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# Backfilling with filtered tailings above an active cave mine: a conceptual study

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## Abstract

*Placing tailings in the subsidence zone of a block or sublevel cave mine during operations would have major benefits, particularly in leaving behind a rehabilitated land surface capable of beneficial use; reducing the area of the subsidence crater and reducing (by at least 50%) the volume of other tailings storage facilities. Surface water flows could be better managed to ameliorate community concerns. In high-rainfall areas underground water management would be simplified; the risks from mud rushes at drawpoints would be reduced and where iron sulphides are present, costs for acid drainage control and treatment would be greatly reduced. With additional oxygen access control measures, tailings backfilling may avoid the need for perpetual treatment of acid drainage after closure.*

*Despite the advantages, the perceived risks have meant that backfilling tailings above an active cave mine has not yet been practiced. The major perceived risk is of a large-scale mud inrush with fatal consequences as at Mufulira in 1970 and Grasberg in 2025. However, filtered tailings, placed so as to prevent surface water accumulation, cannot flow through broken rock and could reduce the risks inherent in an unmanaged subsidence crater.*

*The risks and benefits of progressive backfilling of tailings have been studied in the context of a model mine, based in part on a confidential sublevel cave mining project in tropical South America. The work has involved formal and informal risk reviews, reviews of historical major mud inrush incidents, conceptual engineering analyses and laboratory test work. A 1:100-scale model mine experiment was completed using a block caving withdrawal sequence. The studies to date indicate that backfilling is feasible and risks can be managed. However, experimental work using a new 1:100-scale model mine, structured as a sublevel cave, is continuing. The intention is to obtain quantitative model data on dilution by tailings compared to waste rock fines.*

**Keywords:** *backfill, filtered tailings, cave mining, mud inrush*

## 1 Introduction

The use of caving (sublevel and block) methods for underground mining is widespread for orebodies with suitable geometry and geotechnical properties, but as currently practiced has two significant disadvantages:

- All tailings must be stored in separate impoundments on surface.
- A subsidence cone remains, which is difficult to rehabilitate to any beneficial use and is a safety hazard in populated areas.

Depending on climate and geography, the following additional serious impacts are noted:

- Where mine haulage drives exit to surface below the orebody, acid drainage flows may occur and persist for centuries (Koehnken 1997).

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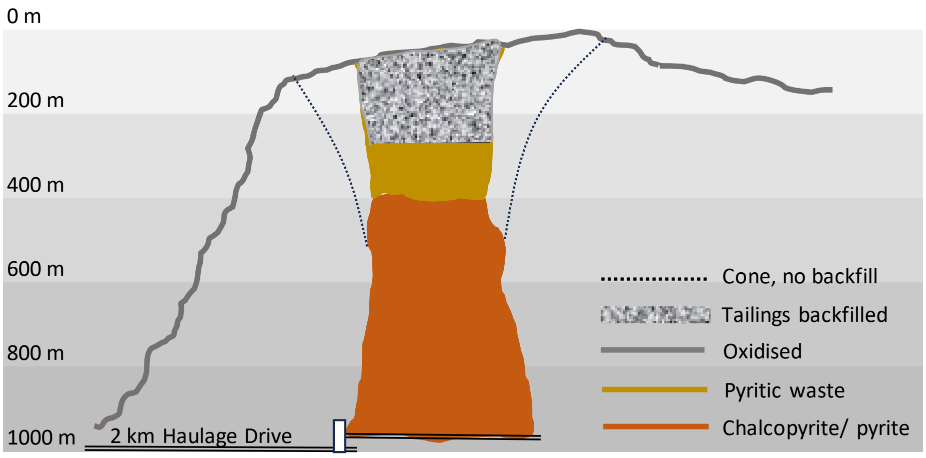
- Surface water flows may be diverted into the cave, affecting other users.

As mines become larger, these deleterious effects become more important and may make permitting difficult. They 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 this is not generally economically feasible. Moreover, this approach negates many of the potential advantages of backfilling. However, the alternative of placing tailings over an active underground mine has been considered too risky by mining regulators (E Hamman, pers. comm, 2012), a view strongly influenced by the 1970 disaster at the Mufulira mine in Zambia. In fact, following stabilisation of the tailings, mining recommenced at Mufulira. The purpose of this paper is to argue that with the current ability to control tailings properties, placing tailings above active caving mines would have many benefits and that the identified risks can be managed for a safe outcome. The approach taken is to present a 'Model' mine which is partly generic but draws on a multi-disciplinary, albeit conceptual, confidential case study carried out with SRK Vancouver for a mine in tropical South America.

