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EVALUATION OF THE STRENGTH PROPERTIES OF DESLIMED TAILINGS PASTE BACKFILL

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This paper examines the influence of desliming pre-treatment of mine tailings on the strength development of the resultant paste backfill. The mill tailings were sampled from an underground copper-zinc mine in northeast Turkey (tailings samples A and B). The beaker decantation method was used for tailings desliming in order to determine the proper particle size distribution (PSD). Using this laboratory sedimentation method, the fine particles (-20 μm) amount of the total tailings samples A (52%) and B (54%) were reduced to 15% and 20%, respectively. Deslimed tailings paste backfill samples were then prepared, cured and subjected to uniaxial compression tests using a digital mechanical press so as to understand the relationship between PSD and backfill strength development. It was found that the averaged 28-day uniaxial compressive strength (UCS) values of desliming pre-treatment tailings paste backfill samples were 30% to 60% higher than the ones of total mill tailings paste backfill samples.

1. Introduction

The underground mining process involves the removal and recovery of economically valuable minerals from the earth's crust. The resulting voids are generally filled with a number of waste material processes being known as backfilling. Mine fill has long been an integral part of the overall mining operation to provide a secure working environment for mine operators and to dispose mill tailings in underground mine openings.[1,2] The additional goal of the metalliferous mines that utilize a wide range of backfilling methods is to ensure the selective excavation of ore bodies without enco-

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untering any ore dilution problems.[3-5] The materials used for filling at most underground mines usually consist of total mill/concentrator tailings, riverbed sand and gravel, waste or crushed quarry rock. Total mill tailings and concentrator tailings can be used either as-received or deslimed (either by hydrocycloning or by sedimentation) to meet the desired percolation requirements and for some other benefits.[6-7] The particle size distribution (PSD) of the tailings largely depends on ore processing and desliming method used.

The development and utilisation of paste backfill technology have evolved over the last two decades around the world and especially in Canada. The mining industry is particularly interested by any technique or procedure to reduce costs associated with backfilling large open stopes. These innovation methods contributed to increase the efficiency of backfilling operations and to improve the stability of underground metal mines. A significant environmental benefit of using paste backfill, particularly when tailings are acid generating, is the possibility of placing a large amount of tailings up to 60% underground.[8-11] Basically, paste backfill is a non-segregating, low-plasticity and high-density material consisting of thickened or filtered mine tailings to which binder and water are added to achieve the desired consistency and strength characteristics. To characterize the physical properties and to understand the mechanical behaviour of cemented paste backfill, a considerable amount of study has been done until now.[12-16] The addition of binder to the paste backfill mixtures is essential for cohesion development since uncemented paste backfill is prone to liquefaction. It was also pointed out that the mechanical performance of paste backfill material is strongly dependant on many factors such as tailings fines content, particles shape and distribution, void ratio of final paste material, confining pressure acting on paste backfill in stope and, the solid concentration of paste backfill after placement underground.[17] The main parameters required for any backfill mix design are chiefly the rate of strength increase (short term property), ultimate strength (mid term property) and durability (long term property).[18-19] Fig. 1 illustrates the intrinsic components that can affect the paste backfill quality and/or performance, such as the chemical composition of binders and tailings pore water, tailings PSD, density and mineralogy, and the chemistry of mixing water. However, if mill tailings are fairly fine (>50 wt% of minus 20 μm), a potential problem of backfill block collapsing due to its low strength acquisition at underground conditions can be experienced. Hence, it must be well determined the amount of fine-grained particles (<20 μm) in the tailings for obtaining an ideal paste backfill material. The objective of backfill operators is to get well-graded tailings exhibiting the desired PSD either directly from total concentrator tailings or by desliming total mill tailings which have in general wider PSD and often contains undesirable amount of very fine material.

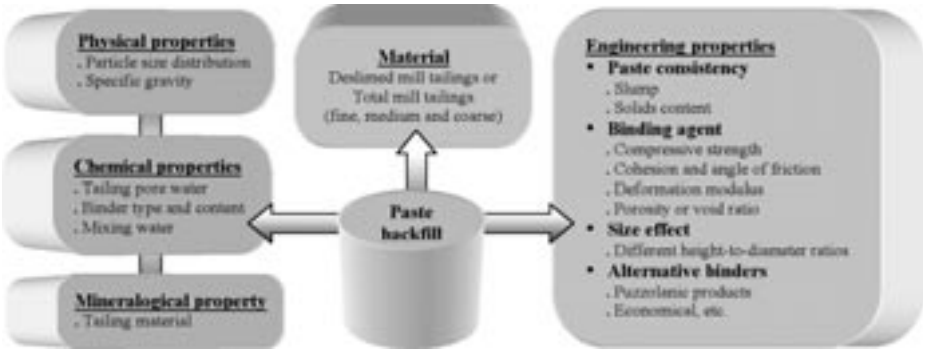


Fig. 1. A schematic diagram of the factors influencing cemented paste backfill quality and performance.

