ores have been showing a long term declining trend. On average, grades of metals have decreased from between 15% and 25% in the 1800s to less than 10% today. Improvements in technology have enabled the exploitation of lower grade deposits although, to do so, the energy requirements per unit of mineral production have had to steadily increase (Mudd [1]). Since the 1980s, the annual growth rate in the reserves (in contrast to resources) of most ores has increased only slightly or levelled out; although the rate for some metals (e.g. iron ore) has been gradually declining.

Increasing population and economic growth, reduction in ore grades and plateauing reserves have necessitated a rapid increase in the volumes of materials that have had to be mined to obtain the necessary increases in ore volumes required to satisfy the increasing demand for minerals. Although some minerals have historically been mined by open-cut methods (e.g. coal, iron ore, bauxite), the trends have led to an increasing use of large scale open-cut mining methods for other minerals (e.g. copper post 1960, coal, nickel). The relatively low cost of energy over this period has encouraged open-cut mines to get larger and deeper. For example, Kennecott's (Rio Tinto) Bingham canyon copper mine in Utah, USA is approximately 1200m deep. The proposed open-cut at BHP Billiton's Olympic Dam copper, uranium and gold

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mine in South Australia may go to 1000m. Codelco's Chuquicamata copper and molybdenum mine in Chile is 900m deep and Minera Escondida's (Rio Tinto, BHP, JECO) Escondida copper mine in Chile is planned to go to 750m. This paper discusses some of the challenges facing the Rock Engineers who manage the risk associated with the wider and higher pit walls.

It is not possible for this paper to discuss or even mention all aspects of this process; in fact an embarrassingly large range of topics must be left out. It can only hope to present some of the issues. Although it focuses on slopes in open-cut mines, the principles discussed as just as applicable to any slope in rock. Examples of these slopes can also be found around the upstream reservoirs of hydroelectric projects and alongside water storage dams.

Challenges

A rock mass is a complex system involving:

- many components, a high proportion of which have characteristics that vary non-linearly;
- high spatial variability in the characteristics of the components;
- characteristics that are unknown at any particular location;
- components that are coupled; hence changes in one components can influence another component (e.g. joint water pressure, discontinuity aperture width and discontinuity shear strength);
- randomness which significantly complicates the ability to make predictions about the performance of any particular area on a slope.
- The combination of these factors creates a system conducive to unpredictable behaviour. As slopes become higher, considering this unpredictability and managing the slope appropriately becomes increasingly more challenging.

The upper 100m+ of elevation of a slope often comprises extremely weathered rocks, saprolites and residual soils. Ensuring benches within these materials remain stable requires management of overland flows, erosion and pore pressures. Doing so effectively can be challenging, especially in moderate to high rainfall locations.

Different hydro-mechanical conditions can exist at different elevations (Sullivan [2]) i.e.

- Unsaturated conditions with no pore pressures at upper elevations.
- Transient saturated conditions in middle elevations, with pore pressures varying according to seasonal fluctuations in rainfall infiltration.
- Partially saturated conditions below the water table, with pore pressures varying according to the depressurization regimen.
- Saturated conditions below the pit floor with full hydrostatic pore pressure.
- These variations influence the relative performances of different sections of a slope with the performance at one elevation influencing the performances at another elevation.

Different elevations of a slope, particularly those that are large, can behave differently even when the sections have similar geology and structure. For example, high biaxial and triaxial stress conditions can develop below the toe of a large slope due to straining following excavation. Under these conditions, confinement dependent strength and dilation become an issue with even high strength rock exhibiting failure mechanisms (e.g. spalling in addition to shearing) that are non-typical in small slopes. Defining what failure mechanism will be applicable can be problematical due to a lack of experience as to the applicability of standard rock mass strength criteria (e.g. Hoek-Brown) for such conditions.

The number of very highly persistent, potentially unstable, structures that daylight out of the slope is likely to increase as the height of a slope increases. Therefore, the likelihood for an event involving very-large volumes of rock increases accordingly.

Financial pressure to increase output can result in the number of shovels and trucks working within a pit increasing. Large pits generally have several areas being worked simultaneously. The increased number of personnel within a pit increases the likelihood that persons will be below the location at which a rapid rockfall event occurs.

