Geotechnical characterisation and stability analysis of BHP Cannington paste backfill

L'analyse géotechnique de caractérisation et de stabilité de la pâte de BHP Cannington remblayent

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ABSTRACT: The paper describes the geotechnical characterisation and stability analysis of paste fills from BHP Cannington Mine. A thorough experimental study was required to fully understand the strength and deformation characteristics of the paste fill. Further, these strength parameters are also required as input parameters to the numerical model developed to undertake the stability analysis. Geotechnical, mineralogical and microfabric studies were made on the paste fill. A numerical model, developed in FLAC ^{3D}, simulated the excavation, filling and curing of the stopes through a full mining sequence. The presence and effect of arching in the backfill mass during the sequential mining of stopes, were investigated and results are reported here.

RÉSUMÉ: : L'article décrit les caractéristiques géotechniques et la stabilité des analyses des pâtes de remplissage des mines BHP Cannington. Une étude expérimentale sérieux a été nécessaire pour bien comprendre la résistance et la déformation des pâtes de remplissage. De plus, ces paramètres de résistance sont nécessaires en temps que paramétres rentrant en jeu pour le modèle numérique fabriqué pour entreprendre l'analyse de stabilité. Des études géotechniques, minéralogiques et microfabric ont été menées sur les pâtes de remplissage. Un modèle numérique, developpé dans la FLAC^{3D}, a simulé l'excavation, le remplissage et le traîtement des gradins à l'aide d'une extraction minière complète. La présence et l'effet du courbement dans la masse de remblai pendant l'exploitation séquentielle des gradins, ont été etudiés et des résultats ont été enregistrés ici.

1 INTRODUCTION

Although a relatively new technology, the use of paste backfill has gained rapid acceptance as an alternative backfill material to the conventional cemented hydraulic fills (Udd, 1989; Udd and Annor, 1993). As mine stopes are removed, the paste fill is used to backfill the empty space. Paste fill provides substantial benefits to mining operations including an effective means of tailings disposal, improvement of local and regional rock stability, greater ore recovery and greatly reduced environmental impacts. BHP Cannington mine has been using paste backfill underground since 1997. Cannington paste fill is simply mine tailings, with typical effective grain size of 5 µm, mixed with a small percentage of cement binder. In order to provide stability, paste fill must remain stable during the extraction of neighboring stopes. If the paste becomes unstable, the adjacent faces may relax and displace into the open stope. High cement quantities (up to 6% typically 3-5% by wet weight) have been used in the past, to ensure the stability of backfilled stopes, especially during blasting. The high cost of cement has placed a greater emphasis on the optimization of fill design for strength with respect to cement usage.

2 LABORATORY INVESTIGATION

Geotechnical, mineralogical and microfabric studies were made on the paste fill. The geotechnical studies include laboratory tests such as unconfined compressive strength (UCS), unconsolidated undrained (UU) triaxial compression tests, and direct tensile tests. X-ray diffraction (XRD) and x-ray fluorescence (XRF) were carried out to determine the mineralogical properties and the scanning electron micrographs (SEM) were used to study the microfabric.

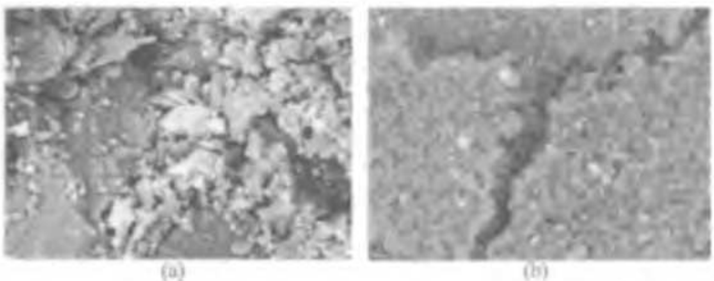
Even a slight reduction in the cement content leads to a substantial cost saving. Therefore, it was necessary to carry out a series of tests using different paste fill mixes, to study the effects of cement content and solids content on the strength characteristics of the paste fill. Laboratory cast samples, with cement contents varying between 0% and 6%, and solids content varying between 74% - 78% were tested at 7, 14, 28 and 56 days after casting.

