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Case study: Holistic test work campaigns for tailings management system designs

J Wickens *Paterson & Cooke, South Africa*

Z Dreyer *Paterson & Cooke, South Africa*

J Gerhardi *Paterson & Cooke, South Africa*

B van der Spuy *Paterson & Cooke, South Africa*

ABSTRACT

A gold mine in East Africa has decided to implement a new paste backfill system. To implement the overall project while ensuring that all the design, environmental and government requirements are satisfied, the mine decided to undertake a comprehensive test work campaign to understand their tailings. The test work program focused on the following:

1. *Transport of tailings to the Tailings Storage Facility (TSF) and*
2. *Tailings used for paste backfill material.*

For the design of the transportation system of tailings to the TSF, the test program focused on the basic material and slurry properties which were used as inputs for the various other tests. Large-scale pipe loop tests were conducted on the tailings to measure the existing system's pressure gradients, stationary deposition velocities and centrifugal pump performances at the operating density range (30%m to 50%m).

For the paste backfill system design, the test program focused on thickening and filtration tests to determine the dewatering characteristics of the tailings. Filter cake bulk material handling tests were conducted to provide input to conveyor and chute designs. In order to understand the material that would be placed underground, unconfined compressive strength and rheology tests were conducted as part of the paste backfill reticulation design, and modified acid-base accounting, net acid generation and shake flask, and tank leach tests were conducted on the cemented paste mixes as part of the environmental tests. This paper presents the laboratory test results on which the overall tailings management design was based.

Keywords: *Thickening, Filtration, Conveyability, Pressure Gradient, Stationary Deposition Velocity, Centrifugal Pump Derating, Unconfined Compressive Strength (UCS), Rheology, Geochemistry.*

1 Introduction

Tailings management is one of the most important aspects of the mining process, and an optimised, well-run tailings disposal system is the key to continued production for any mine. Implementation of a new paste backfill system on an existing mine operation requires a careful understanding of the material properties of the tailings available on the mine as well as the impact on the upstream processes once the paste backfill system starts operating.

This specific project included a conventional tailings disposal system using pipelines, which required a redesign due to a reduced solids throughput required to the TSF (250 dry tph) when the paste backfill system is operational (150 dry tph). However, the tailings disposal system design also needed to be flexible to accommodate the full tailings stream (400 dry tph) when the paste backfill system is not operational.

In addition to the design of the paste backfill system and the upgrades required to the existing tailings system of the project, there were environmental requirements to ensure underground water is not compromised due to paste backfill being placed in the underground stopes. An extensive geochemistry test work campaign was also undertaken as part of the project to assess the leaching potential of the paste backfill material.

The extent of the test work required for both the tailings management and paste backfill system required careful planning before commencing with a test work program of such a scale. Developing a material testing program to support the engineering and project development process was vital.

The data presented within this paper are derived from an extensive, multi-phased test work program undertaken by Paterson & Cooke in its Cape Town laboratory, including the following:

1. Material Property Tests
    a. Solids density,
    b. Particle size distributions,
    c. Slurry pH and temperature and
    d. Mineralogy and bed packing concentrations.
2. Conventional Tailings Disposal Tests
    a. Pressure gradients in 106.0 and 157.0 mm ID pipes,
    b. Stationary deposition velocities in 99.0 and 154.0 mm ID pipes and
    c. Centrifugal pump performance.
3. Paste Backfill Tests
    a. Static and bench scale dynamic thickening,
    b. Thickener underflow rheology,
    c. Pressure filtration,
    d. Filter cake bulk material handling,
    e. Unconfined compressive strength and
    f. Cemented rheology.
4. Cemented Tailings Geochemical (Environmental) Tests
    a. Modified acid-base accounting,
    b. Net acid generation,
    c. Shake flask extraction and
    d. Tank leaching.

2 Material property tests

2.1 Solids density

The solids density of the gold tailings and cement was determined using a helium gas pycnometer (ASTM D5550-06), which measures the skeletal solids density. The gold tailings and cement had solids densities of 2674 and 3030 kg/m^3, respectively. Figure 1 shows the relationship between slurry density, mass solids concentration and volumetric solids concentration at a solids density of 2674 kg/m^3 for the uncemented tailings only.

2.2 Particle size distribution

The material's particle size distribution (PSD) was determined by wet sieving to >25 µm. Figure 2 presents the particle size distribution of the tailings. The tailings had a top size and d_{50} of ~1000 and ~51 µm, respectively.

2.3 Slurry pH and temperature

The slurry's pH and temperature were measured using a hand-held pH metre. The average slurry pH before and after cement addition was 8.7 and 12.5, respectively. The slurry had an average temperature of 24.5°C before and after cement addition.

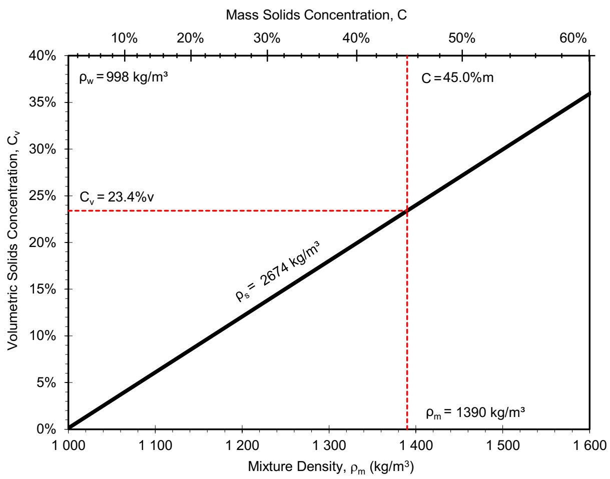

Figure 1 Relationship between ρm, C, and Cv

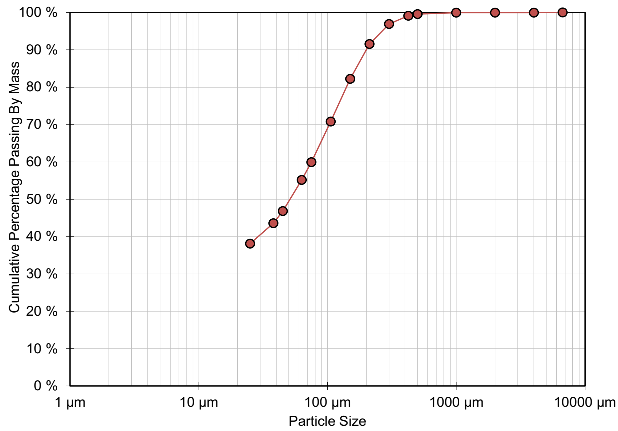

Figure 2 Tailings particle size distribution

2.4 Mineralogy

Mineralogy (XRD) tests were conducted on the tailings to better understand the composition of the material. The mineralogy indicated that the material primarily consists of quartz (45%m), plagioclase (32%m) and muscovite (15%m).

Pyrite at 0.4%m was also detected in the tailings but was low enough not to affect the backfill strength over time. However, when oxidised, pyrite generates sulfuric acid that can affect the long-term strength of the backfill due to sulfate attack. In addition, arsenic levels of 0.052%m were detected, which posed a problem for the leaching of arsenic at high pH values when adding cement to the tailings.

2.5 Bed packing concentrations

The freely settled bed packing concentration by volume is calculated from the volume of the freely settled bed formed by a known volume of solids. A slurry sample is allowed to settle for 24 hours in a measuring flask. The actual solids volume is determined from the dry mass of material and solids density.