# 2 Model mine

The Model mine exploits a copper porphyry orebody which consists of a payable zone carrying chalcopyrite and pyrite, overlain by a waste zone with substantial pyrite which extends almost to surface. 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 surrounding area is farmed for cash crops and has some tourism. Surface streams are captured for use in agriculture and for generating hydro-electric power. The orebody lies within a mountain plateau 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 cone would be profiled to enable placement of a self-draining tailings deposit that does not require pumping. The concept presented in this paper is applicable to a greenfield mine, as a cave mine commencing under an existing open pit would create additional challenges.

Tailings would be deposited as filter cake so there would no liquor release, and substantial shear strength. Clarke (2024) considered placement as thickened tailings, but use of filtered tailings provides greater flexibility; the tailings can be stacked at a steeper slope to suit local topography, albeit at increased cost. Deposition would commence before cave breakthrough to surface, to pre-load the subsidence area and provide 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 limit the formation of depressions and maintain a continuous self-draining beach slope.

# 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, thereby making 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 completely filled, although potentially with a high wall needing to be stabilised, and would be rehabilitated as required.
- Due to the tailings stabilising the surrounding ground, the area of the subsidence cone would be approximately the vertical projection of the area of the orebody. For the case study, the subsidence zone would be reduced in size from close to 40 ha to about 7 ha.
- A substantial proportion of the total tailings can be stored in the subsidence zone, with a corresponding reduction in the size of the separate tailings storage facility. The actual percentage stored depends on the swell factors for both tailings and waste but is estimated to be about 50%. While this will always be environmentally beneficial, any cost benefit will depend on the relative costs of handling tailings to the subsidence zone, vs alternative surface placement.

In the case of the Model mine there are additional advantages which will be common to any cave mine in a high-rainfall climate:

- 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, ameliorating a major community concern.
- Inflows to the mine will be significantly reduced overall and high flows will be eliminated. With no tailings backfill, average inflows to the mine from the subsidence zone are estimated to be 80 L/s with peak flows up to 4,000 L/s by the end of the mine life. With compacted tailings backfill, the infiltration rate reduces to about $6 \times 10^{-7}$ m/s, or less. Ignoring evaporation and assuming the surface is wetted by rainfall for an average of 3.6 h/d, the much-reduced area results in an average flow of about 6 L/s with little variation. While evaporation would further reduce this, groundwater will increase it.
- This reduced and consistent flow will greatly reduce, possibly eliminate, the potential for internal mud rushes.
- With no backfilling, there is an increasing risk over time of fines and water accumulating at the low point of the subsidence zone and leading to a major mud inundation event and loss of life. Safe access to dewater is likely to be difficult or impossible. The mud rush incident at Grasberg on 8 September 2025 illustrates the risk (Fiscor 2025), although in that case the cave mine had commenced under an open pit.

Monis (2026) describes an engineered backfilling strategy that was implemented at the Padcal mine in the Philippines to mitigate "*massive, sometimes fatal mudrush events"* which occurred historically due to the very large volumes of meteoric water collected by the unfilled subsidence crater during the frequent cyclonic high-rainfall events. Backfilling, together with active management of the crater fill, has yielded a demonstrable reduction in both the severity and frequency of mud rush events. It has also had a profound socio-economic benefit by preserving critical surface infrastructure.

For the Model mine, arguably the greatest advantage of backfilling with tailings is mitigation of the costs and regulatory risks of acid drainage production, but this benefit will apply to a smaller proportion of cave mines. The placement of tailings will reduce acid drainage treatment costs as follows:

- The moisture content and fine particle size distribution of the tailings at deposition will result in much lower oxygen diffusion rates than broken ore or waste. With a moisture content as deposited, the oxygen diffusion coefficient is estimated as $6 \times 10^{-8}$ $m^2/s$ using the method of Aachib et al. (2004), compared to $8 \times 10^{-7}$ $m^2/s$ for a waste rock column studied at Woodlawn mine in Australia (Jeffery et al, 1988). Together with the reduction in subsidence cone area, this represents a reduction of 70 times in acid drainage load during operation.

- The tailings deposit will practically eliminate the potential for the mine ventilation system to draw air through the sulphidic rock column. Volumes of leakage air that are very small compared to the total flow of mine ventilation air are very large compared to oxygen diffusion rates, and could therefore drive large increases in acid load.
- By reducing and smoothing volumetric flows, the tailings will reduce the capital and operating costs for treating the acidity.
- It is becoming increasingly common for environmental authorities to limit the sulphate concentration in mine site effluents (Nevatolo et al. 2014; Government of Colombia 2015). Sulphate reduction is difficult and expensive (Leão et al. 2014) compared to pH adjustment and metal precipitation; in cases where this is required, the cost reduction by limiting oxygen access to the mine will therefore be much greater.
- After mine closure, the tailings can be sealed with a cover to further reduce oxygen access and water infiltration. If combined with mitigation measures inside the underground workings, this has the potential to eliminate the need for long-term active water treatment.

