

Lowwall Stabilisation at Riverside Coal Mine

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SUMMARY A combination of geological factors and very weak rocks created adverse mining conditions in one area of Riverside coal mine. A geotechnical study was undertaken to assess ground conditions and recommend stabilisation works which would decrease the risk of instability. The design method, practical implementation and performance for the case is presented.

1 INTRODUCTION

The Riverside Mine is an open strip coal mine operated by BHP-Utah Coal Ltd for Thiess Dampier Mitsui Coal Pty Ltd and is located at the northern end of Queensland's Bowen Basin near the town of Moranbah. The mine is predominantly a single seam operation extracting metallurgical and energy coal from the Goonyella Lower Seam which is part of the Permian age Moranbah coal measures. The Riverside mining lease is on the western edge of the Goonyella mining lease which experienced major pit instability problems when coal extraction began in the early 1970's. Several geotechnical research programmes were completed at Goonyella and that work forms the basis for strip coal mine design and operating practice in Queensland today (e.g. Boyd 1983, Richards et.al. 1981, Philp et.al. 1981 and Dunbavan 1983).

Stripping operations at Riverside Mine rely on large walking draglines digging explosively fragmented overburden. The extended bench method is used with typical overburden depths from 25m to 50m, pit widths of 70m and a coal seam thickness ranging from 5m to 10m. The mine is divided into nine mining areas by pit access ramps for coal haulage. The general layout of the mine is shown in Figure 1. This paper relates to mining conditions in Area E which plays an important role in mine productivity because of its proximity to the coal processing plant and high coal quality.

2 HISTORY OF INSTABILITY

2.1 Geological Features of Area E

Figure 2 shows a cross-section of the coal measures exposed in Area E during mining. The 40m of overburden consists of 15m of variably weathered Permian argillaceous rocks unconformably overlain by Tertiary (major) and Quaternary (minor) materials. The Tertiary sequence is interspersed with deep paleochannels filled with extremely weathered smectite-rich basalt flows and quartzose sand lenses. The sediments in both Tertiary and Permian strata are lithic units weakly cemented with calcite. Some sideritic bands may be found as partings in the coal seam.

Weathering and the inherent weakness of the rocks means that joints are only a minor feature of the geological structure. In common with most coal mines in the Bowen Basin, low angle thrust faulting has a major influence on the stability of

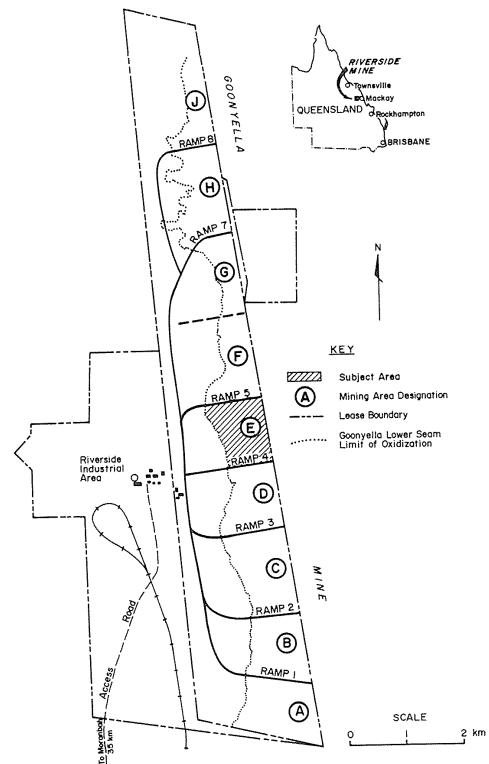


Figure 1 Riverside Mine, Queensland

pit slopes in intact material and lowwall spoil dumps when the thrust fault is located in the coal seam floor (e.g. O'Regan et.al. 1981).