The maximum bed packing is measured as described above for the freely settled bed packing concentration, but with the bed compressed by applying a hydraulic pressure of over 400 kPa across the settled bed with the water allowed to drain through the solids bed. The maximum packing concentration is calculated from the volume of the compressed bed and the known volume of solids.

The freely settled packing concentration (C_{bfree}) indicates the approximate transition between low and high-concentration slurries, while the maximum packing concentration (C_{bmax}) represents an unachievable concentration for conventional slurry pipeline transport. The freely settled bed packing and maximum bed packing concentrations were measured as 43.7%v (67.5%m) and 52.9%v (75.0%m), respectively.

3 Conventional tailings disposal tests

3.1 Pipe loop tests - Equipment

The pipe loop tests aim to collect pressure gradient data over a range of mixture velocities and slurry densities. A 6x4 centrifugal pump fitted with a 90 kW motor (VFD driven) circulated the tailings slurry in the pipe loop while measuring the pressure drop using a differential pressure transducer and controlling the flow rate through the pipe loop. Figure 3 shows the configuration of P&C’s pipe loop used for the test work campaign.

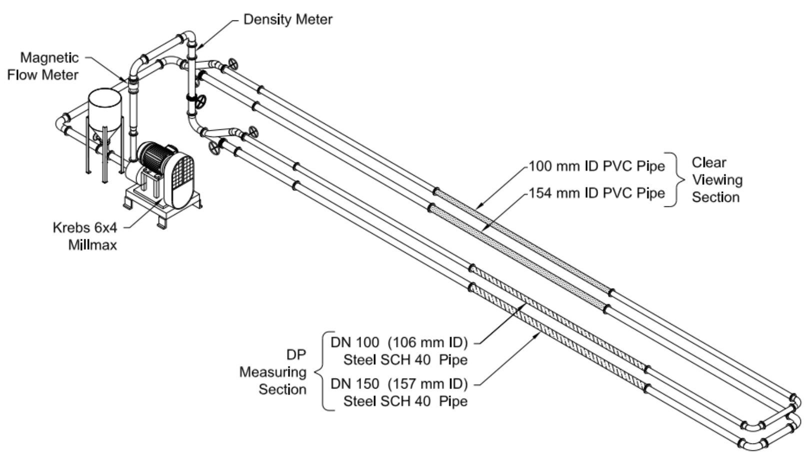

Figure 3 Schematic diagram of P&C’s 106 and 157 mm pipe loop

3.2 Pipe loop tests – Clear water tests

The correct operation of the flowmeter, differential pressure transducer and pressure tappings are confirmed by conducting a clear water test prior to the slurry tests. The average pipe roughness is calculated from the measured water test data points using the Colebrook-White friction factor formulation and the Darcy equation. The measured hydraulic pipe roughness for the 106.0 and 157.0 mm ID pipes were 9 and 8 µm, respectively, as shown in Figures 4 and 5.

3.3 Pipe loop tests – Slurry pressure gradient tests

Slurry pressure gradient tests were conducted in 106.0 and 157.0 mm ID pipes over mass concentrations ranging between 31.3 to 60.7%m and 30.4 to 52.9%m, respectively. Figures 4 and 5 show the measured pressure gradients for the two pipes. The data clearly shows the higher pressure gradient in a 106.0 mm ID pipe compared to the 157.0 mm ID pipe for the same mass concentration and mixture velocity. Below the measured deposition velocity (as per section 3.4), the pipe loop operated with a stationary bed.

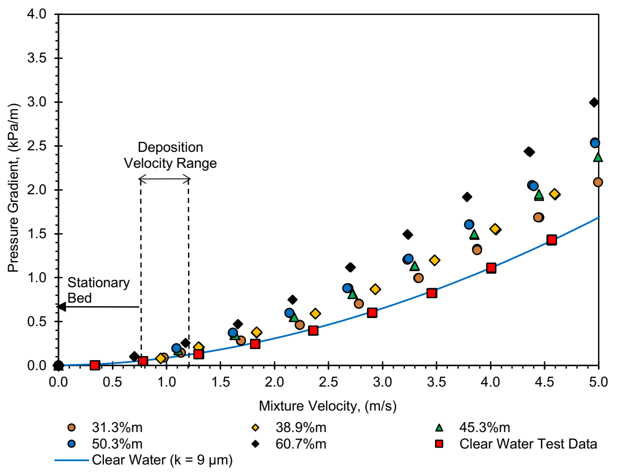

Figure 4 Pressure gradient versus mixture velocity – 106.0 mm ID pipe

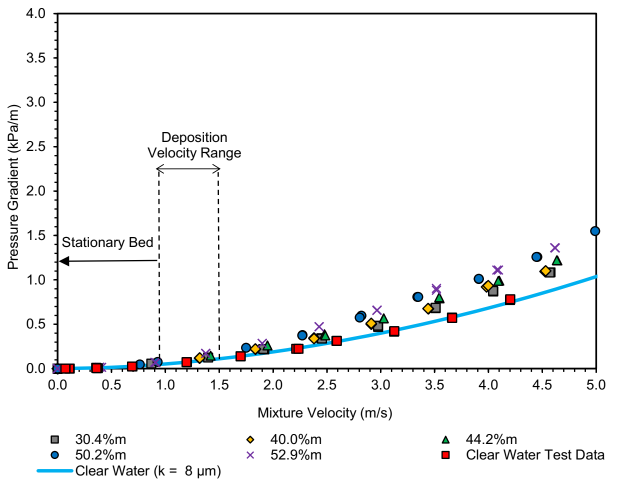

Figure 5 Pressure gradient versus mixture velocity – 157.0 mm ID pipe

3.4 Pipe loop tests - Stationary deposition velocity

This test aims to measure the stationary deposition velocity over a range of slurry densities for the gold tailings sample to enable accurate settling velocity predictions when designing the pipeline.

The stationary deposition velocity is observed visually as the point at which the first particle rests on the invert of the 99.0 and 154.0 mm ID clear pipe sections. Figure 6 shows the observed stationary deposition velocity for the two pipe sizes tested. The data shows a decrease in stationary deposition velocity as the mass concentration increases, typical of settling mixed regime slurries.

3.5 Pipe loop tests - Centrifugal pump performance

These tests aim to determine the head and efficiency ratios of a centrifugal slurry pump to enable scale up or down to the design pump size. The head ratio, H_R, is the ratio of the head developed by the pump when pumping slurry to the head developed when pumping water at a given pump speed and flow rate. The efficiency ratio, E_R, is the ratio of the measured motor power, recorded from the variable speed drive when pumping slurry, to the motor power when pumping water at a given pump speed and flow rate.

The clear water pump head and efficiency curves measured for the 6x4 centrifugal pump are presented in Figure 7. The test was performed at an impeller tip speed of 20 m/s, with the BEP found to be between 60 and 80 l/s. Figures 8 and 9 present the measured slurry pump head and slurry pump efficiency versus volumetric flow rate for the gold tailings. The tests were conducted at mass solids concentrations of 40.0 and 50.0%m, and the maximum head and efficiency derating was calculated at H_R=0.90 and E_R=0.85 at BEP, respectively.

