Paste 2024 – AB Fourie & D Reid (eds)

doi:10.36487/ACG_repo/2455_38

Backfilling tailings above an active cave mine

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Abstract

The potential to backfill tailings above an active cave mining operation has been studied for the model case of a deep, near vertical, porphyry copper orebody in the humid tropics.

The concept has major social and environmental advantages, particularly in leaving behind a rehabilitated land surface capable of beneficial use and in reducing the area of the subsidence cone. Surface water flows could be better managed to ameliorate community concerns. Underground water management would be simplified and the risks from mud rushes at drawpoints reduced, and if iron sulphides are present in the orebody, costs for acid drainage control and treatment would be greatly reduced. A separate tails storage facility would still be required but would be reduced in volume by up to 50%.

Despite the advantages, the perceived risks have meant that backfilling tailings above an active cave mine appears not to be currently practiced. The major perceived safety risk is of a mud inrush with fatal consequences, as at Mufulira in 1970. Other risks are that ore may be contaminated by tailings, either by bulk flow through the broken rock, by mixing, or carried in percolating water; or that the tailings may prevent development of the cave or impede flow through drawpoints and ore passes.

This paper seeks to identify the causes of each risk and argues through a combination of small-scale test work and analysis of documented fatalities caused by mud rushes that all these risks can be effectively mitigated by appropriate design and management practices. The key requirement is that tailings be placed in dewatered form, either thickened to near-paste (yield stress >30 Pa) or filtered, and that a self-draining surface is always maintained, with no ponding of water permitted.

The test work conducted initially investigated migration of solids in seepage and bulk flow of near-paste copper tailings through a high voidage pebble bed. The tailings were observed to rapidly dewater, penetrating the bed to less than one pebble diameter. Liquefaction by shaking for two minutes resulted in flow to a maximum of three pebble diameters. There was almost no entrainment of solids in seepage. The potential for bulk flow of tailings through rock at high pressures, as the depth of tailings increases, was also tested using two different materials. With an applied head equivalent to 174 m of tailings, tailings penetrated the bed to a maximum depth of two particle diameters (20 mm).

A physical scale model of a caving operation was constructed and thickened tailings deposited in layers, followed by drawdown of ore. The test showed no mixing of tailings and rock, even with funnel flow of rock lying immediately below the tailings to the drawpoints. It appears the cohesiveness and lower bulk density of the tailings impede mixing.

Larger scale and project specific testing would be required before implementing the concept presented, but the results provide strong support for considering tails backfill for a new cave mine.

Keywords: *backfill, paste tailings, filtered tailings, cave mining, tropics*

1 Introduction

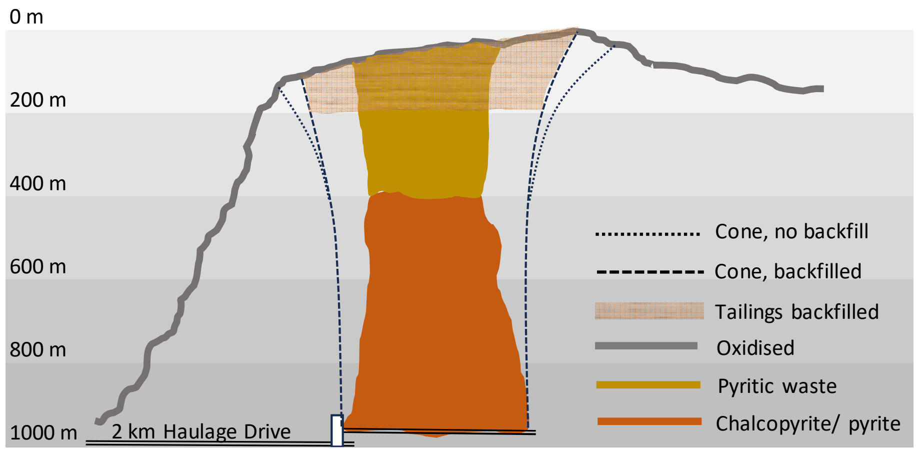
The use of caving (sublevel and block) methods for underground mining is widespread for orebodies with suitable geometry and geotechnical properties due to cost effectiveness, high degree of mechanisation and high ore recovery, albeit at the cost of some dilution (Kvapil 1992). The Australian Centre for Geomechanics currently lists 11 past and present caving projects in Australia alone (acgcaving.com). Although underground mining methods in general can reduce environmental and social impacts compared to open pit mining, cave mining as currently practiced involves:

- A requirement to store all tailings in an impoundment at surface.
- A subsidence cone, which is difficult to rehabilitate to any beneficial use and is a safety hazard in populated areas.
- Depending on climate and geography, additional serious impacts can be:
  - Where mine haulage drives exit to surface below the orebody (Figure 1), acid drainage flows may occur and persist for centuries (Koehnken 1997).
  - Surface water flows may be diverted into the cave, affecting other users.

These impacts could be substantially ameliorated if tailings could be placed in and over the subsidence cone. This could be done after mine closure without risk to operations, but for a project with a single orebody the resulting end-of-mine costs would be very high and not generally economically feasible. However, the alternative of placing tailings over an active underground mine has been considered too risky by mining regulators (Hamman, pers. comm., 2012); a view strongly influenced by the 1970 disaster at the Mufulira mine in Zambia in which 89 men lost their lives due to an inrush of tailings material that had been stored over the mine. In fact, as discussed further, this incident was the result of decades of inappropriate tailings disposal and mining practices and after a detailed investigation (Sandy et al. 1976; Neller et al. 1973), the tailings were stabilised and mining recommenced and continued underneath the area where the failure had occurred. The purpose of this paper is to demonstrate that placing tailings above active caving mines would have many benefits and that the identified risks can be managed for a safe outcome, within the context of stringent draw practices. The approach taken is to present a 'model' mine which is based on real projects the author has personal experience with but with features selected to facilitate discussion of a wide range of risks and benefits.

2 Model mine

The model mine exploits a porphyry copper orebody which consists of a payable zone carrying chalcopyrite and pyrite, overlain by a waste zone with substantial pyrite. The orebody is in the humid tropics, with annual rainfall substantially exceeding evaporation and occurring throughout the year, but at varying intensity. The mine is close to a small town and the whole area is farmed for cash crops and has some tourism. Surface streams are captured for use in agriculture and for generating hydroelectric power. A shallow layer of weathered rock overlies the orebody. The orebody lies within a mountain and is accessed by a haulage drive below the base of the payable ore, with drainage occurring by gravity to the portal. Figure 1 shows a simplified cross-section of the mine. The sublevel cave design parameters are taken from Kvapil (1992).



Sub-level cave design parameters	
Sub-level height	12.5 m
Drift spacing, horizontal	12 m
Drift width	5 m
Drift height	3.5 m
Burden	2 m
Annual production	5 M t
Proportion tails stored	50%
Rate of rise of tails	30 mm/day

Figure 1 Cross-section of model mine with assumed sublevel caving design parameters

After clearing vegetation and storing topsoil, the surface over the expected subsidence zone would be profiled to enable placement of a self-draining tailings deposit. Berms would be constructed to manage supernatant liquor and drainage. Figure 1 shows that the original surface above the deposit is naturally draining and this would usually be the case or could be engineered to be so. The concept presented in this paper is therefore not directly applicable to a caving mine that commences below the base of an existing open pit.