The other major structural features are the discontinuities formed along the Permian-Tertiary unconformity and along the perimeter of the Tertiary palaeochannels. The general dip of the coal measures is 3° to 5° to the east but may be higher (up to 10°) in the vicinity of folds associated with thrust faulting.

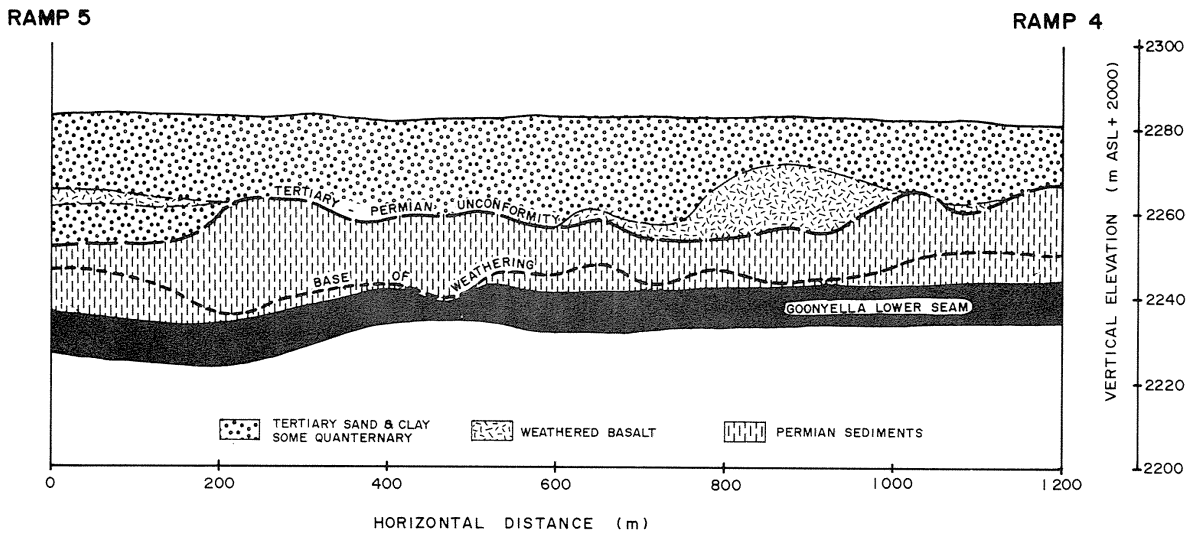


Figure 2 Geological Cross-Section of Coal Measures, Area E

2.2 Instability Related to Geological Features

The instability of boxcut lowwalls is usually due to the presence of a deep Tertiary palaeochannel filled with weathered basalt. The products of weathering include montmorillonite which contributes to the very low shear resistance between the base of the channel and the surrounding Tertiary sediments. When boxcuts were excavated, instability often developed when the base of a Tertiary channel dipped at more than 15° into the pit. Instability was enhanced by disruption of the overburden caused by the boxcut overburden blast (i.e no free face was present), the presence of saturated materials and perched aquifers in the palaeochannels, and surcharge loading of the lowwall slope by the excavated overburden.

In Area E, the initial boxcut stripping was not continuous between Ramps 4 and 5 due to variations in the oxidation limit of the coal. Strip E4 in the southern half and Strip E5 involved boxcut lowwalls (see Figure 3). In both cases, deep Tertiary channels filled with weathered basalt were intersected and instability developed. A minor length of boxcut slope remained to be excavated in the middle of Strip E6. Ground conditions on both sides of this block indicated a very high likelihood of another basalt filled channel being intersected with consequential lowwall instability. The development of a slope failure in the middle region would compound the difficult mining conditions which existed as a result of the previous lowwall instability.