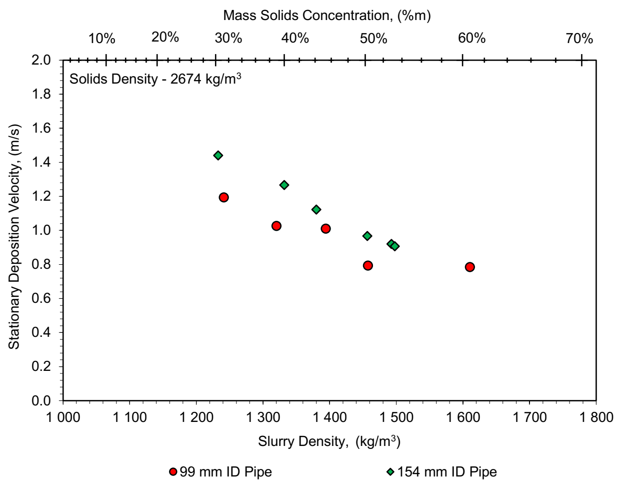

Figure 6 **Observed stationary deposition velocity**

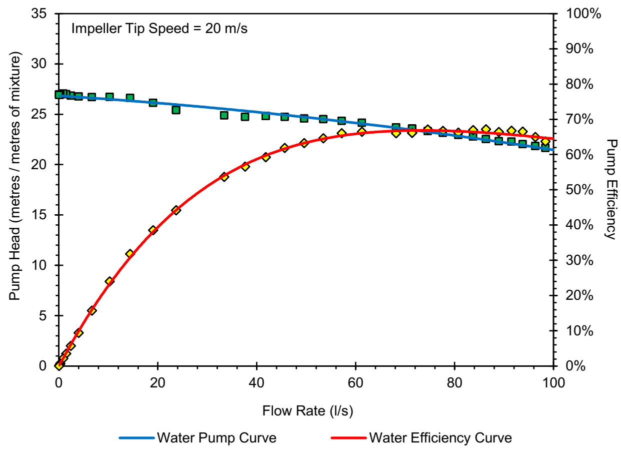

Figure 7 **Pump performance clear water tests**

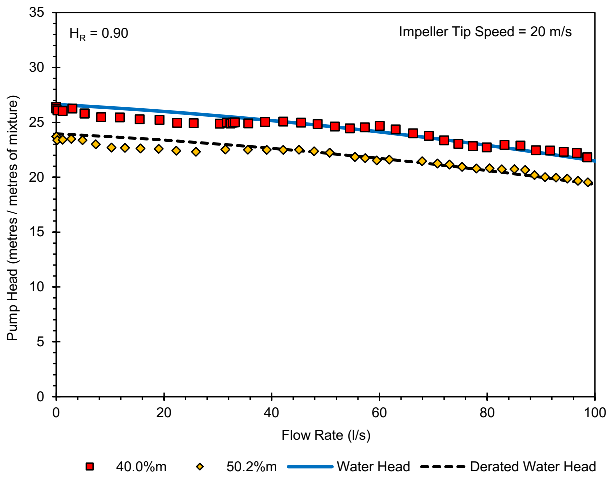

Figure 8 **Pump head vs flow rate for slurry**

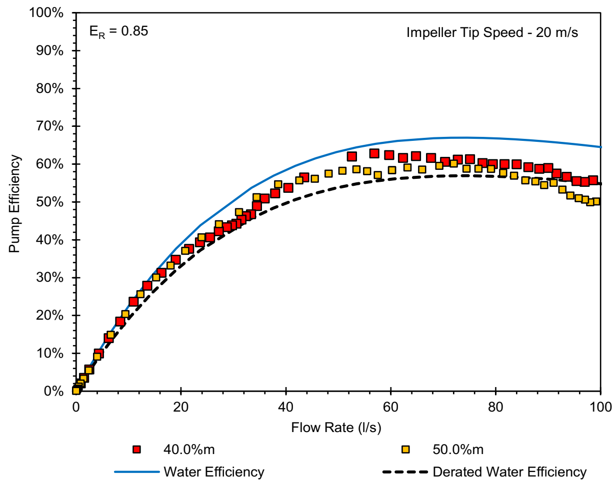

Figure 9 **Pump efficiency vs flow rate for slurry**

4 Paste backfill tests

4.1 Static thickening tests

Firstly, colloidal behaviour tests were conducted on the material. A sample was prepared in a 500 mL graduated cylinder at 4%m solids concentration using process water. The sample was mixed and then allowed to settle over 24 hours. The material was colloidally unstable; therefore, a coagulant is not required for further sedimentation testing.

After the colloidal stability tests, flocculant selection tests were conducted to determine the best-performing flocculant for this material. Figure 10 presents the flocculant selection test results. The data indicated that BASF’s M919, which has an anionic charge with an ultra-high molecular weight, is the best-performing flocculant and was selected for further test work.

Static thickening tests were then conducted to determine the optimum feed mass concentration and flocculant dosage rate. First, feed mass concentrations of 5, 10, 15, 20 and 25%m were made up and dosed with 10, 15, 20, 25 and 30 g/t M919 flocculant, respectively. The settling rates were then measured for each of the permutations. Figure 11 presents the static thickening test results, which indicated that the optimum is 15%m at 25 g/t.

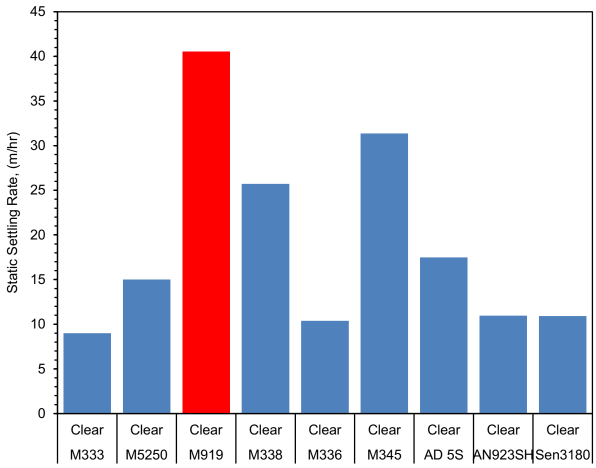

Figure 10 **Flocculant selection tests**

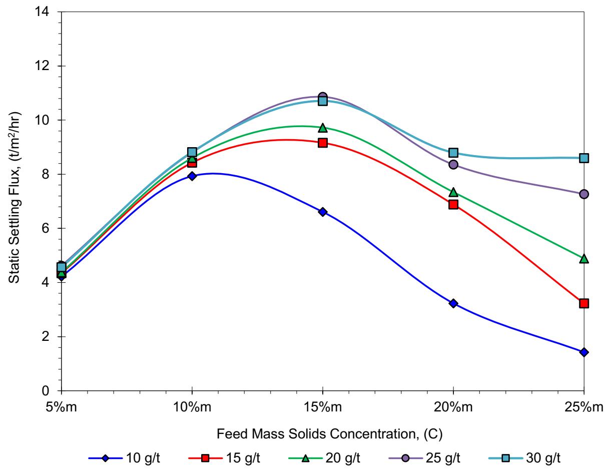

Figure 11 **Static thickening tests**

4.2 Bench scale dynamic thickening tests

Next, bench scale dynamic thickening tests were conducted at a feed mass concentration and target flocculant dosage rate of 15%m and 25 g/t, respectively. Target solids loading rates (dry tonnage to thickener) of 0.5, 0.7, 0.9, 1.0 and 1.2 t/h/m^2 were tested to determine where sliming of the thickener will occur. Figure 12 presents the dynamic thickening test results, indicating that the highest thickener underflow (~62%m) will be achieved at a solids loading rate and liquid rise rate of 0.7 t/m^2/h and 0.4 m/h, respectively, with the thickener overflow having a concentration of 0.45%m.

24-hour consolidation tests were conducted at a solids loading rate of 0.7 t/m^2/h to determine the mass solids concentration that can be achieved at plant scale. The test results indicated that it is likely that ~72%m can be achieved at plant scale.

