Underground Mining with Backfills

R. Rankine, M. Pacheco, N. Sivakugan

The mining industry worldwide has typically not conducted the development of mines with the overall design objective of a safe, environmentally sound and aesthetically satisfactory post-operational mine-site. Mine waste has typically not been engineered to any large degree but has rather been disposed of in the easiest or most cost effective manner with little (if any) regards for the social and/or environmental consequences. The backfilling of mines is an integral part of the mining process and requires the same level of attention generally afforded to the more commonly recognised "profit-producing" parts of the operation. The change in perception of backfilling from an additional cost to mining operations to one of a pre-profit activity will aid the required advancement in technology required for backfills. Backfilling is required for the continuance and efficiency of mining operations. Additional benefits include: improved regional and local rock stability through the support provided by the backfill, reduced costs of building significant tailings disposal structures on the surface, and the reduced environmental impacts by the underground containment of waste material. All these focus the operation towards the overall design objective of a safe, environmentally sound and aesthetically satisfactory post-operational mine-site. With these objectives in mind, the purpose of this paper is to highlight the basic geotechnical issues regarding undergroung mining with backfills, following new developments by the Australian mining industry.

Key words: paste fill, stope, underground mining.

1. Introduction

Discovery of gold in Minas Gerais in 1693 made Brazil the leading gold producer then. A brief historical overview of the mining activities in Brazil was given by Machado & Figueiroa (2001). The mining sector, with more than 1400 active mining companies operating in the country, has an important role to play in the overall economy of Brazil. Brazil is the world's leading producer of iron ore and Latin America's leading producer of manganese, aluminium, ferroalloys, tin, gold and steel. The major minerals recovered from the Brazilian mines include bauxite, gold, iron ore, manganese, nickel, phosphates, platinum, tin and uranium. The states of Minas Gerais (40%), Pará (20%), São Paulo (10%), Bahia (8%) and Goiás (6%) represent 84% of the mining Gross Domestic Product (GDP) in the country. In 2000, the mineral-based industries produced US$50.5 billion, contributing to 8.5% of GDP. In 2004, Brazil opened Sossego mine, it is the largest and a world-class copper mine, in the state of Pará, owned and operated by Companhia Vale do Rio Doce (CVRD).

Australia, by any standards, is extremely well endowed with most minerals even though it has barely scratched the surface of its mineral resources. The nation holds the world's largest known economic resources of bauxite, lead, zinc, silver, uranium, industrial diamonds and mineral sands. The need to ensure the longevity of the nation's economic wealth through the proper and efficient mining operation of mines is then obvious.

South Africa is also particularly rich in mineral resources and is one of the leading raw material exporters in the world. The main mineral raw materials are gold, diamonds, platinum, chromium, vanadium, manganese, iron ore and coal. These goods make up about 60% of the entire export. With platinum, manganese, vanadium and chromium, South Africa is number one globally, as far as mineral resources as well as the actual mining and export volumes are concerned. The mining industry in Canada is strong also. Canada has over 200 producing metal, non-metal and coal mines over 3,000 stone quarries and gravel pits. Diamonds, oil sands and uranium are the main export commodities, with Canada producing over one-third of the worlds global output of Uranium. Canada also has significant mineral deposits of coal, iron ore, nickel, gold and copper. These mines and their economic output accounts for about four percent of Canadian GDP. The USA have abundant natural resources and are the world's leading producer of beryllium, soda ash, molybdenum, phosphate rock and salt. California is the largest producer of non-fuel minerals of any US state, producing approximately 10% of the national mineral product. The concentration of the mining operations located along the western seaboard in Fig. 1 pays testament to the contribution of the Californian mineral reserves.

Since the introduction of favorable mining laws in Chile in the late 1970's, it has become a very attractive minerals target for a number of large national and international mining companies including BHP Billiton, Anglo American, Rio Tinto, Placer Dome, Phelps Dodge, Falconbridge, Barrick Gold, Newmont etc. Chile is the undisputed capital of mining in Latin America and is the world's largest cop-