The underground mine selected for this study is located in the northeast Turkey. The polymetallic volcanogenic massive sulphide ore is mined and processed for copper and zinc recovery. After obtaining the concentrate, a large amount of concentrator mill tailings with a high acid generating potential is produced. At the mine site, a submarine tailings disposal system was used whereby tailings were transported by pipeline along a river to be discharged into the anoxic environment of the Black Sea. Since 2000, this system has partially been replaced by paste backfill operation in order to prevent environmental impact for short and long term perspectives.

The main objective of this study is to optimize the PSD that influences significantly the mechanical strength of paste backfill. This study is presented in two parts. The first part discusses the various criteria to be considered when preparing deslimed tailings paste backfill, including the design of optimum particle size distribution, tailings mineralogy, binder type and proportion, rheological index characterization and, sedimentation by beaker decantation. The second part of the paper presents the results of the effect of desliming pre-treatment tailings (removal of an amount of fine fraction, i.e. minus 20 μm) on the strength gain of paste backfill samples prepared from two mill tailings (samples A and B).

2. Designing the Optimum Particle Size Distribution

One of the most important characteristics of any paste backfill material is the particle size distribution which plays a key role on its resulting mechanical strength. The main purpose of particle size optimization is to produce a paste fill that develops dense packing during placement. This is usually achieved by a well-graded aggregate which allows attaining optimum porosity and therefore reducing binder consumption and mine operating costs.[20] Additionally, the PSD of cemented paste backfill has been shown to be important by studies which show that pipeline pressures and wear are sensitive to the percentage of minus 20 μm size material in paste backfill.[12]

As a general rule of thumb, the tailings material used in paste backfill mixtures

must contain at least 15 wt% of particles finer than 20 μm in order to retain adequate water and therefore to form a paste which will flow easily into pipeline. The preparation of a paste backfill and its corresponding properties not only depend on the PSD of tailings material, but also on its pore water chemistry and mineralogy.[21-27] Some tailings can retain more moisture at a given slump regardless of their particle size. This could be mostly attributed to their mineralogy as a result of high-water-retention minerals.[28] Sericite has been identified as retaining water within paste backfill due to the mineral layers absorbing water. Other minerals that commonly exhibit similar behaviour are micas and clay minerals.[6] The excess capillary pore water could appreciably decrease the paste backfill strength, depending on water to cement ratio (*w*/*c*).[26,29] There are three PSD categories for paste backfill mixture design for most hard rock mine tailings throughout the world.[1] These are coarse, medium and fine tailings (Table 1).

Table 1. Size distribution classification for paste backfill material.[1]

Mine tailings classification	Finer than 20mm content (wt%)	7" slump solid content (wt%)	Backfill characteristic (depending on w/c ratio)
Coarse	15 - 35	78 - 85	High strength acquisition
Medium	35 - 60	70 - 78	Lower strength acquisition
Fine	60 - 90	55 - 70	Poor strength acquisition

Thomas *et al*.[30] suggested that fine particles in a well-graded backfill may fill the voids between larger particles. This reduces the volume occupied by the cement gel probably leading to the formation of a stronger bonding as shown in Fig. 2. Modification of particle size distribution of the tailings is a good method for improving strength development within the produced paste backfill. The correct choice of PSD can help the optimum design of paste backfill mixtures, reducing porosity and thus minimising cement requirements.