4.3 Uncemented thickener underflow rheology tests

Lastly, rotational viscometer rheology tests were conducted on the thickener underflow material. A couette viscometer (rotating bob with stationary cup) was used for the test work to measure flow curves over a mass concentration range that varied between 67.0 to 72.6%m. Only laminar flow data was measured over the concentration range tested. The laminar flow data were analysed by applying the Bingham Plastic Model, and the results are shown in Table 1. Figure 13 shows the rheogram for the thickener underflow.

Table 1 **Bingham plastic yield stress and plastic viscosity**

Mass conc (%m)	67.0	69.1	70.3	71.3	71.9	72.6
Yield Stress (Pa)	12.4	22.2	34.2	48.6	57.1	72.8
Plastic Viscosity (Pa.s)	0.039	0.058	0.075	0.108	0.125	0.158

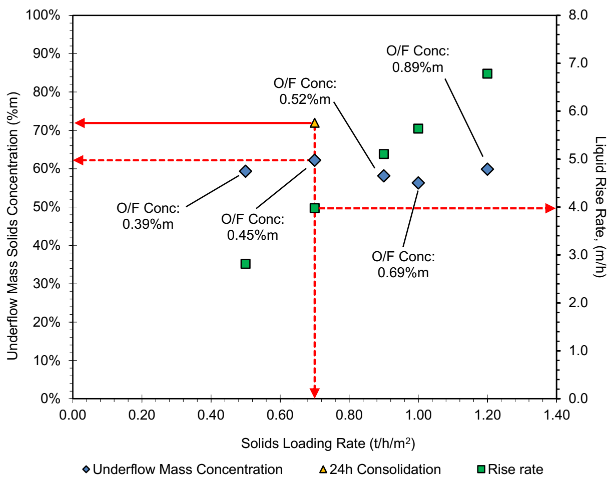

Figure 12 Dynamic thickening tests

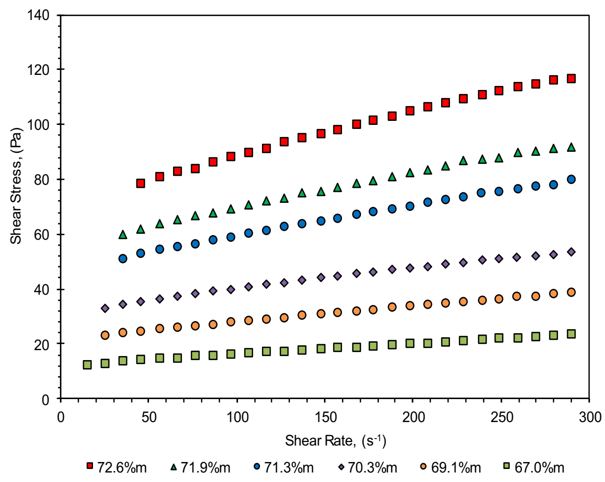

Figure 13 Thickener underflow rheology

4.4 Pressure filtration tests

Pressure filtration tests were conducted to further dewater the thickener underflow material for paste backfill purposes. Vacuum filtration test work was also conducted as part of the test work campaign. The process design of the system, however, requires that dewatered tailings be transported via trucks to the paste backfill plant. The vacuum filtration tests could not achieve the minimum transportable moisture limit (TML) requirement. Therefore, this paper does not present these vacuum filtration tests as they are not part of the overall design. Cloth selection tests were conducted on four different cloths from Clear Edge filtration, and the data showed that the best-performing cloth was the PX657-95 (air permeability of >110 litre/dm²/min), as shown in Table 2.

Table 2 Filter cloth selection tests

Filter cloth	PX 587-11	PX 515-07	PX 657-95	PY 09510
Feed concentration and air pressure	68%m at 600 kPa			
Cake mass conc, (%m)	85.2	84.5	89.9	84.9
Filtrate mass conc. (%m)	0.6	0.6	0.3	0.9

Using the best-performing filter cloth, filtration tests were conducted at a single feed mass concentration of 68%m at 600 kPa and three cakes of different thicknesses were formed. Table 3 shows that the thickener underflow can be filtered to concentrations of ~88%m at equivalent chamber sizes between 30 and 60 mm.

Table 3 Pressure filtration test results

Test number	Test 1	Test 2	Test 3
Feed concentration and air pressure	68%m at 600 kPa		
Cake thickness (equivalent chamber size) (mm)	15 (30)	22 (44)	30 (60)
Cake thickness (mm)	15	22	30
Form time (seconds)	30	40	50
Dry cake loading (kg/m²)	23	36	47
Cake mass solids concentration at form time (%m)	84.8	84.1	82.5

Dry time (seconds)	150	140	130
Final cake mass solids concentration (%m)	89.5	88.8	87.7

Figures 14 and 15 show the cake loading versus cake thickness and cake loading versus cake form time, respectively.

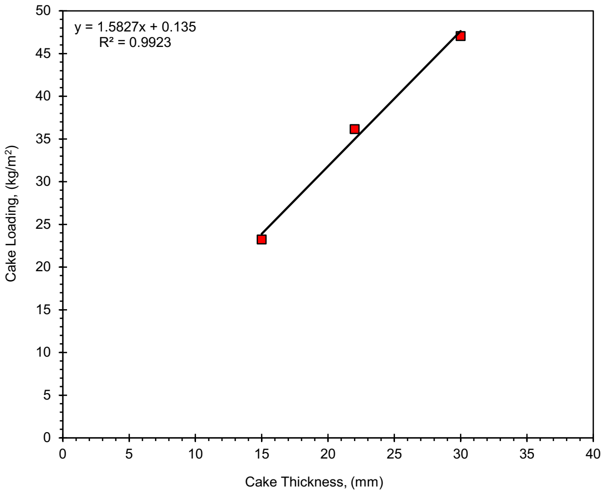

Figure 14 Cake loading vs cake thickness

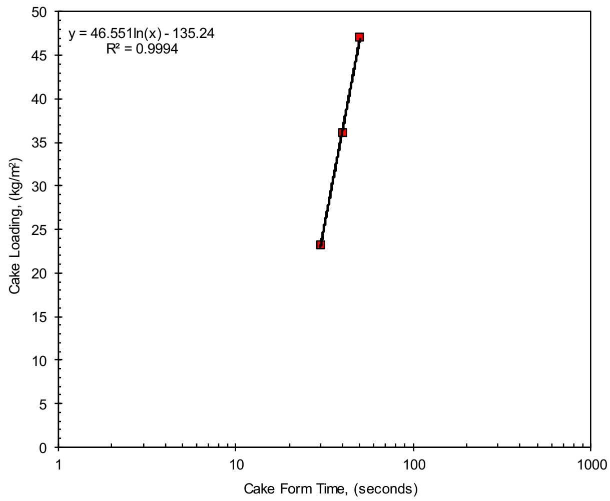

Figure 15 Cake loading vs cake form time

4.5 Filter cake bulk materials handling tests

Bulk material handling tests were conducted on the filter cake material to provide information for the conveyor and chute designs. Table 4 presents the bulk material handling test results. Figures 16 and 17 show the bulk density versus major consolidation force and unconfined yield force versus major consolidation force, respectively. The transportable moisture limit (TML) was determined using a flow table and indicated that the minimum TML is 13% moisture[1] (87%m) to prevent liquefaction from occurring during transportation. This moisture limit could only be achieved using pressure filtration.