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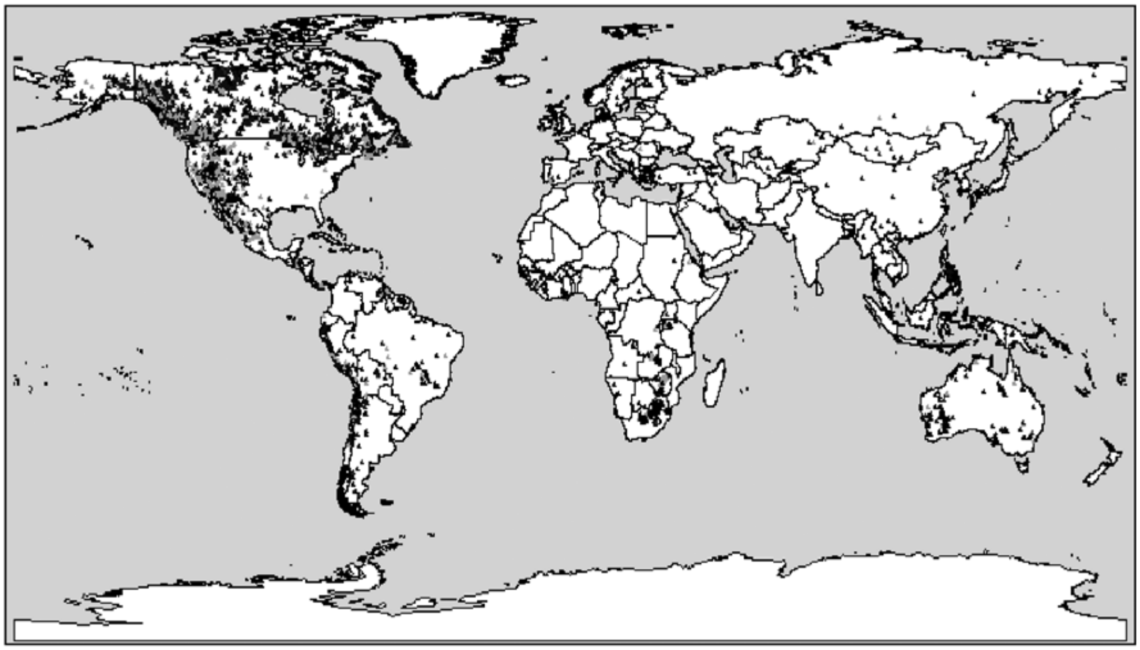

Figure 1 - World-wide mining and exploration activities (Infomining Inc., 2006).

per producer (approximately 20%) as well as exporting gold, silver, molybdenum, zinc, manganese and iron ore. Escondida is the world's largest copper mine and contributes 8% of the world's copper output alone! It is nestled in the cuprous porphyry ore bodies that occur prolifically in the high altitudes and harsh environments of the Andean Cordillera.

Experience gained from the failures of the Ok Tedi Mine (Papua New Guinea) (Kirsch 1996, 2002; Harper and Ravi Rajan 2002) and Marcopper Mine (Phillipines) (UNEP 1996, PDTS 2001), underlines the need to dispose of mine waste in a safe, stable and economically attractive manner. This has highlighted the requirement to be able to accurately predict backfill behaviour and performance. The empirical relationships and operator experience used in yesteryear needs to be replaced by the specific engineering of mine waste. Canada, United States, South Africa, China, Australia, Brazil and Chile are some of the countries that are at the forefront of mining and exploration activities, as shown in Fig. 1. In this paper, a brief description of the main types of backfill is presented, with emphasis on hydraulic fills and paste fills, the most popular backfills used world-wide.

2. Environmental and Safety Issues

The report of the Tribunal (Davies 1967) appointed to inquire into the Disaster at Aberfan (Wales) on October 21st, 1966 detailed the events leading up to and causes of the massive tailings slip from the Merthyr Vale Colliery onto the small mining village of Aberafan, killing 144 people, 116 of whom were school children. The Tribunal was scathing in its appraisal of the competency of those responsible for the stability of the colliery likening them to "moles being asked about the habits of birds". Since then the disposal of mine waste has had a great deal more attention paid to the design and placement of tailings, resulting in a highly engineered "designer waste" (Jones 2000).

Mining activities generally involve several social and economical issues. While millions of tons of soil and rock are removed from the earth's crust, to extract a very small fraction of useful minerals, the rest of the waste material needs to be disposed of. There are strict environmental guidelines stipulating that the mine site is left in good condition on the completion of the mining operation, with all the underground voids backfilled, all toxic material disposed in an environmentally friendly manner, the flora and fauna in the region be protected etc. Mines are required to allocate significant funds to carry out the mine site rehabilitation program, in bringing back the site to a condition similar to what was there before.

Upon extraction of minerals from the ore, there is very large amount of crushed rock, in the form of tailings that has to be disposed of. The most sensible thing to do is to send them back to where they came from. *i.e.*, to backfill the underground voids created in the mining process, using these tailings. With the specific gravity of the parent rock in the order of 2.8-4.0 and the dry unit weight of the backfill of about 15-20 kN/m^3, only about little more than 50% can be placed back into the underground voids. The rest of the tailings have to be sent to the tailings dams or disposed on the surface. Backfills are placed in stopes, which comprise the excavated volumetric unit holding approximately the shape of a rectangular prism, which is subsequently backfilled with some type of waste material. Therefore, any waste materials that are placed into the underground voids (stopes) in the mine are referred to as backfills or minefills. A schematic diagram of a mine stope with access drains is shown in Fig. 2. While serving as an effective means for the tailings disposal, the backfilling process improves the stability of the surroundings, facilitating the mining activities including excavation for ore removal in the nearby areas. Not taking adequate care in the tailing disposal in the underground mine stopes can result in catastrophic accidents that can include fatalities. Mine accidents are reported worldwide, and often these are due to the breach of barricades that prevent fills from flowing into the drives.

