A Review of Underground Mine Backfilling Methods with Emphasis On Cemented Paste Backfill

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ABSTRACT
Mining industry is a demanding sector for human advancement. Scarcity of economic minerals near to ground surface has increased deep underground mining. Safety and environmental factors have caused mining companies consider backfilling of mine wastes to avoid mine collapse in further and deeper extraction phases, ground subsidence in mine abandoning and environmental pollution. In this paper main methods in backfilling underground mines with an emphasis on cemented paste backfill method have been reviewed. It was concluded that selection of backfilling method should be performed based on further goals, available equipment and material and economic factors. Also, cemented paste backfill technology is a developing method that can provide better safety factor for underground mining environments and suitable preventive method for environment pollution by disposing toxic waste minerals underground.

BACKGROUND ON THE DIFFERENT TYPES OF BACKFILL

Underground mining of economic minerals creates voids in different shapes including stope, cave, room, goaf, and gob void forms. These underground voids create instability hazards for extraction of adjacent pillars that contain economic minerals, or subsidence hazard for infrastructures on the ground during operation, or after mines are abandoned. Therefore during historical development of mining technology, several methods of refilling or backfilling for those voids have been developed. Some of the common methods of backfilling which are used according to economic factors and further goals such as development of mine or abandoning the mine are rock backfill, hydraulic backfill, cemented paste backfill, and silica alumina-based backfill methods.

Rock Backfill Method

When extraction of economic minerals considers deeper ground, number and volume of voids including stopes, goaf, and goab increases conventionally, and waste rocks are directly dumped on the ground surface mining fields (Castro-Gomes, 2012, Wang et al. 2013). Disposal of waste rocks has been a challenge due to their large volumes and containment of contaminative metal elements. Winds can spread dust and micro particles, and rain can create leachate from heavy and toxic elements stored in dumped rocks and pollute the hydrologic and hydrogeologic environments (Smuda et al. 2007; Poisson et al. 2009; Wang et al. 2013). The voids created from mining can induce instability and settlement hazards. Collapse of the created voids including stopes, goaf, and etc. is one of main factors in rock burst and potential land subsidence (Jiránková, 2007; Helm et al. 2013; Wang et al. 2013).
Rock backfill can be described as a technology for transportation of backfill forming components such as stone, gravel, soil, industrial solid waste using manpower, gravity or machinery equipment to fill underground mined voids and production of compressed backfill body. Backfill material are usually produced from waste rocks by crushing, sieving and mixing by machinery equipment by taking the particle size distribution pattern into account (Yao et al. 2012). Based on equipment that are used for transfer of filling materials, backfill scheme can be divided into three Schemes. (I) side-dump tramcar, (II) belt conveyor and (III) combination of truck and scrapper, Table 1 (Wang et al. 2013)

<table>
<thead>
<tr>
<th>Scheme</th>
<th>Applicable areas</th>
<th>Advantages</th>
<th>Disadvantages</th>
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<tr>
<td>I</td>
<td>gobs small in size, with good stabilities</td>
<td>(a) easy for operation;</td>
<td>(a) high labor intensity;</td>
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<td></td>
<td>filling workforce near hanging wall</td>
<td>(b) few devices input</td>
<td>(b) low efficiency</td>
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<td>II</td>
<td>nearby hanging wall</td>
<td>(a) high capacity;</td>
<td>(a) expensive devices;</td>
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<td></td>
<td>(b) flexible operation;</td>
<td>(b) complex structure</td>
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<tr>
<td>III</td>
<td>Mines with truck Transportation</td>
<td>(a) high capacity;</td>
<td>(a) critical tire wear;</td>
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<td></td>
<td></td>
<td>(b) wide application range;</td>
<td>(b) gas pollution</td>
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One of the cases that rock backfill is selected as a feasible and economic way and applied to treat excavated gobs underground is White Bull mine in China. Locomotive traction and tramcar haulage were used for waste rock transportation (Wang et al. 2013).

![Figure 1: An example of rock backfill used for underground goaf refilling in China, one of the goafs that intersects with level drift is shown above (Wang et al. 2013).](image)

Rock backfill method is an economical backfilling method which is feasible and applicable in some but not all cases in underground mining. This method decreases waste material on the surface and expands usable land on the ground, decreases environmental pollution by transferring waste rocks to deeper levels out of rain contact, increases stability of mined area and decreases land subsidence and rock bursts due to stress pattern changes.

