Slope Stability 2013 – P.M. Dight (ed)
© 2013 Australian Centre for Geomechanics, Perth, ISBN 978-0-9870937-5-2
doi:10.36487/ACG_rep/1308_61_Zargarbashi

Flow failure in coal stockpiles – how to reduce risk

P. Davies *Golder Associates Pty Ltd, Australia*

S. Zargarbashi *Golder Associates Pty Ltd, Australia*

L. McQueen *Golder Associates Pty Ltd, Australia*

Abstract

This paper presents findings from a geotechnical study undertaken to reduce the risk of coal stockpile instability, following a flow failure observed in New South Wales. To characterise the stockpile behaviour, key input parameters such as permeability, shear strength and compressibility have been assessed for different in situ densities. Excess pore water pressure developed within the stockpile has been estimated by finite element analyses, using 2D PLAXIS, for different stockpile filling rates and target heights and taking into account the stockpile loading/unloading operations adopted in the mine. PLAXIS pore pressures were then incorporated into a series of limit equilibrium stability analyses conducted for three stockpile heights at different filling rates to study variation of Factor of Safety (FS) with the rate of stockpile filling rate, stockpile height and groundwater level. Conclusions from this modelling case study show that loading rates as high as 5 m/day may be adopted to raise the stockpile to a maximum height of about 10 m. Placement rates faster than 3 m/day result in an elevated risk of instability for higher stockpiles. Placement rates faster than 1 m/day increase the risk of flowslides for stockpiles of 15 m or higher. The analysis also shows that loader reclamation from the toe shortly after stockpile filling escalates the risk of instability for stockpiles above 10 m placed faster than 1 m/day. The results were used to update a mine stockpile risk management plan to include consideration of stockpile filling rates, coal placement/reclamation method and stockpile height. A monitoring strategy was also developed to assist with risk management.

1 Introduction

Washed coal is commonly stockpiled and the stability of these fill emplacements can deteriorate due to formation of erosion gullies, shallow slip, and massive or deep seated failures (Eckersley, 2000). There are a number of failure case histories describing the latter, large scale failures in coal and coal wash reject stockpiles (e.g. Dawson et al., 1998; Eckersley, 1985), the most infamous being the Aberfan disaster which killed 144 people in 1966 (Siddle et al., 1996). As in the case of Aberfan, susceptible slip debris can deteriorate into a 'flowslides' in which materials may rapidly flow out to a distance of several times of the stockpile height (Hunter and Fell, 2002).

In many cases initial failure is triggered by some disturbance to the fill material, such as a rotational displacement which then degrades into a debris flow (Dawson et al., 1998). High brittleness in material behaviour can result in dramatic strength reductions as shearing occurs leading to a collapse mechanism involving fluidisation. When the flowslide trigger mechanism is not triggered by earthquake (or dynamic effects), the above phenomenon is also known as 'static liquefaction', in which the trigger shearing event can involve inundation (i.e. rainfall, snow melt, irrigation), toe disturbance (or toe instability) and/or excess pressure generated by rapid filling.

There is a growing body of work which characterises the mechanisms involved in these types of failure; this paper summarises some of the key published data and presents findings from a geotechnical study undertaken to reduce the risk of coal stockpile instability following a flow failure observed in NSW ('the subject site'). To characterise the stockpile behaviour at the subject site, key input parameters such as permeability, shear strength and compressibility have been assessed for different in situ densities. Excess pore water pressure developed within the stockpile has been estimated by finite element analyses, using 2D PLAXIS, for different stockpile filling rates and target heights. PLAXIS pore pressures were then used in a series of Slope/W limit equilibrium stability analyses for three stockpile heights with different filling rates and groundwater conditions to study how stability was affected.

2 Stockpile characteristics

The coal stockpiles at the subject colliery site in NSW, Australia, are divided into three main coal types comprising: coarse coal (CC), undersize coal (USC); and fine, hard coking coal (HCC). The stockpiles cover an area about 250 by 150 m wide and overlap at their margins. The height of the stockpile is typically about 15 m with a maximum batter height of 5 m and minimum bench width of 2 m.

The stockpiles are formed by pushing the coal away from the tipper cones below a conveyor feeder by a dozer and front end loader. The coal is dozed over the edge of the existing stockpile towards a rail line where it is loaded into train wagons by a CAT980 front end loader. When toe materials are periodically excavated, they briefly form near-vertical faces up to about 4.5 m height before slumping.

The stockpile formed using the above methodology exhibits variable in situ densities. The coal on the face of the stockpile is often loose and stands near its angle of repose (between 38° and 42°). The standing time of the coal and compaction effects from overburden or dozers, also influences permeability and the movement of groundwater through the stockpile.

Periodically, oversteep stockpile batters fail by slumping. This is generally not a problem with the USC and CC products, as they are relatively free-draining therefore re-grade in a controlled way to their natural angle of repose, with acceptable risk to operators and other personnel. However, when the fine HCC coal stockpile fails it does not always exhibit the surficial re-grading mechanism of the coarse coal and the observed flowslides can be large and rapid, presenting an increased risk to personnel and infrastructure.

A program of probing and sampling in the lower batter identified a layer of older coal which was finer and more dense than more recently placed coal higher in the stockpile. These basal materials are inferred to be compacted and indurated as fines migrate downward from overlying newer stockpile material, which is more frequently filled and reclaimed. Over time, the permeability of coal remaining at the base of the stockpile is reduced and toe drainage is impeded. This results in perched water developing above the basal layer following infiltration. Pore pressures can also develop in low permeability horizons in response to rapid loading/filling.

Placement techniques also suggest that coal which is deposited directly under stackers is likely to be more compacted (higher strength) than end dumped material, which makes up most of the upper stockpile. As discussed in the following sections of this paper, this layering of loose, contractant and brittle material over denser and less permeable material, which is responsive to stress changes, is pre-disposed to flow-slide type failures.

3 Approach and methodology

Following the coal stockpile failure at the subject site, the authors undertook careful site observations and a review of relevant published literature so as to characterise the failure and plan the material testing and analytical approach. By developing a robust modelling approach in which the geomechanical behaviour of the stockpile was adequately understood, the relationship between stability and triggering factors was identified and provided a basis for risk management.