3. Main Types of Mine Backfills

In mining engineering, backfill refers to any waste material that is placed into the voids mined underground (stopes) for the purposes of either disposal or to perform some engineering function. Backfills that are used only to fill the voids created by mining need only to have sufficient strength to prevent any form of remobilisation through liquefaction, typically caused by dynamic loading. However, where backfills are used as engineering materials, sufficient strength is required to ensure stability during exposure during ore pillar mining in tall vertical faces or undercuts, particularly in the case of paste fills or other cemented fills. In addition to the excavated rock, other forms of backfill are commonly used such as surface placement of hydraulic fill in a tailings dam or discharge of paste fill from a reticulation pipe underground, as shown in Fig. 3.

Backfills can be divided into two broad categories, cemented or uncemented. Cemented backfills generally include a small dosage of pozzolanic binder such as cement, fly ash etc. to improve the strength. This includes cemented rock fills (CRF), cemented aggregate fills (CAF), cemented hydraulic fills (CHF) and paste fills (PF). The uncemented aggregate fills can be in the form of hydraulic fills (HF), rock fills (RF), sand fills (SF) and aggregate fills (AF). Uncemented backfills, as the name suggests, do not use any binding agents mixed in with the filling material. The mechanical behavior and performance of uncemented backfills can thus be studied using soil mechanics theories. Hydraulic fills are the most common uncemented backfills used world-wide. These are sandy silts or silty sands, with no clay fraction, classified in the Unified Soil Classification System (USCS) as ML or SM. The fine fraction is removed by a process known as desliming. Rock fills (RF) are produced by crushing rock to grain sizes of 25- 300 mm. Materials finer than 25 mm that has been rejected from RF production is described as Aggregate Fill (AF).

Cemented backfills incorporate the use of a small amount of binder material, normally Portland cement, or a blend of Portland cement with other pozzolans such as fly-ash, gypsum or blast furnace slag to the parent backfill material to produce a binding agent for the fill. Cement Hydraulic Fill (CHF) is the most common type of cemented backfill. CHF is produced by the addition of 3-5% cement to deslimed mill tailings, which have the grain size distribution very similar to those of hydraulic fills. CHF is the most similar form of backfill to paste fill with the most significant difference being the larger grain size distribution of CHF when compared to paste fill. Typically in CHF all tailings particles are less than 420 μm and have been deslimed (Bloss 1992), whereas paste fill utilizes the very fine frac-

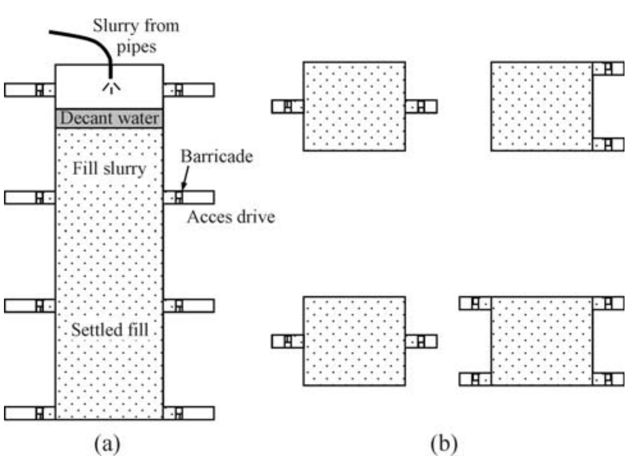

Figure 2 - Schematic diagram of a mine stope with access drains: (a) front view of a stope; (b) possible drain layouts in plan view.