## Hydraulic backfill Method

Hydraulic backfill is one of the refilling technologies that use water as the transportation medium to convey the hydraulic backfill materials, such as waste tailings, water hydrophilic slag, mountain
sand, river sand, and crushing sand to fill underground mined voids like stopes (Yao et al. 2012). An example of hydraulic backfill is shown in Figure 2. The access drives to mine are blocked with bulkheads and a drainage system is inserted into the bulkhead so water from backfill drains out of the stope. Hydraulic backfill material is poured inside the stope from fill hole in the crown of stope. While the height of hydraulic backfill material is increased, free water is accumulated on top of the backfill. All water should be drained through bulkheads or as a seepage water or as decanted water (Potvin et al, 2005). Hydraulic backfill has following properties, maximum particle size is less than 1 µm and very fine particles are removed generally to create permeability in the backfill material. Therefore particles with a size less than or equal to 10 µm should be less than 10 percent and in most cases much lower than that in total (Potvin et al, 2005). Slurries for hydraulic backfill have densities between 40-50% (Solid parts volume). Permeability of hydraulic fill material should be between $10^{-5}$ to $10^{-6}$ m/s. Excess water in backfill material is drained by gravity, and assisted by drainage out of the backfill. Porosity of emplaced hydraulic backfill is 50%, but porosity of 30% has been reported. Hydraulic backfill can be placed cemented or non-cemented, which the later one is one of the cheapest methods in mine when small size waste particles are available (Potvin et al, 2005). Strict rules should be applied in design of backfill and barricades, and controlling of backfill material properties. Increasingly cemented paste backfill is being replaced for hydraulic backfill where strength is required from backfill or waste material contain higher amount of very fine particles (Potvin et al, 2005). Figure 3, shows grain size distribution of 20 hydraulic backfill materials from Australian mines along with cemented hydraulic backfill, and paste backfill materials grain size distribution. It can be seen that hydraulic backfill material fall in narrow band along with cemented hydraulic backfill material. Effect of cement on the grain size distribution is limited. In the paste backfills generally fine fraction is larger than hydraulic backfills or cemented hydraulic backfills, but their colloidal fraction (finer than 2 micrometer) is negligible.

**Figure 2:** Examples of hydraulic backfill in open stope (Potvin et al, 2005; Sivakugan et al, 2006).
Hydraulic backfill grain distribution usually contains Sandy silt, Silty sand (SM-ML). The clay fraction is removed by a process named desliming, in which entire backfill material pass through hydrocyclones by circulation and the clay fraction is removed and then deposited into the tailings dam. Hydrocycloned silty sandy backfill material is transferred in the form of slurry by pipelines to underground voids (Sivakugan et al, 2006). However, more solid material reduces water content in drained status in the hydraulic backfill, but creates problem in smooth transfer through pipes. In current practice 75-80 % of content for solid particles is common but considering 75% solid particle with specific gravity of 3, almost 50% of slurries volume would be water which requires drainage facilities by using special porous bricks in building barricades in front of drawpoints or horizontal drives (Sivakugan et al, 2006). Drainage is the most important factor in designing hydraulic backfills, because negligence in this section has caused several deadly incidents due to liquefaction, rush in, and piping problems, and not using porous barricades (Bloss and Chen 1998; Tolarch, 2000). Threshold for permeability of hydraulic backfill material should be more than 100 mm/h and higher values accelerate drainage of backfilled stope (Herget and De Korompay, 1978).

![Figure 3: Grain size distribution of hydraulic backfill (from 20 mines in Australia) and cemented hydraulic and paste backfills (Sivakugan et al, 2006).](image)

Laboratory tests and field monitoring by other researchers showed that threshold suggested by Herget and De Korompay (1978), is conservative. In other words permeability in the range 7-35 mm/h by Sivakugan et al., (2006) in controlled laboratory environment produced satisfactorily results in the stopes due to much higher values that happen in the in-situ conditions. Kuganathan (2001) and Brady and Brown (2002), suggested that permeability values between 30-50 mm/h are significantly larger than values measured in the controlled laboratory environment for similar backfills. Strength and stiffness of hydraulic backfill is related directly to relative density of backfill material and can be determined by laboratory and in situ tests. Laboratory tests are desirable due to low cost and ease of performance. Maximum and minimum dry density tests, direct shear test, odometer test are other laboratory tests that can be used for strength and stiffness determination of hydraulic backfill material (Sivakugan et al, 2006). Barricade bricks that are used for blocking hydraulic backfill in underground mines are specially constructed from mortar mixed from gravel, sand, cement and water at the ratio of 40:40:5:1 approximately and respectively. Conventionally, the barricade walls constructed in a vertical plane position, but recently construct them in a curved mode to increase its strength, with the convex
position toward the hydraulic backfill (Sivakugan et al, 2006). Two photos from barricade bricks and walls can be seen in figure 4. Although the manufacturers of the barricade bricks often guarantee a minimum strength of 10 MPa, but the average uniaxial compressive strength of dry bricks tested by (Sivakugan et al, 2006) fell between 6 and 10 MPa. Also compressive strength of brick is reduced in order of approximately 25%, after wetting the brick. No important difference was observed between 7 and 90 days wetting, concluding that strength loss occurs immediately upon soaking.

![a](image1)
![b](image2)

Figure 4: (a) Porous barricade brick and (b) Curved barricade wall (Sivakugan et al, 2006).

An example of hydraulic backfill is backfilling voids that are created in room and pillar or bord and pillar underground mining methods in coal mines in Wyoming region in USA. It was applied to prevent surface subsidence in the form of sinkholes or trough subsidence in abandoned coal mines in Wyoming area (Karfakis and Topuz 1991). 100% of the voids in Hannah mine in Wyoming region were filled by hydraulic backfill using granular material from abandoned coal mines spoils. Some results were obtained from hydraulic backfilling in Wyoming area, showed that hydraulic backfilling gives better results in voids with levels under groundwater table than dry voids, and mining areas with lower rubblized zones accept more backfill material by easier water drainage conditions (Karfakis and Topuz 1991). Cemented hydraulic fill shows higher compressive strength for structural support in underground mining (1 MPa for cut and fill, and 5–7 MPa for pillar recovery) (Potvin et al, 2005). Cement can be used up to 16% to reach maximum stiffness. Two methods of emplacement can be applied: I) mixing completely in hopper and emplacing hydraulically, II) percolation of cement slurry over mixture of coarser mineral that previously emplaced, by considering conditions in which backfill contains a mixture of <150 mm coarse grain sizes, and fine <10 mm grains treated with cement slurry and water in a 1.2:1.0 weight ratio (Dudeney et al, 2013).