The UCS and direct tensile test samples were cast into 50 mm diameter by 120 mm long PVC moulds and the UU samples were cast into 38 mm diameter by 90 mm long PVC moulds. Following casting all samples were rodded to remove air voids and then sealed and cured at 100% humidity and 38° C for testing at 7, 14, 28 and 56 days.

2.1 *Material Properties*

A qualitative assessment of tailings mineralogy using x-ray diffraction (XRD) coupled with a semi-quantitative x-ray fluorescence analysis indicated the presence of: - silver minerals (<1%), galena (2.4%), spalerite (1.3%), iron sulphides (39.5%), talc (11.1%) and other silicates (40.7%). The silicates are mostly quartz as well as a small amount of chalcopyrite. The iron sulphides (39.5%) include pyrite, pyrrhotite and arsenopyrite. The remaining 5% consists of aluminium oxides (Chalcopyrite).

Figure 1 shows a scanning electron microscope image of the tailings (a) and cemented backfill (b). The lighter portions in the photo (a) indicate heavier ionic compounds (heavy metals). The filamentous cement bonds can be seen in the cracks shown in photo (b).



(a) (b)

Figure 1. Scanning electron microscope images (a) tailings only (b) paste fill mix (4% cement, 76% solids)

The specific gravity of the tailings was measured as 3.18, reflecting high content of heavy metal. The grain size distribution is shown in Figure 2.

Distribution	1% clay, 8% Sand, 91% Silt
USCS Classification	Sandy Silt (ML)
Coefficient of uniformity	0.82
Coefficient of curvature	11.2
%<20 μm	35.2%
Permeability (Hazen approximation)	$2.6 - 3.9 \times 10^{-7}$ m/s

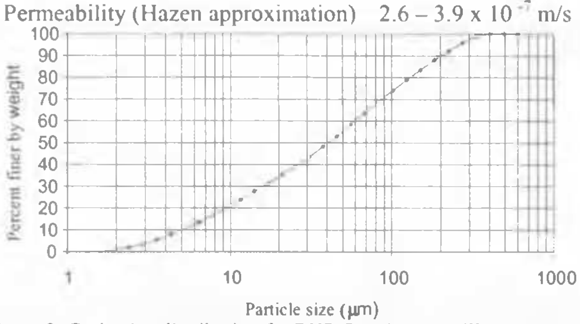


Figure 2. Grain size distribution for BHP Cannington tailings

The physical properties of the fill remained relatively constant throughout curing for all binder contents. The average properties were as follows:

Moisture content:	27.5%
Void Ratio:	0.52
Degree of saturation:	94%
Porosity	34%
Bulk density	2130 kg/m^3

2.2 *Unconfined Compressive Strength*

Table 1 summarizes the progressive strengths obtained from the unconfined strength tests. Results shown are the average of three test samples for each binder content and curing time.

Table 1. Unconfined compressive strength results.

	Average Strength (kPa)			
Days of curing	7	14	28	56
2%C ,74%S*	61	58	78	65
2%C ,76%S	85	97	90	89
2%C ,78%S	138	174	134	133
4%C ,74%S	142	166	193	186
4%C ,76%S	244	233	237	237
4%C ,78%S	386	391	406	335
6%C ,74%S	369	474	489	371
6%C ,76%S	588	623	614	594
6%C ,78%S	822	875	856	828

*2%C, 74%S = 2% cement, 74% solids

Sample strengths increase proportionally with increased cement content, solids content and curing time, as expected. Samples strengths remained reasonably consistent with those obtained for testing after seven days of curing. The attainment of high early strengths may be considered a favourable characteristic when considering the need for strength development in backfill masses before exposure. A slight increase is observed at 14 days returning to the 7-day strength after testing at 56 days. The reduction in strength is possibly due the presence of sulphides in the tailings (pyrite, iron sulphides), which "attack" and weaken the cement bonds. For curing times of up to 56 days this phenomena is not expected to result in any significant reduction in strength. For longer curing times, the effect of the sulphide attack on fill stability could be more severe.

2.3 *Multistage Unconsolidated Undrained triaxial tests*

Paste fill samples were tested in multistage triaxial tests. Samples were consolidated under isotropic stresses of 100, 200 and 400 kPa prior to being sheared. Testing was performed on the strongest and weakest mix variations so that intermediate results may be interpolated.