Tailings would be deposited either thickened or as filter cake so there would be little liquor release and a substantial yield stress. Deposition would commence before cave breakthrough to surface to form a blanket thick enough to withstand damage from traffic and maintain continuity as subsidence commences. Deposition would then continue in line with subsidence and would be managed to prevent the formation of depressions and maintain a continuous self-draining beach slope.

Throughout this paper, the use of the word 'tailings' in the context of placement refers to thickened tailings with a yield stress >30 Pa, unless stated otherwise. The term 'tailings' when referring to previously deposited material means material that has dewatered and consolidated, but the resulting moisture content depends on the conditions.

3 Backfilling advantages

The overarching benefit of backfilling the subsidence cone with tailings is that it would reduce many of the environmental and social impacts of cave mining and make it easier to achieve environmental approvals and social acceptance.

Advantages which would apply to all caving operations are:

- At the end of the mine life, the subsidence cone would be largely or completely filled and would be rehabilitated as required. In some cases, limitations on the tailings slope would mean there would be a remaining highwall which would need to be made safe. Alternatively, it may be better to use filtered tailings in such cases as they can be stacked with a steeper slope.
- Due to swell, not all tailings can be stored in the subsidence cone, but it is estimated about 50% could be which is a major reduction in the area required for a separate tailings storage facility.
- Since the tailings would stabilise the upper sections of the subsidence cone, the area affected by subsidence would be substantially reduced.

In the case of the model mine, there are some additional advantages:

- The reduced area of the subsidence cone, together with improved opportunities for access, may allow pre-existing surface water flows to be maintained or at least reinstated.
- The deposited tailings will release drainage water which will enter the mine. Some stormwater will also infiltrate, but most will drain over the beach slope, particularly in areas of freshly deposited tailings. Water inflows to the mine will be reduced overall and high flows will be eliminated. Based on engineering studies for a similar mine, water inflows to the mine via the subsidence cone would be reduced from an annual average of 80 L/s to 30 L/s. Peak flows for a 1:2 year storm with no tailings backfill would reach 4,000 L/s by the end of the mine life, whereas with backfill, the peak flow would be the same as the average, 30 L/s. These values account for the reduced diameter of the subsidence zone with backfill and assume that all water released during consolidation of the emplaced tailings drains into the mine. This is likely given the free draining base and the constant disturbance of the tailings as caving proceeds which was observed in test work (see Section 5.5) to release moisture.
- The reduction in high inflows will reduce the risk of underground mud rushes at drawpoints.
- The placement of tailings will reduce acid drainage production in two ways:

- The moisture content of the tailings at deposition will result in much lower oxygen diffusion rates than broken ore or waste. The surface layer of tailings will generally be saturated and will have an effective oxygen diffusion rate of about $2 \times 10^{-11} m^2/s$ (David & Nicholson 1995), compared to $8 \times 10^{-7} m^2/s$ for a waste rock column studied at Woodlawn Mine in Australia (Jeffery et al. 1988). This represents a 200,000 times reduction in oxygen flow to the mine which, in the absence of neutralisation capacity in the ore or waste, will translate into a proportional reduction in treatment costs.
- By reducing and smoothing volumetric flows, the tailings will reduce the capital and operating costs for treating any remaining acidity, possibly to the point where passive treatment methods may be applicable, particularly after mine closure.

Once the top of the mine has been sealed, further reductions in acid drainage production could be achieved by sealing access drives on each sublevel as mining is completed, thus preventing oxygen accessing pyritic material through abandoned mine workings.

It is becoming increasingly common for environmental authorities to limit the sulphate concentration in mine site effluents (Nevatalo et al. 2014; Government of Colombia 2015). Sulphate reduction is difficult and expensive (Leão et al. 2014) compared to pH adjustment and metal precipitation so in cases where this is required, the cost reduction by limiting oxygen access to the mine will be much greater.

Backfilling with tailings may also reduce tailings handling costs, but this will be site specific and in some cases costs may be higher.

4 Risks

Potential risks have been identified by considering all aspects of the process of backfilling with tailings and by discussions with colleagues over the past 10 years. The risks identified are first listed and then analysed in more detail:

1. Liquefaction of tailings deposited in the subsidence zone resulting in catastrophic inflows to mine workings and causing multiple fatalities.
2. Localised mud rush.
3. Contamination of ore by tailings by:
    a. Flow of tailings slurry under pressure to drawpoints through the rock mass or via deep tension cracks at the boundaries of the subsidence zone (Laubscher c. 2000).
    b. Tailings migrating in percolating water.
    c. Tailings solids preferentially descending through caved rock movements (Laubscher c. 2000).
4. Sterilisation of resources which might be mineable with a change in economic conditions.
5. Mixture of tailings and rock interfering with operations by:
    a. Preventing caving.
    b. Impeding flow through drawpoints and ore passes.
6. Transport of tailings from the process plant to the discharge point.
7. Cannot operate equipment above an active cave mine.

4.1 Tailings liquefaction

Documented cases of underground mud rushes have been discussed by Butcher et al. (2005). Most of these cases did not involve tailings and will be discussed in Section 4.2. The only relevant case is that discussed in

Section 1, the Mufulira mine disaster in 1970. The situation leading up to this disaster has been described by Neller et al. (1973), Sandy et al. (1976), Tembo (2019) and Vutukuri & Singh (1995).

Production mining commenced at Mufulira in 1933. Between 1933 and 1943, more than 3.9 Mt of slime tailings were used to fill hollows to inhibit mosquito breeding, and then to fill sinkholes up to 25 m deep. In 1945, block caving commenced and became the primary mining method but by the 1960s had transitioned to sublevel caving with a much higher output per drawpoint and less care to prevent chimneying. This led to progressive formation of a surface depression which included the whole of no. 3 dump. By 1956, the entire original ground surface had subsided to form a depression 12 m deep and by 1963, a single large pool of surface water had formed containing 30,000 m^3 of water. As the catchment area was 510 Ha, attempts to pump the depression out failed. In the 1960s, increasing quantities of unthickened tailings were deposited in an unsuccessful endeavour to obtain a gently sloping surface to encourage runoff. By 1970, more than 20 Mt of tailings had been deposited, mainly by open-ended discharge.

From early 1970, mud extrusions were noted underground, but consisted of talc and argillaceous material, resembling material from the hanging wall aquifer. It was recognised that this mud might be followed by tailings but in the event, the tailings broke through without further warning on 25 September 1970 with an estimated 450,000 m^3 flowing into the mine over approximately 15 minutes at a flow rate of 500 m^3/s. The flow entered the mine at various points about 700 m below surface, then flowed into lower levels via the Peterson shaft system, causing 89 fatalities.

A five-man Commission of Enquiry (Commission) was formed (Government of the Republic of Zambia 1971; Vutukuri & Singh 1995) and concluded that the practice of disposing tailings on caving hanging wall ground was 'basically wrong.' However, the Commission also concluded that 'The accident was caused by faulty operational procedure: a (mining) method, a tailings disposal practice, and a drainage scheme, all sound enough in their isolation, became a dangerous combination.'