**Arching effect in Hydraulic backfilled voids**

Arching generally happens when portions of frictional material give up while the adjacent materials remain in place. As the yielding material moves inside the adjacent firm and unyielding walls, the relative movement in the yielding material is contrasted by shear resistance along the boundary and the shear stress accumulated along the contact surface keeps the yielding materials in their original place. By this mechanism, the vertical normal stresses inside the yielding material creates a status which is known as arching (Pirapakaran and Sivakugan, 2007). The FLAC analysis by Pirapakaran and Sivakugan (2007), were performed and compared with analytical solutions developed previously by different researchers, after modification of them to study the arching effects in hydraulic backfilled material in narrow and circular stopes. Generally five different analytical methods exist for
arching phenomenon which includes (I) Free Standing Vertical Face (Grice 2001), (II) Vertical Slope (Grice 2001), (III) 3-D Sliding Wedge Failure (Mitchell et al. 1982), (IV) Simple Arching Theory and its Modifications (Marston 1930; Terzaghi 1943; Aubertin et al. 2003; Li et al. 2003), and (V) Modified Simple Arching (Winch 1999). For numerical modelling of arching effect, Pirapakaran and Sivakugan (2007), assumed that rockmass is homogenous, isotropic and linear elastic material and the backfill material is Mohr-Coulomb material. In construction of the model no interface elements were considered. The stope dimensions, rock and backfill properties can be seen in figure 5. Rock region were allowed in the model to obtain equilibrium under its self-weight, then stope were mined to an equilibrium state using FLAC calculation. Hydraulic backfill is emplaced in different numbers of layers showed an improvement compared to Li et al. (2003) model. For other considerations undertaken by Pirapakaran and Sivakugan (2007), for initial stress and strain, reader can refer to their paper. Stopes with dimension of 10 m width and 60 m height with same material properties for further modeling were considered. The vertical (a) and horizontal (b) stress pattern in the stope and adjacent rocks when the entire backfill was emplaced immediately are shown in figure 6. The results for the computed models are almost identical to those that will be obtained from Li et al. (2003). These models show a non-consistent outline across the stope width (Pirapakaran and Sivakugan, 2007).

![Figure 5: Scheme of stope and rock and backfill material properties used for numerical modeling of arching (Pirapakaran and Sivakugan, 2007)](image-url)
The effect of adding backfill layers in one layer, half portions, quarter portions and 1 m layers on vertical stress change with depth along the stope center line is shown in figure 7. By scrutinizing figure 7, it can be seen that the vertical stress surpasses the overburden stress during backfilling the stope in one layer, particularly in the one-third upper part of stope. This effect has decreased considerably while backfilling is done in two layers and then has entirely disappeared by backfilling in many layers. The results from these models show significantly better improvement compared to Aubertin et al. (2003) & Li et al. (2003) numerical models, in which the vertical stress does not go beyond the overburden pressure while stope is backfilled in more than four layers in the numerical modelling.

Results from other numerical modelling showed that stresses for all friction angles of backfill material were same in the upper third of the stope, but decreased a little for friction angles of 40 and 45 degrees in the range of 1/3 and 4/5 of stope’s height. At the bottom level of stope, the stress change
was entirely different when compared to results obtained from analytical methods and further research is necessary for understanding it (Pirapakaran and Sivakugan, 2007).

**A novel silica alumina-based backfill**

A research was performed by Yao and Sun (2012) to study a novel silica alumina-based backfill material composed of coal refuse and fly ash. Coal industry is one the fast growing fuel industry which produces large amount of waste material in the coal refuse form. The coal refuse and fly ash behave differently in different thermal activation temperatures (20 °C, 150 °C, 350 °C, 550 °C, 750 °C and 950 °C). Flowability and pozzolanic properties of the coal refuse considerably are enhanced with an increase in the thermal activation temperature changing from 20 °C to 950 °C, but flowability of fly ash is reduced in activation temperature more than 550 °C due to intense agglomeration conditions on its surface (Yao and Sun, 2012). An optimized design for the backfill material was obtained i.e. 5% coal refuse at 750 °C and 15% fly ash at 20 °C in activated portion. Research showed that this optimum ratio gives the best flowability, a high compressive strength and a low bleeding rate in backfill material. Figure 8 shows flowability and strength properties of coal refuse and fly ash in different thermal activation temperatures (Yao and Sun, 2012). Mineral phase change of the coal refuse and fly ash in different thermal activation temperature was investigated by microanalysis. Results of this analysis showed decrease in kaolinite peaks in coal refuse at 550 °C, due to kaolinite conversion to metakaolin, disappearance of chloride peaks at 750 °C, decrease of the muscovite peak at 750 °C and totally disappearance of the muscovite peak at 950 °C. Muscovite gradually experiences de-hydroxylation from 2M1 type to muscovite HT type during this temperature. XRD results also showed that the hematite peaks after thermal activation, particularly at 950 °C becomes more apparent. Important mineral phase change in fly ash during the thermal activation was not observed (Yao and Sun, 2012). Phase changes in coal refuse and fly ash minerals are shown in Figure 9.