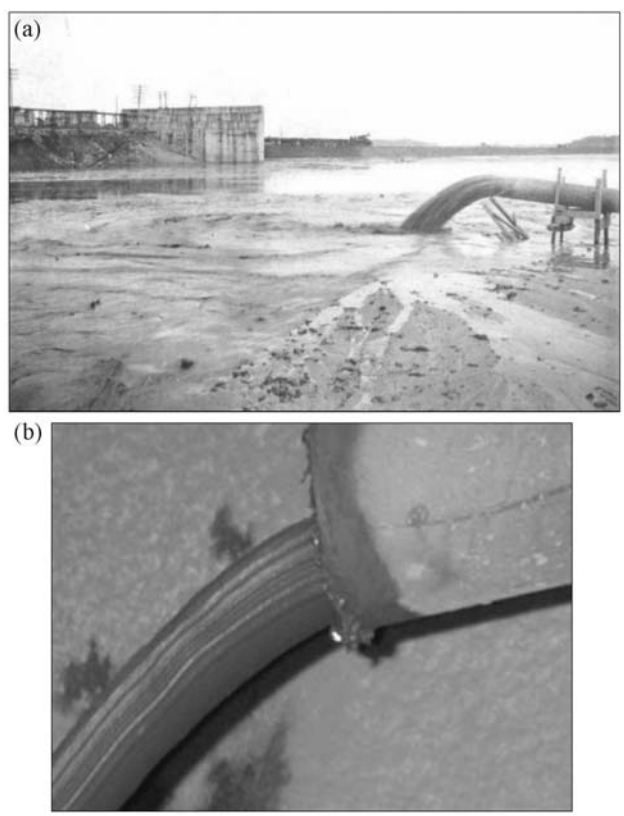

Figure 3 - Various forms of backfill: (a) surface placement of hydraulic fill in a tailings dam; (b) discharge of paste fill from a reticulation pipe underground.

tion of the tailings that provide large specific surface areas and absorb the excess water within the slurry. The grain size distribution of paste fills is significantly finer than CHF and contains a minimum of 15% of particles smaller than 20 μm.

Cemented rock fills (CRF) are prepared by transporting the rock fills to the stope and mixing with cemented hydraulic fills at the ratios of 1:1 to 3:1 (RF:CHF), by weight. The properties of CRFs vary significantly within the stope as the two fills segregate during placement. The ratio of RF:CHF at any location is the dominant factor of the fills behavior at that point (Bloss 1992). As with CRF, AF are mixed with CHF at a ratio of approximately 1:3 AF:CHF by weight. The resulting fill is termed Cemented Aggregate Fill (CAF). CAF typically suffer from segregation during placement, and thus properties at any location within a stope are again dependent on the ratio of AF:CHF at that point as in the case of CRF (Bloss 1992).

Paste fill is the newest form of mining backfill. It is produced from the full mill tailings and has a much finer grain size distribution than any other form of backfill. Typically it has a minimum of 15% of the material smaller than 20 μm, and the maximum grain size in paste fill is between 350-400 μm. Hydraulic fills and paste fills, the two most popular backfills used world-wide, are discussed in detail below.

4. Hydraulic Fills

Hydraulic fills are granular soils with no clay fraction. The fines are removed through the desliming process using hydrocyclones. Rankine *et al.* (2006) and Sivakugan *et al.* (2005) summarised the geotechnical characteristics of the Australian hydraulic fills, based on an extensive laboratory testing program carried out at James Cook University to study more than 25 different hydraulic fills representing all major mines in Australia. It was shown that the grain size distributions for all the fills fall into a narrow band, as shown in Fig. 4. The specific gravity of the grains can be in the range of 2.8-4.5 due to the presence of heavy metals. Since the tailings are fresh from the grinding process, the grains are often very sharp and angular, giving higher friction angles than those for natural soils. A scanning electron micrograph of a hydraulic fill sample is shown in Fig. 5, where the angularities of the grains can be seen. All hydraulic fills, settling only under self-weight, manage to settle to rather high relative densities of 50-80% and porosities of 37-49%, and to dry density (tf/m^3 or gf/cm^3) of 0.57 times the specific gravity or dry unit weight (kN/m^3) of 5.7 times the specific gravity, what is equivalent to dry unit weights in the range of 12 to 25 kN/m^3.

The hydraulic fills are initially transported to the stope in the form of slurry, through pipe lines and bore holes, at solid contents of 65-75%, corresponding to 33-54% water contents. The drives are blocked by a barricade wall, made of special porous concrete bricks, which

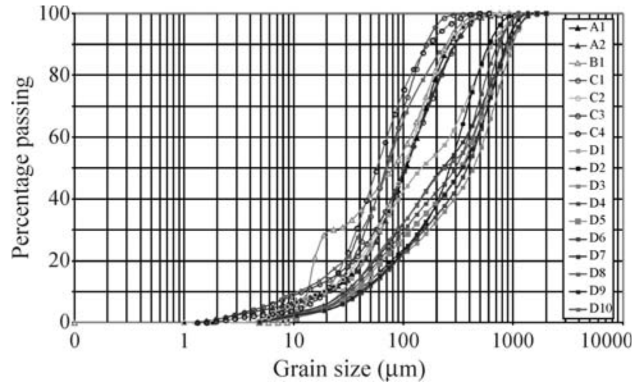

Figure 4 - Grain size distribution curves for various hydraulic fills from Australian mines, Rankine *et al.* (2006) and Sivakugan *et al.* (2005).