The disaster halted production from the richest section of the mine for many months. However, following extensive laboratory and field studies (Sandy et al. 1976), the tailings were dewatered and stabilised using tube-wells and paddocks constructed within which stormwater inflows and evaporation would balance over the year. It was concluded that:

> *'The field work, laboratory tests and basic research work have shown, provided that the moisture content of the tailings in no. 3 dump is reduced below a certain level, that mass mobilisation and continued flow of the material cannot become a hazard. The risk to mining is no greater than with the normal soil cover in the area.'* Sandy et al. (1976)

And that:

> *'There is no doubt that the tailings can be an entirely satisfactory material to form the surface over a caving area. The properties of the material described give it a self-healing ability. The strength derived from the apparent cohesion enables vertical faces of limited height to be formed, but the displacements likely to be caused by underground caving would be deeper than the maximum sustainable height: they would produce slides in the material, and would tend to heal further by creep. This self-healing property, together with low permeability and high strength, make for a tight and resilient cover. The final criterion lies in the water content and density of the material. A completely drained mass is clearly the end-product from the safety viewpoint; but it is likely that safety can be achieved by partial drainage, provided that the density is high enough over the entire mass – particularly when it is remembered that the lower levels, which are the last to drain, are also those which have been subjected to the greatest consolidating load.'* Sandy et al. (1976)

Subsidence mining resumed under the western end of the no. 3 dump by mid-1971 and as of 1976, it was expected that a progressive return to subsidence mining would occur under the rest of the dump.

In summary, it appears that the conclusion of the Commission in 1971 that tails should not be stored on caving hanging wall ground has been taken as inviolable in the decades since, but the context and results of subsequent investigations have been forgotten. In particular, the development of thickeners capable of routinely producing tailings with yield stresses above about 30 Pa, or large-scale pressure filters, dates from the late 1980s (Schoenbrumm 2011) and was not conceivable in 1970.

4.2 Mud rushes

Mud rushes in mines have caused multiple fatalities (Butcher et al. 2005). They require the presence of both mud and water, some disturbance leading to mobilisation and an entry to the mine workings.

A mud rush occurred on 17 January 2014 at the Mt Lyell mine in Tasmania and resulted in the death of one miner, who was the operator of a 60 t Sandvik loader at a drawpoint. A coroner's enquiry was held (Cooper 2021). The incident occurred in the Prince Lyell mine, which is the extension at depth of the historic West Lyell open cut, shown in Figure 2. The Prince Lyell underground mine commenced mining by the sublevel open stoping method, but in 1995 changed to sublevel caving. The column of broken ore extended to the base of the original pit.

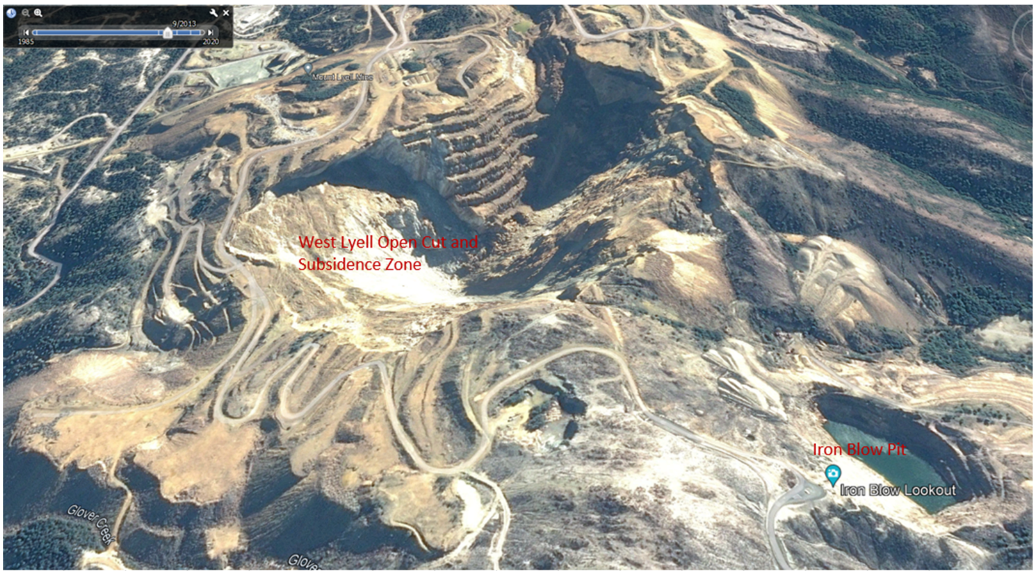


Figure 2 West Lyell and Iron Blow open pits, Queenstown, Tasmania (Google Earth 2013)

Evidence was given by the mine's technical services superintendent and senior mine geologist that mud rushes occurred around once a year and mainly in winter when the rainfall was higher. Warning signs of an incipient mud rush included changes in the ore moisture content, changes in the behaviour of the rill, increased proportion of fine material and the temperature and colouration of the drainage, i.e. green or hot water indicate increased risk. None of these signs were observed immediately ahead of the 2014 event, although instability had been noted in the days leading up to failure.

The coroner stated it was reasonable to conclude that the risk of mud rushes is increased when cave mining is conducted below standing water, but as Figure 2 shows, there is no standing water in the West Lyell open cut and although the Iron Blow pit 750 m to the northeast has permanent water, there is no man-made connection between the Iron Blow and the Prince Lyell (Clyde et al. 2013). From work conducted by the author at Mt Lyell from 1994 to 1998, rainfall does strongly influence inflows to the mine but with a variable response rate that can extend over days, dependent on the rainfall duration and intensity. Annual rainfall at Mt Lyell is high, averaging 2,404 mm/y over the period from 1964 to 1995 (Bureau of Meteorology 2023). The Mt Lyell operations have a long history of acid drainage generation (Koehnken 1997), with both oxidation and neutralisation occurring within the orebody. Operationally, the ore through the process plant varied from being free flowing to extremely sticky and many neutralisation products with clay-like properties are observed in the mine workings. The comments by the mine staff at the inquest support the proposition that mud rushes are related to pockets of high clay material and/or zones of chemical activity which mobilise under high flow conditions.

In this situation, as was shown in Section 3, placement of tailings with a self-draining surface over the caved ore for a new mine would almost eliminate the oxidation–neutralisation cycle and greatly reduce the likelihood of developing pockets of clay-like neutralisation products, as well as reducing peak flow rates. It is worth noting that the Mt Lyell mine has not re-opened following the mud rush in January 2014.

4.3 Contamination of ore by tailings

The potential for contamination of ore by tailings is an economic rather than a safety risk, but nonetheless important. Laubscher (c. 2000) discussed a shallow cave mine in Canada where drawpoints froze during the winter. Dry tailings were placed over the mine to stop the inflow of cold air, but the tailings reported to the drawpoints. Laubscher also notes the possibility that tension cracks forming on the perimeter of the subsidence zone could cause large quantities of surface water to be directed into the caved zone and possibly the same could happen with tailings slurry. Similarly, there might be potential for bulk flow of tailings slurry through the rock mass as the depth of tailings and consequent pressures increased. A third mechanism for contamination could be fine tailings suspended in seepage and flowing to drawpoints.

These risks were examined via a series of small-scale experiments, reported in Section 5.