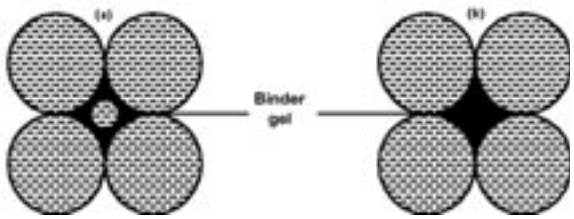


Fig. 2. Model showing benefits of fines in backfill: a) situation with good grading control, b) situation with no grading control.[30]

In theory, a certain gradation of fractions is desirable to produce a denser, minimum void mixture. Fall *et al*.[25] performed laboratory tests to study the influence of fines fraction (minus 20 mm) on strength development within paste backfill. It was shown that there exists an optimum amount of fines fraction (about 50% of -20 mm and correspon-

ding to medium tailings) from which the strength of paste backfill decreases. Yilmaz[19] showed that for different levels of desliming tailings, the highest strength was obtained with the coarser tailings which contain 15% of -20 mm (Fig. 3). Arioglu *et al*.[31] studied the effects of adding coarse aggregates to the cemented tailings backfills to improve their strength. They concluded that although blending coarse aggregates and fine tailings played some role in increased strength properties, the observed increase is mainly attributed to cement content and *w*/*c* ratio. It was concluded that particle gradation had minimal effect on strength development.

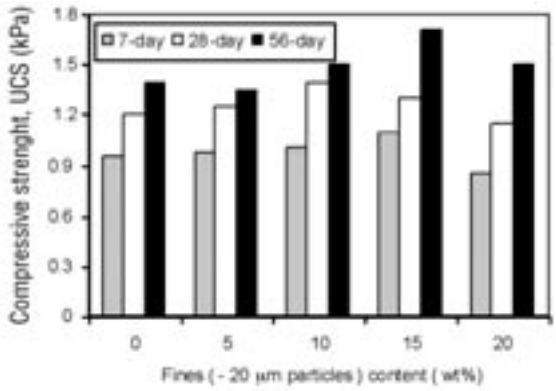


Fig. 3. Effect of fines content (-20 mm particles) on the strength of deslimed tailings backfill.

There is no consensus in the available literature on the optimum size distribution requirements or measurements for backfill materials due to mineralogical considerations and variability of tailings and on cement quality around the world as well as variations in delivery systems (e.g. unlined boreholes allowing groundwater to flow into the paste mixture). PSD curve index such as "coefficient of uniformity (C_u)" and "coefficient of curvature (C_c)" could be correlated to the strength development in paste backfill materials. A well-graded tailings backfill and soil materials have generally $4 < C_u < 6$ and $1 < C_c < 3$.[32]

$$C_u = \frac{D_{60}}{D_{10}} \tag{2.1.}$$

$$C_c = \frac{(D_{30})^2}{D_{10} \times D_{60}} \tag{2.2.}$$

where D_{10} is particle size at 10% passing; D_{30} is particle size at 30% passing; D_{60} is particle size at 60% passing.

The coefficient of uniformity, C_u, is usually used to define the grading of granular materials. Unlike uniformly graded aggregates, the well-graded tailings backfill samples exhibiting a wide range of particle size typically develop high bulk densities and low vo-

id ratios. The resulting compressive strength and deformation modulus are higher compared to that of uniformly distributed tailings (having a narrow range of particle size) backfill.[33]

3. Experimental Program

3.1. Tailings characterization

In order to better understand the effect of desliming pre-treatment of tailings on the paste backfill performance, representative tailings samples were sampled from a disk filter at the pastefill plant. The PSD curves of samples were obtained by laser light diffraction using a Malvern Mastersizer particle size analyser. Total tailings samples A and B were found to contain 52 wt% and 54 wt% of particles finer than 20 μm, respectively. Therefore both tailings sample can be classified as a medium-grained tailings material. Such tailings generally produce a good paste backfill, but typically have lower strength than the coarse-grained tailings because of a higher *w*/*c*.[28]

Fig. 4 shows the PSD curves of the total tailings (as-received) samples A and B. Their fines (-20 μm) content was reduced using the beaker decantation method for tailings desliming from their initial contents of 52% and 54% to 15% and 20%, respectively. This suggests that both deslimed tailings can be classified as a coarse-size tailings as shown in Table 1.