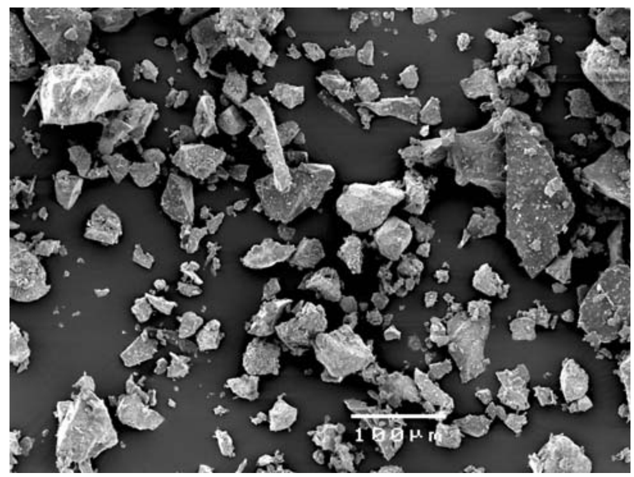

Figure 5 - Scanning electron micrograph of a hydraulic fill.

retains the settled fill and allows the water to drain through. Filling of the stope does not always occur continuously. For example, depending on the site constraints, the filling schedule can be 12 h fill and 12 h rest, continued till the stope is filled. The drainage starts as soon as the filling commences, and continues for weeks, well after the filling ends. At the time when the drainage seems to have finished, the hydraulic fills still have a residual water content, typically about 20-30%, and this residual water remains within the fill indefinitely.

4.1. Drainage issues

Drainage is the prime consideration in the design of hydraulic fill system for a stope. In the absence of good drainage, pore water pressure builds up and reduces the effective stresses within the stope, causing liquefaction, which is one of the prime causes of the barricade failures of hydraulic fill stopes. Any breach of barricades takes place when the hydraulic fill is wet and therefore, every attempt should be taken to get the water out as quickly as possible.

Several trials are generally carried out to arrive at a suitable grain size distribution that will give good drainage characteristics. Herget & de Korompay (1978) suggested the value of 100 mm/h as the minimum hydraulic conductivity required for the hydraulic fill in the mine to perform satisfactorily. Grice (1998) suggested that ensuring D_{10} value greater than 10 μm will ensure adequate drainage throughout the fill. Nevertheless, more than 25 different hydraulic fills, tested at James Cook University laboratory had hydraulic conductivity values in the range of 1-40 mm/h and all these stopes performed satisfactorily. All these fills had D_{10} values in the order of 10-40 μm, satisfying Grice (1998) recommendation. This shows that Herget & de Korompay's threshold value of 100 mm/h is too conservative. Rankine *et al.* (2004) measured the hydraulic conductivity of the barricade bricks using a unique permeameter, under one-dimensional flow conditions as in the barricade wall in the mine. The results showed that the hydraulic conductivity of the special porous brick is 2-3 orders of magnitude greater than that of the hydraulic fills. This enables the fill-barricade boundary to be modelled as free-draining in numerical modelling. The laboratory test procedures and the apparatuses are discussed by Sivakugan *et al.* (2006a).

Isaacs & Carter (1983) developed the first computer model, for two-dimensional stopes, to simulate the hydraulic filling schedule and to monitor the heights of water and tailings within the stope, the pore water pressures, and the discharge through various drains. Rankine *et al.* (2003) extended these to three dimensions and showed that, for the usual stope sizes, two dimension models can simulate satisfactorily the flow in the central portion of the stope. The flow nets of stopes with single and multiple sub-level drains, as obtained by numerical analyses, are shown in Fig. 6. It can be seen in Fig. 6 b that most of the flow takes place through the bottom drain, and this has been observed in the mines. Therefore, for practical purposes, the upper level drains can be neglected, and the stope can be analysed with only one drain at the bottom. Sivakugan *et al.* (2006b) proposed a simpler model with closed form solutions, based on the method of fragments (Harr, 1962), to estimate the maximum pore water pressure and discharge in a two-dimensional stope with a single drain at the bottom. Research is currently underway to extend this closed form solution to 3-dimensions.

5. Paste Fills

Paste fill falls into the broad category of thickened tailings, a concept which was introduced by Dr. Eli Robinsky in the mid 1970's while describing surface disposal of concentrated tailings using pipeline reticulation (Robinsky 1975, 1978). However the first true "paste" backfill was produced at the Bad Grund Mine in Germany in 1979. Acceptance of paste backfill, as a viable alternative to hydraulic slurry and rock fill, did not truly occur until the mid 1990's with the construction and successful operation of several paste backfill systems in Canada and Cannington Mine in Australia in the late 1990's.