# 4 Backfilling risks

Potential risks have been identified by considering all aspects of the process of backfilling with tailings in informal and formal risk reviews. The risks identified are first listed below, then analysed in more detail:

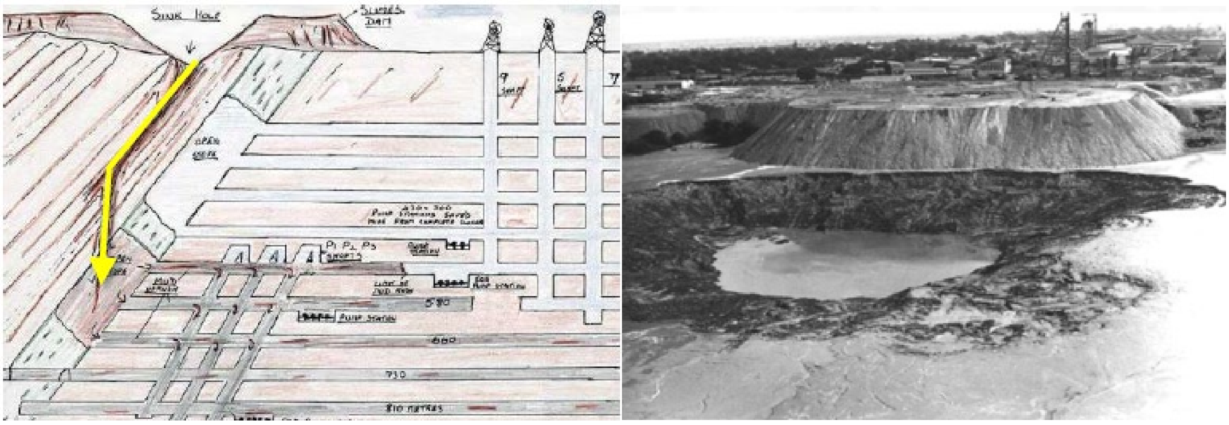
1. Liquefaction of tailings resulting in an external mud rush into mine workings.
2. Internal mud rush.
3. Contamination of ore by tailings, by:
    a. Flow of tailings slurry under pressure to drawpoints through the rock mass.
    b. Tailings migrating in percolating water.
    c. Tailings solids percolating through caved rock movements (Laubscher c. 2000; Hashim et al. 2008).
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. Inability to transport tailings to the subsidence area discharge point.
7. Cannot operate equipment above an active cave mine.

## 4.1 External mud rush

Three cases of major mud rushes originating outside the mine, and causing multiple fatalities, have been found in the literature.

The most directly relevant case is 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) and in the report by the Commission appointed by the Government of the Republic of Zambia (1971).

The Mufulira mine in Zambia was operating a sublevel cave mine beneath a tailings deposit, known as the no. 3 dump. This dump was initially built with 3.9 Mt of slime tailings filling hollows and sinkholes, followed by a total of 20 Mt of unthickened tailings. From early 1970, mud extrusions were noted underground; these consisted of talc and argillaceous material, resembling material from the hanging wall aquifer. It was recognised that this mud might be followed by tailings. On 25 September 1970, the tailings broke through without warning, with an estimated 450,000 $m^3$ flowing into the mine over approximately 15 minutes, 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 (Figure 2).




(a) (b)

**Figure 2 Mufulira mine disaster. (a) Mine section showing tailings entry route; (b) Sinkhole in no. 3 dump**

A five-man Commission of Enquiry was formed and concluded that the practice of disposing tailings on caving hanging wall ground was "*basically wrong*" (Vutukuri & Singh 1995). 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 was dewatered and stabilised using tube-wells, and paddocks were 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 tailings should not be stored on caving hanging wall ground has been taken as absolute in the decades since, but the context and results of subsequent investigations have been forgotten. In particular, the development of thickeners capable of producing tailings with yield stresses above about 30 Pa, or large-scale pressure filters, dates from the late 1980s (Schoenbrumm 2011).

Although the inflow occurred above the sublevel cave workings, Jakubec (2025) noted that tailings migrated through the fractured hanging wall, not through the caved material (Figure 2). The tailings then entered the active workings via the internal shaft system.

Butcher et al. (2005) discuss a disaster at the Bafokeng mine in South Africa in 1974, which resulted in the loss of 12 lives. In this case, a surface tailings dam adjacent to Bafokeng shaft failed following a period of intense rainfall and the fluidised retained slimes flowed through the surface workings and down the mine shaft. The case is relevant in highlighting the importance of water in mobilising tailings, and the fact that entry to the underground mine was by the shaft system.

The third case of an external mud rush is the recent incident at the Grasberg Block Cave (GBC) mine in Indonesia on 8 September 2025 (Fiscor 2025). This incident involved mud formed by rainwater combining with fines from the pit wall collecting at the low point in the pit and then entering the block cave by "*undetected pathways*" in a high-velocity draw zone in the inclined hanging wall of the GBC. Again, entry to the mine appears not to have been through the caved material and the importance of water is emphasised.