2.3 Instability Related to Mining Activity

It may be argued that all instability is related to mining activity. However, this section addresses the situation in which the major causes of lowwall instability could be attributed to or controlled by mining activity. A significant improvement in lowwall spoil dump stability may be achieved by thorough pit floor preparation before spoil from the next strip is placed (Philp et.al. 1981). Pit floor preparation may become impossible in some circumstances due to physical or operational limitations. With the use of the extended bridge stripping method, such problems

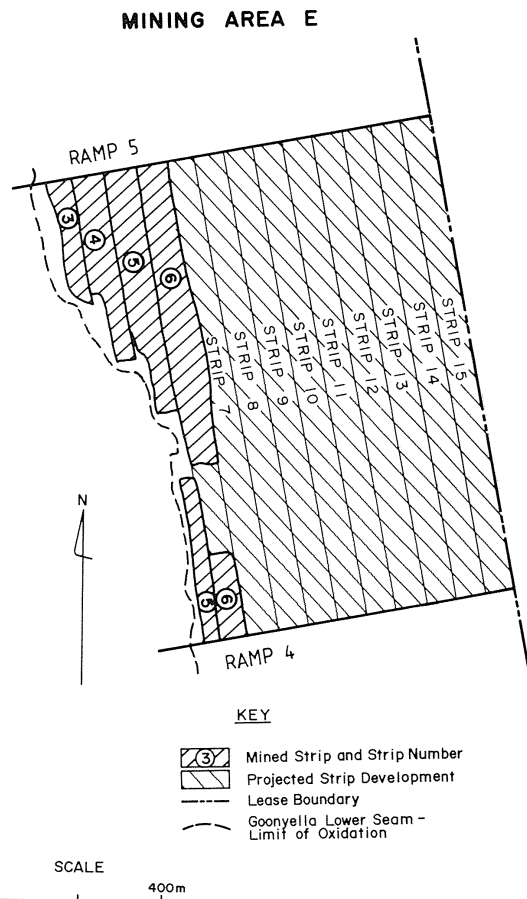


Figure 3 Mining Sequence, Area E

may occur adjacent to pit access ramps and where previous lowwall instability was experienced.

In Area E, the failure of boxcut lowwall slopes associated with weathered basalt channels created extremely difficult mining conditions. Problems were compounded by the barrier of spoil dumps on the lowwall which restricted the capacity for placement of materials involved in a failure away from the pit. Lowwall failures usually result in the worst materials (smectite rich basaltic clays) being deposited in the most unfavourable location (the base of the next lowwall spoil dump). Thus, development of instability in the remaining boxcut slope in the middle of Area E for Strip E6 not only had an impact on immediate mining activity, but also affected the potential for instability in subsequent strips.

3 GEOTECHNICAL INVESTIGATIONS

3.1 Site Investigations

Because of constraints imposed by mining schedules, insufficient time remained for the completion of a site investigation specifically related to the lowwall instability in Area E. Relevant geotechnical properties had to be estimated from existing geotechnical data and results of back analyses. Before the mine commenced operation, an initial geotechnical investigation covering the entire lease was undertaken by Coffey and Partners (confidential report, 1981). One of the ten cored boreholes was located at the northern end of Area E and some classification and shear strength data were obtained for Tertiary and Permian overburden materials.

After mining operations commenced, instability developed in the intact overburden of both highwall and lowwall slopes of the boxcut in Area E. Detailed inspection of the failed areas revealed the main cause of instability was the presence of clay-rich material of low shear resistance in association with Tertiary basalt filled channels dipping into the recently excavated pit (BHP Engineering, confidential report, 1986).

The total amount of geotechnical data available was minimal and supplementary data was required for the design of a slope stabilisation system. Thus, the data had to be inferred from the existing site investigation results, general data published in journals and conference proceedings, and data obtained from back analysis of lowwall failures in Area E.

3.2 Back Analysis of Lowwall Failures

The lowwall in Strip E6 consisted of approximately 1,000m strike length constructed entirely from excavated overburden (a strip cut dump) with a 100m section in the middle of the strip consisting of spoil dumped on intact overburden (a boxcut dump). The different combination of materials meant that two distinct cases existed for back analysis.