Table 4 Bulk material handling test results

Property	Value recorded
Tested mass concentration	88.0%
Loose bulk density	1305 kg/m^3
Angle of repose	32 – 36°
Angle of drawdown	48 – 54°
Angle of surcharge	36 – 42°
Critical chute angle	31°
Coefficient of friction	0.601
Transportable moisture limit (TML)	87%m (13% moisture)

[1] Metallurgical definition i.e. Moisture content = 1 - Mass Concentration

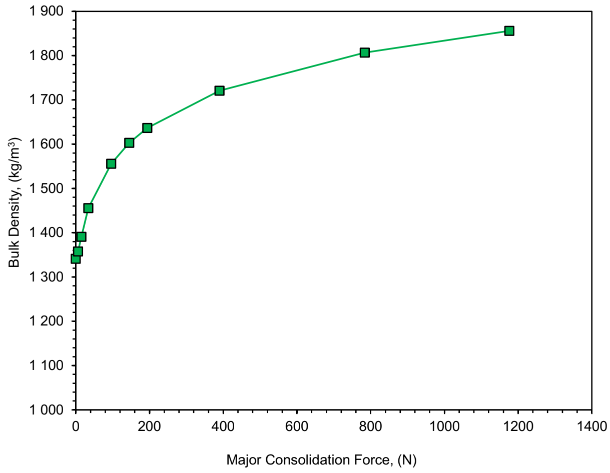

Figure 16 Bulk density vs major consolidation force

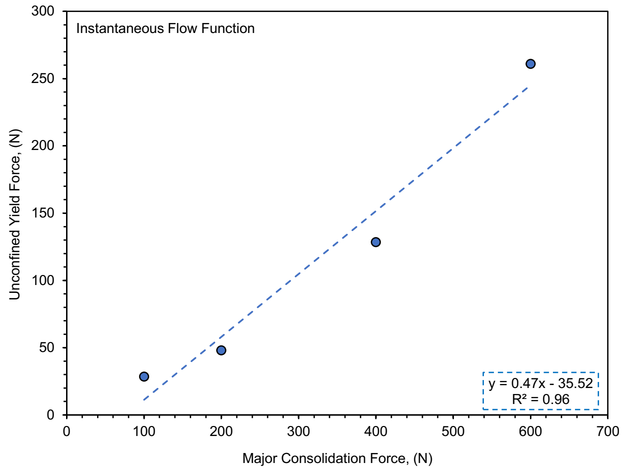

Figure 17 Unconfined yield force vs major consolidation force

4.6 Strength tests

4.6.1 Cement addition, water/cement ratio and mass concentration

The percentage cement addition, water/cement ratio and mass concentration are calculated using the following formulas:

$$Percentage\ Cement = \frac{mass\ of\ cement}{mass\ of\ tailings + mass\ of\ cement} \tag{1}$$

$$Water/Cement\ Ratio = \frac{mass\ of\ water}{mass\ of\ cement} \tag{2}$$

$$Mass\ Conc. = \frac{mass\ of\ tailings + mass\ of\ cement}{mass\ of\ tailings + mass\ of\ cement + mass\ of\ water} \tag{3}$$

4.6.2 Cement quality tests

Before commencing any unconfined compressive strength tests, P&C conducted cement quality tests on the two cement samples delivered for the test work, according to EN 196-1:2005 Edition 2 (European Standard 2005).

Figure 18 presents the ISO bar test results. The data indicate that at 28 days, both cement samples exceed the minimum required compressive strength of 32.5R and 42.5N MPa, indicating that the cement complied with the standard requirements, with type A being a rapid hardening and type B being a normal hardening cement.

4.6.3 *Unconfined compressive strength tests*

UCS cylinders were crushed on 7, 14, 28 and 56 days to determine the change in strength over time, as shown in Figure 19 and Table 5. All mixes were constituted at 70%m with 2, 4 and 6% cement additions, equating to water/cement ratios of 21.44/1, 10.72/1 and 7.15/1.

Figures 20 and 21 present the water/cement ratio versus unconfined compressive strength at 7, 14, 28 and 56 days. The data shows a continuous increase in strength from 7 to 56 days for both tested cement types.

The data also shows that type B cement performs better than type A at the same water/cement ratio. The difference can be attributed to the two types of cement, where type A and type B had compressive strengths of 33.9 and 44.9 MPa after 28 days.

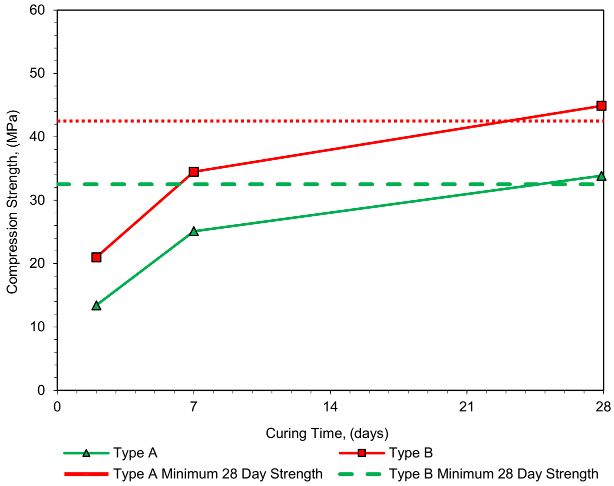

Figure 18 ISO bar test results

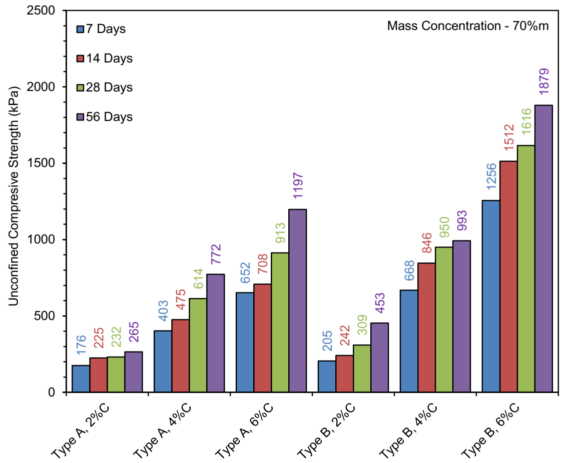

Figure 19 UCS test results

Table 5 UCS Test Results

Mix number	Cement type	Cement addition	Mass concentration	W/C ratio	7 days curing	14 days curing	28 days curing	56 days curing
		%	%m	m/m	kPa			
1.1	Type A	2	70	21.44	176	225	232	265
1.2		4		10.72	403	475	614	772
1.3		6		7.15	652	708	913	1197
2.1	Type B	2	70	21.44	205	242	309	453
2.2		4		10.72	668	846	950	993
2.3		6		7.15	1256	1512	1616	1879