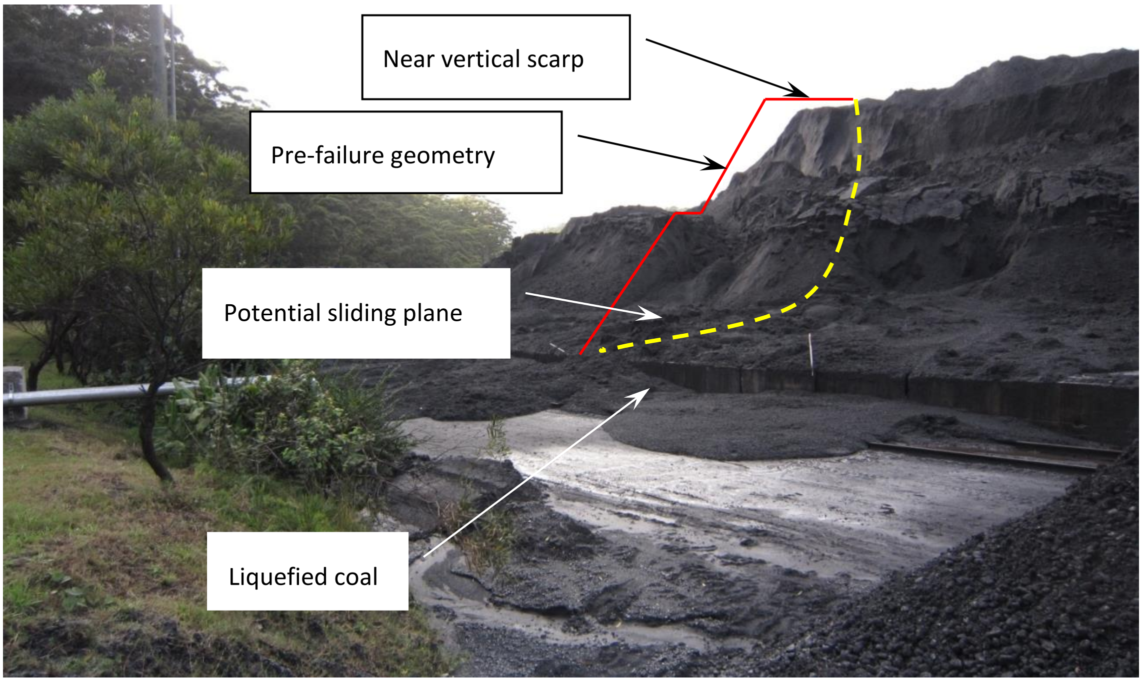


Figure 1 Flowslide occurred within a fine coal stockpile

3.1 Observations, literature review and conceptual model

Key post-failure site observations included:

- Liquefied coal debris around the base of the failed stockpile and residual debris slopes much flatter than those generally associated with incipient failures in granular material (Figure 1).
- Relatively undisturbed 'blocks' of intact spoil conveyed downslope on top of the flowable layer.
- An inferred deep-seated failure plane and near vertical scarps in upper sections of post-failure slope.

These observations suggested that the slip was a rapid, rotational-translational type mechanism, involving a deep layer of liquefied material capable of transporting overlying material. The displaced upper stockpile materials formed blocks of relatively intact material, bounded by shear failure surfaces.

The subject site failure had commonalities with similar failures reported by Hunter and Fell (2001), who described rapid flow slides in 67 coal waste spoil piles and coal stockpiles from three geographic regions including South Wales, British Columbia and Hay Point, Australia. A review of relevant factors from these case histories as they apply to the subject site is provided below in Table 1.

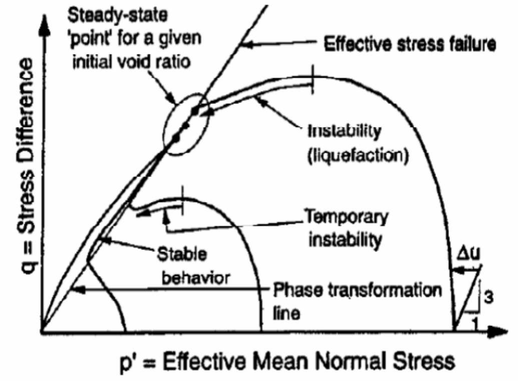
Of the key factors listed above, groundwater, fill rate and fill type are factors common to flowslide behaviour at many case history sites and the subject site.

In addition to these factors, the authors' note that the immediate pre-failure actions which triggered instability were inferred to involve rapid fill placement combined with toe cutting during load-out operations. The inferred conceptual model entails a rapid deterioration of slope stability into a flowslide due to static liquefaction within the loose coal stockpile immediately overlying the basal compacted coal. Liquefaction is associated with the development of a collapse mechanism involving rapid development of excess pore pressures within the loose stockpile materials. This is determined by factors including material grading and density, contractant behaviour on shearing and pore pressures induced by placing fill rapidly.

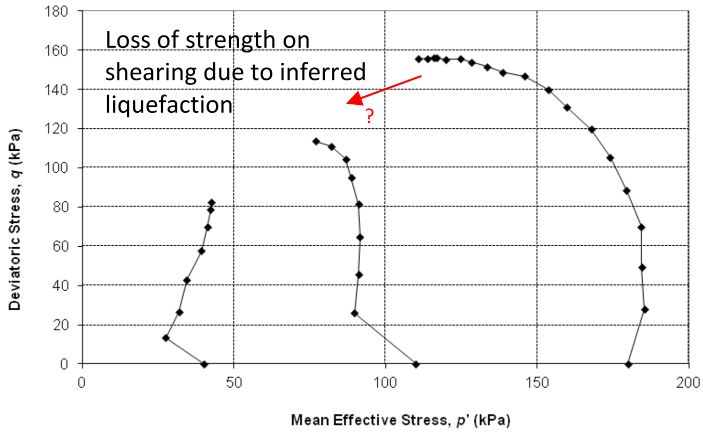
Table 1 Key factors causing flow slides in coal stockpiles, after Hunter and Fell (2001)

Key Factor Reported at Other Coal Stockpile Failures	Key Factor at Subject Site?	Comment
1. Particle size distribution and material density (or placement technique)	✓	Grading characteristics of subject site coal materials are within liquefiable range (Refer Figure 3). Regarding density, void ratios greater than 0.3 are known to be susceptible increasing undrained brittleness on shearing and potential for static liquefaction. The estimated void ratio of loose coal at the subject site is estimated to be in the order of 0.4 (Refer Figure 4).
2. Rainfall or snow/ice melt	✗	At Hay Point, flow sliding was associated with significantly wet periods, whereas heavy rainfall was not a significant factor at the subject site. However, observed poor drainage is likely to contribute to instability (see item 5, below).
3. Foundation slope angle for slides	✗	Hunter and (Fell 2001) reported that flow slides at 46 coal stockpiles in British Columbia occurred where the average foundation slope was about 25°. The foundation material at subject site was near level.
4. Active dumping in progress leading up to the failure	✓	Up to 3 m of additional coal was estimated to have been placed near the crest of the subject stockpile within 48 hours of failure.
5. Springs, streams or high pore pressures in the foundation	✓	Many coal tips in South Wales (Hunter and Fell, 2001) contained groundwater springs in foundation materials. Anecdotal evidence at the subject site suggests that significant ephemeral groundwater flows occurred near less permeable horizons at the toe, analogous to spring conditions.
6. Construction of the tip over pre-existing earth flows	✗	Such cases were restricted to published examples in South Wales and are not applicable to the subject site.

