Evaluating the role of groundwater in the stability of slopes in an open pit coal mine with shallow dipping multi-seam stratigraphy.

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ABSTRACT

The subject of this study is a coal mine in South America, where a series of open pits have been developed as part of a consolidated mining operation. The pits are developed progressively along strike on a sequence of coal seams generally dipping at between 10° and 35°.

Slope failures have occurred, particularly on the footwall. Groundwater is seen as instrumental in triggering the failures and some form of depressurisation is seen as an essential contributor to achieving stable slopes.

This paper reviews the failure modes and describes field and analytical approaches to assess the groundwater conditions and design future dewatering and depressurisation systems. Solutions include deep wells, toe drains and depressurisation wicks.

INTRODUCTION

The subject of this study is a coal mine in South America, where a series of open pits have been developed as part of a consolidated mining operation.

A number of coal seams are mined, but slope failures have occurred, particularly on the footwall, resulting in the potential loss of reserves and a safety issue. The risks and costs of the failures or not achieving the planned footwall are also important in terms of the economic risk. This has been assessed but is not addressed here in any detail.

Groundwater is seen as instrumental in triggering the failures and some form of depressurisation is seen as an essential contributor to achieving stable slopes. Groundwater is important in terms of inflows, slope stability, trafficability and water chemistry, particularly with regard to acid mine drainage and development of pit lakes.

Slope dewatering and depressurisation have been implemented at the largest of the pits, but these measures have not always been successful in preventing failures. The investigations were directed at a more detailed assessment of the causes and locations of possible instability and to devise the required remedial works. This paper will only address the slope stability aspects of groundwater and the difficulty of deriving the correct conceptual model for piezometric pressure distribution.

The present depths of the two principle pits at the site range are approximately 90m and 250m. Planned completion depths are 170m and 300m respectively.

GENERAL GEOLOGY

The mines are located in an valley setting and comprise a thick sequence of alternating sandstone, siltstone, shale and coal of Triassic to mid Miocene age that filled a regional sedimentary basin in the northern part of South America. The sequence has been shortened and duplicated by a series of thrust faults with some extension due to normal faulting.

Strata dip to the southeast at an average of around 15° but locally this increases up to 35° with some local variation due to structural rolls. The total thickness of the coal sequence is about 900m and there is a series of economic coal seams within the operating and planned open pits. The worked seams comprise the thicker seams from around 0.5 to 3m thick with interspersed thin, uneconomic seams.

There are a number of deformation events in the mining area, of which the earliest are numerous low angled thrust faults with a dominantly north west to west tectonic transport direction, thought to be of Eocene age.

These major thrusts are cut by a set of steeply dipping thrusts correlated with the mid-Miocene Andean deformation. The most important is the Rancheria thrust that elevates the coal-bearing formation to economically accessible depths on the eastern side of the valley. This fault is on the western side of the mine lease area whilst a second thrust lies on the eastern side and carries pre-cretaceous basement onto the coal-bearing formation.

There are also a series of pervasive early Eocene thrusts that form zones up to 20m of intensely deformed shale or as barely detectable laminae of crushed coal. Where the thrusts are parallel to the bedding they may be undetectable.

These structures are very important in the stability of the slopes, due to the residual strength of the shear zones and the orientation, which is often sub-parallel to the bedding.

MINING METHOD

The pits are developed progressively along strike on a sequence of coal seams generally dipping at between 10º and 35º.

The present mining method comprises multi bench and multi seam working along strike and backfilling behind using a truck and shovel operation. Although earlier pits were conventionally mined with waste dumped external to the pit, the present method means that future faces could be exposed for a relatively short period. This is important in terms of stability and dewatering requirements.

FOOTWALL SLOPE STABILITY

Due to the geological structure, the mining footwall is on the dip slope of the coal seams and is therefore the most sensitive to failure. The strata dip into the hanging wall and generally, form stable slopes. The focus of slope stability investigations has been on the footwall.

Failures of various sizes have occurred in most of the pits and provide the opportunity for back analysis. However, the complexity of the detail still leaves significant uncertainties in the variables contributing to failure. It is this uncertainty, which requires different approaches to assessing dewatering requirements.

A risk assessment has shown that the Net Present Value (NPV) of the project is not significantly affected by a slightly conservative approach to stabilising the slopes. This is unusual in that steepening slopes in an open pit by a small amount usually has a significant benefit to the NPV. Calculations to optimise the NPV indicates that the loss of potential coal reserve by stepping back in to the footwall is made up by the benefits of fewer slope failures. Furthermore, the significant size and distribution of the coal reserves at the site and availability of working areas, means that estimates of final tonnage is largely insensitive to slope angle. These observations, require careful consideration when assessing the costs and benefits of dewatering.