4.4 Sterilisation of resources

The placement of tailings above an orebody could, in some cases, limit the future extraction of lower grade ore. For the model mine, the placement of tailings in the subsidence cone would have no effect on the future potential to extend mining at depth, or to mine ore offset at depth. If the broken waste at some time became economic to mine, the presence of consolidated tailings might mean that the full depth of waste could not be drawn. Material lying in a halo immediately adjacent to the deposited tailings might also be unrecoverable.

4.5 Mixed tails and rock interfering with caving

The presence of tailings could affect the ability of the cave to develop and the flow behaviour of the ore to drawpoints and in ore passes and bins. It could not affect the initial ability of the cave to propagate to surface as there can be no contact between tailings and rock up to that time other than potentially in tension cracks. Tailings cannot interfere with the formation of the airgap essential to the caving process, as this airgap no longer exists once caving reaches the surface (Lecuyer et al. 2020). Kvapil (1965) studied the flow of coarse granular material in hoppers and bins and identified four classes of material ranging from type 1 with the highest mobility to type 4 with the lowest. Type 4 consists of large rock pieces, chippings, sand and earthy–clayey constituents. These earthy–clayey constituents are liable to plastic deformation at a certain moisture content, stick to large pieces and fill cavities. The greatest operational difficulties with coarse material are caused by arching, which can occur when there is a restriction of flow at the outlet opening. Increasing quantities of finer material require larger openings and in the case of ore passes, steeper slopes. Type 4 material with greater than 10% earthy–clayey constituents cannot be handled through bins and ore passes built of rock, even if vertical.

It is concluded that consolidated tailings mixed with ore would produce a type 4 material and cause operational problems at drawpoints and in the subsequent ore handling system, particularly in ore passes. Consolidated tailings may also reduce the size reduction process that occurs as rock descends through the cave, by a cushioning effect. However, these effects will only occur if there is substantial mixing of tailings solids with ore. The risk of this has been evaluated in small-scale tests (see Section 5.5).

4.6 Transport of tailings to the discharge point

There are many ways by which tailings could be transported to the subsidence cone, as a slurry or as filter cake, so the risk is mainly that the cost of transport is prohibitive. For the model mine scenario, the best option would likely be to use a high-pressure positive displacement pump to pump along the haulage drive and up a raisebore to discharge to an agitated storage tank near the high point of the tailings beach.

Approximate friction losses were estimated, using the method developed by Cowper and Thomas in Rivis et al. (1993), as 703 Pa/m, for 65% solids by weight and a slurry density of 1.718 g/mL. Assuming the design criteria shown in Figure 1 gives a power consumption of 5 kWh/t for a cost of AUD 1.24/t tailings at AUD 0.25/kWh. The pump discharge pressure would be about 19,000 kPa with installed shaft power for the pump of 1,780 kW, for a single lift. A heavy-walled steel pipe would be adequate, probably schedule 140 or 160 to allow for wear. Weir Warman advised that this is within the range of current Geho production pumps, but the combination of high pressure and high throughput would place the duty at the upper end of current experience. Once tailings backfill is committed to, tailings must be available to place whenever underground production is in progress. Hence it would be better to install two pumps and pipelines, each capable of supplying about 70% of design capacity. This would enable tailings to be pumped at a higher rate when necessary to fill in any subsidence areas or to catch up after stoppages. Thus, although the cost of transporting tailings may be substantial, it is unlikely to prevent the adoption of backfill when balanced against other factors.

4.7 Operation of equipment above active cave

Tailings could be placed over a cave mine as a thickened slurry without the need to routinely access the tailings surface. This is normal practice in a central thickened discharge storage facility. However, if tailings are placed as filter cake or possibly a cemented mix then access would be essential. In all cases, the ability to locally manage deposition or drainage would be beneficial. Assuming the model of draw mechanics presented by Hustrulid & Kvapil (2008) and 30 active drawpoints at any time, local subsidence could be up to 12 m above drawpoints if the draw ellipse extended to surface. In practice, this would not occur in the model mine as tens of metres of tailings would lie over the rock by this time. The tailings themselves would provide a self-healing surface and would tend to smooth the subsidence effects (Sandy et al. 1976). The use of large-scale equipment would protect the operators. However, an essential step in advancing the concept of backfilling would be to collect detailed subsidence data from existing cave mines to develop a safe operating strategy. Depending on topography, any high walls that formed around the overall subsidence area would also have to be continually made safe.

5 Test work

Small-scale laboratory testing was used to directly evaluate risks which could not be assessed from publicly available industrial experience, as discussed earlier. These risks were:

- Bulk flow of tailings slurry to drawpoints. This could be both a safety and an economic risk.
- Tailings solids migrating in seepage.
- Consolidated tailings preferentially descending through caved rock movements.
- Mixtures of consolidated tailings and rock impeding flow through drawpoints and ore passes.

Some risks associated with implementation of tailings backfill, particularly safe placement and management, cannot be evaluated in the laboratory. These are further discussed in Section 6. Test work at the scale feasible in this study, whilst very informative, does have some limitations. These are discussed as required in the following sections.

5.1 Tailings materials

Two series of tests were carried out; the first in 2022 and the second in 2023. Tests in 2022 investigated the bulk flow of tailings under low heads, migration of tailings solids in seepage and bulk flow of tailings under high heads. In 2023, testing of bulk flow under high heads was repeated with some improvements to the method and a different tailings sample (although with no material change in the findings), and a physical model was built to investigate the behaviour of emplaced tailings in the ore/waste cave column.

The tailings used in 2022 was from pilot plant rougher flotation for a South American porphyry copper project. The nominal grind size for this ore is 80% <150 µm, but the ground ore has an unusually high content of fines at 44% <38 µm. Some properties are shown in Figure 3 under Tails 1. The pilot plant tails had been filtered and oven dried, and samples split out and stored in sealed bags and drummed for future test work. For the work described here, representative sub-samples were split out dry and water added to generate slurry as required for each test. Tap water from Perth, Western Australia, was used and had previously been shown to produce the same results as site water.

For the tests carried out in 2023 it was intended that tests would be carried out with a fine pilot plant copper tailing similar to Tails 1 (called Tails 1 a) and also with a coarser material, at the coarse end of the range of base metal operations, termed Tails 2. This coarse tailing was made up to have a similar size distribution to the Bougainville Copper tailings (Hinckfuss 1975). The Bougainville plant tailings had a wide size distribution, shown in Figure 3. The simulated Bougainville tailings material was prepared from four source materials: Tails 1 (14%), two size fractions from a vein-hosted fine crushed (not ground) underground gold ore (-500 µm (36%) and -250 µm (40%), and quartz sand (10%). The prepared sample matched the plant distribution reasonably closely, but with less fines.