Figure 2(a) below shows the typical contractive behaviour in undrained shear, as excess pore water pressure increases during shearing; the increase in pore water pressure reduces effective stress and in-turn shear strength, leading to liquefaction and 'rapid' flowsliding. Comparable results from tests on the subject site materials are shown in Figure 2(b).



(a)



(b)

Figure 2 (a) Schematic diagram showing stress paths of samples at the same void ratio and different initial confining pressures (Yamamuro and Lade, 1998, as reproduced in Hunter and Fell, 2001); and (b) Effective stress path during multi stage undrained shear of subject site HCC Sample

Figure 2(b) shows that strength loss occurs in the stage where the initial confining pressure is 180 kPa, which is comparable with the pre-failure overburden pressure under 15 to 18 m of coal overburden. Hunter and Fell (2001) report that relatively high confining pressures can suppress the tendency for dilation, thereby increasing the effects of contractive behaviour and likelihood of liquefaction. Liquefaction can also be triggered in saturated drained samples where pore pressures developed due to contraction in the sample cannot be dissipated rapidly enough in drained conditions.

The grain size distribution of the dump materials are shown in Figure 3 along with other flow liquefaction cases (including Aberfan). As shown on this plot, more than 30% of subject site HCC materials are finer than 0.5 mm particle size. We note that as indicated in ACARP research report (Eckersley, 2000), fine coals with more than 12% finer than 0.5 mm are susceptible to static liquefaction and flow slides. Both subject site and ACARP materials have a low dry density of no more than 0.95 t/m^3 which is also indicative of high voids ratio and hence contractant behaviour susceptible to flow liquefaction, as seen around the base of the stockpile. The ACARP report also notes that loader reclamation from the toe of a stockpile of such uncompacted coal with significant fines (>%12) could initiate a rapidly moving slip.

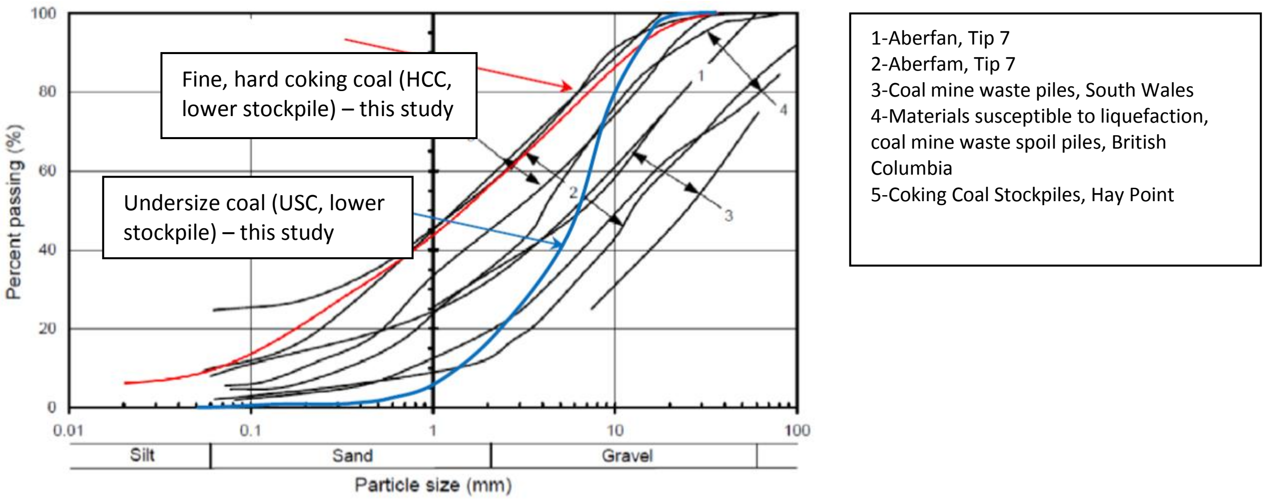


Figure 3 Grading of coal stockpile materials – case histories and subject site (reproduced from Hunter and Fell, 2001)

Dawson et al. (1998) describe liquefaction flow slides in coal mine waste dumps. These occurred in dumps higher than 70–100 m, loose dumped at the angle of repose, onto hillsides with slopes less than 18°. Most failed during operation but failures occurred up to 13 months after placement ceased. Failures usually occurred in response to heavy rain or snowmelt. Laboratory testing showed a tendency for collapse behaviour at void ratios greater than 0.3 with a peak strength (collapse surface) of 28°, compared to the angle of repose of 38° corresponding to an apparent Factor of Safety of 1.0 at the initiation of failure.

Perhaps the most useful and definitive indicator of the potential for static liquefaction and hence rapid flowsliding is the state parameter (ψ), as described by Been and Jefferies (1985). The state parameter is a convenient way of expressing the soil's relative stress and density state relative to the steady state/critical state line (Figure 4). The authors acknowledge the difficulty in estimating the in situ void ratio of coal to the precision required to make this a reliable exercise.

In later work, Jefferies and Been (2006) report that a good rule of thumb is that a soil needs to be denser than about ψ = -0.08 (as shown on Figure 4) to effectively mitigate liquefaction potential in clean sands. For finer materials such as thickened tailings, this value can be increased to ψ = -0.05. More dense materials where ψ < -0.05 tend to dilate on shearing, thereby reducing the risk of flowslides and catastrophic failures. For the coal stockpile materials in this study, samples were recompacted to representative field densities in the range 85% +/- 5% standard maximum dry density ratio before laboratory shearing. Figure 4 shows that the corresponding state parameter was in the range 0< ψ < +0.5 with 'average' compaction conditions expected in the range ψ +0.1 to +0.15.

These site specific values compare with spectacular examples of static liquefaction flowslides reported by Davies et al. (2002) including the tailings failure at the Merriespruit Harmony Mine, South Africa, 1994. The literature states that a large portion of these tailings had ψ > +0.1, and that a flowslide was initiated by toe erosion. The failure involved 600,000 m^3 of material and killed 17 people. Also described is static liquefaction at Sullivan Mine, Canada in 1991. This large scale failure was very brittle, causing sand boils and ejection of water from standpipes. Post failure investigations in the area of the liquefaction failure indicated very contractant in situ states in the range of ψ = +0.1 to +0.12.