BACK ANALYSIS OF SLOPE FAILURES

Small-scale failures occurred in the footwall slopes of two open cast pits during late 2001 and early 2002; only one of these pits was being mined at the time. Both back and forward analyses were carried out using the finite difference modelling programme FLAC-2D (Fast Lagrangian Analysis of Continua) as well as the limiting equilibrium model Slope 2D. The mechanism of failure of the slopes was examined as well as considering the influence of water pressure at the toe and whether buckling of the toe of the slope was important, as well as sliding failure along the clay layers.

The coal seam being mined on the footwall of the active pit at the time of the failure was Seam 7, and was around 60m high (Figure 1). Failure occurred by planar sliding along the clay layers at the top of Seam 6, but also along the top of Seam 5, in places. The layers between the failure surfaces buckled extensively, both at the toe and further up the failed ‘slab’. The main reason for the failure was the steepness of the dip of the strata (up to 35 degrees at the crest). The expected average dip of the footwall in the failed area had been 17 to 18 degrees.

The results showed that the stability of the pit footwall slope is extremely sensitive to dip angle and groundwater conditions. Residual strength parameters are required to be mobilised along the coal seam clay layers and disturbed rock mass strength parameters need to be mobilised in the interburden to allow failure to occur.

This was a shallow failure, only a few metres below the surface of the footwall and that as well as superficial stress relief, disturbance of the immediate footwall by blasting and/or excavation could have played a part in this failure, by allowing pathways for failure from the underlying clay layers. Thus the very low residual strength parameters modelled in the slope may not be applicable for larger scale failure.

Forward analysis was carried out to assist the mining development, as this failure was on the footwall of an operating slope. It was established that stability could be achieved by mining along Seam 4 from the top of the slope to 40m depth, mining along Seam 5 from the to 80m depth and then mining along Seam 7 for the remainder of the slope, thereby creating a series of ‘step ins’. This gave an overall footwall slope angle of approximately 15 degrees. Dewatering down to Seam 3, some 30m below Seam 5, was considered essential for stability.

Additional analysis was done to develop a better understanding of the potential mechanisms and slope geometrical configurations that contributed to the failure of the pit footwalls. Although the principle failure mechanism was planar failure on bedding horizons, it was important to establish whether the method of failure initiation was by shear through the toe of the interburden slab or by buckling.

The analyses were done for a range of rock mass and discontinuity strength parameters as with the previous analyses. Groundwater was modelled by applying a pore pressure to the base of the clay layer equivalent to a 35m head of water. Different ubiquitous joint orientations were used to model the effect of rock mass weakening at the toe.

Failure was generally characterised by creep type movement along the clay layer. The magnitude of slope instability was less than that observed in the actual failure, though it is considered that this was an effect of processing time on such a large model.

The results showed that failure was by a combination of mechanisms with the failure models showing a component of toe uplift, being indicative of buckling, and a component of shear failure through the base of the slope.

The results of the analyses also showed that:

- For the smaller failures, pore pressure must exist below the slab with the full hydrostatic pressure head being mobilised behind the toe of the slope,
- Residual strength parameters must be mobilised along the sliding surface,
- The mass strength of the interburden must approach ‘disturbed’ strength conditions or must contain low angle discontinuities at the toe of the slope to initiate failure,
- For the larger failures, excess pore pressures were not necessarily required.

It is recognised that the model had a number of limitations in terms of the analytical method, the failure mechanism and the strength parameters used. Static ground water pressures were used in the analysis. The potentially complex interaction between slope movement and groundwater pressure accumulation and dissipation could be simulated using the dynamic groundwater flow module available in FLAC.

The situational slope stability analysis has produced design charts of slope height versus dip angle for a range of water conditions, representing the majority of the structural conditions present (Figure 2). Water pressure is important and the slopes need to be depressurised and drained to achieve stability. The situational analysis enabled design charts to be developed, which allow stabilisation strategies to be analysed using the assessment of the various components of straight slabs.

In general the parameters selected for the main stability analysis fitted well with the larger failures: however, the smaller scale (shallow) slab failures only occurred with extreme water conditions. The shallow slab failures are probably triggered by some specific, small to medium scale event, such as blasting, putting in drain holes, disturbing the toe during mining, small scale irregularities in the surface of the failure plane, small scale structures, local variations in bedding strength, etc., which makes them more difficult to predict, or back analyse. These results also emphasize that the applicability of the rules, derived from the analysis based on the currently selected parameters, is limited to general, large-scale situations.

APPROACH TO DEWATERING

It was clear from the stability analysis and previous experience that the focus of attention had to be on depressurising the slopes. Inflows to the pit were not a major issue. Trafficability on haul roads is an issue due to the clay, but the difficulties are primarily caused by rainfall.

Consideration was given to pre-dewatering using deep wells behind the slope crest, or forms of drilled toe drain in the pit slopes. The former would benefit stability in that the slopes can be dewatered before excavation. Toe drains can only be installed into the excavated slope.