## 4.2 Mud rushes

Internal mud rushes, originating within the cave column, have caused multiple fatalities (Butcher et al. 2005, Jakubec et al. 2016). They require the presence of both mud and water, some disturbance leading to mobilisation, and an entry to the mine workings.

Placement of tailings above a cave mine would be expected to *reduce* internal mud rushes in two ways:

- a reduction of total water flow and particularly the elimination of high peak flows
- a reduction in clay-like secondary minerals formed from sulphide oxidation and subsequent neutralisation (Clarke 2024; Tyler et al. 2021).

However, if a substantial volume of tailings was to migrate to the drawpoints, then the increased fines content could increase the internal mud rush risk. The risks associated with various mechanisms of tailings migration are discussed in subsequent sections. However, it is worth noting that the same risk would arise where highly weathered saprolitic material exists and is not removed.

## 4.3 Contamination of ore by tailings

The potential for contamination of ore from backfilled tailings is primarily an economic risk, although as noted in Section 4.2 it could also increase the risk of internal mud rushes. The potential causes of contamination by tailings were investigated in a series of small-scale experiments, reported previously in Clarke (2024). The work and the conclusions are summarised in the following sections.

### *4.3.1 Flow of tailings through the rock mass*

Test work was carried out in 2022 and 2023 using thickened tailings slurry, not filter cake. Thickened tailings can be produced and placed at lower cost than filter cake and results using thickened tailings will be conservative compared to filtered tailings.

The initial test involved pouring a thickened slurry at 72% solids w/w and a yield stress of 55 Pa over a bed of river pebbles with a top size of 20 mm and no fines. The experiment was carried out in a transparent Imhof cone. The tailings used was pilot plant rougher tailings from a South American porphyry copper project and had a nominal grind size of 80% <150 µm and an unusually high content of fines at 44% <38 µm.

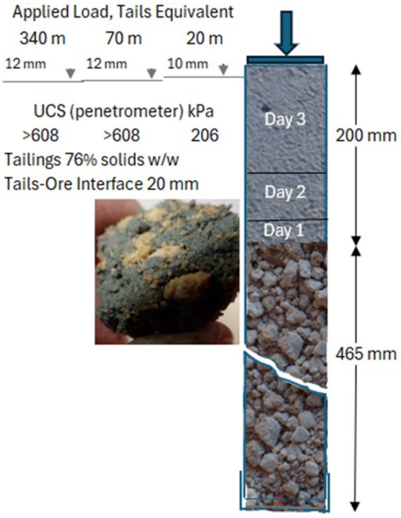
The tailings flowed to a depth of 15 mm (slightly less than one pebble diameter) in the centre of the cone and to 30 mm at the walls. Close observation suggests that when the slurry reaches a contact point between the pebbles, a small quantity of tailings passes but the increase in surface area and short drainage path leads to rapid water loss and increase in yield stress, thus preventing further movement. As the pressure head increases, more moisture is lost and the effective viscosity increases, preventing flow. It is important to note that placement of tailings when backfilling would be in relatively thin layers, so the pressure would build gradually, allowing time for resistance to flow to increase.

The cone was then subjected to two rounds of vigorous manual horizontal shaking and vertical tapping for 2 minutes each time, to approximate a severe earthquake. Horizontal acceleration was measured as >2 g using the Arduino Science Journal 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 depth of 185 mm along the walls, but only to 60 mm within the pebble bed (three maximum pebble diameters). These depths (scaled) would not appear to be a concern in a full-scale operation, although further studies of potential flow under design earthquake conditions and at a larger scale are required. The results are shown in Figure 3.

(a) (b)

**Figure 3 (a) Side and top views of drainage water collected from tailings; (b) Two pours of thickened tailings slurry**

A further test was carried out to investigate the flow of tailings through the rock mass under high heads. The test was carried out in a 600 mm length of 25 mm standard galvanised steel pipe loaded to a depth of 400 mm with quartz pebbles crushed to −10 mm. Thickened tailings slurry was added to the top 200 mm of pipe in a series of lifts. Pressure was then applied by a length of threaded rod and measured with a Fluke pressure sensor. A pressure of 3,200 kPa was achieved but could not be maintained so it was thought the tailings was being forced through the pebble bed. In fact, the problem was undetected water leakage, and the tailings had formed a plug at a density of 87% solids, rising to 90% solids immediately above the pebble bed. The equipment was redesigned to be able to achieve higher pressures and continuously measure the water balance by weighing the assembly. A coarser tailings slurry, mixed at a pulp density to achieve a yield stress of about 50 Pa, was used. The new arrangement estimated the pressure applied from bolt torque and on that basis achieved 6,050 kPa, equivalent to a tailings' depth at an assumed bulk density of 1.6 t/m$^3$ of 385 m. A diagram of the column is shown in Figure 4a. After leaving under pressure overnight, the contents of the column were removed. The average per cent solids of the tailings was 76.7% (30% geotechnical moisture). However, the tailings had penetrated the crushed pebble bed to a depth of only 20 mm, or two particle diameters as shown in Figure 4b.