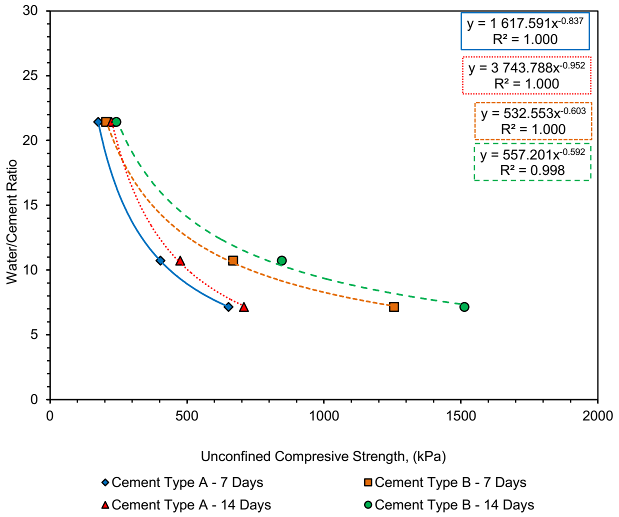

Figure 20 W/C ratio vs UCS at 7 and 14 days

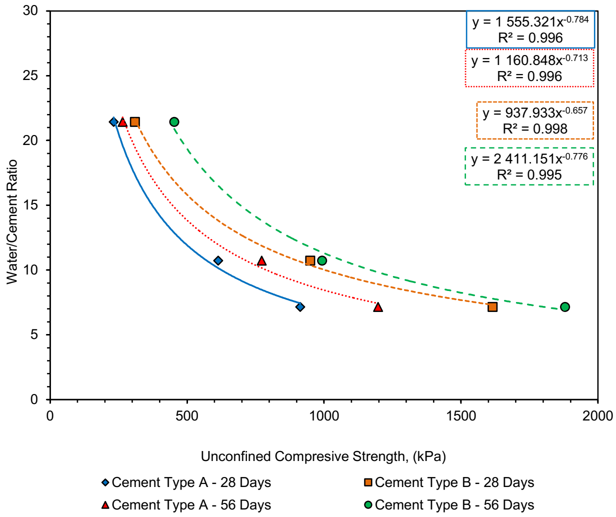

Figure 21 W/C ratio vs UCS at 28 and 56 days

4.7 Cemented rheology tests

Due to the viscous nature of the material, a couette rotational viscometer with a bob and infinite cup instead of a typical bob and cup method was selected for the paste backfill rheology test work. The measured data were corrected for end effects, secondary and undeveloped flow.

Figure 22 presents the rheogram for the cylinders cast for both the type A and B cement. The data shows a slight difference between the yield stress and plastic viscosity for the type A and B cement and that the minimum and maximum expected values will range from 90 to 108 Pa and 0.120 to 0.134 Pa, respectively, at a mass concentration of 70%m.

Based on these results, rotational viscometer rheology tests were conducted for a range of mass concentrations at 4% cement addition for type B cement to provide input for the paste backfill reticulation design, as shown in Figure 23.

The laminar flow data were analysed by applying the Bingham Plastic Model, and the results are shown in Figures 24 and 25. The data show a sharp rise in the plastic viscosity from 70.4 to 72.5%m, which is most likely the onset of slip at the wall of the bob which will most likely result in lower yield stress and higher plastic viscosities being measured at concentrations above 72.5%m.

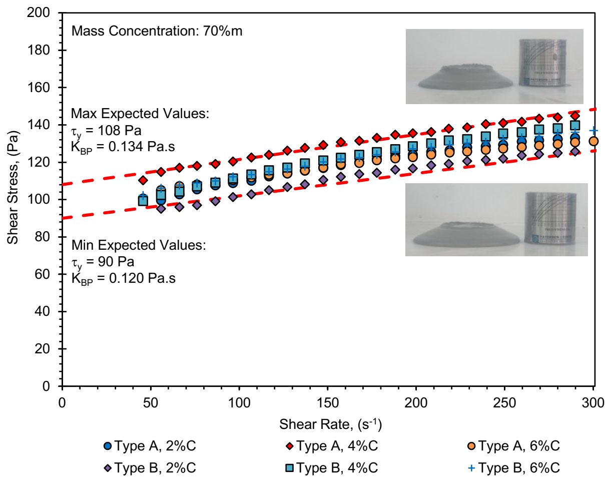

Figure 22 Rheogram for cylinder castings

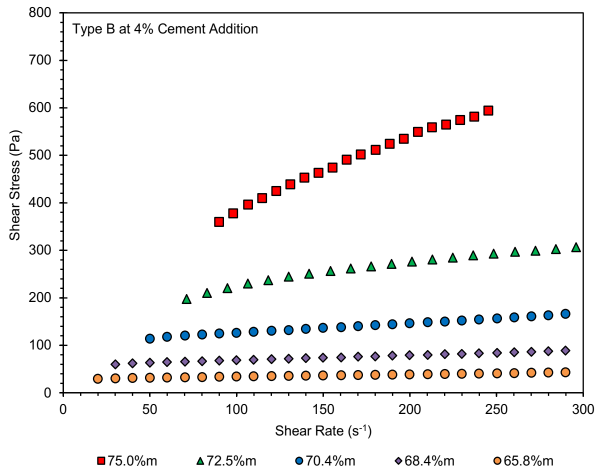

Figure 23 Rheogram for Type B cement

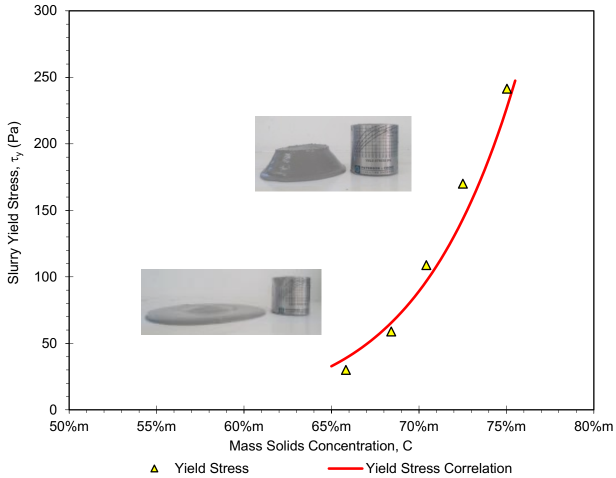

Figure 24 Yield stress vs mass concentration

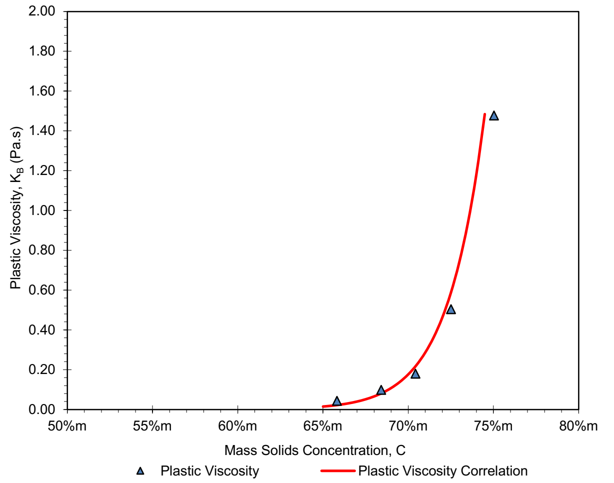

Figure 25 Plastic viscosity vs mass concentration

5 Cemented tailings geochemical tests

Geochemical tests on tailings and cemented paste fill are conducted to investigate the possible environmental impact paste fill might have. Two categories of geochemical testing were conducted:

1. Acid rock drainage (ARD) tests and
2. Leachability tests

ARD tests include modified acid-base accounting (ABA) and NET acid generation (NAG) tests. These tests evaluate the possibility of acidic drainage from weathered tailings material. Low pH drainage is of environmental concern and often mobilises numerous heavy metals (Pb, Se, Cd, As, Hg, Cr, Cu, Ni, Sb) from the solid tailings matrix (leaching). Stabilising tailings with cement is very effective at preventing ARD. However, ARD testing on the tailings material is still conducted to determine if the cemented backfill might be subject to acid attack from the tailings.

The leaching of arsenic (As) from the paste fill (possibly contaminating groundwater supplies) is of great concern at the mine site. Arsenic (unlike many other metals) shows increased mobility at both high and low pH. The cemented material creates an elevated pH aqueous environment in which encapsulated arsenic might be rendered mobile.