The spoil dump at the southern end of Area E was unstable in Strip E5 and provided a case for the back analysis of a strip cut dump. As slope displacements increased, the slip in the boxcut dump from Strip E4 was reactivated. Figure 4 shows a simplified cross-section of the strip cut dump slip mechanism. Measurements of the dip angle on the main slip scarp were made and the locations of other slip planes were noted. A two-wedge analysis based on the principle of virtual work (Dunbavan, 1980) was used for

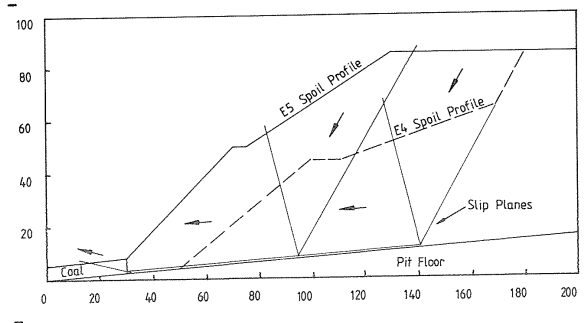


Figure 4 Strip Cut Lowwall Profile

calculations. The shear resistance along the basal slip plane giving a factor of safety of unity was found to be practically independent of realistic variations in the shear strength values for bulk spoil material. The combinations of cohesion and friction angle for the strip cut dump slip and the reactivated boxcut dump slip are shown in Figure 5. The common shear strength values were 35kPa cohesion and 11° friction angle. The values were similar to those obtained for smectite-rich materials and reported in earlier site investigations.

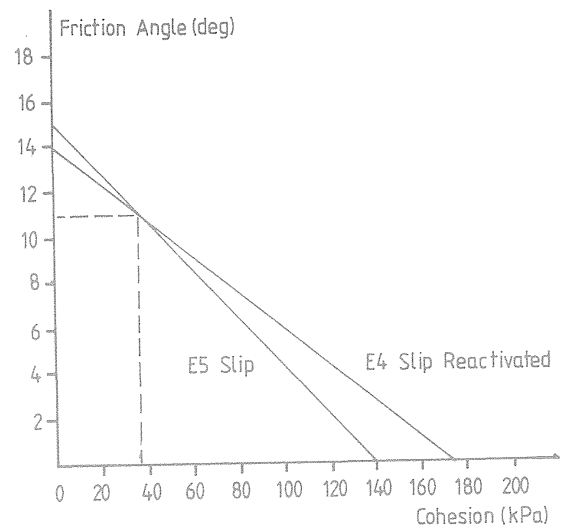


Figure 5 Results from Strip Cut Lowwall Back Analysis

For the boxcut dump, a similar procedure was adopted with a multi-wedge slip mechanism and virtual work calculations. Initially, a field example was not available for measurement of slip plane orientations and locations. Results from the strip cut dump analysis were used as a starting point for the boxcut analysis. By chance, industrial action stopped operations when a boxcut slip was being excavated in Area E. A cross-section of the slip was exposed and measurements were taken. Figure 6 shows the position of critical slip planes estimated from back analysis together with the measured slip plane angles.

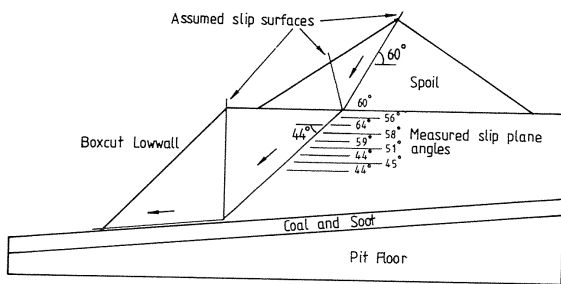


Figure 6 Boxcut Lowwall Profile

TABLE I
REPRESENTATIVE GEOTECHNICAL PROPERTIES

Material	Unit Wt. Constant kN/m ³	Friction Angle		Cohesion	
		Ave. deg.	StdDev deg.	Ave. kPa	StdDev kPa
Slip Planes in Intact Overburden	25	14	3	100	25
Spoil in Dump	20	28	3	15	10
Basal Slip Plane in Spoil Dump	-	14	3	0	0