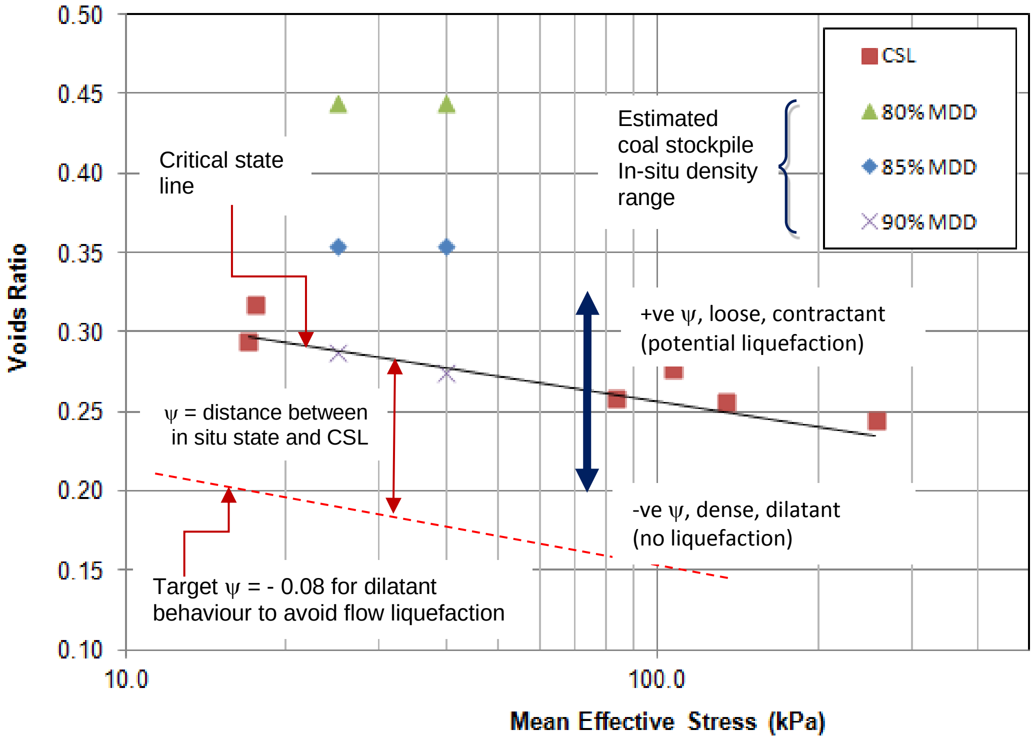


Figure 4 Void ratio-effective confining stress state diagram

Unless fills are mechanically compacted, subjected to high downward seepage gradients, and/or exposed to evaporative drying and desiccation (such as occurs in tailings), then stockpiles or waste fills rarely achieve dilatant states (e.g. ψ <0). Hence, such loose fills often exhibit contractant behaviour upon shear loading.

The stress path involved in the shear loading is also important as instability can be triggered by pore pressure change, vertical loading or loss of horizontal confinement. At rest (K_0) conditions create a static shear bias condition that is typically close to the collapse surface. Initiating a spontaneous liquefaction event in a slope typically requires only small additional shear stress beyond the at-rest soil condition. Other authors (Ulrich and Fourie, 2003) report that only 1% axial strain was required to trigger strength loss in laboratory samples and that this could occur through wetting up alone.

In summary, static liquefaction of the type observed at the subject site and other case histories may occur in fill material susceptible to spontaneous collapse due to one or more of the following circumstances:

- Increased pore pressures induced by rising piezometric surface.
- Excessive rate of loading due to rapid filling.
- Removal of toe support such as an overtopping event Davies et al. (2002), lateral erosion or toe unloading.
- Foundation movements rapid enough to create undrained loading in fill (such as occurred in a high waste rock fill at Carstenzweide (after Hustrulid et al., 2000).

Any or all of these actions can cause shear stresses within the body of the fill in excess of the collapse surface, leading to ‘spontaneous’ liquefaction. Given this tendency for brittle strength loss on shearing in slopes susceptible to static liquefaction, it is worth considering the cumulative probability of these potential triggering mechanisms. In other words, a granular fill slope with an apparently satisfactory Factor of Safety against deep seated failure based on conventional drained strength properties provides no such recognition of static liquefaction potential and could provide a false sense of security.

3.2 Analytical approach

Having reviewed the site observations and pre and post-failure survey data, additional bulk coal samples were collected from the subject stockpile for laboratory testing (refer following section). Once the subject material properties and problem geometry were assessed, a series of analyses were developed to model the onset of rapid flow failure. The following calculation approach was adopted:

- Two dimensional finite element analyses of the pre-failure stockpile were undertaken using PLAXIS software, adopting material parameters obtained from site specific and published laboratory test values. The modelled construction steps were estimated from site records, photographs and anecdotal evidence. This analysis was used to evaluate how excess pore water pressures built up in materials just before the failure. A ‘Safety Factor’ analysis was also undertaken using PLAXIS to evaluate stability behaviour under rapid filling and to explore initiating failure mechanisms.
- Conventional limit equilibrium slope stability analyses using SLOPE/W software were also carried out. This provided an alternative examination the initial failure mechanism, using the same pore pressure regime generated in PLAXIS. This approach enabled a comparison against the PLAXIS derived Factor of Safety (FS) and provided further calibration of the inferred failure mechanism prior to strength loss and liquefaction.

The PLAXIS finite element analyses were undertaken with the main purpose of assessing the pore water pressure response within the stockpile immediately prior to the flow failure and to investigate where yielding occurred as a precursor to a collapse mechanism. Four construction stages comprising one toe cut and three filling steps were used to simulate actual loading history of the stockpile.

The model indicates that excess pore water pressure builds mainly due to the final rapid fill lift before failure. As shown in Figure 5, this increases to over 50 kPa in lower permeability basal coal material and critically, extends upwards into overlying contractant materials. In other situations, such as the tailings failure at the Merriespruit Harmony Mine reported by Davies et al. (2002), fully drained conditions can exist up until the very moment of liquefaction. Here, failure was initiated not because of increasing shear stresses and undrained loading, but because of decreasing effective stress due to the rising phreatic surface under drained conditions.