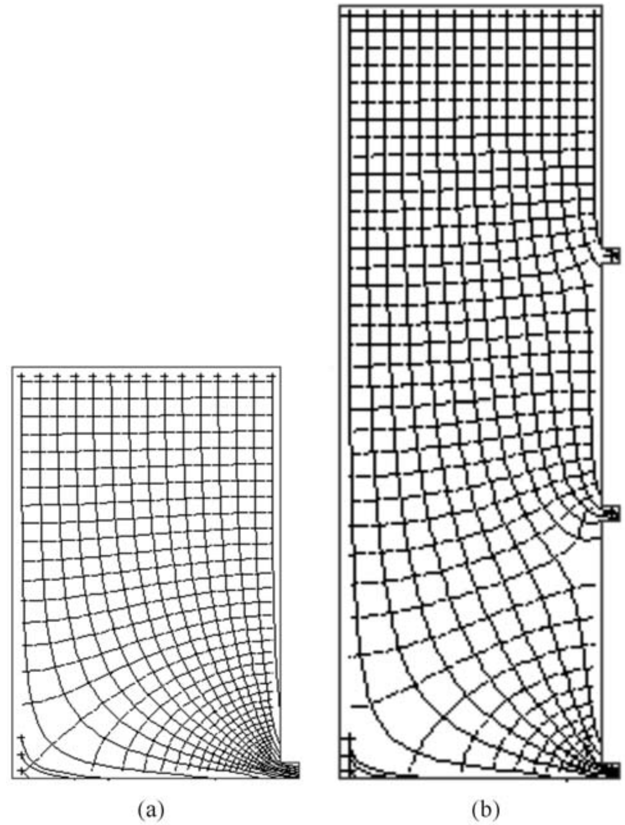

Figure 6 - Flow nets in 30 m wide stopes with one and three drains: (a) single drain; (b) multiple drains.

The definition of "paste backfill" has been one of great debate since its inception in the late 1980's. Primarily because a number of different industries were involved in the evolution of paste and the definition which is adopted by each industry reflects their respective needs and experiences. In an attempt to unify the various definitions of "paste", Fig. 7 was devised and is formally recognised by a number of industry experts and academics (Jewell *et al.* 2002). It should be noted that the boundaries

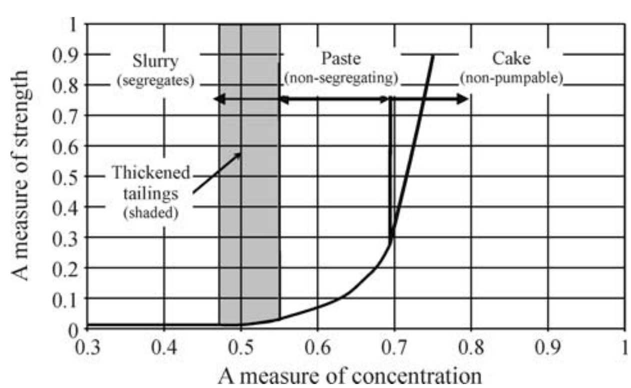

Figure 7 - Concept of thickened tailings continuum (Jewell *et al.*, 2002).

between the mediums are not defined at specific levels or concentrations. Rather they depend on a number of physical and material characteristics of the tailings materials. The shaded area represents the backfills that are commonly referred to as "thickened tailings". Thickened tailings are a special case of slurry tailings and tend to show many similar characteristics to paste. The similarity of thickened tailings and paste, while moving, is the basis for thickened tailings being commonly confused with or wrongly identified as paste. The primary difference between thickened tailings and paste is that thickened tailings will segregate or settle out once a minimum velocity is reached. A more detailed description of the characteristics which define the difference between slurry, thickened tailings and paste is given in Table 1. When referring to Fig. 7 "thickened tailings" is typically referring to the shaded portion of the graph.

Paste fill rheology closely conforms to the Bingham plastic flow model (Rankine 2004), which is strongly non-Newtonian in its behaviour. Fig. 8 shows a number of fluid models and plots the change in shear stress that is experienced as a function of the shear rate. Fluids which exhibit

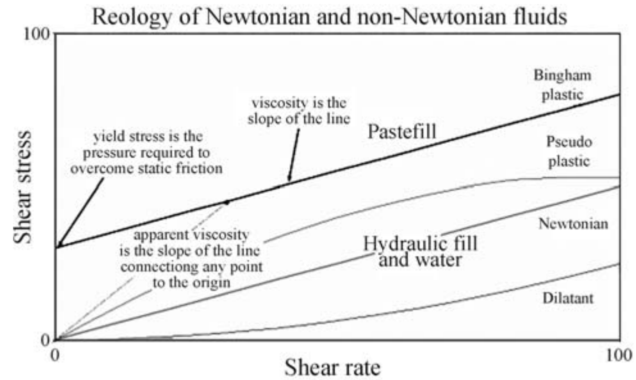

Figure 8 - Rheological curves for particulate fluids.