3.3 Representative Geotechnical Properties

The combination of limited direct site investigation data, careful field observations and appropriate back analysis of unstable lowwall slopes resulted in the adoption of the representative geotechnical property values listed in Table I. Assuming that the Mohr-Coulomb strength criterion was relevant and that the parameters (friction angle and cohesion) were independent and normally distributed, the necessary data for a probabilistic analysis of slope instability was available. Values for standard deviation were estimated using the coefficients of variation published in Lee et.al. (1983).

4 DESIGN OF STABILISATION SYSTEMS

4.1 Accepting a Risk of Instability

Geotechnical engineers are always facing the fact that soils and rocks are inherently variable materials. The incorporation of this characteristic into designs is compounded by the very low number of samples usually available for the reliable determination of the distribution functions for geotechnical properties. The broad base of observed lowwall instability in Queensland's strip coal mines and geometrical uniformity of lowwall slopes prompted the decision to limit the "uncontrolled" variables to the material shear strength parameters. That is, for a given lowwall profile, the critical location of the slip surfaces was determined for average shear strength values. To determine the risk of instability, shear strength values were varied for a constant set of critical slip surfaces.

The factor of safety, F, for the analysis based on virtual work was defined as:

$$F = \frac{\text{work done against shear resistance}}{\text{potential energy released in slip}}$$

The risk of instability was defined as the probability of $F < 1.0$. Monte Carlo simulation is the usual method for determining the probability distribution of F in slope stability analyses (e.g. Rosenbaum and Jarvis, 1985). Because time constraints did not permit the development of a computer program to combine a two-wedge (or multi-wedge) stability analysis with the Monte Carlo data generating procedure, one of the authors (Dunbavan) made the assumption that the control limit theorem could be applied (for details, see Walpole and Myers, 1972). Simply, by using average shear strength values, the average F value could be calculated, and so on. The symmetry of the distribution of F was applied as a simple check on the assumption involving the control limit theorem.

For no changes to slope profile and minimal removal of existing lowwall slip material, the average factor of safety and associated risk of instability were:

	Average F	Risk of Instability
Boxcut Lowwall	0.97	65%
Strip Cut Lowwall	0.87	99.5%

The high risks of instability were not acceptable for continued mining in Area E. A subjective value made by mine planning staff for an acceptable risk of instability was 30%. One way in which this risk may be interpreted is the percentage of lowwall which could fail, i.e. 300m slip in 1,000m strip length for 30% risk. Thus, some form of lowwall stabilisation work would be required for an acceptable risk of instability to be achieved.

4.2 Practical Reinforcement Options

4.2.1 Boxcut Lowwall

Tensioned cable bolts appear to be the most successful form of artificial reinforcement for slopes. Shear pins (untensioned dowels) have been used successfully in some cases (see Rosengren, 1986). The extremely weak rocks of Riverside Mine would not be suitable for the installation of tensioned cables. Rosengren described the conditions for economical use of shear pins as:

- failure occurs along a well defined surface,
- use of scrap steel for reinforcing members, and
- ready accessibility to the installation area.

Because these conditions were met for the boxcut lowwall section in the middle of Area E, the use of shear pins consisting of scrap rail line was adopted for design purposes. This innovative approach to ground support was first introduced by BHP Engineering at the surface iron ore mines at Mount Whaleback and Koolan Island, Western Australia.