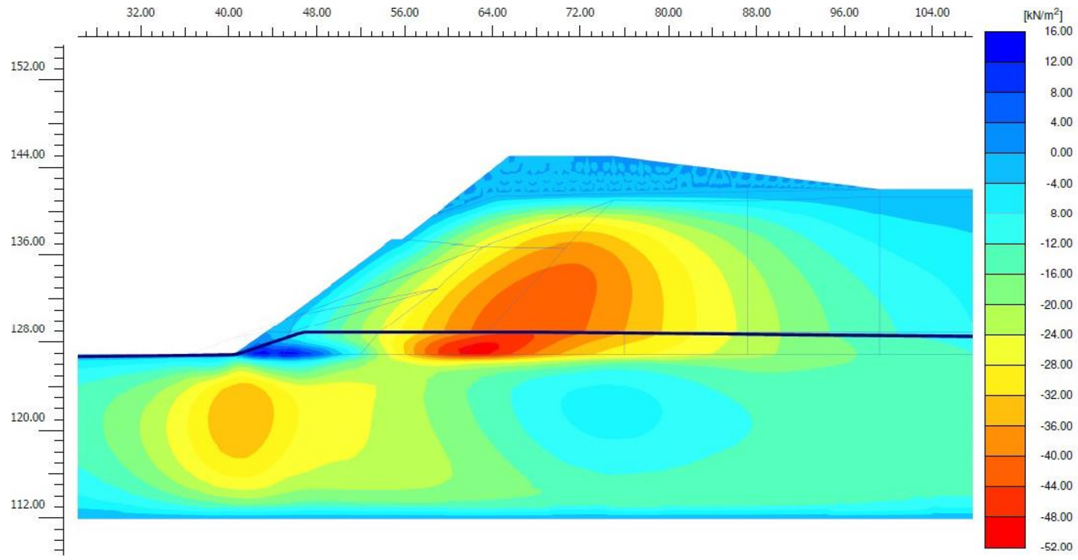


Figure 5 Excess pore water pressure developed in stockpile prior to failure

Initial yielding within the PLAXIS model was assessed at the final filling stage and also after development of a full scale failure mechanism by performing a strength reduction safety factor analysis. The latter provided a measure of FS expressed as the degree of strength reduction required to induce a failure mechanism as defined by a pre-set strain criterion. The excess pore water pressures from PLAXIS at failure were then used as inputs in a series of limit equilibrium slope stability analyses undertaken using SLOPE/W software. This was done by specifying pore pressures in specific regions in terms of equivalent 'Ru' values, where Ru is the ratio of excess pore water pressure developed to initial overburden pressure at each point. These user-defined Ru pore pressures were applied in SLOPE/W only in those regions where excess pressure was identified in PLAXIS. Failure mechanisms and FS values identified by the two types of analysis were also compared and were generally found to agree within 10%. By exporting the pore pressure regime from PLAXIS to SLOPE/W, it was possible to investigate the failure hypothesis and confirm the failure trigger mechanisms. The 'calibrated' back-analysis models were then used to identify critical factors for future stockpile risk management including stockpile height, fill placement rate, pore pressure and slope geometry so that these could be safely managed during operation.

4 Analysis input parameters

4.1 Material properties

Bulk samples were collected from the stockpile. The samples were taken from the top of the stockpile (Upper Stockpile) and near the base of the stockpiles (Lower Stockpile), and were sent to a NATA accredited Golder laboratory in Brisbane for geotechnical testing, as follows:

- Particle size distribution (PSD) for classification and to allow assessment of susceptibility to flow slide/liquefaction. PSD was also used to assist in estimating material strength and permeability.
- Field moisture content to estimate in situ degree of saturation and to provide a basis for adopting similar moisture content for remoulded test samples.

- Dry density/moisture relationship (Standard Compaction). The maximum dry density obtained, along with field Dynamic Cone Penetrometer (DCP) test results, is used to estimate in situ dry density.
- Permeability to estimate coefficients of permeability for numerical modelling to assess excess pore water pressures.
- Unsaturated unconsolidated undrained triaxial tests (UUU) to estimate shear strength parameters of unsaturated coal.
- Saturated unconsolidated undrained triaxial tests with pore water pressure measurement (SCU) to estimate effective shear strength parameters and also to assess whether samples are contractive or dilative under stress ranges similar to in situ stress.

Laboratory test results are presented in Tables 2 and 3.

Table 2 Fine coal (hard cooking coal) properties

Sample Reference	Basal Layer	Upper Layer
Field moisture content (%)	12.1–18.4	5.6–6.6
Std maximum dry density (kN/m^3)	11.34	11.23
Optimum moisture content (%)	12.9	12.8
Coefficient of permeability* (m/sec)	1.8×10^{-7}	2.5×10^{-7}
Grain size 10% finer than d_{10} (mm)	0.04	0.03
Percentage finer than 0.5 mm (%)	35	34

Note: Coefficient of permeability was measured at a recompacted dry density of 0.95 t/m^3, which is the estimated average field condition.

Table 3 Shear strength parameters of fine coal

Sample ID	Test Type	Initial Dry Density (kN/m^3)	Initial Moisture Content(2) (%)	Effective Friction Angle (°)	Cohesion (kPa)
1–L(1)	SCU	0.96	12.1	27	18
2–L	UUU	0.96	12	33	0
3–U	SCU	0.96	6.6	34	12
4–U	UUU	0.96	6	32	0
5	SCU	1.01	6	35	10
6	SCU	0.9	6	35	10
7	UUU	0.9	6	35	5
8	UUU	1.01	6	33	27
9	SCU	0.9	12	33	20
10	SCU	1.01	12	39	10
11	UUU	0.9	12	23	6
12	UUU	1.02	12	30	10

Notes: 1. L Sample has been collected from the lower layers around the toe.
U Samples has been collected from the upper layers.
2. Moisture content used to compact samples to the initial target dry density.

4.2 Stockpile in situ density

Dynamic Cone Penetrometer (DCP) tests were undertaken to assess the in situ density of stockpile. The test results indicated that the stockpile comprised of consecutive layers of loose to medium dense or denser coal underlain by a medium dense lower basal layer. A thickness of 1.5 m was considered for the low permeability basal fill layer in the stability analyses.

Table 4 presents the coal parameters adopted in this assessment. These parameters are based on the test results and interpretation of DCP test results undertaken on the stockpile, in consideration of the published data for similar materials (e.g. Eckersley, 2000).

Table 4 Material parameters adopted for analysis

Identification	Coal (Loose)	Coal (Medium Dense)	Coal Wash Gravel
Unsaturated unit weight (kN/m^3)	10	13	19
Saturated unit weight (kN/m^3)	12	14.5	20
Coefficient of permeability (m/day)	0.016	0.001	0.0001
Internal friction angle, ϕ’ (°)	30	33	40
Effective cohesion, c’ (kPa)	5	10	10

5 Excess pore water pressure versus loading rate

As discussed in Section 3.1, to estimate the excess pore water pressure developed within the stockpile under different filling rates, the stockpile formation was simulated using PLAXIS. Different cases were considered where the height of the stockpile was raised from 5 to 11 m, 13 and 15 m under specified loading rates of 1 m/day, 3 m/day and 5 m/day for each height. 2 m wide benches were included at 5 m vertical intervals in accordance with the mine slope stability management plan (Figure 6).