Sieve	Sieve Sizing Cumulative % Passing			
	Tails 1		Tails 2	
µm	Plant	Test	Plant	Test
710	100.0	100.0	98.0	97.7
500	100.0	99.4	95.5	95.1
300	100.0	99.1	86.0	83.7
212	98.8	95.7	77.5	74.7
106	75.6	72.5	60.4	54.4
75	63.7	61.8	51.7	44.9
38	43.7	42.7	38.0	29.6
	Hydrometer Sizing			
Sand	37	50	ND	63
Silt	51	41	ND	34
Clay	12	9	ND	3
Class	Loam	Loam	ND	Silt-loam
$SG_{hydrated}$		2.67		2.97
$SG_{anhydrous}$		2.70		2.70
Plastic limit		17%		NP
Liquid limit		20%		c30%

ND Not determined or unknown

NP Non plastic

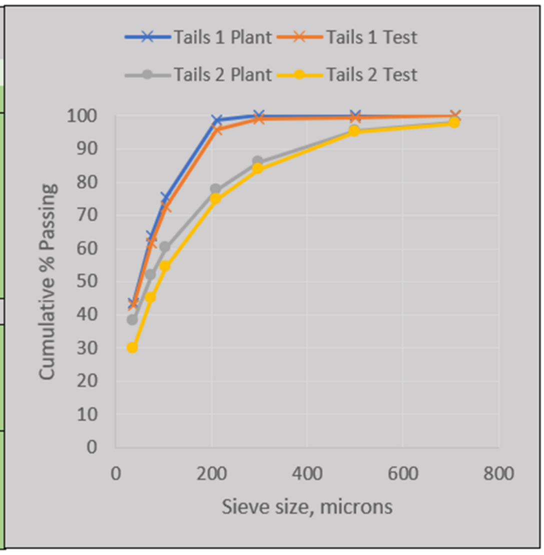


Figure 3 Properties of tailings materials used in test work

For the tests in 2023, a target yield stress of 50 Pa was adopted for the slurry, rather than a specific pulp density, as this is within the operating range for a high compression thickener and can be pumped with appropriately specified centrifugal pumps but offers most of the benefits of a higher yield stress paste. A series of 12 small samples of Tails 1 a were representatively split out dry and made up to pulp densities between 71% and 92% solids by weight. Yield stresses were measured using the Boger 50c rheometer method (Pashias et al. 1996). All yield stresses reported in this work were determined using a PVC plastic cylinder 42.3 mm internal diameter and 40.9 mm high. For samples with no measurable slump, the unconfined compressive strength (UCS) was estimated using a geotester pocket penetrometer with a ¼ inch tip. These tests were carried out without first ageing the samples in contact with water, although the material had been in prolonged contact (days to weeks) with water before filtering and drying. The pulp density required for a yield stress of about 50 Pa on Tails 1 a was established as 72.5% solids. A bulk sample was then prepared for test work at that density, with an actual measured yield stress of 77 Pa. Overnight, it hardened to a concrete-like material with UCS >600 kPa (measured in situ by pocket penetrometer) which could not be removed from the mixing vessel. No binders had been added to this material and it was in a sealed container. Testing with this fine tailing material was unable to proceed.

The coarse Tails 2 material exhibited similar properties despite 86% of the components being from a different orebody, so the target density was established by mixing samples over a range of pulp densities and ageing overnight in a closed container before measuring yield stress. Throughout this work, precautions were taken to minimise moisture loss during processing as this was known from previous work to be a significant problem in the dry Perth climate. This established a target of 62.5% solids for the bulk test work samples. A series of representative dry samples were prepared so the bulk slurry samples required for test work could be made individually, about 24 hours before required. Thus, all samples were aged in contact with water for the same time.

In practice, the target 62.5% solids proved too low and the aged bulk samples were adjusted by evaporation in a low temperature oven to an average of 69% solids for 50 Pa yield stress. It is not known why the bulk samples needed a higher density than the initial small test samples to achieve the target yield stress. One reason could be the formation of agglomerated lumps, which could not be fully dispersed, with ageing of the bulk samples.

Some additional tests were carried out to investigate the ageing, or hydration, process. Figure 3 shows that both Tails 1 and 2 had the same solids true specific gravity (SG) when measured on dry solids (by helium pycnometer or in acetone). After hydrating, however, the SG for Tails 1 was measured in water as 2.67 whilst that for Tails 2 increased to 2.97. Tails 1 had been ground wet then dried while 86% of the components in Tails 2 had never been in contact with water. The small reduction in SG with Tails 1 in water may be due to some air entrainment which is difficult to completely eliminate. Samples of Tails 2 were also made up to a carefully measured % solids by weight, then dried to determine how much water had reacted with the solids and was not removed by drying. The work is continuing but preliminary results show 3.2% water reacted with solids after 24 hours, rising to 4.0% after 72 hours. For thickened tailings, such changes in moisture content can substantially increase yield stress. Since run of mine ore entering a plant is typically only in contact with water for a few hours before discharging to the tailings storage facility, these results imply that the length of time thickened tailings samples are left in contact with water during testing, before adjusting the slurry density, can substantially affect the outcomes.

5.2 Migration of tailings solids in seepage

It was shown in Section 3 that tailings placed over the subsidence cone could be expected to generate close to 900,000 m^3/y of water draining into the mine workings. There would also be some additional flow from stormwater infiltration into non-active tailings deposition areas but based on previous test work by the author on tailings samples, this would be a much lesser amount. Two tests were carried out to determine how much solids this seepage water might be expected to carry.

Rounded quartz pebbles with a top size of 20 mm were taken and half were crushed to produce a mix with a low packing density and high void space, such as might occur in the upper part of the cave. The measured dry density was 1.5 g/mL and the void ratio was 0.67. The pebbles were loaded into an Imhof sedimentation cone and water run through to remove fines. A 224 mL sample of freshly prepared tailings slurry was poured onto the pebble bed with density 72.0 % solids by weight and a yield stress of 55 Pa. Water drained down over the subsequent 19 h with 19 mL collected. The water was visually clear (Figure 4a). A further sample of 240 mL of tailings was poured, followed by water to a depth of 7 mm and again the drainage water was clear.

This test was not repeated with the much coarser Tail 2, but drainage from the column test (Section 5.4) had a suspended solids content of 3 mg/L (estimated using a Hach DR/2010 spectrophotometer and method 8006), which represents a solids load of less than 3 t/y.

It is concluded that the cohesive nature of consolidated tailings is likely to prevent mobilisation of fine solids from deposited tailings. This result is consistent with observations of industrial filtration where concentrations of suspended solids are normally low once a filter bed has formed. However, it should be confirmed at larger scale with project specific test work before application.

5.3 Bulk flow of tailings under low heads

The potential for tailings to flow through the caved rock and contaminate ore was investigated in 2022 through a series of experiments. The first experiment was described in Section 5.2 and entailed two successive pours of tailings over a bed of pebbles in a transparent Imhof cone. A photograph is shown in Figure 4b.