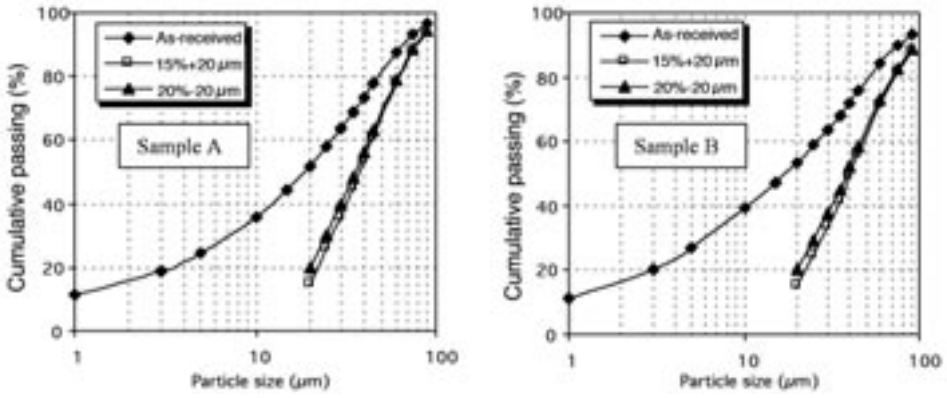


Fig. 4. The PSD curves of as-received and deslimed tailings for samples A (left) and B (right).

As-received tailings samples A and B had a C_u value of about 28, while deslimed ones had a C_u value of 2.9. According to criteria mentioned above[32], C_u values of both tailings give an indication that the overall range of particle sizes are not considered to be well graded. The C_c value for as-received tailings sample A had a value of 7.4 while as-received tailings sample B had a C_c value of 1.3.

The specific gravity of as-received tailings samples A and B was measured by helium gas pycnometer and determined to be 4.82 and 4.10, respectively. The measurements were performed in triplicate. The specific gravity of the tailings used to make the paste backfill will influence the solid mass concentration that can be achieved with regard to the desired slump. A higher specific gravity will yield a higher solid mass concentration. The mineralogical composition of tailings samples A and B was determined by X-ray diffraction (XRD) analysis (Table 2), which shows only the crystalline mineral phases in the tailings samples. The relative proportion of major, minor and trace mineral is based on XRD peak height. The major mineral phase identified in the tailings samples A and B is pyrite.

Table 2. Mineralogical composition of mill tailings.

Tailings sample	Major	Minor	Trace
A	pyrite	dolomite	sphalerite, barite
B	pyrite	kaolinite, dolomite	barite, sphalerite

The chemical composition of both tailings samples was determined by atomic absorption spectrometry (AAS), spectrophotometer (K_2O and Na_2O), and wet chemical analysis (Table 3). Tailings sample A is dominated by iron oxide, Fe_2O_3 (58.64%). Minor quantities of silicon dioxide, SiO_2 (3.36%) and aluminium oxide, Al_2O_3 (1.48%) were detected as well as trace amounts of magnesium, calcium, potassium, sodium, nickel, titanium, chromium, manganese, and phosphorous oxides (all less than 2%).

Table 3. Chemical composition of mill tailings (%).

Tailings Sample	MgO	Al_2O_3	SiO_2	CaO	Fe_2O_3	S^{2-}	Na_2O	TiO_2	Cr_2O_3	Mn_2O_3	P_2O_5	LOI
A	0.48	1.48	3.36	0.94	58.64	2.29	0.24	1.16	0.08	0.04	0.12	31.17
B	1.25	3.89	10.96	1.47	44.86	3.78	0.26	0.24	0.05	0.15	0.16	32.91

Tailings sample B is also dominated by iron oxide, Fe_2O_3 (44.86%) and minor quantities of silicon dioxide, SiO_2 (10.96%) and aluminium oxide, Al_2O_3 (3.89%), together with trace amounts of magnesium, calcium, potassium, sodium, nickel, titanium, chromium, manganese and phosphorous oxides (all less than 2%). The LOI (loss on ignition) value of 31.17% for sample A and 32.91% for sample B is indicative of loss of sulphur (at 500°C) as pyrite is burned off to reveal the high iron oxide reading.

3.2. Binding agent

Ordinary Portland cement and some mineral additives (e.g. fly ash, blast furnace slag) are commonly used as the binder for paste backfill mixtures. The cemented backfills attain strength over the curing period. Temperature and humidity can affect the hydration

reactions by creating strong bonds in backfill matrix. Previous studies showed that paste backfills could produce adequate strength with relatively low binder dosages of 3-5wt%.[2,12] In this study, Turkish Portland composite cement[34] was used to prepare paste backfill samples at a binder content of 5wt% for tailing samples A and B. Further information about the cement used can be found in Yilmaz.[19]