4.2.2 Strip Cut Lowwall

The construction of the strip cut lowwall by dragline using the extended bridge excavation method meant that no system of reinforcement could

be installed within the slope without unacceptable disruption to dragline operations. The impracticality of placing the excavated overburden anywhere other than in the previous pit meant that only minor changes could be made to lowwall slope geometry. Thus, the practical option for improving lowwall stability was to increase the resistance to down dip movement at the toe of the slope.

After the overburden is removed, the intact coal seam creates a natural buttress against spoil dump slip. Options were considered for the replacement of this coal buttress as soon as practicable after the coal was extracted. The most effective scheme was back filling the full width of the pit with compacted spoil material to a design depth.

4.3 Recommended Designs

4.3.1 Boxcut Lowwall

The installation of a regular pattern of vertical shear pins was recommended. Limitations on locally available materials and machinery resulted in the choice for each pin of two lengths of 47kg/m scrap rail line placed in a 310mm diameter hole back filled with 15MPa concrete. The drilling pattern was offset with holes at 10m centres. The spoil surcharge on the lowwall was to be placed no closer than 15m from the excavation. The predicted risk of instability was 30% with an average factor of safety of 1.05.

4.3.2 Strip Cut Lowwall

The placement of a 3m (minimum) compacted fill buttress over the entire pit area after coal extraction would reduce the risk of instability to 30% with an average factor of safety of 1.10.

The fill material should consist of the fresh Permian overburden (low degradability) occupying the first 5 to 10m above the coal seam. Construction of the buttress would be in thick layers (up to 1m) from rear dump trucks. The same trucks would be continually driven over the fill to achieve compaction. If possible the buttress adjacent to the toe of the lowwall should be raised to 6m, tapering to 3m at 20m from the toe. The cross-pit buttress should be kept as close as practicable to the coal face so that the opportunity for instability to develop in the unsupported area would be minimized.

5 PREDICTION VERSUS PERFORMANCE

5.1 Realities of Reinforcement Installation

A total of 41 large diameter holes were drilled in an offset pattern at 10m centres for placement of shear pins. Two holes at the northern and of the area collapsed due to the slumping of unconsolidated wet Tertiary quartzose sand. Similar sandy strata were indicated in many holes, but no others collapsed. Supply problems occurred with the scrap rail lines. Within the time available, only 31.5kg/m rail was obtained. To compensate for the lower shear resistance of the lighter rail, the installation of three rails per hole was attempted. Once again, practice departed from theory to allow only two rails per hole increasing the risk of instability to 40% (average F of 1.02).

Although the support measures for the strip cut lowwall were less sophisticated than those for the boxcut lowwall, the recommended design could not

be achieved. Approximately 200m south from Ramp 5 (see Figure 3), a prominent thrust faulted structure in the form of a 10m high fold in the floor use was encountered in the pit.

The magnitude of the thrust fold was not evident in the previous strip. Associated with the fold was an increase in depth of weathering in the overburden and a lithological change where the Permian grey siltstone was replaced by brown mudstone. These geological changes eliminated the source of competent back fill material for the cross-pit buttress. Mixed Tertiary and Permian overburden spoil material with a minimum of basaltic clay was used instead. The increased risk of instability for the strip cut lowwall could not be quantified because it related to the higher degradability of the spoil should the pit become wet. Saturation of the spoil would not only reduce its effectiveness to carry horizontal loads from the current lowwall, but would also create a low shear strength base for the next lowwall slope.

5.2 Performance Monitoring

The performance of a slope is usually measured by the development of instability. In the mining context, slope instability may develop but not impare the mining operations. Thus, the performance of a slope at a mine is related to its impact on mining operations. For the lowwall in area E, the purposes of the stabilisation work were to restrain the spoil dump so that in the subsequent strip the volume available for spoil placement would not be significantly reduced and to maintain good pit conditions to ensure future lowwall stability.

The design of the stabilisation work was based on an accepted risk of instability. Should instability develop, the effectiveness of the overall slope design would be gauged by the proportion of spoil volume lost compared with the overall volume available should no instability occur. Changes in spoil volume may be estimated from surveys of the slope profile.