There is a layer of alluvium overlying the coal measures and in the particular case investigated at the actively mined pit previously described, there was a river passing within 100m of the pit crest that flooded during winter. A protective berm has been constructed along the pit crest to prevent direct flooding, but the lower coal seams subcrop the alluvium and in the stream bed in places. This could provide a source of direct recharge to the coal seams.

The stability assessment showed that it was important to understand the detailed distribution of piezometric pressures within various horizons in the slopes and to assess the best means of relieving those pressures. Ingress of rainwater to the weaker horizons will also help to reduce material strength, so management of surface water runoff will also be important.

SITE INVESTIGATION

A programme of test dewatering wells and monitoring holes was developed. All boreholes were sited and designed so that, where possible, they could form part of the ongoing dewatering and monitoring requirements.

The two sites analysed were selected to represent pits:

- Where no active mining was taking place; and,
- Where there was active mining.

DRILLING

HORIZONTAL DRAINS

At the actively mined pit, three holes were drilled at about 5 degrees upwards to depths ranging between 113 and 149m. The final target seam was originally Seam 3. However, detailed scrutiny of the geological logs from the drainage holes and from the nearby vertical monitoring hole indicated that the drainage holes probably terminated just beyond Seam 2 (Figure 3).

Ten metres of casing were grouted in to the start of the hole to seal against leakage. The hole was also fitted with a 50mm plastic sleeve to the full depth of the hole to keep it open. The sleeve consisted of alternating 3m lengths of slotted and blank casing down the hole.

Each hole was completed with a ‘T’ piece and valves such that a pressure gauge could be set, but the water could also be drained as required. The valves were closed to minimise drainage until the tests were carried out. However, seepage of water occurred around the exterior of the grout seal and along horizontal bedding surfaces adjacent to each hole. The footwall was very broken and fragile in appearance and this may account for why the seal length was inadequate to stem flow.

At the inactive pit, three sub horizontal holes were drilled 60 and 70m apart and were driven about 100m in to the footwall with the terminus of each hole targeted at the vertical monitoring holes being constructed on the upper slopes.

In this case, the first 20m of each hole was cased and grouted and this appeared adequate for the prevention of the uncontrolled seepages seen at the active pit.

While drilling the final hole, a failure occurred in the footwall above the drainage holes and all of the sub-horizontal boreholes were lost. The vertical monitoring holes were also damaged.

The drains in the footwall of the active pit, which had been closed in January, were opened to prevent the build up of pressure that could cause a similar failure there.

VERTICAL BOREHOLES

The vertical pumping wells were developed in two stages. The first stage involved the drilling of a pilot hole to assess the presence of sufficient water for a pumping test. Holes where yields were deemed adequate on the basis of simple hydraulic tests (slug tests, airlift) were then reamed to the full size (Stage 2), whilst those that failed were either abandoned, or converted to monitoring wells.

The pumping wells were then cased and screened using steel to a minimum diameter of 200mm. The screen sections were positioned adjacent to water inflow zones identified as a result of the geological and geophysical logging. At the active pit, the indicated yields were low but the selected hole was completed as a pumping well with 8'' casing installed to 116m.

The vertical observation holes were constructed in a similar fashion at all of the sites, but with different target depths according to the depth of the target coal seams. The completion depths for the piezometers included the final target coal seam as well as at least one other at about 50 to 75m above the lower piezometer. Each piezometer was set in a gravel section with 3m of bentonite above and below the gravel.

The depth of the observation holes ranged between 62m and 125m and intersected the Seams 1, 2 and 3 (as illustrated in Figure 3). The topmost piezometer in several of these wells was installed in the alluvial aquifer within the near surface water table.

Observation holes were drilled behind the face with the toe drains at inactive pit, but they were abandoned after the footwall failure on the 28th January 2002.

PUMPING AND DRAINAGE TESTS

VERTICAL PUMPING WELLS

The hydrogeological testing of the vertical pumping well consisted of a step test comprising only two steps due to the low yield. Discharge during the first step was 0.18l/s and lasted for 1 hour. The rate was increased to 0.32l/s during the second step, which lasted for a further 1.67 hours.

The constant rate discharge test was set conservatively low at 0.2l/s but was increased to 0.35l/s after 32 hours to maximise the drawdown and increase the radius of influence of the test. The test was continued for a total of 3 days followed by recovery.

The borehole responses are summarised in Table 1.

Table 1. Borehole Responses to the Pumping Test in the active pit.

<table>
<tr><th>Well Type</th><th>Well No.</th><th>Distance to Source well (m)</th><th>Seam No.</th><th>Maximum Drawdown (m)</th></tr>
<tr><td>Source Well</td><td>Q1</td><td>N/A</td><td>N/A</td><td>49.92</td></tr>
<tr><td rowspan="5">Observation Well</td><td>Ob1</td><td>54</td><td>Seam 2</td><td>2.3</td></tr>
<tr><td>Ob2</td><td>22</td><td>Seam 2</td><td>2.94</td></tr>
<tr><td rowspan="3">Ob3</td><td rowspan="3">236</td><td>Seam 1</td><td>0.06</td></tr>
<tr><td>Seam 2</td><td>0.25</td></tr>
<tr><td>Seam 3</td><td>0.17</td></tr>
</table>

The steady state water level immediately prior to the test was 31.13m bGL. The final drawdown in the pumping well after 73 hours was 49.92m.