(a)

(b)

**Figure 4 (a) Diagram of high-pressure test column; (b) Tailings–pebble contact zone in high-pressure test**

### *4.3.2 Tailings migrating in percolating water*

A risk was identified that tailings solids could migrate with water percolating through the tailings bed. Figure 3a shows the drainage water collected from the low-pressure drainage test in the Imhof cone. The drainage was visually clear. The drainage from the high-pressure column test contained a small amount of suspended solids, measured by Hach DR/2010 spectrophotometer and Hach method 8006 as 3 mg/L. At the reduced average annual flow of 6 L/s expected with tailings backfill, this is a total solids load of less than 1 t/year. Without tailings backfill, the much higher average estimated flow of 80 L/s, flowing though weathered rock, would be expected to entrain much more solids, although still likely negligible in economic terms.

### *4.3.3 Tailings solids percolating through caved rock movements*

Dilution of ore through mixing with waste is a major concern and a disadvantage of cave methods of mining. Dilution occurs through waste material entering the draw zone and flowing to the drawpoints. This can happen in a fluctuating manner and is affected by many factors (Laubscher et al. 2017, p 244). In sublevel cave mining, a grade-bearing blanket is frequently left above the uppermost sublevel so that much of the dilution material is actually ore grade. The risk of tailings contributing to ore dilution by reporting in bulk to the drawpoints will depend on the thickness of other waste between the ore and the base of the tailings. For the Model mine, this risk is probably negligible. Where the top of the ore is shallow, dilution by mixing with tailings may only replace dilution by material from the collapsing walls of the subsidence zone.

Dilution also occurs due to fines preferentially percolating down through the cave and reaching the drawpoints (Hashim et al. 2008). Johanson et al. (2005) found that if the fines material is free flowing and less than one-third the size of coarse particles, they may percolate through the coarse matrix of particles. Fines migration can also be modelled in software such as PCSLC which uses a series of rules based on calibration (MaxGT 2025). The model uses a fines migration parameter which is the probability for a void to be filled by fine versus coarse material, and the percentage of fines present. In this model, 'fines' is considered as about 100 mm or smaller, or about 10% of the maximum rock particle size. Tailings, however, is typically about 100 µm, or 1,000 times smaller again.

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 excessive tailings reported to the drawpoints and the practice had to be stopped.

For the Model mine, applying the PCSLC model predicts a high, although not necessarily unacceptable, migration of tailings fines to drawpoints by the end of mining. It is worth noting that tailings fines entering the plant would not incur full processing costs, nor necessarily displace its weight in new ore. However, the modelling assumes the fine tailings particles are free flowing, as specified by Johanson *op. cit*. This is not the case when tailings are deposited as paste or filter cake.

An experiment was carried out to study the behaviour of tailings in a 1:100-scale physical model. Full details of the test are given in Clarke (2024). A glass-fronted box was built 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 at drawpoints. 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 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 the Ernest Henry mine (G Dunstan, pers. comm., 2019), by using the portion of the distribution below 10 mm (i.e. 1,000 mm top size, full scale). 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 (Figure 5). The tailings size distribution was not scaled – the test used the natural size distribution. Although this means that similitude is not maintained between ore and tailings, if the tailings was scaled, the top size would be 1–2 µm and the rheological properties would be very different to full-scale tailings. The change in size relationship between ore and tailings is very unlikely to affect results since, at full scale, the tailings top size is about 1/10,000 of the ore top size and in the model, it is 1/1,000.

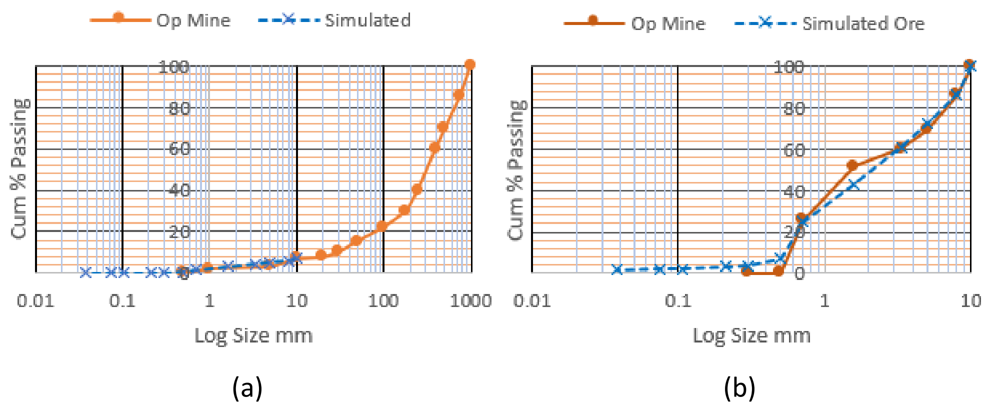



**Figure 5 Size distributions for operating mine and simulated waste–ore. (a) Plotted with operating mine 100%; (b) Plotted with simulated ore 100%**

The ore was loaded in nine layers to minimise vertical segregation, and the last bucket was sprayed with orange paint to form a marker layer beneath the deposited tailings. Figure 6a shows a photograph of the box loaded with ore and tailings.