5.3 Development of Instability

Two major geological features combined resulting in the development of instability along a 300m strike length of the pit. One feature was the thrust fold with its axis diagonally across the pit in a northwest-southeast direction. Pit floor dips in the vicinity of the fold were steeper than the regional dip and probably contributed to the lowwall instability.

Associated with the same thrust faulting and folding was deeper weathering and deeper Tertiary strata. The overburden from which the lowwall spoil material originated consisted predominantly of smectite rich basaltic clays interbedded with un cemented quartzose sands. The sands formed perched aquifers so that the clays were usually very moist. The shear resistance of the bulk of the lowwall material would have been lower than in other locations and would have more readily formed slip planes with low shear resistance.

The development of substantial slip movements was identified during regular geotechnical pit inspections which serve to alert mine staff to geotechnical hazards potentially affecting operations. A line of survey points was established along the lowwall at bench level so that slope movements could be monitored

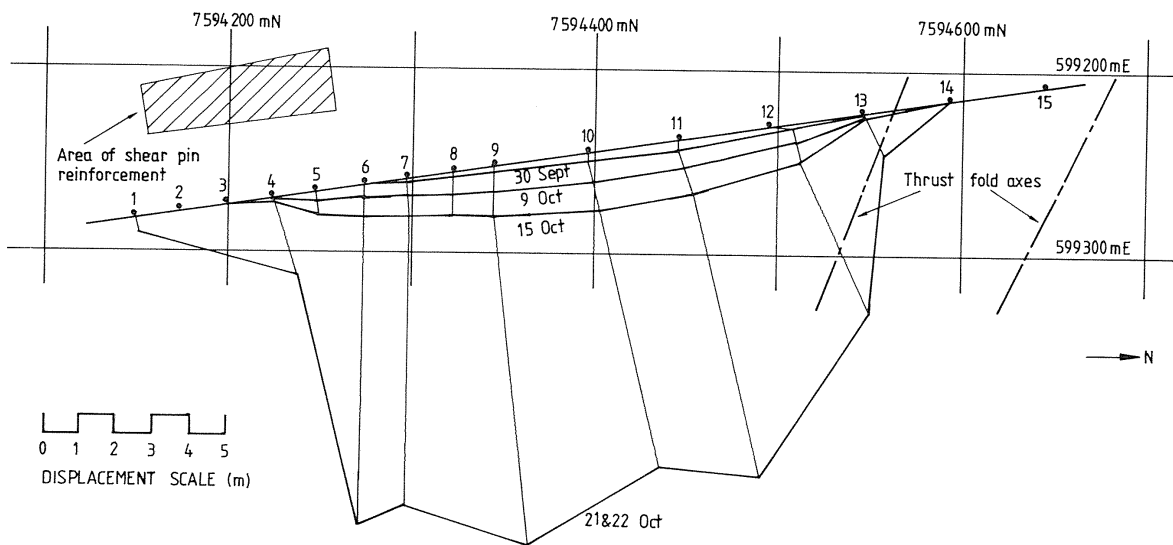


Figure 7 Displacement Measurements, Strip E6 Lowwall

(Figure 7). The interpretation of the data was in the form of variation of displacement rate with time. To reduce the influence of the systematic errors inherent in the simple measurement system used, displacement rate was averaged over two successive sets of readings. This procedure tended to mask rapid changes in displacement rate, but this disadvantage was outweighed by the improved data reduction at low levels of displacement rate. The cumulative slope displacements over a period of one month are shown on Figure 7 and a plot of the displacement rate versus time for selected points is shown in Figure 8.