The results from the triaxial tests indicate that the friction angle remains reasonably consistent throughout curing. A slight increase in the friction angle is observed for each of the samples at 14 days. It appears that the hydration of the cement added to the paste mixture utilizes a significant portion of the water content leaving the paste fill samples at below full saturation. Structural change appears to occur within the paste fill between 14 and 28 days and reduce the friction angle. Cohesion was shown to increase linearly with time, cement content and solids content. Test results are summarized in Table 2.

Table 2. Multistage unconsolidated undrained triaxial test results

CuringTime	7-Day		14-Day		28-Day		56-Day	
Mix Design	ϕ_u (deg)	c_u (kPa)	ϕ_u (deg)	c_u (kPa)	ϕ_u (deg)	c_u (kPa)	ϕ_u (deg)	c_u (kPa)
2%C ,74%S	2.3	50.4	4.0	55.3	1.3	47.2	3.7	60.9
2%C ,78%S	4.3	143.4	7.1	136.0	5.0	168.5	3.5	154.0
6%C ,74%S	14.3	139.0	16.2	157.4	12.8	173.9	14.4	177.7
6%C ,78%S	21.5	259.0	22.0	246.7	16.4	255.2	17.0	269.1

The poisons ratio (ν) used for the analysis of paste fill was 0.25, which was based on experimental measurements and comparison with reported values in literature. Typical E (Young's Modulus) values for the paste ranged between 14×10^6 Pa and 60×10^6 Pa and were found from laboratory testing.

2.4 *Direct Tensile Strength*

Figure 3 summarizes the averaged tensile strength for the tested paste fill mixes. The higher 14 day strength is attributed to the development of cement bonds, and the re-alignment of the paste's soil matrix.

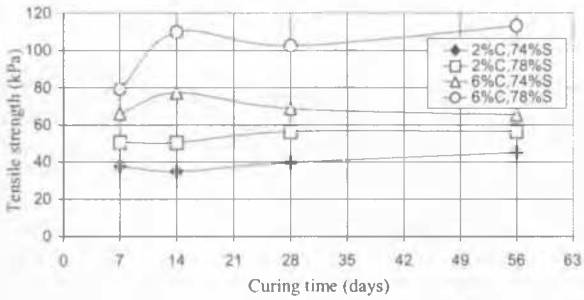


Figure 3. Progressive tensile strength results

3 STABILITY ANALYSIS

Cannington Mine is an underground lead-silver-zinc mine in North West Queensland, and is the world's largest single mine producer of silver and lead. Cannington mine is the first mine in Australia to use the open stoping mining method in conjunction with post placed paste backfill. To achieve complete ore extraction cemented fill is used to fill the voids left by mining. The design of the paste fill mix is based on the requirement of stable exposures during the mining sequence. An idealized stope extraction/mining sequence is shown in Figure 4.

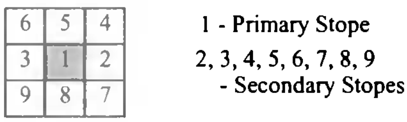


Figure 4. Plan view of idealized extraction sequence around a given fill mass

Exposures are created when an adjacent fill mass is removed, leaving the backfill mass self supporting. The Cannington stope dimensions selected for analysis are 25 m x 25 m in plan and 50 m high, and exposures are one full face (25 m x 50 m tall). Stability of the paste is critical. If either a full or partial failure of the fill mass occurs, extraction of the ore may become impossible or unacceptable levels of dilution may occur. In previous backfill analysis throughout the world, limit equilibrium methods were used to obtain an indication as to the stability of the fill mass, by calculating a factor of safety against failure. Mitchell et al. (1982) developed a three-dimensional limit equilibrium solution for the stability of exposed vertical faces. Failure was assumed to occur in the form of a confined block mechanism as shown in Figure 5.