Three layers of tailings were poured over a 3-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 unconfined compressive strength (UCS) on the top surface of 90 kPa overnight after each pour, moderately firm but insufficient to support normal vehicular movements (Knight 1961). On the third day, 8.5 cm of 'ore' was withdrawn from the three drawpoints, taking one scoop from each drawpoint in turn. This followed the practice in previous physical models (Campbell 2020) which investigated interactive draw to minimise dilution entry. It is not representative of current sublevel cave practice where drawpoints, once started, are dawn to completion and a limited number of drawpoints are in operation at any time (Laubscher et al. 2017). This sublevel cave practice is likely to lead to greater bulk mixing, but further work is needed, and is planned, to determine whether it might increase migration of tailings fines through the rock mass.

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 to simulate the vibrations from blasting and cave movements from adjacent draw, 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 was commenced immediately, without allowing the last pour to dewater. This was to determine if unconsolidated tailings might mix with the caved rock.

(a) (b) (c)

**Figure 6 (a) Box loaded with 'ore' and tailings; (b) End of drawdown; (c) Tailings–ore interface**

A time-lapse video was prepared from the photographs and is available on Vimeo (Clarke 2026).

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, as in Figure 6b. 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 shown in Figure 6c. 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. While the tailings layers fractured across the width of the box, the depth of the box (from front to back) was smaller, and the tailings bridged this and tended to hang-up. The problem was controlled using vibration from hammer blows on the box side. This would not be expected to occur in full-scale operation and future work should increase the depth of the model to avoid this.

Figure 6c shows a close-up of the ore–tailings interface, showing that penetration was less than one particle diameter.

It appears that because the tailings behave as a cohesive mass with a relatively low density, the tailings does not penetrate the ore or waste which has a higher particle density. An analogy would be the behaviour of the ragging in a mineral jig, where only particles with specific gravity higher than the bed of ragging can pass through. Similar behaviour would be expected from highly weathered saprolite or clays at surface, provided water was not allowed to accumulate. There may be experience from operating mines that would provide insights on this.

## 4.4 Sterilisation of resources

In some cases, the placement of tailings above an orebody could 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 could 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

Two risks relating to tailings interfering with the caving process were identified above.

### *4.5.1 Tailings prevent caving*

Placement of tailings in the subsidence area cannot affect the initial ability of the cave to propagate to surface as there is no contact between tailings and rock up to that time. The air gap essential to the caving process no longer exists once caving reaches the surface (Lecuyeer et al. 2020).

### *4.5.2 Tailings impeding flow through drawpoints and ore passes*

If tailings mix with the ore, it could affect the ability of the cave to develop as well as the flow behaviour of the ore to drawpoints and in ore passes and bins. 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 requires 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.

Consolidated tailings mixed with ore could produce a Type 4 material and cause operational problems at drawpoints and in the subsequent ore-handling system, particularly 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 and the test described in Section 4.3.3 strongly indicates that there will be minimal mixing of tailings and ore or waste.

## 4.6 Inability to transport tailings to the subsidence area discharge point

One risk identified is an inability to transport tailings to the subsidence area discharge point. There are many ways by which tailings could be transported, 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 is to use high-pressure pumps and a pipeline along the haulage drive and up a raisebore to discharge to an agitated storage tank feeding a filter plant near the subsidence zone. An alternative could be to filter the slurry at the main process plant at the portal and transport by truck or aerial conveyor, which would avoid the need for a filter plant at the subsidence area.

For pumping, a system has been designed at the conceptual level for the Model mine case. It includes two parallel positive-displacement piston diaphragm pumps with 2,500 kW installed power each, pre-charge centrifugal pumps and gland seal water pumps. Each pump feeds a separate Schedule 120 steel pipeline DN 10”. This is mature technology, albeit at the upper end of the envelope for volumes pumped at the required pressure. Although there is no doubt that positive-displacement pumps are more complex than centrifugal pumps, with appropriately trained maintenance personnel and sound operating procedures, they have been operated successfully on critical long distance pipelines in rugged terrain. Examples include Savage River (iron concentrate) from 1968 until the present (Jackson 2016), the Bougainville copper mine from 1974 to 1986, the Irian Jaya Ertsberg project and the Ok Tedi project (Jackson 2015).