The final drawdown in the observation wells was generally small indicating poor connectivity and low permeability. The drawdown response in Ob3, positioned 236m away from the pumping well was particularly attenuated.

Recovery was monitored for 99 hours and the water level in the pumping well recovered by 96%.

The test data were analysed using various confined aquifer models to achieve the best possible match. The results indicated a transmissivity for a 94m saturated interval was 9.30E-06 m^2/sec and a storativity in the region of 1.53E-05; the overall permeability for the pumping well is therefore quite low (1E-07 m/sec).

However, applying an overall permeability for the coal measures is misleading, since the great bulk of flow appears to occur at the margins of individual coal seams and rarely elsewhere. This much was evident from the geophysical logs, which tended to show the most marked temperature inflections when passing the coal seams. In addition, analysis of the observation zones indicated that permeability of individual coal seams ranged between 2.0E-06 and 4.0E-06 m/s.

HORIZONTAL GRAVITY DRAINAGE WELLS

Testing of the horizontal wells in the active pit consisted initially of draining individual wells whilst observing the pressure drops in the adjoining drains and the decline in water levels in a vertical well positioned behind the crest of the footwall.

In practice, the pressure drops in the neighbouring drains could not be monitored because leakage from around the casing had already substantially reduced water levels in the formation. Consequently, there was much greater dependence on measuring the change in flux or discharge from the open drain, the leakage rates and the water level changes in the vertical monitoring holes.

A final test was done with all three horizontal drains open. The holes were still draining two months later although yields had significantly reduced.

The following general observations can be made about the behaviour of the groundwater regime in the footwall area of the active pit:

1. The water levels in Seams 1 and 3 are 5 to 6m lower than in Seam 2. A lower piezometric surface is expected of Seam 1; this is evident when groundwater equipotentials are plotted and compared. However, lower heads in Seam 3 are likely to have arisen as a direct consequence of the dewatering by the three horizontal drains that intersect this seam;
2. Throughout this period the only coal seam to show relatively rapid responses to dewatering was Seam 2. An exception to this general observation appears to have occurred during the first dewatering trial with all drains open, when water levels in Seams 2 and 3 rose quite suddenly (15^{th} January 2002). This anomaly almost certainly corresponds to a blasting event that occurred in the active pit in the middle of the day. This suggests a possible sensitivity of the slopes to blasting suddenly raising water pressures;
3. Seam 2 not only underwent the most substantial decline in water level over the course of the monitoring period, but the range and standard deviation of measured water levels were greatest for this seam. These observations would suggest that Seam 2 is likely to be the best connected and most conductive of the seams on the the active pit footwall;
4. In support of this observation, it was noted that the water levels in the vertical monitoring well behind the crest of the footwall (Figure 3) all appeared to react to pumping from the pumping hole, but that this response was fastest in Seam 2, followed by Seam 3, and lastly, Seam 1;
5. Certain of the responses in Seams 1 and 2 are anomalous, or difficult to explain in the context of activities on site. In particular, water levels appeared to rise by between 1m and 1.5m in the period from 5th and 18th February when all three horizontal wells were open and draining freely. The water level in Seam 1 rose throughout this episode. However, although the water level rise was more pronounced in Seam 3, it declined again after 10th February. It should be noted that the horizontal drains do not penetrate Seam 1, whereas they do intersect Seams 2 and 3. During the initial part of this period the water levels in Seams 1 and 3 behaved in a diametrically opposite fashion to the water level in Seam 2;
6. Seepage was observed coming from the toe of the footwall mid-way between two of the drains at the outset of the dewatering trials. However, it was noted that this seepage had ceased by the end of February; an outcome clearly brought on by the opening of all three drainage holes at the end of January;

7. Steady state flow rates from the three drainage holes ranged from 0.46 l/sec and 0.15 l/sec on the lower bench, down to 0.02 l/sec on the upper bench. It was clear from these flows that the permeability of the coal is generally quite low , though local heterogeneities mean that zones yielding more water exist, particularly in proximity to structural discontinuities.

BACK ANALYSIS OF THE FOOTWALL FAILURE IN THE INACTIVE COAL PIT

The failure of the footwall in the inactive coal pit occurred during the preparation for the dewatering trial in that slope. Three horizontal drains had been drilled into the footwall intersecting seams located between 40 and 75m behind the face. These holes had been grouted a distance of 20m at the collar and control valves inserted, which were then closed. It appears that the main (target) seam in this area had high permeability and it caused a rapid build up of water pressure behind the footwall slope, which caused the slope to fail in the interburden just below the footwall surface. The slope height was 50m and had an overall angle of approximately 20 degrees, steepening up to 25 degrees at the toe.