3.3. Rheological index

The rheological index (slump) of paste backfill are important and influenced by many factors which include the solids content, particle size, the particle surface chemistry, and binder proportion. Rheological index tests are usually used to create paste flow properties when transported through a borehole or pipeline.[29] In practice, the standard slump test is widely used for determining paste backfill consistency. Slump is a measure of the drop in height of a material when it is released from a truncated cone. Determination of the slump provides a way to characterize a material's consistency that can be related to its transportability. Fig. 5 shows the results of water separation tests performed for each paste backfill at two different slump values (6" and 7"). The uncemented sample A has bled apparently more water than the uncemented sample B. The amount of water separated from tailings sample A is ~3 times higher compared with tailings sample B. Within 24 hours, the difference between the 6" and 7" slump values of tailings sample A was ~10 times higher water separation than tailings sample B. This refers to variations in rheological characteristics and could be mainly attributed to their mineralogical composition as the water retention of tailings generally increases in the presence of calcite, clay (kaolinite, etc.) or similar highly colloidal, high-water-retention minerals.

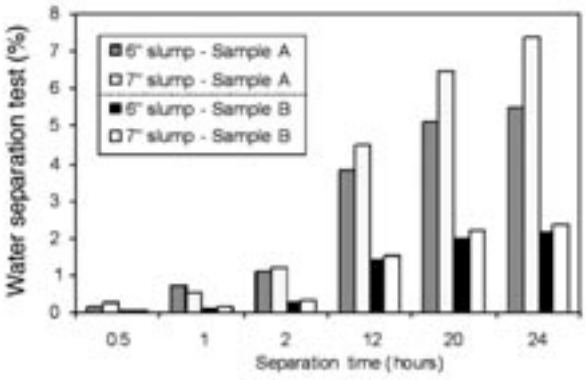


Fig. 5. Water separation versus time for tailings samples A and B.

3.4. Desliming pre-treatment of mill tailings

In the present study, the tailings samples A and B desliming (partial removal of fines size fraction) were carried out by sedimentation method. The sedimentation method is ba-

sed on the measurement of the rate of settling of the particles uniformly dispersed in a fluid, and the principle is well illustrated by the common laboratory method of "beaker decantation" (Fig. 6). The tailings material is uniformly dispersed in water contained in a beaker. A syphon tube is then immersed into the water to a depth of h, corresponding to about 90% the liquid depth L. The particles in the siphoned fraction will have size less than a desired size d (-20 mm in our case). The terminal velocity v is given by the Stokes law:

$$v = \frac{d^2 g(D_s - D_f)}{18\eta} \tag{3.1}$$

where v is the terminal velocity of the particle (m/s), d is the particle diameter (m), g is the gravitational acceleration (m/s^2), D_S is the particle density (kg/m^3), D_f is the fluid density (kg/m^3), and h is the fluid viscosity (Pa.s) (h = 0.001 Pa.s for bi-distilled water at 20°C under atmospheric pressure).

The times required for different particles to settle from water level to the bottom of the siphon tube (t) is the ratio of immersion depth h and the terminal velocity v calculated from Eq. (3). Fresh water is then added repeatedly (typically 7 to 10 times) to obtain clear water above the settled solids for each size fraction, until the entire particles smaller than the given size are removed.[35]

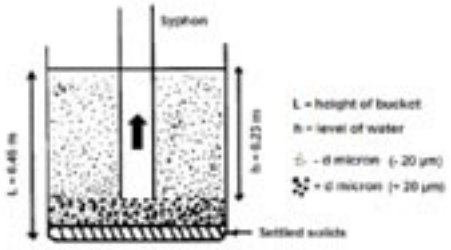


Fig. 6. Beaker decantation of tailings samples A and B.[35]

A total amount of 15 kg tailings was put into 60-litre bucket filled with sufficient amount of water. The pulp was thoroughly mixed and the particles were allowed to settle over the time required to deslime. The terminal velocity, v was calculated from the Stokes' Eq. (3) for the 20 μm size of tailings particle (samples A and B). A 20 μm particle size of tailings sample A had a settling terminal velocity v of 83 x 10^{-5} m/s, and the time required for 20 μm particle to settle from the water level to the bottom of the siphon tube (h = *0.23 m*) was about 5 minutes ($t = h/v$). For tailings sample B, these values were v = 675 x 10^{-6} m/s and t = 6 minutes. For each tailings sample (A and B), eight separate decantation tests were performed to ensure a reasonably clear decant water. Therefore, a required total settling time of about 40 and 48 minutes were calculated, respectively.