Depending on the size, depth, mining rate and the value of the mineral, steepening the wall of a pit can add an additional 5+ years to the life of the pit. Even a 1^{o} increase in the overall slope angle of a wall of a large pit can add hundreds of millions of dollars of additional NPV to the project. These factors place significant pressure on those designing the walls to accept designs with a higher than desirable likelihood for instability. Doing so is only acceptable if wall management procedures are established that ensure that the risk remains tolerable.

One final challenge with the management of slopes, particularly those that are large, is that there is a global shortage of qualified Rock Engineers who are experienced in designing and managing these slopes.

The only way to address these challenges is to have a sound, comprehensive pit slope management programme. At the heart of such programme is the concept of risk.

RISK, $R_{(LOL)}$

Risk management requires consideration of the components of the risk equation. For loss of life this equation can be expressed as follows (AGS [3]):

$$R_{(lol)} = P_{(H)} \times P_{(S:H)} \times P_{(T:S)} \times V_{(D:T)} \qquad (1)$$

Where,

$R_{(lol)}$ is the risk (i.e. annual probability of death of an individual).

$P_{(H)}$ is the annual probability of a hazardous event (e.g. rockfall).

$P_{(S:H)}$ is the probability of spatial impact (i.e. probability that a person occupies the trajectory of the hazard).

$P_{(T:S)}$ is the temporal probability (i.e. probability that a person does so at the exact time of the hazard).

$V_{(D:T)}$ is the vulnerability of the person to impact (i.e. likelihood that impact will cause death). Finlay et. al. [4] suggested that $V_{(D:T)}$ be quantified as listed in Table 1.

Table 1: Vulnerability $V_{(D:T)}$ ranges for persons. Adapted from Finlay et.al [3]

Scenario	Range in Data	Recommended value	Comments
	Person in Open Space		
Hit by rockfall	0.1–0.7	0.5	May be injured but unlikely to cause death
Hit by debris and buried	0.8–1.0	1.0	Death by asphyxia almost certain
Hit by debris yet not buried	0.1–0.5	0.1	High chance of survival
	Person in Mine Vehicle		
Mine vehicle is crushed	0.9–1.0	1.0	Death is almost certain
Mine vehicle is damaged	0–0.3	0.3	High chance of survival

The risk to persons can be reduced by reducing any one of the four components of Equation 1. Good risk management requires that all components be considered and each reduced where possible. For example, there is no risk if a rockfall event

- does not occur;
- occurs but persons are not within the trajectory;
- occurs but persons within the trajectory are isolated from impact.

The goal of risk management should be to reduce the risk to a level as Low As Reasonably Achievable (ALARA) but certainly below that considered tolerable. The tolerable risk for loss of life for engineered mine slopes can be considered to be 1×10^{-5} per annum (AGS [3]). The following sections present some of the systems commonly used for achieving this goal. These systems presented are not exhaustive; they represent a snapshot of some of the practical techniques that the author has found useful for a range of scenarios.

Reducing Annual Probability of Instability $P_{(H)}$

Effective Design

Discontinuity spatial data required as input into stability analyses can only be obtained if a thorough programme of discontinuity mapping has been carried out in each geotechnical domain. This process may involve core logging, scanline mapping, photogrammetry and/or borehole scanning with tele-viewers. A rigorous series of discontinuity analyses are then carried out to assign discontinuities into sets and define the distributions of the:

- orientations of the discontinuities in each set in terms of a mean orientation and a measure of variability (e.g. Fisher's constant);
- spacings between adjacent discontinuities from the same set;
- proportion of random discontinuities within the population (sometimes referred to as the "isotropic component");
- block sizes within the rockmass and
- areal extents (persistences) of the discontinuities.

Quantifying the areal extents of the discontinuities is particularly important in large pits as discontinuities having very-high persistences can underlie kinematically unstable blocks having very-large volumes. The persistences cannot be ascertained from borecole core. Neither are they the same as the distribution of trace lengths observable on exposed surfaces although they are a function of this distribution. A discussion of this issue is included in Priest [5].

Also to be ascertained is the proportion of random discontinuities within the sampled population. The characteristic of randomness is assigned to those discontinuities that cannot be allocated into one of the sets. It is not unusual for a rockmass that has undergone some deformation to have a 30% random component. A significant proportion (i.e. >20%) of instabilities that occur within many pits comprise one or more random discontinuities. However, the influence of these discontinuities on the stability of pit wall is often ignored during the design process. Proprietary rigid block analysis software does not generally have the capability to consider the random component. Until such capability becomes available, some allowance for their presence can be made by manually modifying the parameters that define the variability of the discontinuities within each set (e.g. reducing Fisher's constants) although doing so is statistically incorrect.