The runs were carried out on models of the pre-failure slope with gradually decreasing strengths to cause the slope to fail. Failure occurred when the rock mass strength was reduced to 'disturbed' values (c=100kPa, φ=18 degrees) and the ubiquitous joint strength had been reduced to 'residual' values (c=0, φ=12 degrees) with full water pressures. Additional runs with a lower slope height were carried out to refine the strengths. As a result the cohesion of the ubiquitous joints was increased 15kPa.

An additional element to this failure was the presence of two small faults close to the toe of the slope, which had displacements of less than 1m, but nevertheless could have played an important role in the reduction of shear strength at the toe, acting as a failure pathway as the stresses built up from the increase in water pressure behind the face.

CONCEPTUAL HYDROGEOLOGICAL MODEL FOR THE FOOTWALL.

The responses seen in the three monitoring zones of the vertical well behind the footwall indicate that the three coal seams (Seams 1, 2 and 3) are acting independently (Figure 3). There does not appear to be any widespread connection across the intervening layers of mudstone, silt and sandstone, except where possibly the formation is cross-cut by sub-vertical faults. However, the evidence for this is only circumstantial and mainly based on the markedly higher flow rates that were measured in one of the gravity drains located on the lower bench.

In general, the conductivity of the coal measures is uniformly low with the highest permeabilities in the Seam 2 (2.0E-06 – 4.0E-06 m/s). Exceptions are likely to exist where the formation is faulted, but surface mapping and evidence from testing indicate that faulting is not a widespread phenomenon in the NW corner of the active pit.

It is difficult to attribute the anomalous behaviour observed during the dewatering tests to any known activity on site. One conceivable explanation is progressive slope movement due to increased pore pressure at the toe of the footwall. It is likely that the more permeable seams i.e. Seam 2, will tend to dissipate pore pressure more readily than those, like Seams 1 and 3, where conductivity is generally lower. Hence, the declining water level in Seam 2 and the rising water levels in the other two seams throughout the first half of February.

The focal point of any slope failure may well be in those seams where pore pressure increases are difficult to dissipate on account of their relatively low permeability i.e. Seams 1 and 3. Note also that the pressure rise was more pronounced in Seam 3 than in Seam 1 and that the water level in Seam 3 declined prior to the onset of trial pumping in Borehole Q1 (17th and 18th February). This may be because the shallow 3 Seam was where the pressure build-up was most acute and where it was consequently most readily alleviated through partial failure in the slope.

Failure, or movement, appears to be initiated once a certain pore pressure and rock mass strength threshold is exceeded for a particular slope geometry. The pore pressure is usually relieved as the failure occurs, but in time can resume its climb as new strains are developed in the footwall. Contributory factors like blasting could increase the incidence of failure by momentarily lifting the pressure above the critical threshold. Another potential factor in promoting failure is likely to be recharge, which could be due to the nearby river, or be seasonally driven. The slope stability analysis shows that overpressure is an important trigger in slope failure where pore pressures are higher than hydrostatic due possibly to local infiltration through tension cracks or joints during rainfall.

DEPRESSURISATION

It is clear that slope stability in the footwall is sensitive to rock mass strength, slope geometry, structure, and hydrostatic water pressure due to groundwater and water overpressure due to local recharge. In some areas pre-dewatering using deep wells will not achieve the depressurisation required and toe drains will be required. The difficulty is to ensure that sufficient drainage can be achieved as slopes are developed.

In order to develop an approach to production depressurisation and dewatering, it was necessary to carry out some preliminary modelling, using the available test information, to evaluate various possible combinations of drains.

This study used the programme SEEP/W to investigate the effect of dewatering the pit slope on the distribution of pore water pressure using different combinations of wick drain and gravity drain. SEEP/W is a commercially available 2-D finite element software product that models both saturated and unsaturated flow within porous materials such as soil and rock.

There were two stages in the development of models for this study. The first of these concerned the design of models to calibrate field derived physical input parameters like hydraulic conductivity and storage. The second stage involved the use of these values in a series of generic models intended to mimic conditions in the footwall generated as part of the SRK pit slope stability study.

The slope stability analysis identified five key conditions that required analysis (Figure 4), namely:

- WC0: Fully saturated slope for analysis of stepped slopes;
- WC1: Untreated slope with anisotropic permeability;
- WC2: Untreated slope with isotropic permeability;
- WC3: Partially treated slope using inclined wick drains; and
- WC4: Pre-dewatered slope.

The stability analysis shows that the improvement of the Factor of Safety (FOS) in some cases is fairly small and may not warrant the cost of the depressurisation because other stabilisation methods will need to be applied. These conditions should be evaluated in more detail once calibrated ground water pressure and drainage measurements are available.