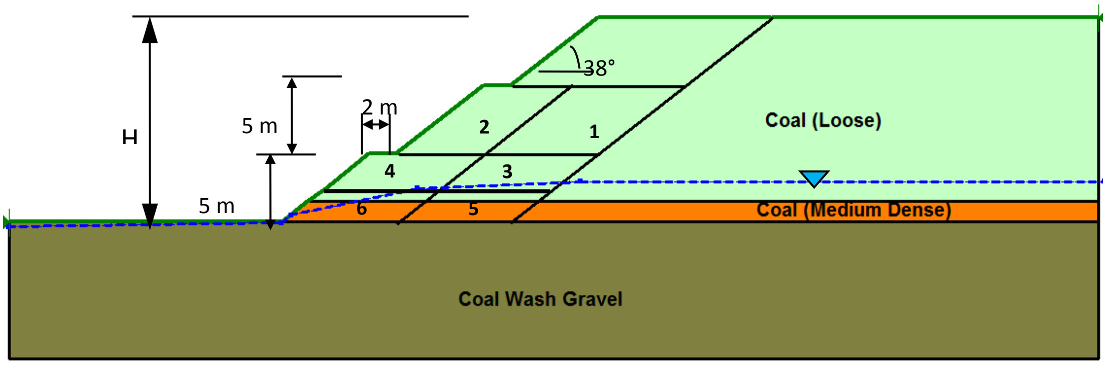


Figure 6 Model geometry showing excess pore water pressure zones

Six zones of excess pore water pressure were considered within the stockpile for stability analysis, as illustrated in Figure 6. The average excess pore water pressure ratio, R_u, was estimated in each zone for different loading rates calculated from the PLAXIS outputs. The average R_u adopted in the Slope/W stability analyses are presented in Table 5.

Table 5 Average Ru values adopted in Slope/W stability analyses

Stockpile Height (m)	Loading Rate (m/day)	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5	Zone 6
11	1	0	0	0	0	0.15	0.10
	3	0	0	0.15	0.15	0.25	0.15
	5	0	0	0.25	0.30	0.30	0.30
13	1	0.17	0	0.13	0	0.20	0.15
	3	0.30	0	0.25	0.20	0.30	0.20
	5	0.40	0	0.30	0.30	0.40	0.30
15	1	0.25	0	0.20	0.13	0.20	0.17
	3	0.50	0	0.20	0.13	0.50	0.25
	5	0.50	0.25	0.50	0.40	0.50	0.40

6 Stability analyses

Stockpile slope stability analyses were completed using Slope/W software to assess the influence of loading rate and water level within the fine coal stockpile, expressed as a limiting equilibrium Factor of Safety. The analyses were carried out on two types of the stockpile geometry based on the on-site observations and history of the stockpile instability in the mine:

- The original stockpile before toe excavation and after loading up to a specified height (Figure 7(a)).
- The stockpile after loader reclamation from the toe leaving a subvertical cut of maximum 4.5 m high (Figure 7(b)).

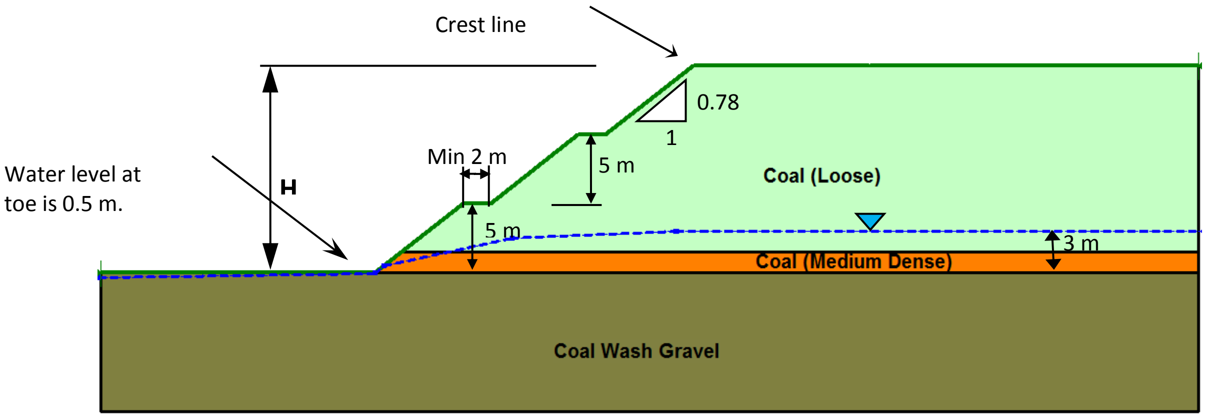
The water level in the stockpile was considered to be at 3 m height above foundation level in the centre of the stockpile and gradually reduced to approximately 0.5 m at the toe, based on the maximum water level recorded during the last year before this assessment at two piezometers installed within the stockpile.

Furthermore, slope stability analyses were carried out on stockpiles with three different heights and varying hydrostatic groundwater levels above the foundation level, ranging between 1 and 7 m. These analyses were performed without considering excess pore pressure as a base case.

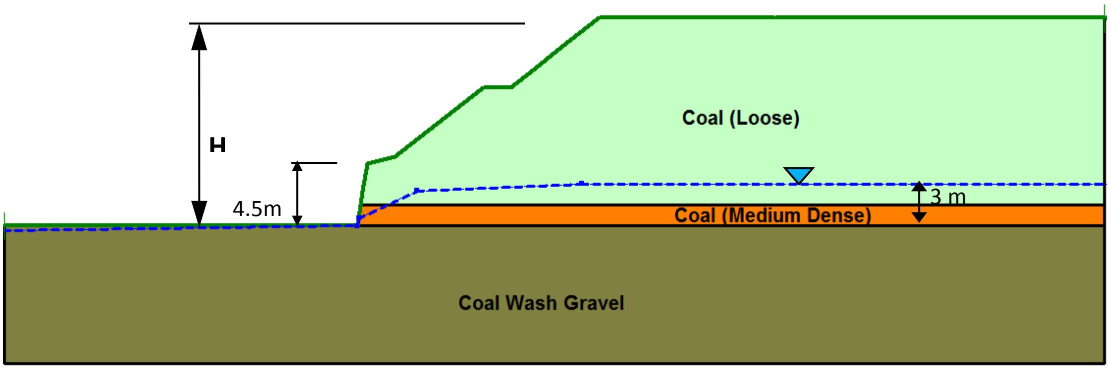
7 Analysis results and discussion

Figure 8 presents the Factor of Safety values achieved in the cases where the groundwater level varied within the stockpile (without excess pore water pressure). As indicated in this figure, FS dramatically reduces as water level increases beyond a specific value from the base of the stockpile, which ranges from less than 2 m for the 15 m high stockpile to about 3 m for the 11 m high stockpile. Considering a minimum FS = 1.2 for temporary conditions, it was shown that risk of stockpile instability increase to high level as the water level rises to about 6 m for stockpiles 13 high or higher.