All analyses carried out for bench and wall scale designs are to be probabilistic based; deterministic analyses are inappropriate.

At sites where there is a significant depth of extremely weathered rock and residual soils, the influence of pore and joint water pressures needs to be considered as the phreatic surface will normally be above the pit floor. Hydrologic modelling will be necessary to ascertain pressure heads behind the walls as these pressures can influence differently the shear strengths of the materials in the saturated and unsaturated zones. Modelling is also required to predict the influence of slope depressurization and groundwater discharge into the pit. Input data for the models will require field and laboratory testing to ascertain hydraulic conductivities of the various stratigraphic units and the location of the natural phreatic surface.

Depending on the scale of the design component being assessed, rigid block, continuum, discontinuum or hybrid methods of numerical modelling may be appropriate. There is not scope in this paper to discuss the merits and drawbacks of each method however a thorough discussion is provided by Read and Stacey [6].

Worthy of mention is one of the more recent and ongoing developments in discontinuum modelling; ITASCA's Slope Model code (Emam et.al. [7]). The code enables a 3D pit to be modelled complete with discontinuity structure. The model does not impose the unrealistic assumptions about discontinuity orientation implied in 2D discontinuum codes. It also does not impose the unrealistic limitation on block numbers required in 3D discontinuum codes or require their extremely long time run times. The code does not require the rock mass strength to be based on any rock mass classification scheme or failure criterion, assume a failure mechanism or a shape for a failure surface. The model is based around the micromechanics principles coded in ITASCA's Synthetic Rock Mass (SRM) models. However, whereas the SRM can be computationally intensive, Slope Model uses a simplified discrete approach which makes it significantly more efficient. The entire failure process is generated by the model itself in response to the changing stresses induced by mining (Hoek, [8]).

Results of analyses are generally expressed in terms of minimum factors of safety and maximum probabilities of failure. These values are compared to acceptance criteria to assist with the selection of appropriate design specifications. A thorough discussion on these criteria is provided by Read and Stacey [6] and is summarised in Table 2.

The design specifications should also consider the significant cost imposed by reducing batter angles and hence overall slope angles (assuming berm widths remain constant) compared to the cost to remediate a failed bench and the possibly negligible effect on the operation of leaving a failed bench in place. Also valid is the potentially significant cost to the operation if a failed bench is located above or below a ramp or haul-road.

Table 2: Typical FoS and PoF acceptance criteria values (Read and Stacey [6])

Slope scale	Consequence of failure[b]	Acceptance criteria[a]		
		FoS_{min} (static)	FoS_{min} (dynamic)	PoF_{max}
Bench	Low-High	1.1	n/a	25-50%
Inter-ramp	Low	1.15-1.2	1.0	25%
	Medium	1.2	1.0	25%
	High	1.2-1.3	1.1	10%
Overall	Low	1.2-1.3	1.0	15-20%
	Medium	1.3	1.05	5-10%
	High	1.3-1.5	1.1	≤5%

a: needs to meet all acceptance criteria
b: Semi-semi-quantitatively evaluated

Once a design is established at the planning stage, there should not be any hesitation in changing a design at a later stage if the performance of the slope once excavation begins warrants doing so. Over the past 30 years, more than 90% of medium to large scale pits to 1000m depth have changed overall slope angle by $\pm 3^o$ to 16^o (Sullivan [9]).

Scaling Batters

Thorough scaling of all batters, whether they are at the final limits or not, is the single-most effective technique for reducing the risk to personnel associated with rockfalls. Scaling generally occurs in two stages; primary and secondary.

Primary scaling begins as soon as the excavator reaches the blasted muckpile. It continues while the excavator waits for trucks. The operator removes all loose rock from the crest to the toe taking care to preserve the crest. To access the crest, the excavator can create and stand upon a ramp of broken rock. However, as bench heights should not be specified that are greater than the maximum reach of the excavator, doing so should not be necessary.

Attention must be given to achieving competent crests. An overhanging crest comprising a secure massive boulder may be less of a concern than is a crest comprising a mass of small rocks. If a crest will be damaged, or a potentially unstable cavern created, by removing an overhang, it is often better to avoid doing so as long as the overhang is reasonably secure.