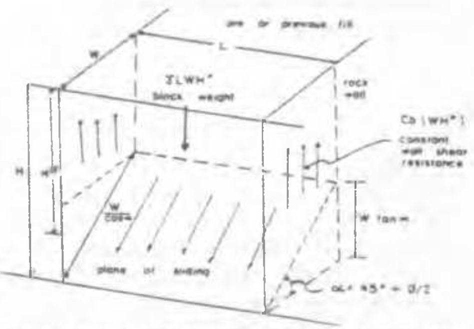


Figure 5. Confined Block Mechanism (Mitchell et al. 1982)

Here H = fill height; H^* = effective sliding block height = H-(w.tanα)/2, W_n = net weight of the sliding block, α = angle of failure plane from horizontal = 45+ϕ/2, ϕ = fill friction angle, c = fill cohesion, L = distance between the hanging wall and foot wall, γ = fill unit weight; K_0 =1-sinϕ and w = strike length.

The stope was assumed to fail along a plane of sliding. Furthermore, it was assumed that constant wall shear strength, equal to the cement bond shear strength (cohesion), was mobilized to reduce the net effective load acting on the slip plane. Simplifications were made which allowed the height to be much larger than the length. The assumption that the cemented sand backfill was a frictionless material allowed for the derivation of a simpler and somewhat more conservative equation (Mitchell et al., 1982) as

$$\sigma_{1F} = \gamma H\left(1+\frac{H}{L}\right) \tag{1}$$

where σ_{1F} = major principal stress at failure (kPa) and γ = bulk unit weight of fill (kN/m^3).

A preliminary analysis indicated that shearing of cement bonds occur at small strain, thus supporting the use of a friction angle of zero for cemented fill. Additional analysis concluded that the constant wall shear assumption used in equation (1) was satisfactory, although conservative.

Winch (1999) proposed an analytical solution to the total vertical stress within a three dimensional backfill mass. The results from the model give a vertical stress at a specified distance from the top of the fill. The calculation of the vertical stresses is based on the assumption that full mobilization of shear strength occurs along the walls. This assumption is incorrect, as the full mobilization of the shear only occurs only at the limit of stability. Consequently the arching will not occur to the extent that has been assumed by the model and calculated vertical stresses will be underestimated. This is of little consequence as the stresses are only underestimated when the fill mass is stable.

Numerical modeling, using FLAC^{3D}, was used to examine the stability of backfilled stopes in more detail. FLAC^{3D} was able to provide more accurate solutions by not imposing the simplifying assumptions required for solutions of analytical methods. FLAC^{3D} was used to develop a model capable of accurately modeling the excavation, filling and curing of a stope throughout the complete extraction sequence, as shown in Figure 4.

3.1 Model Development

The geometry of the problem is defined using a finite difference grid, the constitutive behaviour and associated material properties dictate the type and response of the model, and the in-situ state is defined by the boundary and initial conditions.

The finite difference grid was generated using a predefined "block" element shape from the element library in FLAC ^{3D}. The block extended 75 m (3 x 25 m) in each direction in plan view and 50 m vertically. The nodes along the perimeter and base of the block were fixed in all directions. This was considered reasonable as the stiffness of the rock at the boundaries is considered infinitely stiffer than the paste and would not move during the sequential mining of stopes. The initial conditions were found by excavating the primary stope, applying gravity forces, and then solving the system to equilibrium. Alterations were then made (e.g. stopes are excavated/ backfilled) and the resulting response of the model recalculated. By saving the solution at each step, the initial conditions for the proceeding step could be defined.

To model the sequential excavation and filling of the stopes it was necessary to be able to define the change of strength properties of each zone. Initially all zones in the model were assigned the properties of rock. When the primary stope was excavated the zones within stope 1 were assigned the properties of a void. When filling, the zones contained within each lift are sequentially activated and assigned material properties of curing paste. Each lift was 5 m tall and was assumed to cure for 7 days prior to the application of the next layer. Thus when the second lift is activated, the zones were assigned material properties for 7-day paste and the initial lift assigned 14-day strength characteristics. This filling process is cycled through in seven-day increments until the stope has been completely filled and the top lift has acquired 56-day strength properties. This process of filling and curing of the backfill is applied throughout the mining sequence.

Rock was assumed to behave elastically, as the applied loads were not considered to be significant enough to force the surrounding rock into a plastic state. Voids were assumed to have the properties of the "null" constitutive model. A linear regression analysis was performed on the failure of paste fill during the multistage triaxial testing. Correlation coefficients of greater than 0.9 were achieved from the linear regression analysis for all triaxial tests indicating that the Mohr-Coulomb failure criterion applies to paste fill, regardless of mix proportions.