The latter plot is on a logarithmic natural scale

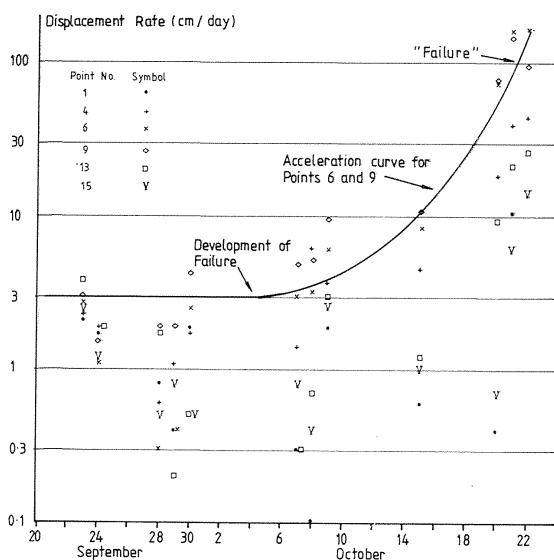


Figure 8 Displacement Rate Showing Acceleration to Failure

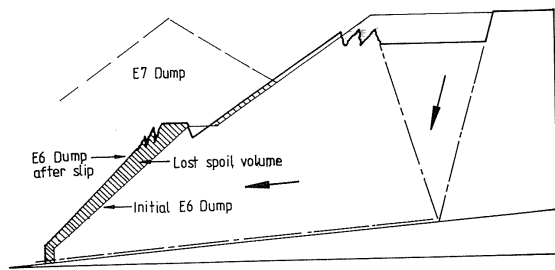
because the creep of slopes in coal mines usually accelerates toward slip (Cruden and Masoumzadeh, 1987). By comparing points which remained practically stationary (numbers 1 and 15) with those which underwent significant movement (numbers 6 and 9) in Figure 8, the threshold rate signifying the onset of slip was approximately 3cm/day. Significant slope displacements were associated with a rate of 100cm/day. The value for onset of slip compares very well with that suggested by Cruden and Masoumzadeh (1987) of 0.02 mm/min (2.9 cm/day).

The slope displacements shown in Figure 7 indicate that the lowwall "failed" and therefore the performance of the stabilisation works was unsatisfactory. Figure 9 may be used as a guide for the evaluation of slope performance. Figure 9 (a) shows the actual slip plane locations with the stabilisation work completed. The approximate spoil volume reduction per metre strike length of the slope was 340m³. Figure 9 (b) shows the assumed slip plane locations if no stabilisation work was completed. The spoil volume reduction would have been 540m³ per metre strike length. Although some spoil volume was lost with the stabilisation work completed, the effectiveness of the work is clear.

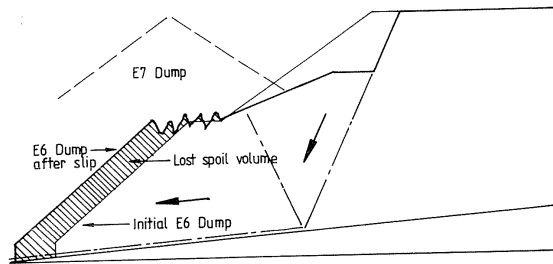
6 CONCLUSIONS

The use of a risk analysis approach for slope design in open strip coal mines has significant advantages. Different design options may be compared on the basis of the same risk of instability. When a data base suitable for rigorous probabilistic analysis is not available, restriction of the number of variables and subjective choice of property values may allow useful comparative analysis provided the assumptions on which the risk analysis was based are validated.

The performance of a slope at a mine site is very different from a similar slope in a civil engineering project. In strip coal mines, the accepted risk of instability is high because slope failure is not synonymous with unsatisfactory



a) Spoil slip with stabilisation work



b) Spoil slip without stabilisation work

Figure 9 Performance Evaluation by Comparing Spoil Volume Losses

design performance. The same design procedures may be used for slopes in civil and mining engineering, but the criteria on which design performance is evaluated are dissimilar.

7 ACKNOWLEDGEMENTS

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