The loading excavator is generally fitted with a bucket that is too large to effectively remove individual loose rocks. It is for this reason that secondary scaling is carried out after the muckpile has been removed and before the final cleanup of the berm.

A dedicated scaling excavator may be assigned the time consuming task of secondary scaling. Doing so frees the loading excavator to be used elsewhere. The excavator is often fitted with a longer reach boom, than that used on the loading excavator, to ensure it can access the crest while standing a safe distance back from the batter. It will either be fitted with a small toothed bucket or alternatively configured as a backhoe and fitted with a ripper tyne to enable individual loose rocks to be removed.

If the reach of the available excavator is insufficient for effective secondary scaling to be carried out, it may instead be done by “chaining” with a dozer. Chaining can be more effective for scaling than doing so with an excavator.

Chaining uses a heavy link chain, such as a ship’s anchor chain, that is at least 10m longer than the height of a bench. Its effectiveness can be improved if dozer track plates are attached to its far end.

The chain is attached to the drawbar of a dozer on the berm above the batter to be scaled. The operator positions the dozer sufficiently far back from the crest so as not to be at risk if a damaged crest collapses. The end of the chain is pushed over the batter and falls to the toe. The dozer then moves forward, allowing the chain to drag over the batter. A slow speed is essential to ensure that the dozer is not pulled off-balance if the chain is caught on large rocks. The process can be repeated if necessary until no loose rocks remain on the batter.

Control of Overland Flows of Water

Overland flows towards the crests should be diverted to contour lows by the use of diversion ditches or bunds. These infrastructures should be at least 1m wide and installed at least 10m behind the crests so they do not obscure tensions cracks that develop behind a crest. Clay linings and facings increase their longevity and effectiveness. They should be repaired and cleaned as required.

Water must not be allowed to pool on berms, ramps or haul roads. To prevent this occurrence, if benches are cut in hard rock, they can be graded at 5% towards the adjacent crests thereby enabling water to be removed down batters. Alternatively, berms can be graded at 5% back from the crests into V-drains running along strike. Drains must have a cross-section with an area sufficient to cater for peak flows. They should grade at a minimum of 1% towards vertical batter drains or to the ends of the slope.

Achieving effective drainage on berms that are cut in extremely-weathered rock or sediments is critical and challenging. Water cannot be allowed to flow directly off the berms as the crests and batters will erode thereby causing silt to build up on the berms below. Over time, water flowing on the berms will preferentially gravitate into channels which will erode and form gullies. If not repaired, the gullies will eventually cause the benches to deteriorate which will trigger bench scale failures. Water on these berms should be directed into lined V-drains running along strike. The drains can be installed along the centrelines of the berms to reduce the likelihood for the drains to silt up with sediments that erode from adjacent batters. A sufficient flow velocity can generally be achieved if the drains are graded at 1%. The drains can terminate at vertical batter drains.

Shotcrete or fibrecrete can form an effective lining for V-drains where alternative lining materials are not available. However, as often over 30km of drains need to be installed, the significant cost to mobilise a concrete batching plant and shotcreting rig and to purchase cement can be a significant cost to the project.

An alternatively technique to drain berms cut in extremely weathered rock or sediments is to use subsurface drains. These drains comprise perforated pipes placed within backfilled trenches. The pipes can be either flexible corrugated pipes having circular or rectangular cross sections or rigid uPVC. Both types are socked in geotextile filter fabric. Flexible pipes require trenches to have a minimum grade of at least 1% whereas uPVC pipes require trenches with a minimum grade of 0.3%. The trenches are backfilled with medium gravel sized (i.e. -6mm) hard crushed rock. They can terminate at batter drains.

Batter drains are used where water needs to be directed down batters that would otherwise be erodible. The drains comprise 2m wide chain-link mesh pinned to the batter with spikes and then sprayed with a 25mm layer of shotcrete. The mesh may not be required if fibrecrete is used. A maximum 100m spacing between drains is often efficient as it limits the depth of fall of the berm drains. Although batter drains can cost approximately US$1000/m to install, the cost can be insignificant compared to that required to repair benches that have failed due to excessive erosion.

Water in drains must be directed to the pit floor as rapidly and efficiently as possible from where it can be pumped out of the pit. They require regular cleaning as required to prevent being blocked by siltation.

Water pumped out of a pit must be channelled into contour lows, clay lined raw water dams or the tailings dam. It must not be discharged anywhere near the pit to prevent it permeating into the pit walls.