Newtonian flow characteristics have no "yield stress" to be overcome to initiate movement and have a constant viscosity. Viscosity is defined as the rate of rise of shear stress with the increase in shear rate. Water is an example of a Newtonian fluid. Fluids exhibiting non-Newtonian characteristics are the Power Law (pseudo plastic and dilatant) flow and Bingham plastic flow regimes. Pseudo-plastic fluids are characterised by the reduction of the viscosity with

Table 1 - Material properties for thickened tailings continuum (Jones 2000).

Material property	Slurry	Thickened tailings	Paste
Particle size	Coarse fraction only. No particles less than 20 µm. Segregation during transportation and or placement is dependent only on the coarse fraction	Some fines included (typically < 15%), Fines content tends to modify behaviour from slurry - *i.e.* rheological characteristics more similar to paste, however does segregate when bought to rest. Segregation during transportation and or placement is dependent only on the coarse fraction	Additional / most fines (typically 15%(min) > 20 µm
Pulp density	60%-72%	70%-78%	78%-82%
Flow regimes/ line velocities	Critical flow velocity. To maintain flow must have turbulent flow ($v > 2$ m/s). If $v < 2$ m/s settling occurs. Newtonian flow	Critical flow velocity. To maintain flow must have turbulent flow ($v > 2$ m/s). If $v < 2$ m/s partial settling occurs. Newtonian flow	No critical pipeline flow velocity. *i.e.* no settling in pipe. Laminar/ plug flow
Yield stress	No minimum yield stress	No minimum yield stress	Minimum yield stress
Preparation	Cyclone	Cyclone end elutriation	Filter/ centrifuge
Segregation in stope	Yes/high	Slight/partial	None
Drainage from stope	Yes	Partial/limited	None/insignificant
Final density	Low	Medium/high	High
Supernatant water	High	Some	None
Post placement shrinkage	High	Insignificant	Insignificant
Rehabilitation	Delayed	Immediate	Immediate
Permeability	Medium/low	Low	Very low
Application	Above ground	Above ground	Above ground and underground
Water consumption	High	Medium	Low
Reagent recovery	Low	Medium	High

an increased shear rate (shear thinning), whereas the viscosity for dilatant fluids increases with an increased shear rate (shear thickening).

Bingham plastics exhibit a significant shear stress that must be overcome before movement (shearing) commences. This value of shear stress is commonly referred to as "yield stress". Once shearing has commenced, the viscosity remains approximately linear. The key determinant of the rheological properties of paste fill is the yield stress and can be shown to be exponentially proportional to the solids density of the mix before the effect of cement is realized.

Pullum (2003) suggests that there are effectively two separate forms of paste: homogeneous and heterogeneous paste. The heterogeneous pastes satisfy the minimum rule of thumb of 15% finer than 20 μm. However, during transport, Pullum (2003) has shown stratification of paste during pipe flow with all paste fills with a maximum grain size of over 20 μm. Paste fills with a maximum grain size of under 20 μm tend to form homogenous paste fills during both transportation and deposition.

6. Stress Developement in Backfilled Stopes

The most important issues regarding the stability of backfilled stopes are the failure of the barricade walls and the self supporting ability of the fill mass when exposed.

6.1. Failure of the barricade walls

Catastrophic failure of a barricade results in the inrush of material into the mine workings, which is commonly referred to by miners as a "mud rush" (*i.e.* liquefaction). Also, the barricade can fail under static loading, when the paste lateral stress overcomes the barricade yielding strength. Prior to the curing of the cement, backfill materials have very little self supporting ability. Subsequently, during the backfilling process, an isotropic stress condition equal to the product of the fill's unit weight and gravity develops. As the cement cures, fibrous bonds form between the fill particles and the shear strength of the fill increases. The ability of the fill to sustain and transfer loads through shear stress increases until such time as the cemented fill is strong enough to support the self weight and any additional loads placed on top of the fill. At this point in time the vertical shear stresses acting in the stope walls reach the maximum value, as a result of arching. Accordingly, the vertical normal stresses at a point within the hydraulic fill stope can be substantially less than what is estimated as the product of the depth and unit weight. Therefore, a significant fraction of the fill weight is carried by the rock walls in the form of shear stresses. Similarly, arching causes the lateral stress at the barricade to reach its maximum value when the paste cures. To ensure structural integrity of the barricade, its ultimate strength must be greater than the applied lateral earth pressures at all times.