3.2 Model Verification

To verify the model used to assess stability at BHP Cannington mine, it was first necessary to develop a numerical model that had previously been verified by comparison with in-situ data. The modeling of the underground stability of cemented hydraulic fill (CHF) at Mount Isa Mine (Bloss, 1993) was considered to be the most appropriate problem to validate the numerical model for Cannington mine due to the physical similarity between backfills. The vertical stress profile down the center and across the primary stope (fully confined) at mid-height, were used to compare and validate the FLAC^{3D} model against the previously validated TVIS modelling package. Figures 6 and 7 show the comparison between results from the FLAC ^{3D} and TVIS models.

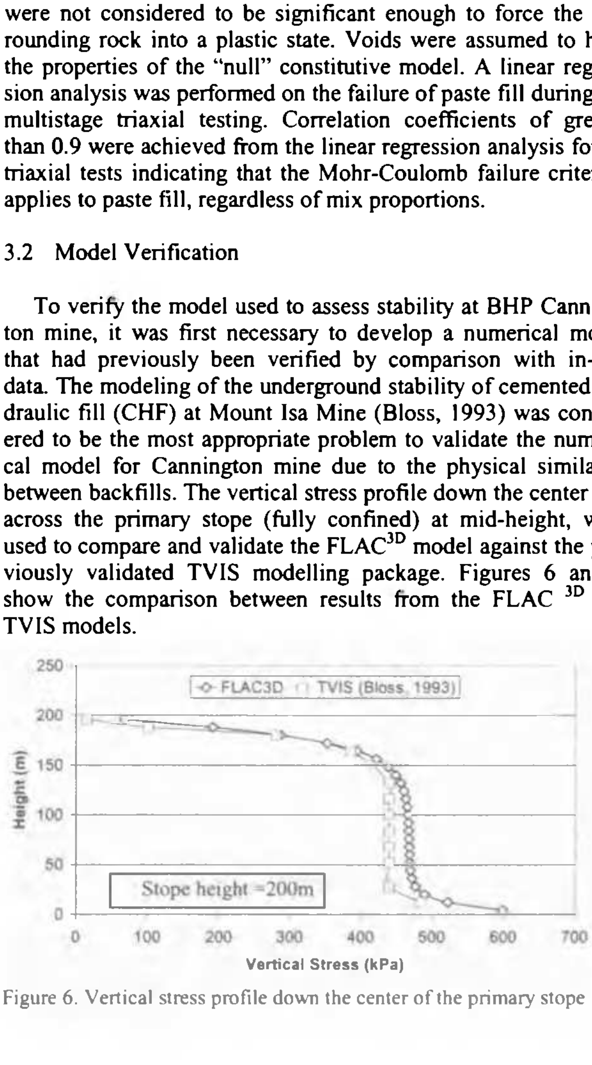


Figure 6. Vertical stress profile down the center of the primary stope

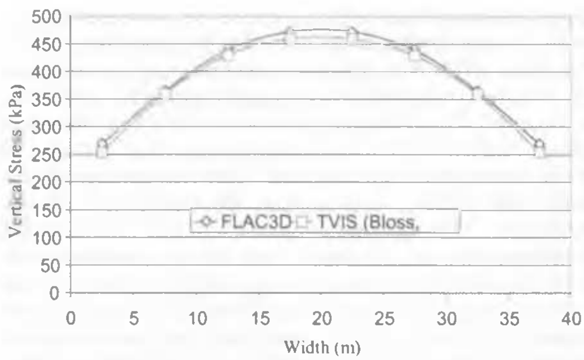


Figure 7. Vertical stress profile across the center of the primary stope.

The slight variations between the TVIS and $FLAC^{3D}$ models could be due to the way in which the initial stresses / conditions were defined. Bloss (1993) calculated the horizontal confining stresses based on the Poissons ratio for the surrounding ore, where as $FLAC^{3D}$ solved the equations of motions for the whole modeling region.

3.3 *Complete Extraction Sequence Model*

To assess the progressive stability of the backfill mass through the mining sequence, a numerical model was developed and the effect of the sequential exposure and filling of stopes on the vertical stresses in the primary stope was studied.