![Figure 8: Presentation of the backfill material properties. (a) Flowability of the coal refuse-based backfill material, (b) Flowability of fly ash-based backfill material, (c) compressive strength of the coal refuse-based backfill material, and (d) compressive strength of fly ash-based backfill material.](image-url)
strength of the coal refuse-based backfill material and (d) compressive strength of the fly ash-based backfill material (Yao and Sun, 2012).

![Figure 9: XRD of (a) coal refuse and (b) fly ash at different activation temperatures (Yao and Sun, 2012).](image)

Hardened backfill structure was tested in different places for toxicity leaching attributes, and none of the tested elements went above the EPA limitations, representing that the silica alumina-based backfill material is environmentally friendly. Just 1% cement added to experimented silica–alumina backfill material, and the rest of components were industrial solid waste, which shows vast potential for saving in capital expenses in the backfill industry (Yao and Sun, 2012).

**Cemented paste backfill**

Cemented paste backfill (CPB) is a non-homogenous material made by mixing waste tailings, water, and cement. Waste solid proportion is between 70% and 85%, water is either clean water or mine processed water, and a hydraulic binder which is added usually between 3–7% of total weight. CPB is one of new and fast growing waste management and backfill technologies in mining industry, which created economical way of providing support and safe working place in underground mining development beside its environmental benefits (Grice, 2001; Kesimal et al., 2003; Fall et al., 2005; Fall et al., 2008; Ercikdi et al, 2009; Pokharel and Fall, 2013; Ghirian and Fall, 2014). Cured and hardened CPBs in stopes create strengthened pillar type ground support for the adjacent mining operation in underground mines and safe working surroundings for workers in the underground mines (Grice, 2001; Kesimal et al., 2003; Yilmaz et al, 2004; Fall and Pokharel, 2010; Mahlab et al, 2011; Pokharel and Fall, 2013; Ghirian and Fall, 2013).

**PREPARATION OF CPB AND TRANSPORT OF CPB**

Underground backfill materials can be from different types of aggregate and binders with different preparation, transportation, and placing methods, depending on the fill type, mining method, and infrastructure (Helmes, 1988). Cemented paste backfill materials are usually prepared using dewatered slurries of mill tailings by thickening or filtering to get a filter cake (70–85 wt.% solids). Then these
main solid particles are mixed with one or more types of cement as a hydraulic binder such as common type of portland cement, sulfate-resistance cement, and ground granulated blast furnace slag (fine grained smelter slag), pulverized fly ash, if these materials show puzzolanic behavior plus water to produce high slump (18–25 cm) for transportation of the paste to the underground voids (Helmes, 1988; Brackebusch, 1994; Benzaazoua et al., 1999; Fall et al, 2005; Orejarena and Fall, 2011; Yilmaz et al., 2014). The use of sulfide containing material as a binder for backfill is no longer popular. Other agents can be added to CPB to improve its pumping ability. The mixture components of CPB such as type and grain-size distribution of the aggregate, type and ratio of the binder agent and the water plus other additives depend on the required strength, density, permeability of the hardened backfill, and the planned transportation system. Also, availability of solid particles and binder are also very important (Helmes, 1988). The percentage of binder and tailings in the CPB mix is usually between 3–7% by weight for binder and 70–85% for tailings, correspondingly (Landriault 1995; Orejarena and Fall 2008; Nasir and Fall 2009; Orejarena and Fall, 2011). CPB should contain some enough fine particles to prevent settlement and particle segregation while transported through pipeline for disposal (Landriault, 1995; Klein and Simon, 2006; Tariq and Yanful, 2013). The unique feature of CPB is its high water content (w/c ratio between 2.5 and 7%) which is usually much more than hydration requirements of cementitious materials, and is necessary for its consistency and flowability in transportation through pipelines by pumping (Ramlochan et al., 2004; Tariq and Yanful, 2013). An example of CPB preparation plant is shown in figure 10. CPB transported by gravity or pumped through pipelines to underground voids. The transportability of fresh CPB is one of its main operational factor and is related to CPB fluidity or flowability (Wu et al, 2013). CPB operation costs 20% on average of total mine operations (Fall et al., 2008). Flowability of fresh CPB is not only important in efficient pumping and delivery to stopes, but also in prevention of pipe clogging, which can cause significant financial loss for the mine.