6.2. Self supporting ability of the fill mass when exposed

The backfill must have sufficient strength to prevent collapse when exposed during the mining sequence, as the surrounding ore is sequentially removed and the backfill is unsupported. Static failure is deemed to have occurred when the principal applied stress is greater than the unconfined compressive strength of the fill mix. This may be conservative for the confined fills within the center of the stope, but provides a reasonable approximation to the fill on the face of the exposure.

To evaluate the risk associated with either form of failures, it is necessary to develop a thorough understanding of the stress developments within the minefills. A closed form solution for estimating the average vertical normal stress ant any depth within a narrow stope containing a cohesionless soil was developed originally by Marston (1930), and later extended by Terzaghi (1943) to include cohesive soils. The general equation to determine the average vertical normal stress (σ_v) at a depth of h within a fill contained in a narrow stope is:

$$\sigma_v = \frac{(\gamma B - 2c)}{2K \tan\delta}\left[1 - \exp\left(\frac{2Kh\tan\delta}{B}\right)\right] \tag{1}$$

B in Equ. (1) is the stope width, γ is the unit weight of the fill, c is the cohesion at the fill-wall interface, δ is the friction angle between the fill and the wall, and K is the ratio of horizontal to vertical normal stress. The corresponding horizontal normal stress is given by:

$$\sigma_h = K\sigma_v . \tag{2}$$

Marston (1930) assumed K is the same as Rankine's active earth pressure coefficient, and δ ranging from 0.33 to 0.67 times ϕ, where ϕ is the friction angle of the fill. Terazghi assumed $\delta = \phi$, and $K = 1/(1+2\tan^2\phi)$, which gives a slightly higher value for K than the coefficient of earth pressure at rest K_0. Aubertin *et al.* (2003) suggested a wide range of values for K, from the active to passive earth pressure coefficients. Intuitively, neglecting the yielding of the rock walls, the coefficient of earth pressure at rest seems more plausible. Accordingly, Pirapakaran & Sivakugan (2006) compared the above values with numerical model predictions, and suggested that $\delta = 0.67\phi$ and $K = K_0$ in the above equations would match numerical predictions and measured results better. They also extended the above equations for rectangular and circular stopes, containing cohesionless fills.

Numerical models appear to be quite effective in predicting the stress developments within the minefills. The variation of the vertical normal stress with depth, along the vertical centre line, as obtained from numerical modeling, for a 30 m wide very long stope and a 30 m diameter circular stope of 150 m height is shown in Fig. 9. Pirapakaran & Sivakugan (2006) compared the vertical stress values

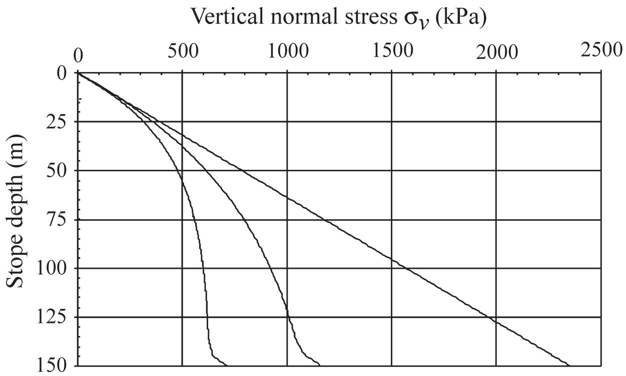

Figure 9 - Variation of vertical normal stresses along the centre line of narrow, circular and square stopes (Pirapakaran & Sivakugan, 2006).

within a square and circular stope of same dimensions and found them to be very close. They used numerical, analytical and laboratory model tests to demonstrate this point.

7. Summary and Conclusions

The use of backfills in underground mining has become a more critical issue in the modern era of the "resource boom". Increased commodity prices and production demands have increased mining rates and required filling rates to record levels. Backfill technology has more capital investment focused into it now than ever before. To ensure that backfilling carried out in underground mines is conducted in a technically proficient and economically attractive manner, a review of the current practice was deemed appropriate, in addition to a brief review of the mining industry in the wider global sense. In addition to this the overall environmental and safety issues in relation to backfilling were reviewed, highlighting the need to learn from historical events.

A variety of backfill types were reviewed and specific focus placed on the two most popular forms of backfill: hydraulic and paste fill. Both were reviewed and the critical issues of stability, drainage, and rheology identified for the hydraulic and paste fills respectively.

The vertical stress development in the fill masses was investigated in relation to the stability of backfill barricades and the static stability of backfilled stopes when exposed. The current analytical solutions were compared against numerical modelling techniques which were found to be an effective means of predicting vertical stress within a backfill stope.

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