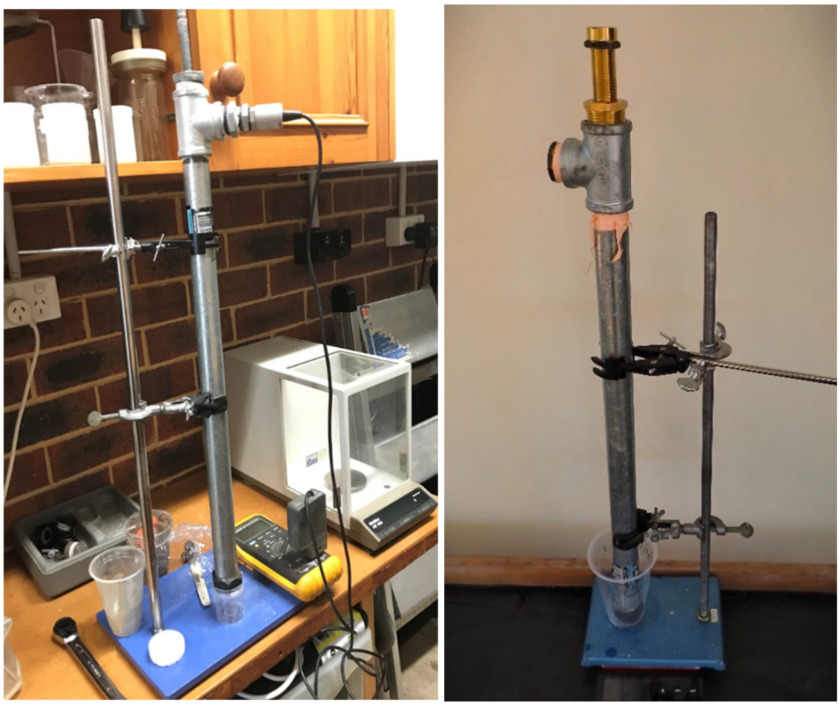
(a) (b)

Figure 4 (a) Side and top views of drainage water collected from tailing; (b) Two pours of thickened tailings

The tailings flowed to a depth of 15 mm (slightly less than the pebble diameter) in the centre of the cone and to 30 mm at the walls. Close observation suggests that when the slurry reaches a contact point between the pebbles, a small quantity of tailings passes but the increase in surface area and short drainage path leads to rapid water loss and increase in yield stress, thus preventing further movement. The cone was then subjected to two rounds of vigorous manual horizontal shaking and vertical tapping for two minutes each time, to approximate a severe earthquake. Horizontal acceleration was measured as >2 g using the Arduino SJ app on an Apple iPhone 7, which equates to extreme shaking (X) on the Modified Mercalli intensity scale (Wikipedia 2024; New York Hall of Science 2024). Tailings flowed to a greatest depth of 185 mm along the walls, but only 60 mm within the pebble bed (three maximum pebble diameters). These depths would not appear to be a concern in a full-scale operation, although more formal studies of potential flow under design earthquake conditions and at a larger scale are required.

5.4 Bulk flow of tailings under high heads

The experiments described in Section 5.3 demonstrate that the pebble bed acted as an effective barrier to the flow of tailings under low heads. However, for the model mine, the column of consolidated tailings would eventually have a maximum height of about 180 m, resulting in a pressure of about 3,500 kPa at the interface with the rock column. This much higher pressure might result in tailings being forced through the caved bed resulting in mass flow of tailings into the ore and to the active sections of the mine. To test this, in 2022 a 600 mm length of 25 mm standard galvanised steel pipe was filled to within 200 mm of the top with quartz pebbles crushed to -10 mm. A pipe cap of HDPE with 3 mm drainage holes drilled in it was used to retain the pebbles in the pipe. A galvanised tee piece was fitted to the top of the pipe, with a Fluke pressure sensor model PV350 fitted in the side arm. This sensor can measure up to 3,400 kPa. A photograph of the set-up is shown in Figure 5a. Tails 1 material was freshly mixed each day and loaded at 72% solids at a rise rate initially of 30 mm/day for three days, to match the estimated production rate for the model mine and give time for a realistic consolidated bed to form. After the three days, tailings were loaded up to the tee and pressure was applied by driving in a 10 mm diameter by 100 mm long bolt. This bolt did not directly apply pressure to the tailings, but by taking up volume exerted pressure indirectly.



(a) (b)

Figure 5 (a) Initial experimental arrangement to test flow of tailings under high heads; (b) Revised arrangement, 2023

Despite several cycles of driving in the bolt then withdrawing and topping up with tailings, the required pressures could not be achieved so the bolt was replaced by a length of ½ inch Whitworth threaded bar and using this a pressure of 3,200 kPa was achieved. However, these high pressures could not be maintained even after continually topping the column up with tailings added at 60% solids. The lower density used for topping up was to minimise air entrapment. After 13 days of cycles of pressure loading and adding tailings it was calculated from a running water balance that the tailings were still at 77.8% solids which is higher than the approximate liquid limit. That and the volume balance implied that tailings were being forced through the pebble column. It was decided to halt the test and inspect the column contents.

The column was carefully dismantled to recover the contents in sections. The tailings were observed to be highly compacted. The tailings recovered from the tee piece were at 87% solids, rising to 90% solids immediately above the pebble bed. This is higher than the best achievable in pressure filtration testing. Tailings only penetrated the pebbles to a depth of about 20 mm. The estimated saturation in the tailings in the upper part of the pipe was 73%. It was concluded that water had been lost from the column throughout the test, mainly by leaking along the threaded rod, but was not detected due to evaporation. Almost no flow of tailings had occurred.

The column was redesigned to enable the mass balance to be more accurately tracked and the test was repeated in 2023. The pressure gauge was removed, and the side arm sealed. Pressure was applied directly to the tailings column using a length of 15 mm threaded brass pipe inserted through a reducing bush and a steel piston which was a sliding fit in the pipe. The disadvantage of this system compared to the first design is that the pressure applied to the ore can only be estimated by calculation using bolt torque approximations (Engineers Edge 2024), not measured. The advantage is that the ore and tailings are contained in the pipe thus eliminating any chance of leakage, and the assembly can be accurately weighed. The pipe was filled to approximately 200 mm below the top with a simulated ore/waste mix ('ore'), discussed in Section 5.5.

The assembly was weighed at all stages to track the water balance. Figure 5b shows the assembly on the balance. Water was passed through the column and allowed to drain to pre-wet the ‘ore’ so that drainage from the tailings would be fully recovered. The column was then filled over three days with Tails 2 slurry samples, prepared as described in Section 5.1, with an average yield stress of 50 Pa. Two charges of tailings were initially placed over two days, the first about 30 mm thick and the second 60 mm. On the third day, the column was filled to the top and was allowed to partially drain and consolidate. This is less consolidation time than would occur in operation.

After 18 hours a torque of about 490 N-mm was applied to the threaded pipe, equivalent to an estimated pressure of 335 kPa. The tee piece was removed and it was found that the top of the tailings was 10 mm lower and the tailings had a UCS (by pocket penetrometer) of 206 kPa. The tee piece was replaced and the pressure on the tailings increased to an estimated 1,209 kPa and maintained overnight. The tailings surface was again inspected and was found to have descended just 2 mm and increased in UCS to >608 kPa. The estimated pressure on the tailings was increased to 6,047 kPa, which substantially exceeds the expected maximum pressure in the model mine. After leaving overnight, the column was dismantled. The ‘ore’ was able to be poured out of the column, but the tailings were recovered using a long-fluted drill bit.

Inspection of the products showed that the tailings only penetrated about the top 20 mm of ore, as shown in the photograph of the contact zone in Figure 6a. The average percent solids of the tailings was 76.7%, or 30% moisture dry basis and the saturation was estimated as 83%. The data in Figure 6b shows that 95% of the solids loaded was recovered, with some tailings cemented to the inside of the pipe and not able to be removed. However, 12.4% of the water added was not accounted for. The assembly was weighed before and after every addition and weights agreed closely. Moisture samples were taken immediately before loading. The unaccounted moisture represents 2.8% of the mass of material loaded in the column so it is probable that at least some of that loss was due to irreversible reaction with the solids.