Once tailings backfill is committed to, tailings must be available to place whenever underground production is in progress. The exact quantity of tailings required depends on the actual rock swell and tailings dry density achieved and is uncertain. Hence, the conceptual system is designed to accommodate a range of flow rates while maintaining velocities higher than the required minimum to prevent settling and to keep discharge pressures below 300 bar, limited by current pump technology. Detailed design must include sufficient redundancy to ensure tailings is available to place whenever mining is active, although interruptions of a few days could be dealt with by maintaining stockpiled filtered tailings at the subsidence area. Maintenance and operational continuity of the backfill tailings placement system must have the same priority as operation of the plant itself.

The pumps are fed from tanks at the main process plant and discharge to a filter plant adjacent to the subsidence zone. The capital cost for the pump system is about USD 25 M, and the operating cost is predominantly for power. Assuming power at USD 0.10/kWh, the operating cost is about USD 1.1/t.

As tailings must be disposed of anyway, the additional cost for backfilling tailings in the subsidence area depends on the costs for the alternative. If the alternative tailings disposal method is filter stacking, there may be little or no additional cost for placement and a saving on construction.

Although the cost of pumping tailings is substantial, it is unlikely to be a critical factor in adopting backfill.

## 4.7 Operation of equipment above active cave

Placing filtered tailings over an active cave mine requires that heavy equipment can access the area. The tailings itself will aid in this process by continuously compensating for material withdrawal and so preventing the formation of potentially unstable walls, except perhaps where tailings cannot be stacked as steeply as the original up-slope topography. The tailings will also form a flexible and even surface for traffic. Given that the backfilled tailings do not require the same structural strength as in a filter stack, there may be benefit in dewatering only to the extent necessary for trafficability so that the tailings more readily adapts to and smoothens sudden depressions.

There are no known operations currently placing tailings above a cave mine, but, as discussed in Section 3, the Philex Mining Corporation has backfilled an area above the block cave operations at the Padcal mine using various sources of waste, including sediments dredged from silt ponds and canals (Monis 2026; Philexmonitor.synthasite.com n.d.). The purpose of the backfilling is to divert surface water flows and reduce the risk of underground mud rushes. Backfilling is carried out using truck dumping and spreading with bulldozers. The slope of the fill is maintained at 4–7% with precision floor profiling using a grader, ensuring zero water ponding. The subsidence rate is tracked using strategic post markers. Twice-daily monitoring of all drawpoints is caried out to obtain early warning of excess sediments or water reporting to the muck pile and prevent ‘rat holes’ where vertical channels form and connect the surface directly to the extraction horizon. When a rat hole or surface depression forms, immediate action is taken to pause ore withdrawal in the affected area and to fill the depression. The resulting backfill is sufficiently stable that a road runs across the filled surface to offices and schools.

For backfilling with tailings, the tailings would be truck dumped then bulldozed into the craters above the active drawpoints as they form. Trucks would only be operated in dry weather but bulldozing would need to be all-weather so low ground-bearing pressure equipment might be required. Remotely operated bulldozers would remove the risk to operators. Close monitoring of cave progression would be required, to the extent possible. Detailed information from existing mines on the progression of surface subsidence would also be valuable. Although the placement of filtered tailings in a wet tropical environment is challenging, it is proven technology (Cobos et al. 2025). The main challenge is usually ensuring that filter cake moisture meets the requirement to compact to 95% of Proctor maximum dry density, but as the backfilled tailings will mostly be eventually buried below ground level, the main requirement for backfill is that the surface is trafficable.

# 5 Discussion

The major safety risk from tailings backfilling is an external mud rush, as discussed in Section 4.1. However, published studies and the laboratory test work discussed in Section 4.3 clearly show that thickened or filtered tailings will not flow through the caved rock. Although details on the recent Grasberg disaster are scarce, it appears that two of the three documented fatal external mud rush events involved tailings or mud flowing down a pathway through the hanging wall, which in the case of Mufulira was enlarged through earlier passage of mud and debris. In the third case, Bafokeng, flow was down a shaft. This raises the question then regarding the risk of a similar external mud rush occurring in the Model mine.

Engineering studies for filtered tailings disposal for the Model mine case assume that until the tailings is covered at closure, it will saturate due to the high rainfall compared to evaporation. If placed at 95% of Proctor maximum dry density, saturation is at 20% geotechnical moisture or 83% solids w/w, compared to target filter cake moisture of 12.3% and a liquid limit of 22%. To represent a hazard to the mine, this material would have to connect to presumably a large tension crack in the hanging wall with open passage to horizontal workings at depth, or a vertical channel would have to form, as discussed by Monis (2026). The saturated tailings could then flow down the channel. However, applying the definition of flow behaviour developed for dam breach analyses (Martin et al. n.d.), mud at 83% solids is on the boundary between failure by block sliding and slow creep. Thus, the mud would extrude rather than flow into the mine and while it might cause the loss of a drawpoint, it would not represent a hazard to life (Jakubec et al. 2016). Moreover, the column of deposited tailings would be likely to prevent large continuous cracks from forming in the hanging wall, as the tailings provides support.