The effect of a gravity drain on deeper portions of the formation was also investigated in this study. The model design was based on the arrangement of drains on the active pit footwall during the SRK site investigation in 2002.

Two forms of depressurisation were considered appropriate:

- Wick drains inserted on the slopes perpendicular to the strata to depths of around 30m for immediate relief of shallow pressures;
- Gravity toe drains installed sub-horizontally to around 100m depth to relieve deeper pressures.

Figure 5 shows wick drain spacing down slope against time. The criterion for the three model scenarios illustrated on the chart are summarised in Table 2 below. They have been derived through interpolation of the curve for WC3 in Figure 2.

Table 2. Design Criteria For Dewatering Scenarios

Scenario No.	Slope Height (m)	Slope angle (degrees)	Slab T (m)
1	35	27	10.75
2	77	19.5	15
3	156	15	35

Slab thickness, slope height and slope angle do not affect drainage as much as drain spacing and time. The stability analysis shows that the FOS decreases rapidly for increasing slab thickness for the first 5 to 15m. However, increasing slab thickness above 15m does not alter the FOS significantly.

In the analysis, we have assumed that a 75% reduction of the head can be achieved and that this will represent a sufficiently low ru factor to achieve the required FOS. In order to achieve a FOS of 1.25, along the complete section of footwall about 20 days will be required for a drain spacing of about 50m. For closer spacing the time will be less. In practical mining terms, provided the drains are installed as mining proceeds, there will be sufficient time for drainage to occur before the slopes are fully developed.

In the vertical direction, Figure 5 shows that a drain spacing of 70m down the slope, will require 160 days of drainage and a spacing of 50m will require 70 days. Again, this will be readily achievable based on the rate of deepening of the mine.

It should be noted that the analysis has focussed on three main elements of dewatering to achieve stability. These are:

- Release of the overpressure which triggers many of the failures;
- Drainage of the stored water to reduce the hydrostatic head; and,
- Deeper dewatering to drain the head to below any potential failure plane.

The wick drains are designed to penetrate to the depth of the critical slab thickness and immediately relieve any over pressure, to do preliminary drainage of groundwater in storage and to assist in preventing future build up of pressure after rainfall. They will also help to remove any gas pressures that have been observed in places.

To achieve deeper drainage, gravity toe drains are proposed.

Modelling based on the drainage tests with the toe drains demonstrated that toe drains should be effective in draining the deeper portions of the formation. Over 90% of the initial head in the coal seams above the drain is dissipated within two to three weeks.

This rapid drainage to a steady state level is due to the intersection of the higher permeability coal seams. Full drainage of the slope can only be achieved by developing the interference effects of adjacent drains similar to the analysis for the wick drains.

FUTURE DEVELOPMENT REQUIREMENTS

The stability analysis has shown that reduction of groundwater pressure and overpressure is critical in improving stability of the footwall slopes. However, in some cases of steep dips or potential larger, deeper seated failure, the improvement by drainage will be limited and other means of stabilisation will also be required.

The focus of depressurisation is the use of wick drains to reduce over pressure and longer toe drains are used to promote deeper and longer term drainage and reduction of hydrostatic head.

Vertical wells for pre-dewatering the footwall outside the pit perimeter, will have limited benefit due to low permeability and lack of hydraulic connectivity with the critical horizons. Higher yields are obtained where geological structures enhance the local permeability and vertical wells may be of local benefit.

Toe drains will promote a more widespread and deeper zone of depressurisation and dewatering to lower the phreatic surface. These should be up to 100m deep, spaced at 100m intervals along the footwall and located at 100m intervals down slope. They will be interspersed with the wick drain spacing. However, because the objective of the toe drains is to drain the deeper coal seams, they should be cased and grouted to a depth of about 20m at present depths of the pit, to prevent water from the deeper aquifers recharging the shallow seams or potential failure surfaces. As the pit deepens, the depth of grouting should increase to about 30m because the potential failure surface is deeper into the slope. At this stage additional wick drains should be installed, to ensure that the potential shallow failure surface is also depressurised.