Although the diameter of the column used in this test work was small and wall effects are possible, the results from test work in a larger box, reported in the next section, also show negligible penetration of tailings into the ‘ore’ bed.

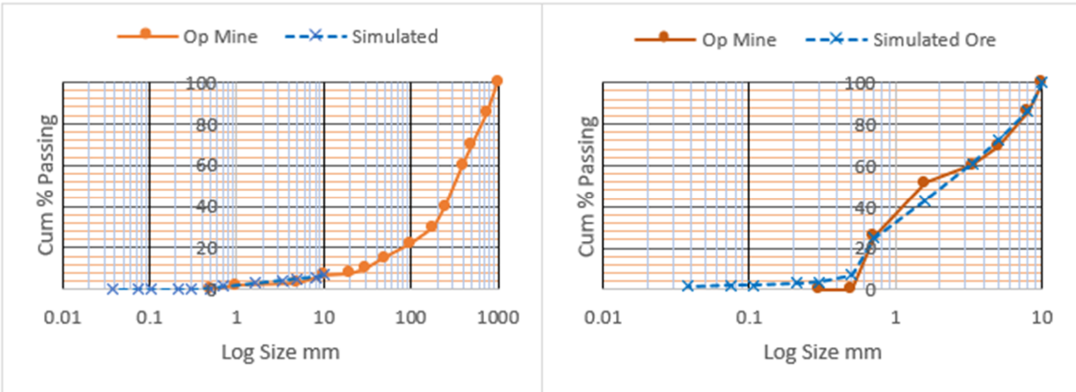
Material	Loaded	Recovered	Lost
	g	g	%
All solids	470	449	4.6
"Ore" solids	321	318	0.9
Tails solids	130	148	12.2
Water	112	99	12.4
Moisture Contents - % solids w/w			
"Ore" solids	89.2%		
Contact zone	76.5%		
Tails	76.7%		

(a) (b)

Figure 6 (a) Contact zone; (b) Mass balance and moisture data for mass flow test on Tailings 2

5.5 Mixing of tailings and caved rock

The extent to which tailings deposited over caved rock will mix and move with or through the rock affects the extent to which tailings can impede the flow of caved rock to and through drawpoints and ore passes, and the amount of ore dilution. Heslop (2010) summarised studies on the flow of caved rock and the influence on recovery and dilution. Three methods have been used: physical models most often using sand, but sometimes scaled ore and waste, marker tests in operating mines, and computer modelling. Heslop has shown that full-scale behaviour does not necessarily match the results of small-scale studies and it is beyond the scope of this study to try to predict overall waste flow and dilution effects. However, Kvapil (1965, 1992) showed that the formation of arches impeding material flow is related mainly to particle top size, with secondary effects of size distribution and material characteristics. A laminated glass fronted box was built at a scale of 1:100 based on the model mine dimensions and filled with broken rock also scaled at 1:100, to maintain the correct ratio for flow interruptions from arch formation. Heslop (2010) reports that tests were carried out at the Climax Molybdenum Mine in 1945, at a scale of 1:120 and at the Shabanie Mine between 1974 and 1980, at a scale of 1:100. No distinction was made in the test between ore and waste which are referred to here as 'ore'. The size distribution was assumed to be constant with depth. The size distribution of the 'ore' was based on test data from an Australian sublevel cave mine (Dunstan G, pers. comm., 2019). A question arose whether the size should be scaled by scaling the entire distribution, or by using the part of the full-scale distribution that is less than the scaled top size. The distributions are similar in both cases, but scaling the entire distribution down would result in a lot more ultra-fine material which would be likely to give unrealistic flow properties, so the part of the full-scale distribution below 10 mm was used. Figure 7 shows the full-scale and 1:100 scale distributions. The 'ore' was prepared from size fractions of commercially available crushed limestone and crushed red lateritic rock. These products were chosen after testing several materials, as they could most easily be modified to match the required size distribution and the colours made it easier to track material movements. The size distribution of the prepared 'ore' is a close match to the -10 mm range of the operating mine, but the simulated mix data includes finer sieve sizes.



(a) (b)

Figure 7 Size distributions for operating mine and simulated waste/ore. (a) Plotted with operating mine 100%; (b) Plotted with simulated ore 100%

The 'ore' was split into nine representative samples which were loaded consecutively, to minimise size segregation within the 'ore' column. Strong segregation still occurred within each layer of 'ore' depending on the direction of pour so this was varied. The last bucket of 'ore' was sprayed with orange paint to form a marker layer beneath the deposited tailings. Figure 8a shows a photograph of the box loaded with 'ore' as well as the first tailings pour, which was fine grained Tails 1 a. Figure 8b shows rapid migration of moisture from the base of the tailings into the 'ore'. However, this first pour set hard overnight (>600 kPa UCS) and bridged across the box so had to be chipped out. This shows one aspect of the test which cannot be scaled. In full-scale operation, the larger dimensions of the cave and higher pressures would break up slabs of tailings. However, this removed slab of tailings material did enable the interface with the 'ore' to be inspected and showed there had been essentially no penetration of 'ore' by tailings , as seen in Figure 8c.

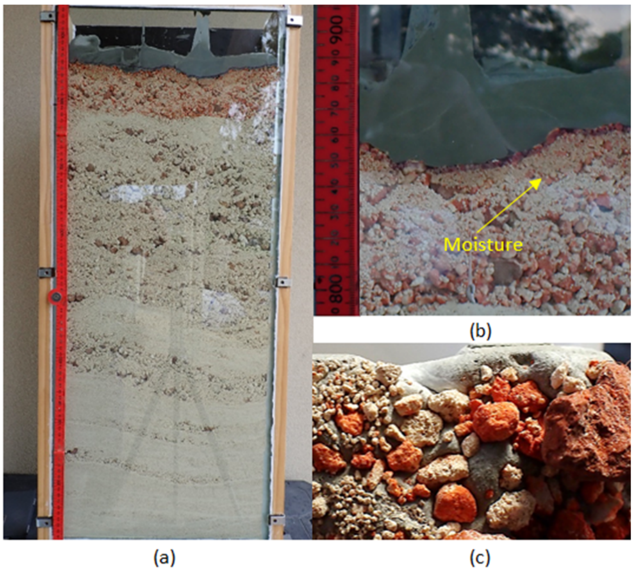


Figure 8 (a) Box loaded with 'ore' with first tailings pour; (b) Tailings 'ore' interface; (c) Close-up of tailings after removal from box, showing interface

As the remainder of the Tails 1 a material was unusable, work proceeded with the coarser Tails 2. As discussed in Section 5.1, this was hydrated for 24 hours before adjusting the pulp density. Three layers of tailings were poured over a three-day period, each about 30 mm as for the full-scale estimated average rise rate. The box was kept covered to minimise moisture losses by evaporation. The tailings nonetheless developed average UCS on the top surface of 90 kPa overnight after each pour, moderately firm but insufficient to support normal vehicular movements (Knight 1961). On day three, 8.5 cm of 'ore' was withdrawn from the three drawpoints, taking one scoop from each drawpoint in turn. A photograph was taken every scoop to record the 'ore' movement. The side of the box was given 10 blows with a light mallet every 4 cm of drawdown, to simulate the vibrations from blasting helping to consolidate the 'ore' column. A further 30 mm of tailings was poured and allowed to consolidate for 24 hours, then a last pour following which 'ore' withdrawal commenced immediately, without allowing the last pour to dewater. This was to determine if unconsolidated tailings might mix with the caved rock.