Controlled Blasting

In the last decade, economics has driving the increased use of high-energy mass blasts. For example, blasts involving in excess of 4million tonnes of rock using 1400 holes drilled along 1.2 km of berm and containing 2000 tonnes of explosive are not uncommon. The energy resulting from blasts of these sizes has the potential to damage final walls and production walls containing infrastructure such as ramps. In an effort to reduce damage, controlled blasting techniques have become more widespread, primarily for limit blasts but also for production blasts. Some of the techniques being used are as follows (Orica [10]):

- Buffer (cushion) blasts have the lowest cost of the controlled blasting techniques. They can be used in low-strength rocks and/or when design batter angles are less than 60^{o}. They involve using a standard production pattern with width three to six rows deep and no sub-drilling into the design berm below. The rear row of holes is drilled on a smaller pattern and stands off from the location of the design batter by 5m. Holes in this row contains lighter charges than do those in other rows and have a delay sequence modified to reduce vibration levels and displacement.
- Trim blasting (post-splitting) can be effective in medium to high-strength rock particularly when design batter angles are 60^{o} to 75^{o}. The blasts often involve 3 to 5 rows of holes with no sub-drilling into the design berm below. The rear row of holes is closely spaced and stand-off from the toe of the design batter by 1m. The burden on these holes is made greater than the spacing between the holes to encourage the web between the holes to split. The rear holes are charged with a light, continuous column of highly decoupled explosive (e.g. 25mm explosive for 89mm blastholes). Often they will contain an air-deck between the charge and the stemming to reduce pressure. They are fired after the production holes in front have detonated. If successful, the webs between the holes split producing a smooth face with minimal over-break.
- Pre-splitting is most effective in massive high-strength rock. It involves a row of closely spaced holes drilled along the design limit. The holes are usually 76-102mm diameter but can be up to

250mm. In average rock conditions, the charge load increases with the hole diameter, but is generally less than that used for the production holes in front. Optimum charge load varies considerably with rock characteristics; weak and/or highly discontinuous rocks requiring less charge and reduced hole spacing than do high-strength massive rocks which requires a greater charge load. Holes are often air-decked. The holes are detonated simultaneously, before the holes in front of them, which can produce high vibration and air-blast levels. The webs between the holes split thereby producing a smooth surface. Strain wave produced from the subsequently fired holes in front are partially reflected by the surface which also acts as a vent for gasses from the holes. If successful, the result is a relatively undisturbed batter with minimal damage.

Pore Pressure Management

Maintaining the stability of the upper benches often requires minimising pore pressures in the sediments within which the benches have been excavated. The phreatic surface must be lowered to a level below the sediments by depressurizing the walls. Doing so may involve pumping from vertical wells located behind the crests and from sumps within the pit floor and gravity drainage from horizontal drains in the benches. The success of these measures depends on the hydrogeological characteristics of each stratum within the rockmass.

Ensuring depressurization is successful requires ongoing monitoring of the pressure heads at various locations behind the crests using piezometers. Various manual, pneumatic and electrical piezometers are available to suit the hydrogeology and site accessibility. The most basic and low-cost device for use when time-lag is not an issue is a standpipe piezometer. This device comprises a filter element fitted to the end of a series of thick-walled PVC piezo-tubes. The tubes are placed within a vertical borehole with the element located at the depth required. A pea-gravel filter pack is placed around the element and sealed off from the annulus above with a bentonite layer. The annulus is then sealed off with a weak cementitious grout. The depth to water in the tubes is measured with a water level indicator enabling the pressure head at the element to be calculated.

Reducing Temporal and Spatial Probability of Impact $P_{(T:S)}$

Dynamic loading by relatively small rocks can cause considerable injury to persons. Impact of the head can cause the scull to deform and fracture. It can damage the brain if it is distorted, stretched, compressed or torn away from the skull. The amount of damage is related to the kinetic energy dissipated on impact; damage occurring at between 45 J and 102 J (avg. 68 J) of energy. The lower range can be dissipated by a 570 g (60 mm) rock released from a 10 m high crest and launching out from the batter. Personal head protective equipment (i.e. hard hat or helmet) is designed to dissipate up to 120 J of energy. This energy can be dissipated by a 1.5 kg (84 mm) rock releasing from the same height. Falling Object Protection (FOP) equipment installed in mining vehicles is designed to absorb approximately 50 kJ of energy. This energy can be dissipated by a 320 kg (0.5 m) rock falling from the crest of a 20m high batter. None of these rocks of fall heights is particularly great indicating how dangerous any rockfall can be. Avoiding impacts can be achieved by either being:

- within the trajectory of a rock but not present at the time of the event (achieving temporal separation) or present at the time of the event but not within the trajectory of the rock (achieving spatial separation).