Figure 9 illustrates how stockpile slope stability (as measured by Factor of Safety against failure) varies with the fine coal loading rate, at key stockpile heights. In these analyses, FS was considered only for failure mechanisms that extend over the whole height of the stockpile. Lower factors of safety were observed for small slumps occurring at the surface, although the consequences of failure are relatively low.



(a) 'Design' geometry before toe load-out



(b) Subvertical cut at toe after toe load-out

Figure 7 Typical geometries considered in the analyses

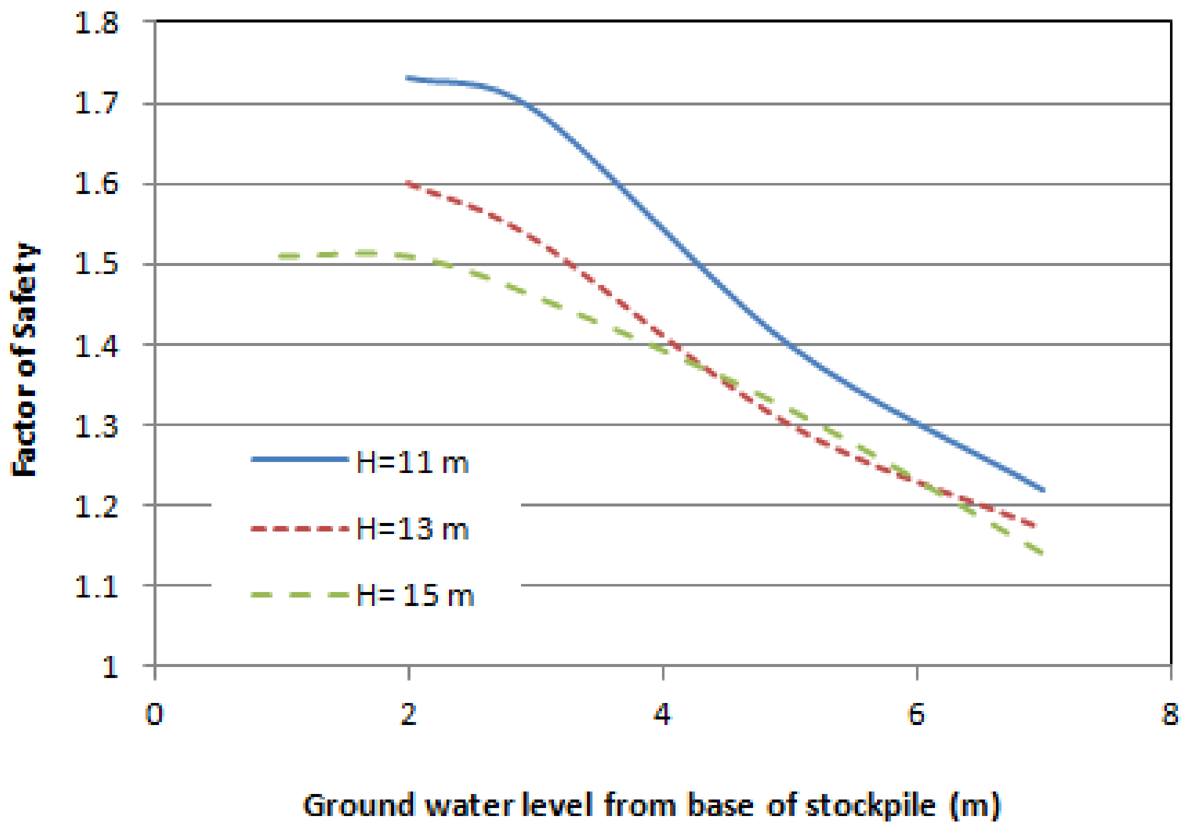


Figure 8 Variation of FS with steady groundwater level for stockpile at different heights

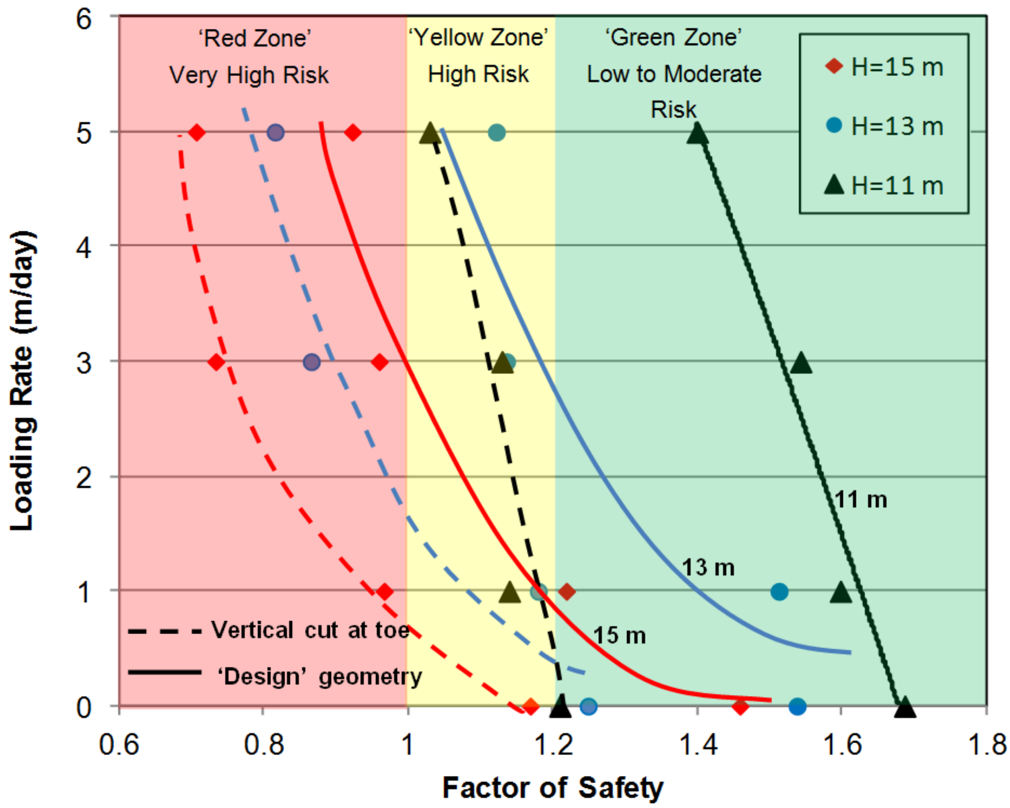


Figure 9 FS versus loading rate for the HCC stockpile with different height and geometry

As indicated in Figure 9, as loading rate increases excess pore water pressure develops around the basal layer. Consequently, effective stress and therefore frictional resistance within the stockpile is reduced, resulting in yielding, which is predicted to occur along a deep-seated failure surface near the top of the basal layer. In the course of a detailed study on the stockpile failure, it was observed that when a full scale failure was allowed to develop using the PLAXIS 'Safety Factor' analysis, a rotational failure surface was evident. However, this large strain model does not reflect the true behaviour of collapse on shearing, liquefaction and development of flow failure (having very low strength) which controls the mass movement of material at high strains. Potential theoretical simulations of this flowslide behaviour are discussed in Finay and Fell (1997) and include consideration of factors such as plastic flow, laminar and turbulent flow, degrees of confinement, non-hydrostatic stress and entrainment/deposition of material during motion.