The beaker decantation tests allows to separate all the fines (-20 mm) fraction (equ-

ivalent to overflow particles) of tailings samples A and B from the coarse (+20 mm) fraction (equivalent to underflow particles). Then the separated fines (-20 mm) fraction of tailings samples A and B was mixed thoroughly in a Hobart model A200 mixer until it would be homogenized. Appropriate quantities of fines particles are then added to the coarse fraction (+20 mm). For each tailings sample (A and B), two subsequent tailings samples containing 15% and 20% of their particles size less than 20 μm were prepared. According to the tailings PSD classification given in Table 1, the deslimed tailings samples (A-15%, A-20%, B-15%, B-20% of -20 mm particles) are considered as coarse-size tailings.

3.5. Paste backfill preparation

From as-received and deslimed tailings samples A (15% and 20% of -20 mm particles) and B (15% and 20% of -20 mm particles), a series of paste backfill mixtures were prepared. The amount of the binder agent (5 wt%) and tailings were weighed and mixed homogenously with a measured volume of water in a 4.73-litre bucket. The paste backfill solids content was set to 82.495 wt% for tailings sample A, and 77.375 wt% for tailings sample B. The paste material was then filled into the standard slump cone with 1/3 volume increments. After the cone was filled, the paste was tamped 25 times with a small rod. The final slump which corresponds to the height between the top of an initial state of the paste (truncated cone height) and its final state (after removing the cone) was measured following the ASTM standards.[36] Two slump values, namely 6" and 7" were measured.

After setting the desired slump value, the paste mixture is cast into plastic cylinders having 10.16 cm (4 in.) diameter and 20.32 cm (8 in.) height. Seven holes with 2-mm diameter were drilled at the bottom surface of the plastic cylinders to allow water drainage. For each tailings sample, three cylinders were cast, sealed and cured in a foggy room maintained at 95% (25ºC) to mimic the underground conditions for a curing time of 28 days.

3.6. Uniaxial compression tests

The uniaxial compression tests were performed on the 28-day cured paste backfill samples to get their uniaxial compressive strength (UCS). The UCS value corresponds to the maximum stress observed during the test. The compression tests were carried out using a computer-controlled mechanical press (ELE Multiplex 5.0) having a nominal load capacity of 50 kN and a displacement rate of 1 mm/min. All the experiments were carried out in triplicate and the mean UCS values were presented in the results. The height to diameter ratio for samples was 2. The two ends of the paste backfill samples were first rectified to get plane surfaces before running a test.

4. Results and Discussion

In this study, the paste backfill mixture design criterion was to produce a UCS ≥ 1 MPa. However, the backfill samples produced from tailings samples A and B failed to meet this design criterion, producing UCS values below 1 MPa (Fig. 7). These low UCS values could be attributed to the chemical and mineralogical composition of the two types of tailings used (Table 2) and the quality of binding agent. Tailings samples A and B are high sulphides content mainly pyrite and other sulphide minerals.

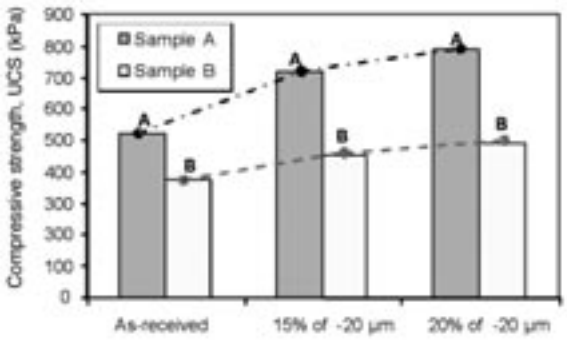
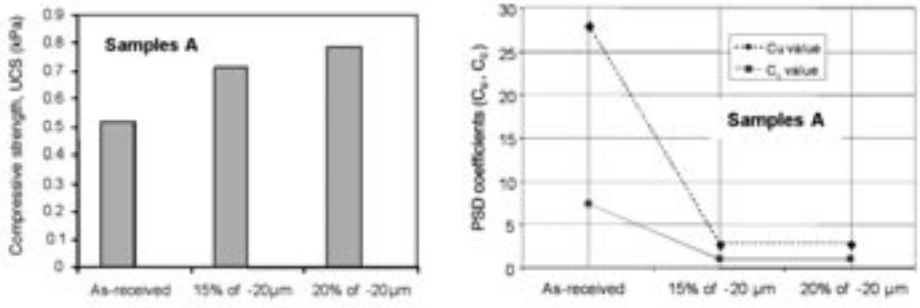


Fig. 7. Compressive strength of as-received and deslimed tailings samples A (15% and 20% of -20mm) and B (15% and 20% of -20mm).