The other main risk that requires further work, due to its importance and some limitations in the ore box experiment discussed in Section 4.3.3, is dilution of ore by tailings solids percolating through the cave. A larger ore box experiment has been constructed, still at a scale of 1:100, but with a series of six sublevels and provision for multiple material types as typically found in a sublevel cave mine. The dimensions from front to back wall have been increased to reduce hang-up of tailings as occurred in the original experiment. Ore will be withdrawn using a sublevel cave withdrawal pattern, with sublevels drawn to completion and no more than six drawpoints over three levels drawn at once. The test materials selected to represent the full-scale rock or tailings have distinct chemical signatures, such that by separating the ore withdrawals into size fractions and analysing by pXRF, dilution can be quantified by source. This should enable more precise modelling using PCSLC. The withdrawal system has been successfully commissioned, and the test is in progress.

Other work that is recommended is a detailed study of the behaviour of subsidence zone surfaces, to assist in planning management and tailings placement, and a detailed seepage analysis, including the effect of any layer of highly weathered layer of saprolite. Fines migration down the edges of the cave is a possible risk for which there may be data from existing cave mines. However, the placement of drained filtered tailings overlapping the subsidence zone edges should reduce this problem.

Where a weathered and fractured rock layer lies over highly impermeable fresh rock, groundwater will concentrate at the interface between weathered and fresh rock. If, as is likely, the placed tailings eventually descend below this level, the interaction between groundwater and tailings requires study. It is likely in this case that tailings would be entrained in the groundwater flowing down the walls of the cave zone to the working levels. Similarly, fines would be entrained if water flows through any channels that form in the tailings. The importance of this mechanism depends on the flow rate of water and the length of the contact path but an approximate calculation for the Model mine suggests it could result in recycled tailings contributing about 1% of plant feed. The effect on operating costs and throughput rate would be less because tailings would pass straight through the primary grinding and rougher recovery circuits.

A full-scale trial could be carried out in a disused ore pass filled with cave rock and with tailings placed in layers on top. The ore would be drawn from the bottom of the pass and the degree of mixing with tailings assessed. It would be important to ensure that the tailings moisture content was such that the tailings would descend the ore pass – some tailings has very poor materials handling properties.

# 6 Conclusion

The major benefits of backfilling tailings over an active cave mine area are a potential reduction (~80%) in the area of the subsidence zone, the ability to rehabilitate for beneficial use at the end of the mine life and a reduction in surface tailings storage. Stormwater inflows to the mine could be greatly reduced and large peaks eliminated. Where iron sulphides are present and the mine drains by gravity, backfilling will at least markedly reduce the quantity of acid water needing treatment, and with other management measures has the potential to eliminate the need for long-term active water treatment.

The perceived major safety risk of backfilling tailings over an active cave mine is the occurrence of an external mud rush. The work completed has shown that conditioned tailings placed to form a self-draining surface cannot give rise to an external mud rush and in fact will greatly **reduce** the risk of external and internal mud rushes by preventing the accumulation of water, reducing infiltration rates and reducing the formation of large, continuous tension cracks.

The perceived major economic risk from backfilling tailings is of tailings mixing with ore to an extent that significantly affects economics or the flow behaviour of ore. While further work is in progress to provide more quantitative data on this, the evidence from test work is that there is no mechanism by which tailings can significantly dilute ore, except in the situation where the tailings is deposited close to the top of the ore. Even then, the experimental evidence is that the tailings would not mix with the ore.

Although completed test work address both main risks discussed above, the authors recognise the fact that many block or sublevel cave operations in the past have not had a stellar record of following the plans. It is the authors' experience that where strict draw control is breached, localised overdraw creates preferential rat holes reaching the surface, and sink holes within the subsidence areas. Such a fundamental breach of 'caving rules' cannot be taken as the base case scenario and is a dangerous, economically damaging and unacceptable practice regardless of tailings backfilling.

The authors also recognise that specific site conditions such as presence of large-scale structures, topography and groundwater situation need to be taken into consideration. Jakubec et al. (2026) discussed a suggested methodology for evaluating subsidence backfilling.

From a practical perspective, the transport of tailings slurry or filter cake to the subsidence area can be done within the scope of proven current industrial technology. Further work would be required to develop subsidence management procedures in detail for the placement of tailings, but the technology for stacking filtered tailings, even in a wet tropical environment, is proven. Operation of equipment over an active subsidence zone is also proven.

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