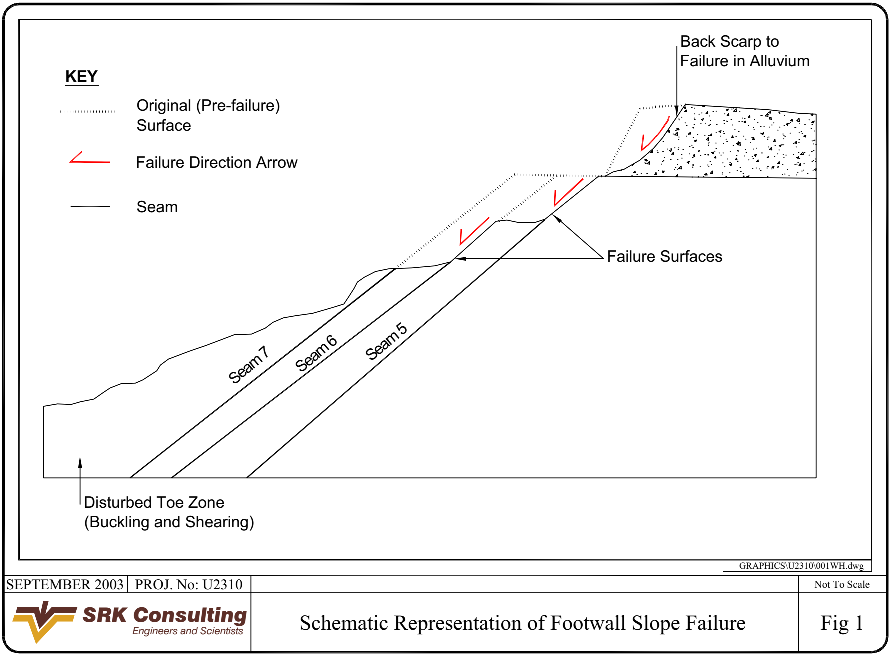
From the modelling, slab thickness, slope height and slope angle do not affect drainage as much as drain spacing and time. Slab thickness does significantly influence the time for drainage and the thickness has a significant influence on the stability.

Wick drains are designed to penetrate to the depth of the critical slab thickness and immediately relieve any over pressure, to do preliminary drainage of groundwater in storage and to assist in preventing future build up of pressure after rainfall. They will also help to remove any gas pressures that have been observed in places.

Gravity toe drains are required to dissipate pressure for potential deeper seated failures.

Future investigation will require the precise installation of pressure transducers in carefully selected horizons and to monitor the effects of blasting, mining and active depressurisation. This information can then be built in to the slope stability risk matrix, which will be used as an ongoing slope design tool. In the past, the piezometer installation has not been able to assess or monitor the piezometric pressures that influence stability of the relatively thin slabs,

Once the detailed piezometric model is understood, a hazard plan will be developed using GIS, from which slope drainage requirements and drain design, can be planned ahead of mining.



Schematic Representation of Footwall Slope Failure Fig 1

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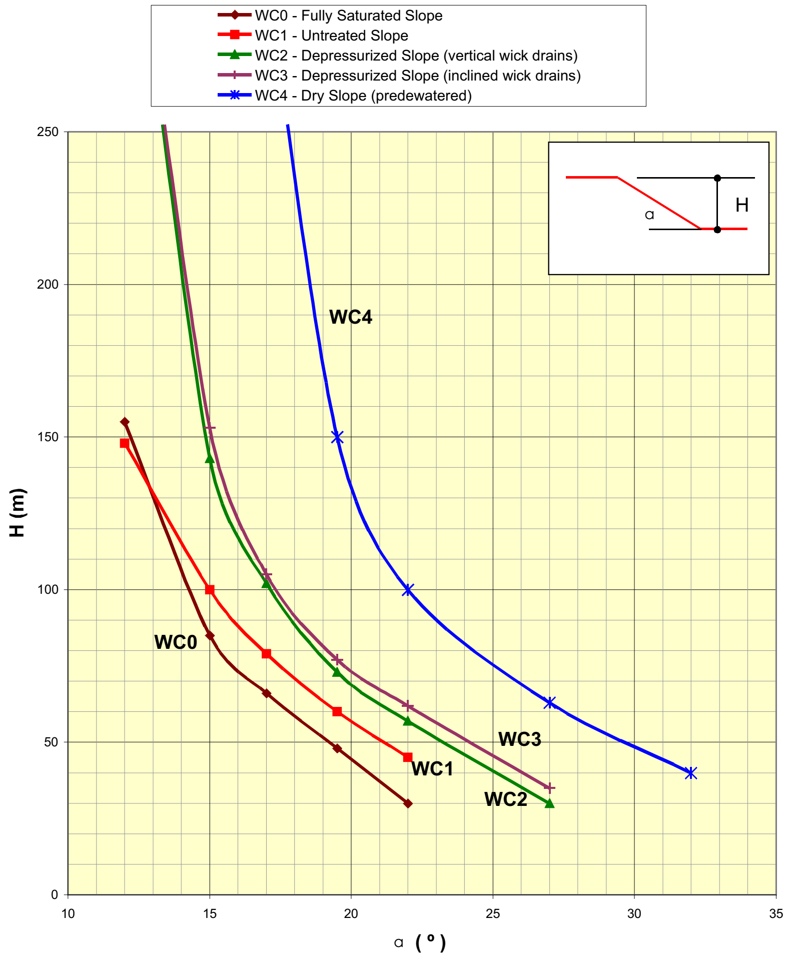