If clogging occurs, the transportation system should be halted and the pipeline network be disassembled to clear the pipes, which finally cause delay in production process and more operating costs (Wu et al, 2013). Rheological properties or behavior of CPB is main controller of CPB fluidity or flowability which itself depends on factors such as density and concentration of the CPB mixtures, characteristics of the CPB mixture components, and pH which are internal elements. External elements
such as effect of temperature and the coupled effects of temperature and time in progress of cement hydration and shear time are important on the rheological behavior of fresh CPB (Wang et al., 2004; Huynh et al., 2006; Mahlaba et al., 2011; Yin et al., 2012). In various thermal loading conditions and their changes with time due to friction between CPB and pipeline walls during short or long transportation distance, and in different underground mining environments and depths, rheological properties of CPB can be changed, figure 11 (Wu et al, 2013). In each single underground mine, backfill transportation system is unique considering differences in temperatures (Fall and Samb, 2008, 2009)

![Figure 11: CPB transportation system to underground voids and influence of thermal factors (Wu et al, 2013).](image)

**Design criteria of CPB structures**

Design criteria of CPB structures can be attributed to mechanical performance, stability of barricades, environmental performance and durability. After emplacing in underground stopes or other voids, CPBs have to sustain certain loading stresses to provide a safe underground working environment for personnel. Good mechanical properties for the CPB, while good flowability for transfer should be provided. Appropriate hydraulic conductivity of CPB for drainage while drained water not destroy barricades, and the permeable free draining retaining walls, or bulkhead and the impermeable retaining walls are important in designing CPBs (Fall et al, 2009; Abdul-Hussain and Fall, 2012; Ghirian and Fall, 2013, 2014). Furthermore, another important factor in design criteria is environmental performance and durability that is controlled by CPBs permeability. After mine flooding CPBs can be susceptible to acid mine drainage (AMD) into the mine environment or groundwater. CPBs potential to produce AMD mainly depends on the reactivity (oxidation potential) of the mining wastes that build matrix of CPBs. The reactivity potential can be increased with types and quantity of sulphide minerals components of the CPB system, and can also be controlled by permeability of CPB matrix which influences conductivity of fluids, such as oxygen and water inside the backfill (Levens et al., 1996; Fall et al, 2009; Wu et al., 2013). The required strength for CPB design is controlled by type of its function underground. CPB for ground support require uniaxial or unconfined compressive strength (UCS) to be at least 5MPa while, for free-standing CPB applications, the UCS of CPB can be commonly lower than 1 MPa (Stone 1993; Belem and Benzaazoua, 2004; Sheshpari 2015). Researchers indicated that the required UCS for CPB can be between 0.2 MPa and 5
MPa, depending on application when the surrounding rock mass has UCS of 5 MPa to 240 MPa (Belem and Benzaazoua, 2004).

Arching effect in CPB

Arching effect is a universal feature that appears in geomaterial both in field and in the laboratory. Change in stress pattern in geomaterial happens and the shearing resistance keep the yielding mass in its original position leading to change of stress on yielding part and adjacent part of soil (Terzaghi, 1943). If the yielding section goes downward, shear resistance will act upward and decreases the stress at the base of yielding section. In reverse action if yielding mass goes upward, the shear resistance will prevent its movement by acting downward and lead to stress increase at the support of yielding section. This phenomenon is most visible in underground structures (Figure 12), in which arching action reduces overburden pressure. Stress redistribution affects the load from overburden geomaterial, surface surcharge, or lateral earth pressure on the structure. Rocks or geomaterial adjacent to an underground structure can increase structure load-carrying potential compared to similar unburied structure (Tien, 1996).

Figure 12: Diagram showing arching effect in backfilled stopes in mine, (Pirapakaran, 2008)

Pirapakaran and Sivakugan (2007) and Pirapakaran (2008) studied arching effect in backfilled mines using numerical methods by FLAC, and his research results for hydraulic filled stopes has been shown in figure 6 in previous sections. Arching effect by horizontal pressure transfer to the sidewalls should be considered in required fill strength for backfill design. Horizontal pressures under arching impact can be calculated from five analytical or semi-analytical methods which take cohesion at the backfill-sidewall interface and/or frictional sliding along the sidewalls into account. The mentioned methods are Martson’s model, its modified version, Terzaghi’s model, Van Horn’s model and Belem and Benzaazoua, proposed model, (Belem and Benzaazoua, 2008). Belem et al. (2004) introduced a three-dimensional model that implicitly considers arching impact to calculate horizontal stresses at the stope floor ($\sigma_h$) composed from both longitudinal stress ($\sigma_x$) which acts across ore body, and transverse stress ($\sigma_y$) which acts along ore body. Equation for this model is:
Vertical stress $\sigma_y (= \sigma_z)$ at the bottom of backfilled stope is obtained from equation 2.

$$\sigma_y = 0.185 \gamma H \left( \frac{H}{B + L} \right) \times \left[ 1 - \exp \left( -\frac{2H}{B} \right) \right] = \sigma_x$$  \[2\]

where

- $\gamma =$ backfill bulk unit weight (kN/m$^3$); $H =$ Backfill height in stope (m)
- $B =$ stope width (m); $L =$ Length of stope in strike direction (m)
- $\omega =$ directional constant which is equal to 1 for stress across ore body ($\sigma_h = \sigma_x$), and 0.185 for stress along ore body ($\sigma_h = \sigma_y$)

In the case of narrowly exposed backfill face, arching affects confined backfill through adjacent stope walls (figure 13). Using Terzaghi’s arching model Askew et al. (1978), based on 2D finite element modeling, suggest the following equation to calculate compressive strength for backfill design (Belem and Benzaazoua, 2008):

$$UCS_{\text{design}} = \frac{1.25B}{2K \tan \phi} \frac{(\gamma - 2c)}{B} \times \left[ 1 - \exp \left( -\frac{2HK \tan \phi}{B} \right) \right] FS$$  \[3\]