Figure 8 shows the progressive increase in vertical stress in the middle (at width of 12.5 meters, and depth of 12.5 meters) of the backfill in the primary stope at a height of 25 meters.

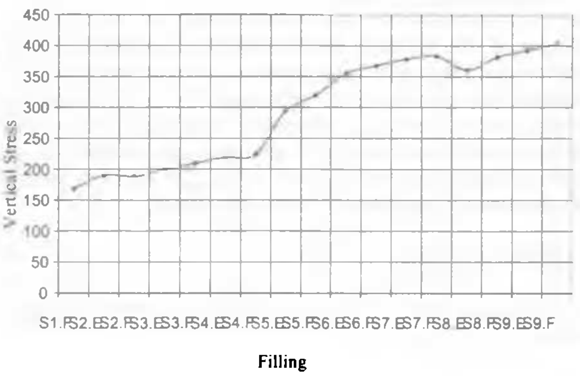


Figure 8. Vertical stress measured in the center of the primary stope (stope 1) at a height of 25m during the mining sequence. S1F = Stope 1 Filled, S2.E = Stope 2 Excavated.

The vertical stress continuously increases through the mining sequence, with the progressive removal of the ore and sequential filling of the surrounding stopes. A significant increase in the vertical stress in stope 1 is observed with the excavation of stope 5. This corresponds to the removal of a wall, critical to the essentially two-dimensional arch, remaining in stope 1.

When fully surrounded by rock, full three-dimensional arching occurs. When a wall is exposed the arching potential of the wall goes to zero and the corresponding vertical stresses increase accordingly. When the newly created void is backfilled, the support ability of the interface is returned, but at a significantly reduced rate (approximately one quarter of the original). The reduction of arching is due to: 1) the reduced strength / support ability of the fill-to-fill contact and 2) the relaxation of the horizontal confining stresses during the excavation of the ore. Again once the stope has been backfilled, confining stresses are applied, but are at a reduced rate.

Considering Figure 4 to be the mining cycle, when stope 2 is excavated the arching mechanism transfers the support of the overlying loads through a predominantly two-dimensional arch spanning between the two opposing rock faces (stopes 5 and 8)., The transfer of the vertical loads to the primary two-dimensional arch is observed to continue through the mining cycle. In the case of stope 5 being excavated, a significant increase in the vertical stress results, as it provides a primary support to the two dimensional arch. Figure 9 shows the effect of the progressive disintegration of the support provided by the arch through the mining sequence. The loss of support is identified by the increase of vertical stresses to those that would be obtained by hydrostatic forces (no arching).

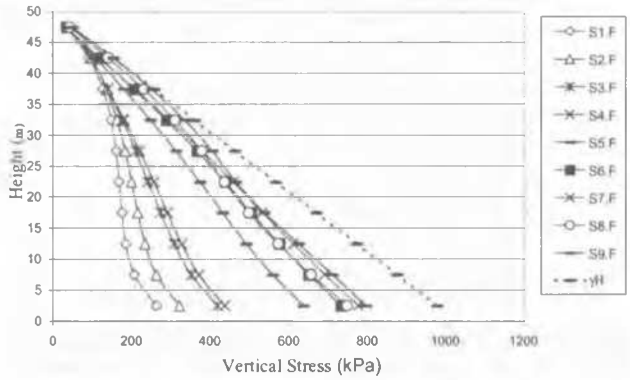


Figure 9 Vertical stress profile down the center of the primary stope, during the filling sequence. γH = ore unit weight *height – used as a comparative value to assess the extent of support provided by arching.

4 CONCLUSION

A series of laboratory tests were carried out to determine the behavior of paste fill in unconfined compression, confined triaxial compression and tension. Total stress parameters were obtained from sample testing and used as inputs to the stability analysis undertaken using $FLAC^{3D}$.

$FLAC^{3D}$ was used to model the verification problem (Bloss 1993) to ensure the integrity of the calculations and then modified and applied to underground mining operations at Cannington. The model was extended to include a 3-dimensional analysis of the induced stresses in the primary stope throughout the mining sequence.

5 ACKNOWLEDGEMENTS

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