Cutting into the toe during load out was also shown to dramatically reduce stability of the stockpiles. For instance, a 4.5 m high vertical cut made into the toe of the stockpile immediately after final filling increased the risk of instability to high or very high levels, when filled above 10 m with a loading rate of 1 m/day.

Based on the above observations, specific risk mitigation measures were considered for this site such as providing drainage at the base of the HCC stockpile, controls on the stockpile size and shape, placement rates and loading operations. In particular, groundwater level measurement was identified as a key management tool for assessing the stockpile instability risk. A groundwater monitoring system was recommended as part of an updated stockpile risk management plan. Embedded earth pressure cells were also installed to enable monitoring of overburden load and rate of loading, for use with Figures 8 and 9 to manage instability risk. Table 6 shows a summary of the risk management recommendations.

Table 6 Recommended risk control measures for Trigger Action Response Plan (TARP)

Item	Criteria	Control Measurement
Loading rates	H≤11 m then loading rate ≤5 m/day. 11 m<H≤13 m then loading rate ≤2.5 m/day. 13 m<H≤15m then loading rate ≤1 m/day.	Daily survey and fill depth marking on poles. Installation of earth pressure cells below the stockpile base.
Groundwater	Pore water pressure to be ≤50 kPa within 10 m offset from crest line (equals a total water head of 5 m above the foundation). Seepage at the toe less than 0.5 above foundation.	Installation of piezometers at the base to measure groundwater level. Trial: a drainage blanket at the foundation of the stockpile comprising coarse coal, about 300 mm and monitor the effectiveness with the water and earth pressure measurement.
Geometry	Maintain design geometry (or flatter) during stockpile loading. Minimum 15 m clearance between toe and adjacent rail track. Maximum height of sub-vertical cut at the toe is limited to 4.5 m. A minimum of 1, 2 and 3 days are allowed between loading and unloading for stockpiles up to 11, 13 and 15 m high, respectively, unless monitoring confirms pore water pressures are <50 kPa (i.e. an excess pressure of 20 kPa) within the stockpile.	Survey and documentation.

8 Conclusions

Outcomes of a geotechnical assessment have been presented to manage the risk of deep-seated flowslide failures for a fine coal stockpile in NSW. The key factors controlling the stockpile stability were identified as: 1) Elevated pore water pressures induced by rising water levels within the stockpile, 2) Excess pore water pressure developed due to rapid loading rate of the stockpile, 3) Removal of toe support during stockpile unloading, and 4) stockpile height.

PLAXIS finite element analyses were conducted to estimate excess pore water pressure developed in different zones under different stockpile filling rates. Adopting the PLAXIS excess pore water pressures, a series of limit equilibrium analysis were carried out to study stockpile stability at different heights and filling configurations. The outcomes of this assessment indicate that a loading rate as high as 5 m/day may be adopted up to about 10 m height, however, it should be reduced to less than 1 m/day for stockpiles 15 m or higher. It was also shown that groundwater level more than about 6 m above the base of the stockpile could reduce the short-term stability of the stockpile to FS≤1.2. The analyses also suggested that loader reclamation from the toe shortly after stockpile filling escalated the risk of instability for stockpiles above 10 m placed faster than 1 m/day. Based on the results, recommendations were provided to control instability risk by monitoring loading rates, groundwater level and stockpile geometry.

Acknowledgement

The authors gratefully acknowledge the permission of the site owner and project manager in permitting the writing of this paper for the benefit of the technical community. Also, this paper could not have been written without the input, energy and technical guidance provided by our colleague Chris Meikle.

References

Been, K. and Jefferies, M. (1985) A state parameter for sands, Geotechnique, Vol. 35, No. 2, pp. 99–112.

Davies, M.P., McRoberts, E. and Martin, T. (2002) Static Liquefaction of Tailings – Fundamentals and Case Histories, in Proceedings Tailings Dams, 29 April – 1 May 2002, Las Vegas, USA, Association of State Dam Officials /United States Committee on Large Dams.

Dawson, R.F., Morgenstern, N.R. and Stokes, A.W. (1998) Liquefaction flowslides in Rocky Mountain coal mine waste dumps, Canadian Geotechnical Journal, Vol. 35, pp. 328–343.

Eckersley, J.D. (1985) Flowslides in stockpile coal, Engineering Geology, 22, pp. 13–22.

Eckersley, J.D. (2000) Moisture changes and stability problems in coal stockpiles-final report, ACARP Project C4057.

Finlay, P.J. and Fell, R. (1997) Landslides: risk perception and acceptance, Canadian Geotechnical Journal, Vol. 34(2), pp. 169–188.

Hunter, G. and Fell, R. (2001) Rapid Failure of Soil Slopes, Uniciv Report No. R-400 June 2001, UNSW, ISBN: 85841 367 1.

Hunter, G. and Fell, R. (2002) Estimation of travel distance for landslides in soil slopes, Australian Geomechanics, Journal and News of the Australian Geomechanics Society, May 2002, Vol. 37, Issue 2, pp. 65–82.

Hustrulid, W.A., McCarter, K. and Van Zyl, D.J.A (2000) Slope stability in surface mining, ISBN 0873351940.

Jefferies, M.G. and Been, K. (2006) Soil liquefaction, a critical state approach, Taylor & Francis, London.

Siddle, H.J., Wright, M.D. and Hutchinson, J.N. (1996) Rapid failures of colliery spoil heaps in the South Wales Coalfield, Quarterly Journal of Engineering Geology, Vol. 29, pp. 103–132.

Ulrich, B.F. and Fourie, A.B. (2003) Assessment of the potential for static liquefaction of a tailings dam using laboratory and field testing, in Proceedings Tailings and Mine Waste 2003, 12–15 October 2003, Vail, USA, Balkema, Rotterdam.

Yamamuro, J.A. and Lade, P.V. (1998) Steady state concepts and static liquefaction of silty sands, Journal of Geotechnical and Geoenvironmental Engineering, American Society of Civil Engineers, Vol. 124 (9), pp. 868–877.