It is known that the presence of sulphur compounds within cementitious materials can cause the deterioration of construction works due to the phenomenon known as sulphate attack. In this respect, the low UCS values produced by tailings samples A and B paste backfill could be a result of oxidation of sulphide minerals such as pyrite under the curing conditions in the presence of oxygen and water.[11] The oxidation of pyrite would lead to the formation of sulphate and H^+ that produce acidy which will in turn inhibit the hydration reactions (strength increase) and may dissolve the hydration products. It is also relevant to note that calcium-rich cements including ordinary Portland cement used in this study are particularly susceptible to sulphate attack and hence perform poorly in the case of the tailings used in the present study.[14]

The paste backfill samples obtained from tailings samples A were found to produce consistently higher UCS than those obtained from tailings sample B. This could be related to the mineralogy of both tailings types with respect to their silicate content (Table 2). Tailings sample B have apparently higher silicate content than that of tailings sample A, which was consistent with the high water retention of tailings sample B. The high *w/c* ratio could have adversely affected the strength gain of paste backfill samples obtained from tailings sample B. It can also be observed from Fig. 7 that desliming of both tailings produced a positive effect on the UCS of the paste backfill samples. A 1.38- to 1.52-fold increase in the strength with reducing the amount of -20 mm fraction to 15-20 % was noted to occur for paste backfill samples prepared from tailings sample A. Si-

milarly, the strength of paste backfill samples obtained from tailings sample B increased by 21-31% on decreasing the fines content from 54% (as-received) to 15 and 20% (deslimed). It was also observed that the UCS for both tailings sample peaked at 20% fines that could be interpreted as the optimum fines (-20 mm) particles for the paste backfill mix design. Similar test results were also obtained by Cayouette.[7] He conducted an *in-situ* test work at the Louvicourt Mine on partially deslimed paste backfill having 4.5 wt% binder in a mixture of 20% Portland cement and 80% ferrous slag. The results revealed that deslimed paste backfill with 12% minus 20 mm fines content provided for average UCS increases in the order of 52% after 28 days, 17% after 56 days and 10% after 90 days. Fig. 8 shows a variation of UCS with fines content (<20 mm particles) and a comparison of C_u and C_c values for paste backfill samples with deslimed and as-received (total) tailings sample A.



(a) (b)

Fig. 8. Properties of total and deslimed tailings sample A: a) variation of UCS with fines content, b) comparison of C_u and C_c values.

The C_u and C_c values were determined to be 2.9 and 1 respectively for the deslimed tailings sample A (15% and 20% of -20 mm particles) compared with as-received tailings sample A with the C_u and C_c values of 28 and 7.4 respectively. The C_u and C_c values of deslimed tailings being close to the optimum range[32] are in agreement with the findings of a significant improvement within the paste backfill strength on the deslimed tailings sample A. Similar results were also obtained for tailings sample B.

5. Conclusion

The effect of fines (<20mm particle size) contained in the tailings on the compressive strength of paste backfill was investigated using deslimed and total tailings. The results showed that the decrease in the fines content of tailings via desliming produced a positive effect on the UCS of paste backfill samples with the highest UCS obtained for the 20% fine particles content for tailings samples A and B. The positive effect of desliming could be attributed to the improvement in the gradation of tailings as indicated by the co-

efficients of uniformity and curvature determined for as-received and deslimed tailings samples. The paste backfill samples prepared from tailings samples A and B were found to have a UCS value ≤ 785 MPa, which was below the design criterion for this paste backfill (target UCS of 1 MPa). The low UCS values, despite the positive effect of desliming, could be related to the mineralogical composition of the two types of tailings used (high sulphide content, mainly pyrite). In other words, the oxidation of sulphide minerals under the curing conditions could have led to the formation of acidity and sulphates, and thus adversely affecting the binding properties of Portland cement used. Besides, tailings sample B was observed to produce the paste backfill samples with consistently low UCS compared with tailings sample A. This could be attributed to the relatively high water-retention ability of tailings sample B presumably due to its high silicate content as indicated by the results of chemical and mineralogical analyses. Finally, this study places the emphasis on the importance of fines content and particle size, and mineralogical composition of tailings to be used for paste backfill.

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