A strategy for limiting arsenic leaching is iron sulfate (FeSO4) stabilisation. When FeSO4 is added, it co-precipitates with the arsenic to form a stable, insoluble precipitate of iron arsenate (FeAsO4), which is less mobile. The test work was aimed at addressing all the above points. Table 6 presents the test methods.

Table 6 Test methods used

Test name	Test method	Comments
Acid-Base Accounting	MEND 1.20.1 (Price 2009)	Modified Sobek Method
NET Acid Generation (NAG)	MEND 1.20.1 (Price 2009)	A single addition NAG was required
Shake Flask Extraction	MEND 1.20.1 (Price 2009)	24 hours, 3:1 liquid-solids mass ratio
Monolithic Tank Leach	NEN7345 / IS EN 15863-2015	64-day total duration, 28-day cured backfill

5.1 Acid generation tests

Determining the sulphur content in tailings is critical to testing the acid generation potential. Acid drainage is formed by the oxidation of sulphide minerals to form sulphuric acid. The tailings material sulphide content was determined via combustion and infrared absorption with a LECO-style instrument. The tailings were found to contain sulphide of 0.62%m.

5.1.1 Modified acid-base accounting (ABA)

Acid-base accounting (ABA) showed that the tailings material has an average neutralisation potential of 52.3 kg $CaCO_3$ per tonne, with a NET neutralisation potential (NNP) of 32.9 kg $CaCO_3$ per tonne.

5.1.2 NET acid generation (NAG)

Effective NET acid generation (NAG) was found to be 1.4 kg H_2SO_4 per tonne of tailings. An equivalent of 1.4 kg $CaCO_3$ per tonne of tails is needed to neutralise the produced acid. During testing, the NAG pH was measured as 5.17. Interpreting the NAG pH and ABA neutralisation potential ratio (2.75) together suggests that the tailings material is non-acid forming.

5.2 Leachabaility tests

The total arsenic content in the tailings was determined with hot acid digestion and an ICP-OES instrument finish. The tailings arsenic content was measured as 520 mg per kg (0.052%m). Mineralogical analysis could not ascertain in which phase the arsenic was present, but the arsenic source is likely to be arsenopyrite (FeAsS).

5.2.1 Shake flask extraction (SFE)

All SFE tests were extracted for 24 hours. The SFE testing parameters for pure and treated tailings are presented in Table 6. Treated tailings were dosed with $FeSO_4{\cdot}7H_2O(s)$ at 11.5kg per tonne solid tailings (equivalent to Fe:As molar ratio of 6).

Table 7 Shake flask extraction test parameters for tailings

Sample type	Sample mass	Particle size	Extractant
Air-dried tailings as received	100 g tails	As received PSD (Figure 2)	Deionised water, 300 g
Air-dried tailings with $FeSO_4{\cdot}7H_2O$ (s) added at Fe:As molar ratio of 6			

SFE on tailings alone suggests an apparent arsenic solubility of 1.5 mg per kg tailings with a final eluate pH of 7.9. Adding $FeSO_4 \cdot 7H_2O$ to the tailings (at an Fe:As molar ratio of 6) brought the apparent solubility of arsenic down to 0.04 mg/kg, which is a 97% decrease. The final eluate pH for the salt-stabilised SFE test was 7.4.

Paste backfill (cemented tailings) comprised 6% binder by mass (dry solids basis) to a final paste solids mass concentration of 74%. The binder was a CEM III B-SR 42.5N (Type B, section 4.6.2). The water used for batch A-06 was groundwater from the mine site; groundwater spiked with ferrous sulphate salt (FeSO4·7H2O) was used for batch S-06 with the salt dosed at 11.5 kg/t tails content. Both mixtures were allowed to cure for 28 days before testing. The arsenic added from the groundwater is negligible with respect to the arsenic present in the tailings. The SFE testing parameters for the above cemented tailings are presented in Table 8.

Table 8 Shake flask extraction test parameters for cemented mixtures

Sample type	Sample mass	Particle size	Extractant
Backfill batch A-06	100 g cured backfill (tailings + cement)	Crushed to +4-5 mm	Deionised water, 300 g
Backfill batch S-06			

SFE on cured backfill suggests an apparent arsenic solubility of 0.15 mg per kg backfill with a final eluate pH of 11.6. Treating the tailings with salt before cementing brought the apparent solubility of arsenic down to 0.08 mg/kg, which is a 47% decrease. The final eluate pH for salt-stabilised backfill was 11.5. Some hydroxide ions are consumed by the iron ions to form iron hydroxides, thus lowering pH slightly and consuming iron needed for arsenic co-precipitation.

The $FeSO_4$ salt stabilises the arsenic in cemented tailings, but the extent is not as dramatic because arsenic (unlike most other metals) shows increased mobility at both high and low pH. The final eluate pH for salt-stabilised backfill was 11.5 compared to 7.4 for salt-stabilised tailings. The cemented material creates a high pH aqueous environment in which encapsulated arsenic is more mobile and leachable. Previous testing has suggested that arsenic is most effectively stabilised by $FeSO_4$ salt at a pH range of 6.5 to 8.

Nevertheless, cement stabilisation of arsenic in tailings is effective. It supplies calcium for calcium arsenate precipitation and encapsulates the arsenic containing tailings in a lower permeable matrix that limits oxidation and arsenic dissolution.

5.2.2 Tank leach test (TLT)

Since the backfill was crushed prior to SFE testing, tank leach tests were conducted to better represent site conditions. Backfill will mostly be present as a monolithic material, thus the monolithic tank leach testing selection. SFE tests generally show elevated leach levels due to increased surface area exposure (compared to monolithic material) and agitation.

Cemented paste backfill was prepared as described in section 5.2.1. The backfill was cast into 46 mm internal diameter and 60 mm high PVC moulds and left to cure for 28 days. No releasing agent was added to the moulds. The extractant was 1 litre of deionised water to yield a liquid-solid ratio of 8 ml per square centimetre of solid. Extractant is renewed at specified intervals until a total leach time of 64 days. Eluates were analysed for dissolved ions, metals and pH. Arsenic was the only inorganic specie of concern. Figure 26 presents the eluate pH & cumulative arsenic release results.