Ensuring that at least one of these requirements is applicable is a key component of risk management. However, doing so must not be considered to be a solution for poor slope design, inadequate scaling of batters or damage to benches due to poor blast design or inadequate water management.

Ensuring Persons within the Trajectory of a Falling Rock are Isolated from Impact

Systems that can prevent persons within the trajectory of a falling rock from being impacted are as follows:

Specifying Effective Berm Widths. Berms are the primary component in a slope design for limiting the trajectory of falling rocks. As a "rule-of-thumb" production berms should have a minimum width 2/3^{rds} the height of benches. Final berms should have a minimum width defined by width (m) = 0.2 x bench height + 4.5m (Ryan & Prior [11]). Additional width (i.e. 1m to 2m) should be specified to allow for overbreak. On large slopes, a "geotechnical berm" having additional width may be specified for every 10 benches. For berms to remain effective, drainage must be maintained, damage must be repaired and rockfall debris removed.

Bunds. Bunds constructed on production berms can temporarily reduce the risk to persons who must spend time in front of production batters e.g. blast-hole drillers and shot-firers. Bunds are normally constructed from blasted rock and are at least 2m high. They should be placed at least 1/3rd the height of a bench out from the toe of the batter. Their effectiveness can be improved by excavating a trench between the batter and the bund. However, trenches should not be used if they will create a location for water to pool. If a permanent structure is required at the toe of final walls it can be achieved with a reinforced soil embankment which can dissipate energies in the order of 10MJ. Such structures are not however commonly used in mining.

Barriers. Over the past decade, the use of proprietary flexible rockfall barriers (e.g. Geobrugg, Maccaferri, Pfeifer-Isofer) in mining has been steadily increasing. Barriers are generally placed on the berms of final slopes or out from their toes on the pit floor where the risk to persons or infrastructure is deemed unacceptable. Their placement enables rockfalls from a large area to be controlled. Proprietary barriers are 1:1 tested and certified. Most have wide post spacing resulting in short installation times. They can easily be cleared out after a rockfall event. The unit cost of barriers is a function of their energy dissipation capacity and their height. Although barriers having 500kJ capacity are available, 500kJ capacity is common in mines although there are sites that have barriers with 1500kJ capacities. Barrier heights from 3m to 6m are common in mining although heights to 9m are available. The cost to supply and install a barrier can be significant (e.g. \$1500 to \$3000+ per metre) hence a barrier that has too high or too low a capacity or height is a waste of money. Too low a capacity or height is dangerous as it can provide personnel with a belief that the risk is less than it actually is. Correct barriers selection requires a statistically valid assessment as to the distribution of rock masses and an assessment as to the potential trajectories of rocks. The latter will normally require numerical modelling.

Proprietary 2D rockfall analysis software is commonplace and 3D software is becoming available. The results from most numerical analyses are generally highly sensitive to the characteristics assumed for the rocks and the material comprising the slopes, in particular the coefficients of restitution. Selecting appropriate values is best done by assessing the results from physical field tests or by back-analysis of actual rockfalls.

Draped Mesh. Rockfall mesh (e.g. Geobrugg, Maccaferri) can effectively reduce the risk associated with rockfalls from final batters by controlling the horizontal displacement of rocks thereby preventing them from launching out from a batter. Mesh is generally suitable for controlling individual rocks up to 0.6m diameter or ravelling masses of small rocks up to 10m^3 in volume. Multiple batters can be covered in a single drapery although it is more common for individual batters to be draped. Mesh aperture sizes range from 45mm to 80mm and tensile strengths from 48kN/m to 85kN/m. Construction generally involves lacing the upper and lower edge of the mesh to steel cables. The upper cable is attached to grouted dowels located 2m behind a crest at 2m to 4m centres. The majority of the load carried by the mesh is transferred to these dowels therefore their length and the diameters of the dowels and the grout columns must be designed appropriately. The lower cable is attached to grouted dowels at 5m centres above the toe. The rope is held tightly against the toe if sufficient area is not for rocks to accumulate safely. Otherwise, the rope is allowed to move up to 1m out from the toe, which enables rocks to exit the mesh. Doing so reduces the loading on the mesh by accumulated rocks. Adjacent panels of mesh are either laced together with steel rope s or, more often, clipped together with steel “hog rings”.