A time-lapse video has been prepared from the photographs and is posted on YouTube (Clarke 2024).

It was noted that ore drew down preferentially in narrow zones, more funnel flow than ellipse, directly above the drawpoints and measured as extending about 40 cm (equivalent to 40 m full-scale) or three model sublevels vertically, above which drawdown was more even across the box. In consequence, material from the marker layer did not start to enter the drawpoints until the surface of the 'ore' had descended from the initial 80 cm to about 30 cm above the left hand drawpoint, where the surface was most depressed. At no stage did any tailings mix with the ore. As the tailings descended, the successive pours tended to separate along pour surfaces and the slabs broke up, enhancing drainage and loss of moisture via the box surfaces such that from about 40 cm down the tailings largely separated from the 'ore', as seen in Figure 9a. This would not be expected to occur in full scale operation. However, Figure 9b shows that the tailings lumps floated on the surface of the 'ore' and did not mix. It seems likely that even under the much greater pressures in full-scale operation, momentary separation of tailings layers and break up into lumps would occur during downwards movement.

(a) (b)

Figure 9 (a) Separation of tailings from 'ore' at late stages of drawdown; (b) Enlarged section showing tailings lumps floating on 'ore'

Thus, this test work showed that at this scale and minimising mixing in the cave column by maintaining even draw rates across the three drawpoints, the tailings did not mix with the caved rock and would not interfere with the behaviour of the cave or the flow of caved rock until the tailings interface reached the drawpoints. Larger scale work is required to confirm this, with box dimensions large enough to ensure the consolidated tails slabs break up as the caved rock collapses and descend with the rock A scale of, for example, 1:10 would also confirm the behaviour of the tailings lumps in tending to float on the ore/waste bed.

6 Implementation

The test work that has been completed shows that thickened tailings with a moderate yield stress will not bleed fines and will not significantly penetrate a bed of broken rock, even under pressure. Thus, this work supports the conclusions from test work at Mufulira that dewatered tailings can be an entirely satisfactory material to form the surface over a caving area. However, this requires that the tailings can be safely placed over an active cave without allowing appreciable bodies of water to form and that drawpoint management maintains even drawdown. The concepts presented here apply to the case where caving commences under virgin ground and where the topography enables tailings deposition to be arranged with a self-draining surface directing stormwater and supernatant liquor clear of the subsidence area.

Tailings could be deposited in any of three forms, in order of decreasing cost:

- Cement amended tailings.
- Filtered tailings.
- Thickened tailings.

Although no test work with cement amended tailings was carried out in this program, previous test work with the same materials has confirmed that low additions of cement to thickened tailings produce a moderate strength product that does not liquefy in the presence of water. Cement amended tailings would have the advantage of creating a trafficable surface within hours of pouring. Lecuyer (2020) states that the

New Afton mine was planning to place cement amended tailings in the historic Afton Open Pit at the boundary with the subsidence zone in 2022, after mining ceases in the East Zone (which is closest to the pit). However, the use of cement adds substantially to operating cost. The brittle nature of the amended tailings loses the advantage of tailings alone which can flow plastically or potentially be moved by earthmoving equipment to adapt to changing ground levels. Cement amended tailings could also increase the risk of bodies of water being stored within the cave if the compressive strength of the amended material was less than the pressures at depth so a low permeability layer could form below layers of fractured cemented tailings. Thus, the use of cemented amended tailings does not appear beneficial where backfilling can commence before a subsidence cone is formed and where pressures at the base of the final tailings deposit may cause compaction of the amended tailings.

Filtered tailings have the benefit that they would generally be immediately trafficable at least by tracked equipment, which would allow active management of the deposited tailings to maintain a self-draining surface and rapidly fill any depressions. Filtered tailings could also be stacked more steeply and potentially eliminate remaining high walls when mining finishes. Drainage berms could be built to divert stormwater away from the surface above active drawpoints so preventing flooding of sinkholes. Filtered tailings would also eliminate the risk of bulk flow of slurry down deep tension cracks, which could potentially reach the active mining zone. However, filtration of tailings and transport and spreading of filter cake is a costly process to be avoided if it is not essential. It would also require that heavy equipment could be used safely over the subsidence zone, although operation would not be required directly over localised areas of high subsidence activity.

Thickened tailings offer most of the benefits of filtered tailings at lower cost. The major uncertainty is whether tailings deposition can be managed so that local areas of subsidence can be rapidly filled, displacing any collected stormwater before large volumes are created. If thickened tailings are placed, very low ground bearing pressure equipment should be available to take localised remedial action if necessary.

The most important area for further investigation before backfilling of tailings above an active cave mine could be implemented would be to acquire detailed information on subsidence behaviour and so determine feasible strategies for maintaining a self-draining surface with no long-term depressions. While the results of the work completed to date look promising, further generic investigations would be valuable to demonstrate that results can be repeated at larger scale. This could include model testing at probably 1:10 scale as well as full-scale testing which could potentially be carried out in a disused orepass. Earthquake flow risk should be evaluated in a more controlled way for the relevant design earthquake. The lower limit of slurry yield stress required to prevent any flow through rock should be evaluated. Project specific tailings testing would be required. The risk of perched water tables within the cave would need to be evaluated and action plans for potential inrush established.

Some preparation of the ground over the cave before mining commences may reduce risks. Clay-rich saprolitic soils that are substantially less permeable than the tails could trap water in the tailings column, particularly as the saprolitic material came under pressure at depth. Removal of the saprolite would eliminate this risk. Blasting a layer of rock after saprolite removal may help to provide more even subsidence at surface resulting in safer working conditions, as well as creating an effective starter filter bed. Deep tension cracks reaching surface would fill first with blasted rock and so reduce the risk of bulk flow of tailings to drawpoints.

7 Conclusion

The risks from placing thickened tailings or filter cake over an active cave mine fall into three categories: fluid flow of tailings through the caved rock, mixing of dewatered tailings with the caved rock and implementing tailings placement. Test work and analysis of published data have shown that broken rock acts as an effective barrier to the movement of thickened tailings which possesses a yield stress and that the cohesive properties and lower density of consolidated tailings inhibit mixing with broken rock or migration of tailings. Thus, with appropriate management, the results indicate that the identified risks relating to backfilling of tailings above an active cave mine can be reduced to an acceptable residual level. Whilst larger scale test work is required to confirm the findings, the main uncertainty relates to the magnitude of localised movements in the subsidence zone, particularly before a deep layer of tailings has been established, and the consequent ability to safely place tailings without allowing significant ponded water to occur.

Acknowledgement

The author takes responsibility for all concepts presented in this paper but would like to acknowledge valuable conversations with the following people which enabled a wide range of possible risks to be identified and gave encouragement by not ruling the idea out *a priori*: E Hammam, A McCauley, A Fourie, N Steward, A Barros and J Wesseloo. The constructive comments and additional information from the reviewers of the paper are also gratefully acknowledged.

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