In which: $B =$ stope width; $K =$ Backfill pressure coefficient

$$K = \frac{1 + \sin^2 \phi}{\cos^2 \phi + 4 \tan^2 \phi} = \frac{1}{1 + 2 \tan^2 \phi}$$  \[4\]

$C =$ Backfill cohesive strength (kPa), obtained from triaxial test of backfill sample in laboratory

$\phi =$ Backfill internal friction angle in degree obtained from triaxial test of backfill sample in laboratory

$\gamma =$ backfill bulk unit weight (kN/m$^3$); $H =$ Backfill height (m); and $FS =$ Factor of safety

**Figure 13:** Diagram showing stability analysis of narrowly exposed backfill face (Belem and Benzaazoua, 2008)
Li et al, (2003), conducted analytical and numerical studies of arching effects in narrow backfilled stopes. In numerical modelling of narrow backfilled stopes excavation and filling sequence was considered as a separated calculation procedure. Calculations were performed by FLAC-2D to an equilibrium state, then backfill was placed in the excavated stope setting initial displacement field value to zero in calculation resulting that wall convergence before backfilling not to be taken in calculation process. Figure 14 shows the pattern of distribution of vertical and horizontal stress within backfilled stope. It is visible that, the vertical and horizontal stresses have a non-uniform distribution. In any depth, both horizontal and vertical overburden stresses grow faster with depth than stresses along the central line of backfilled stope due to arching effect. Stresses along the wall are lower than the central ones. The Modeling outcomes for stresses along the full height, with the Marston theory and overburden stress were shown in figures 15 and 16. As can be seen, the overburden stress is located fairly close to analytical and numerical calculations, when the backfill is emplaced in lower heights. In higher backfill at larger depths, arching phenomena happens, and the vertical and horizontal stresses are lower than those that can be obtained from the overburden weight of the backfill (Li et al, 2003).

**Figure 14:** Pattern of stress distribution in backfilled stope, a) vertical stress, b) horizontal stress (Li et al, 2003).

Usually Marston theory overestimates the value of stress transfer than numerical results, but it can be seen from figure 15, that Marston theory underestimates the stresses. By inspecting figure 6, it can be seen that Marston theory, underestimates the horizontal stress along the walls, while the vertical stress component $\sigma_{yy}$ is overestimated in status of being active or at rest that were attributed by $K = 1/2$ or $1/3$, respectively. Also $\sigma_{yy}$ is underestimated by attributed $K = 3$ and it should be considered that the passive case is not represented in this system (Li et al, 2003).
In this section, views of other authors on effect of arching especially on CPB are discussed. Due to effect of arching, the vertical stress at the bottom of the backfilled stope is considerably less than stress from overburden pressure (Marston, 1930; Terzaghi, 1943; Pirapakaran and Sivakugan, 2007; Nasir and Fall, 2008). Let us consider an example of arching effect on stability of CPB in stope. For example, a CPB with UCS strength of 500 KPa can stand in over 100 m height. If arching phenomenon did not exist such a CPB needed 2 MPa of UCS strength, which therefore, required more cement addition and led to more costs for operations (Kuganathan, 2005). The shear performance of the CPB-rock interface, specially the frictional angle is one of important factors in the assessment and calculation of arching effects in CPB (Marston, 1930). There is not enough information on this behavior and conducting field shear test is cumbersome and expensive. Therefore the friction angle between the CPB and rock wall in stope is often taken as the friction angle of the CPB materials (Terzaghi, 1943; Li et al., 2003) considering following assumptions: (I) the frictional angle of the CPB-rock interface is usually higher than frictional angle of the CPB material because of frequent high roughness for the rock wall; (II) In major of cases, the stope walls have large-scale roughness that shear failure surface at a CPB-rock interface, passes through the backfill rather than along the CPB rock interface (Nasir and Fall, 2008). Therefore, the friction angle of CPB-rock boundary is usually underestimated, which leads to underestimation of the arching effect in the CPB structure and thus, the overall stability of the CPB is underestimated (Mitchell et al., 1982), and hence design of

**Figure 15:** Evaluation of the calculated stresses with the analytical and numerical methods along the vertical central line of stope, at different depths \( h \), : a) vertical stress; b) horizontal stress (Li et al, 2003).

**Figure 16:** Evaluation of the calculated stresses with the analytical and numerical methods on the walls of stope, at different depths \( h \), : a) vertical stress \( \sigma_{yy} \); b) horizontal stress \( \sigma_{xx} \) (Li et al, 2003).
CPB structures will be conservative (Nasir and Fall, 2008). When smooth rock interfaces exist like in foliated rock, the shear failure usually will pass along CPB-rock interfaces. Overestimation of arching effect and shear behavior of CPB-rock in the stability analysis of CPB structures should be considered in these cases. It has serious technical, economical and human effects. A study was conducted by (Nasir and Fall, 2008) to study the shear behavior of the interface between the CPB and rock using a direct shear test method. Results from this study showed that the shear strength of the CPB materials is larger than shear strength of the CPB-rock interface in similar stress conditions leading to unsafe design in smooth rock walls. The magnitude of the normal stress highly affects shear performance of CPB-rock, i.e. in high normal stresses (≥200 kPa) the CPB-rock interface shows a different behavior despite of the curing time effect (Nasir and Fall, 2008).