Ensuring Persons are not within the Trajectory of a Falling Rock

Exclusion zones remove persons from areas generating falls of rocks (e.g. adjacent to production digging and below cutbacks) and areas that are potentially unstable. If zones are to be effective they must be rigorously enforced with persons breaching restriction being reprimanded. Various levels of zones would normally be created e.g.

- No entry.
- Entry by heavy vehicle with suitable FOP systems.
- Entry by light vehicle only – no foot traffic.
- Entry by foot traffic allowed.

Visual Inspection. Benches above all active mining and dumping areas should be inspected daily by a trained Ground Control Supervisor and visually monitored throughout the mining cycle. All non-active mining and dumping areas should be inspected thoroughly weekly. Features of interest include fresh cracks, lowering of the ground surface; bulging on a batter or a berm, water running over a crest or entering cracks, pooling of water, water issuing from a batter and rocks on berms that have detached from batters.

Surface Extensometers. Any cracks or surface movement on a berm or behind a crest should be monitored with a surface extensometer. Proprietary devices having digital readout and data logging capability are readily available. However proprietary devices with manual readout can be just as effective, reliable and, being of low cost, can be used regularly. Manual devices are simple to operate so non-technical personnel can be trained to use them correctly. Readout units are generally attached to a steel stake installed on stable ground. They are connected by a thin stainless-steel cable to an anchor installed within the potentially unstable ground. If movement is indicated to be occurring, systems can be connected by electrical cable to a remote alarm system consisting of a limit switch, solar-charged 12V battery, flashing xenon beacon and 120dB pulsed tone siren. The Ground Control Supervisor or their appointee read each instrument daily. The readings are plotted in terms of the date whenever displacement occurs.

Laser scanning. Prismless laser scanners (e.g. Riegl, Leica, Maptek’s I-Site) enable pit walls to be scanned from distances up to 6km with an accuracy of 25mm to 50mm. As prisms are not required, there are not the safety and time issue applicable when installing new prisms or replacing damaged or lost prisms. There is also not the significant limitation that only those locations with prisms can be scanned; failures generally seem to occur between prisms. The scanners are often fixed permanently within a rigid steel enclosure located behind the crest of a wall opposite the wall to be monitored. The latter wall would initially be divided into a grid and, automatically when required, each point on the grid is scanned; the scanner being controlled by inbuilt software controlled motors. The scanner operates by emitting pulses of laser light even under bad visibility conditions. The distance to the target is calculated based on the time taken for the reflected light to return (Riegl, 2011). The devices can be combined with a high resolution digital camera which provides data that enables coloured point-clouds, textured triangulated surfaces and high resolution 3D images to be obtained. These images are digitally compared to those obtained from previous surveys; areas of potential instability are highlighted and the rate of movement at these areas can be plotted.

Photogrammetry (e.g. SiroVision, AdamTech, JointMetrix) involves taking a minimum of two overlapping digital photos of a pit wall and digitally processing and manipulating the photos. A standard digital camera is set up on a levelled tripod and a photograph is taken of the wall. The location of the camera and an identifiable reference point on the wall is surveyed in using a handheld GPS. The tripod is then moved a sufficient distance to the left to ensure that there is at least 90% overlap between photographs. The distance depends on the distance between the camera and the wall. The process is then repeated (Little [12]). The photographs and survey locations are then uploaded into software which creates a 3D photograph and a 3D point cloud from the images. The results can be used to determine quickly and safely the spatial characteristics of large structures that would otherwise have been inaccessible yet may have a significant influence on the stability of a wall. Having the ability to overlay the structures on a pit design, view where they intersect the wall, where they daylight from it and how they may influence its performance is particularly useful. The results also enable 3D changes in the characteristics of the wall over time to be highlighted. These changes may be the precursor for instability. Knowing their whereabouts enables instrumentation and radars to be located efficiently.

In-Pit Radar. In-pit radar (e.g. GroundProbe, Reutech) has been one of the most significant developments in the past 30 years for reducing the risk associate with instability of slopes. The reliability of early radars was poor (<60% availability) however significant improvements now provides much better reliability.