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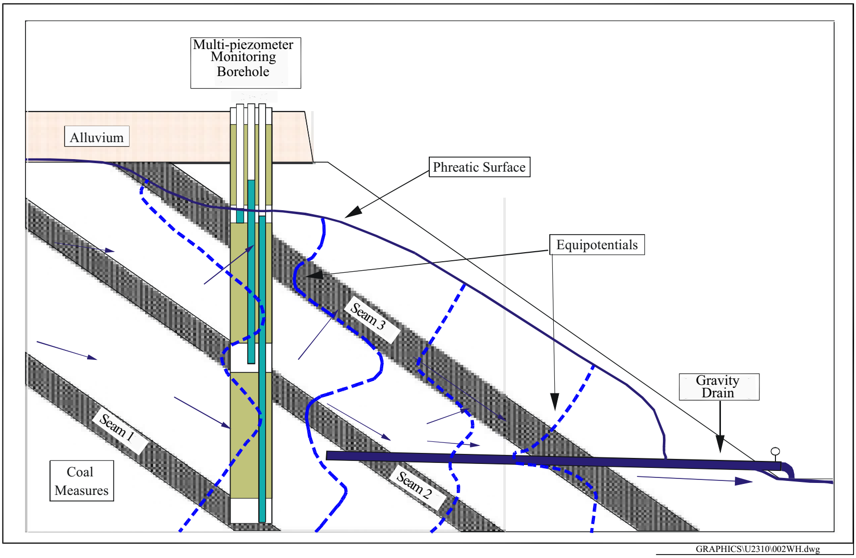
SRK Consulting Engineers and Scientists		SLOPE ANGLE (α) vs MAX. ACC. SLOPE HEIGTH (H) STRAIGHT SLOPES - FOS = 1.25	Fig 2



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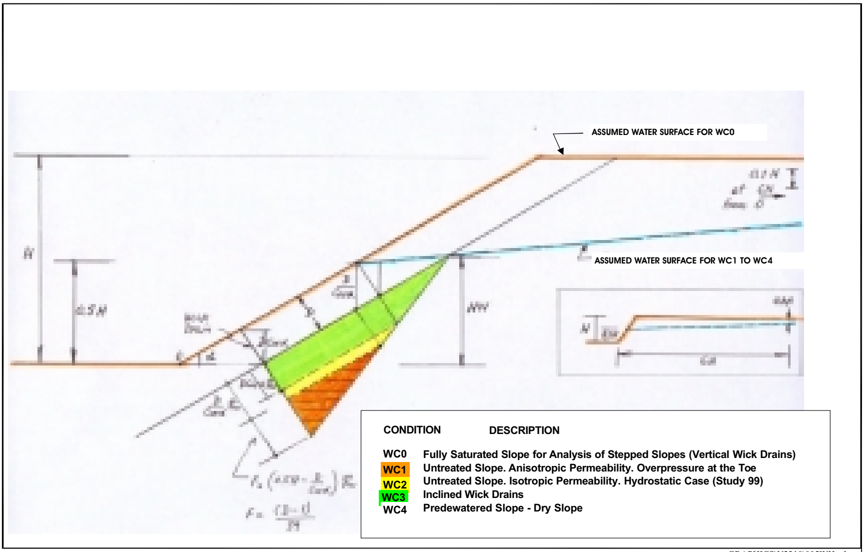
Schematic Cross-Section of Pit Footwall Showing Boreholes, Water Table and Distribution of Equipotentials

Fig 3



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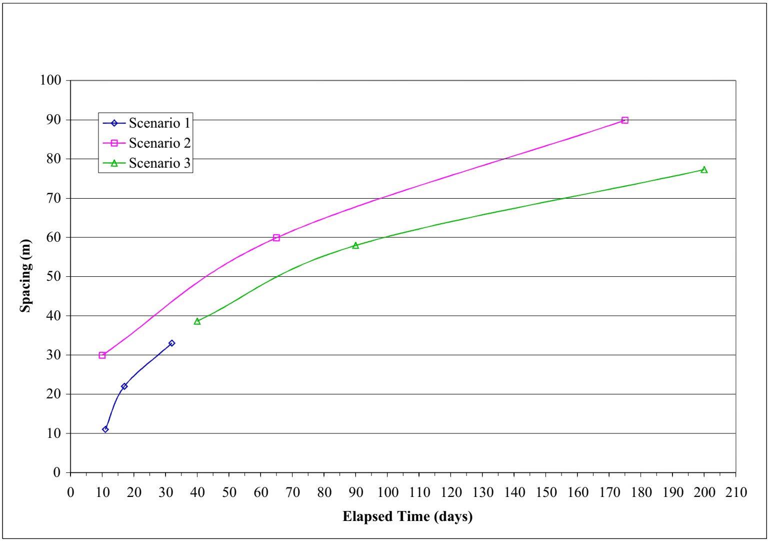
SRK Consulting Engineers and Scientists		Proposed Water Conditions For Footwall Slopes SRK Study 2002 In Terms Of Water Pressure Distribution	Fig 4



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Wick Drain Spacing Down Slope

Fig 5