**Rate of backfilling of cemented paste backfill**

The rate of the pumping the cemented paste into the stope can be defined as the rate of backfilling. Rate of pumping by equation [4] can be converted to rate of increase in the height of backfill structure which mainly depends on the cross sectional area (A) of the stope.

\[
\frac{\Delta H}{\Delta t} = \frac{\Delta P / \Delta t}{\gamma A_r}
\]

where \( H \) is the height of backfill, \( D_p/D_t \) is the pumping rate (t/tonne), and \( A_r \) is the cross section area of the stope. It requires long time to fill large stopes, which affects hydration rate of cemented paste in different layers (Nasir and Fall 2009).

The rate of backfilling in voids depends on several parameters including void geometry, strength, and cement content ratio in backfill material. Another important factor is curing time which itself depends on other factors such as temperature, gradient and environmental temperature, water content, permeability or hydraulic conductivity of backfill material and or barricade material, height of final backfilled material, hydraulic conductivity and internal friction of adjacent rocks. In next section some examples from two mines are presented regarding rate of filling. In KB gold mine in Western Australia, the filling procedure consisted of filling the first 10 m, which was done in the range of 0.2–0.5 m/h rise in vertical status and giving a 24-h rest period after filling sequence. Filling continued with increasing height with the rate of 0.3–0.6 m/m after the rest period, until the stope was full, which took 184 h after the start of the filling. Prepared CPB at this case had a density of 75% solids with cement content of Cc=3.1% by weight. Geometry of stope in KB gold mine was a rectangular prism with dimensions of 40 m in height, 18 m length, 15 m width approximately (Helsinki et al, 2011). In SNM, formerly known as the Sally Malay Mine in Western Australia, a stope with the height of 23 m and floor plan of 10 by 12 meters in dimensions was filled 6 m at a vertical increase rate of about 0.04 m/h in one step and then the remainder of the stope was filled at a constant rate of 0.1 m/h and whole process of filling took about 300 h. CPB mixture components were same as KB mine (Helsinki et al, 2011). There is a term in backfilling which is called sequential filling and it includes adding the saturated backfill in layers on top of the former deposited backfill, with drainage, until the stope is totally backfilled (El Mkadmi et al. 2013). El Mkadmi et al., (2013) conducted a study on modeling of backfilling of stopes. They found out that the initial hydraulic conductivity for backfill layers deposited in previous steps go under modification after each backfill layer is added. Results driven for the non-cemented cases (C0, C1, and C2 layers), with similar properties are as follows. Saturated unit weight and frictional angle equal to 20 and 35 KN/m, respectively, poisson’s ratio of 0.25, and hydraulic conductivity of 10⁻⁷ m/s are shown in figure 17. The lowest level of backfill, case C0 represents the filling equal to 5m thick layers, which are placed quickly after 5 days of drainage, in
other words 1 m/day filling rate, and 50 days for entire stope by 10 layers. Rate of 1 m/day is considered very slow rate and applied for evaluation purposes to observe the effect of filling rate on the stress pattern allocation in the backfilled stope and it is compared with other realistic rates such as C1 case for 1 m/5 h, and 1 m/h (case C2) (El Mkadmi et al. 2013).

**Figure 17:** Different rates of filling and change of the total (a) vertical and (b) horizontal stresses close to the lower level of the stope (h = 47.5 m) (El Mkadmi et al. 2013).

Figure 18 specially shows the change of pore water pressure in emplacement of the third, sixth, and 10th layers in the stope. In other words pore water change when the height range changes from 35 m ≤ h ≤ 40 m, 20 m ≤ h ≤ 25 m, and 0 m ≤ h ≤ 5 m, correspondingly in the 5 m/5 days slow backfilling rate (El Mkadmi et al. 2013).
Figure 18: All graphs are related to case C0 in this figure. (a) change of pore water pressure; (b) effective vertical stress within different backfill heights in stope at different emplacement times of layers 3, 6, and 10 filled in 50 days (filling rate: 5 m in 5 days). (c) Effective vertical stress in vertical center line in 5 days after filling each of 10 layers. The overburden pressure calculated for base of stope and presented in graph b and c (El Mkadmi et al. 2013).
SOURCE OF TEMPERATURE IN CPB

Generally sources of temperature in CPB can be due to hydration, earth gradient temperature, geographical location of mine and friction between transferring pipe, and cemented paste and friction between cemented paste and hosting rock walls. Each of these factors can become first source of temperature rise in CPB according to mine location, depth, transport type and length, roughness of hosting rock walls, and start time of hydration. The greater depth of mines going under CPB backfilling not only have longer transport distance and longer time for the fresh CPB for initiation of hydration, but also accompanied with natural heat influx due to the geothermal gradient (Orejarena and Fall, 2008, 2011). The rock mass face in deep underground mines is the primary factor for continuous heat for CPB. The geographical location of the mine is also considerable aspect that can impact transported CPB temperature in shallow depth mines. In permafrost parts of the earth, temperature is very low during the year which can influence CPB temperature in surface mixing plants, transportation, and hydration thermal load. Change of temperature because of seasonal change can significantly impact the temperature of the water component of CPB originating from surface lakes (Wu et al., 2012). These variations in CPB temperature due to mentioned sources can considerably affect rheological behavior of CPB (Wu et al., 2013).