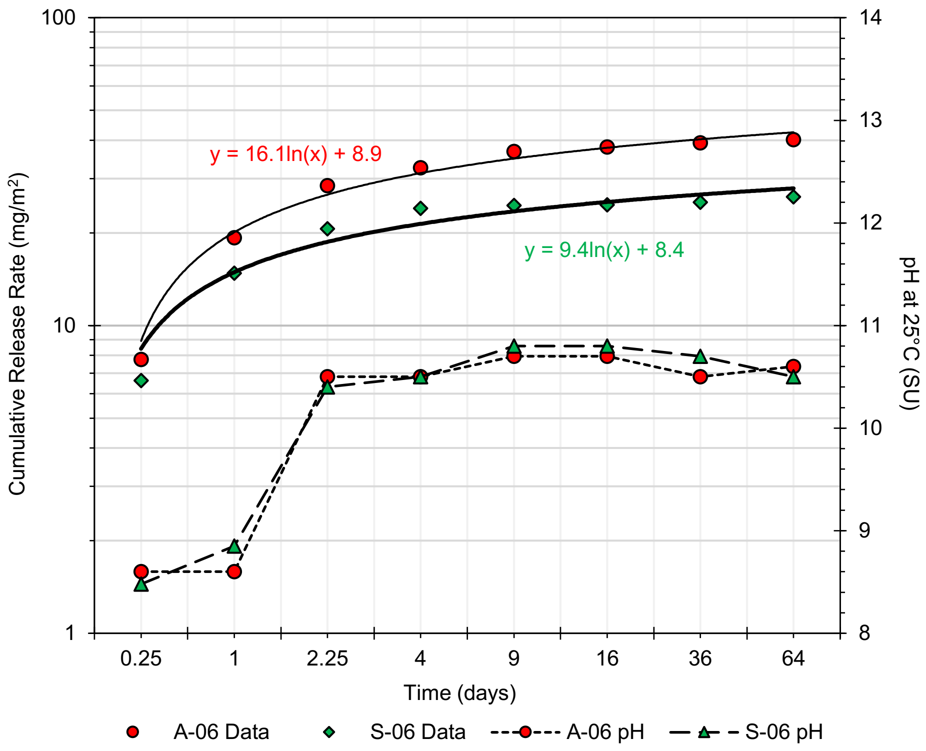

Figure 26 Eluate pH & cumulative arsenic release from cemented tailings

Backfill samples A-06 and S-06 displayed cumulative arsenic leaching greater than 10 mg per square metre of exposed backfill. The data interpretation per IS EN 15863-2015 grouped the release mechanism as 'unidentified with depletion influence'. The porous cemented matrix is continuously changing due to curing and saturation. It is thus difficult to establish a leaching mechanism. Additionally, arsenic leaching is a complex process involving oxidation, dissolution and precipitation.

Sample A-06 displayed a cumulative release over 64 days of 45.2 mg As/m^2. Sulphate cumulative release was 35.4 g/m^2 (also with an unidentified leaching mechanism). Sample S-06 displayed a cumulative release over 64 days of 29.5 mg As/m^2. Sulphate cumulative release was 36.5 g/m^2 (also with an unidentified leaching mechanism).

Adding ferrous sulphate salt thus resulted in a 34.8% reduction in Arsenic leached per unit area of backfill. Sulphate leaching increased for salt-treated backfill, but sulphate has a low environmental impact.

The added sulphate is, however, a concern for sulphate attack on the cement. Early ettringite formation (resulting from sulphate and tricalcium aluminate, C3A) benefits the strength and durability of backfill. Still, excessive ettringite formation can lead to expansion and cracking, thus leading to strength loss and increased permeability. Therefore, sulphate-resistant cement (low C3A content) was proposed and used in all the testing. The added $FeSO_4$ salt resulted in a ~20% decrease in the 28-day UCS of the backfill (1.62 MPa for mix A-06 versus 1.30 MPa for mix S-06).

6 Final designs

All the various aspects of the tailings management system on the mine were designed based on the actual results of the test work campaign completed, thereby providing confidence in the final process, mechanical, piping, electrical and equipment designs.

6.1 Tailings pipeline design

The results of the conventional tailings disposal tests were used to baseline the design models used by Paterson & Cooke to select the final tailings pipeline size that would prevent settling in the pipeline during operation while limiting the pumping pressures required. The centrifugal pump derating tests were used to accurately determine the expected efficiency and head ratings experienced by the pumping system and ensure that the selected pumps and motors are suitably sized with sufficient design margin for fluctuations in the process.

6.2 Paste backfill system design

The results of the paste backfill tests were used to design the complete backfill process, from tailings dewatering to disposal underground. The results allowed the accurate sizing and selection of all process equipment, such as the thickener, flocculant system and pressure filters.

The filter cake bulk materials handling tests provided inputs to the design of all conveyors, bins and feeders that form part of the paste backfill system. Bin angles, conveyor speeds, etc., could be selected confidently to accommodate the complete design criteria provided by the mine.

The UCS results provided the required dosing requirements for sizing the binder system to achieve the range of target strengths provided by the mine. In addition, rheology test work results were used to complete the paste reticulation design, including the positive displacement pump selection, paste pipeline sizing, pipeline required pressure ratings, coupling selection and underground support designs.

6.3 Cemented tailings geochemical (environmental) tests

The geochemistry test work provided input to environmental impact studies for paste backfill. In addition, the sizing for an external salt dosing system could be conducted for different salt dose rates. Lastly, the arsenic leach rate and contamination can be estimated with dimensional data on site backfill and groundwater volumes.

7 Conclusions

The results obtained from these tests were used as inputs to the next set of tests and ultimately will dictate the type and size of equipment required based on the overall design. This paper presents the results of a series of tests conducted as part of the mine's conventional tailings disposal redesign as well as the design of the paste backfill system. The main conclusions are as follows:

1. The material properties indicated that the material is a typical gold tailings with a solids density of 2674 kg/m^3 and a particle size distribution with a top size and d_{50} of ~1000 and ~51 µm, respectively. The mineralogy indicated that the material primarily consists of quartz (45%m), plagioclase (32%m) and muscovite (15%m). Pyrite (0.4%m) was also detected in the tailings but was low enough not to affect the backfill strength over time.

2. The conventional tailings disposal test results were used to baseline the design models used by Paterson & Cooke and indicated that the final selected tailings pipeline size (~254 mm) would operate in turbulent flow at the design concentration of 45%m. The stationary deposition velocity tests showed a decrease in stationary deposition velocity as the mass concentration increased, which is typical of settling mixed regime slurries. The centrifugal pump derating tests showed that at BEP, the maximum head and efficiency derating was calculated at H_R=0.90 and E_R=0.85, respectively.

3. The thickening tests indicated that the optimum feed concentration, flocculant dosage rate, and solids loading rate to the thickener were 15%m, 25 g/t and 0.7 t/h/m^{2,} and an underflow of ~72%m can likely be achieved at plant scale. To further dewater the material, the test work indicates that

only pressure filters will achieve the TML requirements of 87%m (13% moisture) and that ~88%m can be achieved at equivalent chamber sizes between 30 and 60 mm.

4. The strength tests indicated a continuous increase in strength from 7 to 56 days, which supports the mineralogy findings that the 0.4%m pyrite presents, does not affect the backfill strength over time. Type B cement, a 42.5N cement, performs better than type A cement, a 32.5R cement, for the same water/cement ratio.

   The cemented rheology tests indicated that there is only a slight difference between the yield stress and plastic viscosity for the type A and B cement and that the minimum and maximum expected values will range from 90 to 108 Pa and 0.120 to 0.134 Pa.s, respectively, at a mass concentration of 70%m. The rheology data for the type B cement indicated that there is a sharp rise in the plastic viscosity from 70.4 to 72.5%m which is most likely the onset of slip at the wall of the bob which will most likely result in lower yield stress and higher plastic viscosities being measured at concentrations above 72.5%m.

5. Acid-rock drainage tests suggest that the tailings material is non-acid forming, given the low sulphide content (0.62%m). Additionally, leachate test work indicated that $FeSO_4$ salt effectively reduces arsenic leaching from both the tailings and backfill mixtures.

   Dosing $FeSO_4$ at 11.5 kg per tonne of tailings decreased cumulative arsenic leaching from backfill by 34.8%. However, the added salt significantly impacts backfill strength at low binder doses (<6% binder), causing strength losses of up to 45%. At higher binder doses (>6%), the strength reduction reduces down to only 8%. At 6% binder, the strength reduction is roughly 20%. Test work suggests the optimal $FeSO_4$ dose is between 4 and 11 kg per tonne tails.

6. All the various aspects of the tailings management system on the mine were designed based on the actual results of the test work campaign completed, thereby providing confidence in the final process, mechanical, piping, electrical and equipment designs. Construction of the paste plant and tailings system is underway, with commissioning targeted for Q3 in 2023.

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