Radars enables 270^o horizontal and 100^o vertical scanning of pit walls in real time from a distance of up to 2.8km without the need for reflectors (GroundProbe, 2011). Units can have 0.1mm accuracy depending on range and can operate in fog and darkness. A radar can be set up in about an hour. It initially takes 14 photographs of the area it can scan and converts these to a mosaic. The operator then indicates the 2D area to be scanned and scanning begins. It can scan $10{,}000m^2$ of a wall in one minute and provide early warning of any pending failure. It does this by recording the time for a signal to be sent, reflected and received from a point on the wall. A signal processing technique called differential interferometry is used to achieve the desired level of accuracy. It compares the phases of the radar signal it receives from one scan to the next. Any phase difference is converted to a measurement in millimetres and the information is displayed as colours on a computer screen; hot colours indicate movement towards the radar, cool colours indicate movement away, as would occur if a rock fell from a batter. Flashing beacons, sirens and SMS warnings can be activated if movements exceed a pre-set amount.

Microseismic monitoring (e.g. ISSI) has been used successfully for over a decade in underground mines. It detects the onset of brittle fractures deep within a rockmass before there is any evidence of instability on the surface of a nearby excavation. The system involves geophones suited to operating in the 10-

400Hz range typical of slope seismic events. The geophones are installed near the collar and at the base of each of a series of short vertical and long inclined boreholes located behind a pit wall. Sensors are normally installed 100m to 200m apart. These devices are sufficiently sensitive to detect accelerations associated with movements to 0.001mm. The low amplitude (1µm/sec) signals that typically are generated are sampled at 6000 Hz by seismic stations. Event data is transferred off site via internet transfer for processing by seismological specialists. The arrival times of seismic events at different geophones are compared and triangulation locates the event within the rockmass. A series of events from the same area can enable a surface to be identified along which movement may be occurring. Increases in the frequency of events can indicate the development of a possible failure scenario involving the surface, long before any movement is detected on the wall. Microseismic monitoring has only been applied to a small number of surface mines in the past 5 years, primarily as a research tool. However, the systems are expected to become more common when the significant costs to implement them, compared to alternative monitoring systems, reduce.

Ground Control Management Plans

A Ground Control Management Plan (GCMP) is required to formalise the above systems and processes. By doing so it provides a systematic approach to the planning, design, management and review of all aspects of work associated with ground control at the mine. Each site must have its own GCMP because:

- each site is different, operates differently and has different ground conditions;
- regulations vary in different localities, jurisdiction and countries;
- different companies operate different mines, have different policies, accept different risks and have different management styles and procedures; and
- different sites use different suppliers and products for ground control and ground management.

There are however some components of a GCMP that are relevant to all sites:

- Identification of hazards and management of perceived risks;
- Assignment of responsibilities and authorities;
- Data collection;
- Stability assessments (engineering design);
- Specification of ground control systems and products;
- Monitoring programmes and feedback into design;
- QA / QC, competency and training;
- Documentation and communication.

Having a GCMP is useless if it is not going to be used and updated regularly. To facilitate these actions the structure used for the plan should be one agreed on by site personnel, company executives and regulators as being the most useful possible for the site.

The GCMP must be written in an easy to read style so that non-technical site personnel and executives don't find it too difficult to understand or cumbersome. Readability ensures that it becomes a source of practical information that personnel will refer to for all geotechnical related issues. It must however be sufficiently thorough to enable new technical personnel (engineers, managers, geologists, surveyors, technicians etc.) or geotechnical engineering consultant to read through the document and gain an understanding of all geotechnical issues at the site and the procedures used for managing them.

Rather than being all encompassing, the GCMP should reference "associated documents" and provides hyperlinks to the company's intranet site where these documents are stored. These "associated documents" discuss in detail the procedures, especially technical details, for each activity and system. If all of these details were included within the main body of the GCMP, it would be bulky and unreadable. By keeping the associated documents separate, each document can be updated as required without the necessity to rewrite sections of the GCMP. In addition, keeping each documents separate allows personnel to only have access to those documents relevant to their activities.

Conclusions

Managing risk associated with mining slopes is challenging. Particularly challenging are high slopes in large mines due to the general lack of understating or experience as to how these slopes perform and the inevitable hydro-mechanical coupling. Effective risk management requires addressing each of the components that contribute to the risk, the likelihood of instability, the likely of impact and the consequence of the event.

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