SOURCE OF SULPHATE IN CPB

The main source of sulphate in the cemented paste backfill material is the tailings material from sulphide bearing minerals in polymetallic mines that can constitute of low to high amounts (2% to 60%) of sulphides that can change to sulphate as a result of reaction. One of the main sulphide bearing minerals in tailing fragments is Pyrite (FeS2) (Fall and Benzaazoua, 2005). Several authors (Anderson, 1930; Berard, 1970; Flann, and Lukaszewiski, 1970; Ninteman, 1978; Moses and Herman, 1991; Prein, 1994) suggested the equation 5 as the chemical reaction for pyrite oxidation, however it includes some complex reactions that are not comprehensively understood (Herbert, 1997).

\[
FeS_{2(s)} + \frac{7}{2}O_{2(g)} + H_2O_{(aq)} \rightarrow Fe^{2+}_{(aq)} + 2SO_4^{2-}_{(aq)} + 2H^+_{(aq)}
\] [5]

When sulphide oxidation reaction has been done actively in the tailings at the surface, it would be small and insignificant in the cemented paste backfill due to water saturated status of CPB and limitation of oxygen availability. Binder hydration products that act as a physical barrier to oxygen and porosity reduction, and bacterial activity due to high Ph > 9 are other factors (Benzaazoua et al, 2003; Ouellet et al, 2003; Levens et al, 1996). Another considerable source of sulphate is destruction of cyanides using sulfur dioxide/air in gold mining (Akcil, 2003). Also some sulphate exist in some mill tailings and/or in mine water from processed minerals in mining operation in tailings pore water and/or in water used for mixing CPB material. Using a cement that has added gypsum (CaSO4.2H2O) or anhydrite (CaSO4) in the clinker for flash setting control can enter a small fraction of sulphate to the CPB (Fall and Benzaazoua, 2005).
THE MECHANICAL PROPERTIES AND BEHAVIOR OF CPB AND THE FACTORS THAT CAN AFFECT IT

Generally speaking, cemented paste backfill is a geotechnical material, whose mechanical properties and behavior such as compressive and shear strength obey its main influencing factors like other geotechnical materials like cohesion, internal friction angle, pore water pressure. These factors themselves can change due to CPB components such as percentage of solids, binder, water content, heat load due to hydration or geothermal gradient, curing time, surcharge load, and hydraulic conductivity (Sheshpari, 2015).

Currently it is obvious that in situ properties of CPB are considerably different from the laboratory CPB material. Getting comprehensive knowledge from behavior of CPB in field is essential to optimize the CPB mix design. The differences in field and laboratory values of UCS can be due to the coupled thermo-hydro-mechanical-chemical (THMC) phenomenon in CPB (Abdul-Hussain and Fall, 2012; Ghirian and Fall, 2013, 2014). Laboratory tests performed by Ghirian and Fall (2013, 2014) showed that UCS values of CPB samples for different column heights increase with time due to combined impact of the advancement of cement and suction development inside the CPB material. Ghirian and Fall (2013) find out that the UCS value improves when the effective void ratio decreases. There is direct relation between the UCS value, the hydraulic conductivity, cement content ratio, curing time. In other words with increase of cement content ratio, or curing time due to cement hydration or both, microstructure of CPB changes, hydraulic conductivity decreases and UCS of CPB increases (Fall et al, 2009). Modulus of elasticity for CPB increases with curing time and hydration. In the current experiments run by Ghirian and Fall (2013, 2014) approximately all cohesion values resulted from the direct shear tests were around the half of the UCS values. Also, no important change was seen in internal friction angle (φ) values with time, which can be concluded that development of shear strength is primarily controlled by the improvement of cohesion due to bonding between the tailings grains, and in turn because of cement hydration (Ghirian and Fall 2013,2014). Fall et al. (2010) conducted a research on mechanical properties of CPBs with different Portland cement content, temperature and water to cement ratio, and concluded that increase of cement percentage in CPB improves modulus of elasticity and UCS. Increasing water to cement ratio decreases modulus of elasticity and UCS. Also, increasing the temperature in any cement content increases UCS and modulus of elasticity.

CONCLUSION

It can be concluded that mine backfill technologies are used increasingly to achieve required safety factors for hazard mitigations and prevention of environmental pollution. Rock backfill, hydraulic backfill, new silica alumina-based backfill, and cemented paste backfill are main backfill methods that are currently used. Application of any of above methods depends on safety goals such as mine collapse or land subsidence prevention in further and deeper mining or abandoning the mine, available equipment, material, and budget, and also environmental factors. Cemented paste backfill is an increasingly used in backfilling underground mines due to its better safety factor for working surroundings and preventive environmental measures against toxic waste minerals. Different parameters play role in mechanical strength and stiffness of CPB including chemical, thermal, and hydraulic factors. Arching effect is a strength improving phenomenon especially in hydraulic and cemented paste backfills that should be considered in designing backfill structures.
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