

Evaluating the Benefits of Water Covers as a Rehabilitation Strategy in Temperate Climates: A Tasmanian Perspective

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ABSTRACT

Environmentally safe disposal of reactive sulphidic tailings is one of the major management challenges facing the mining industry. Consequently, there are a broad range of tailings rehabilitation strategies suitable for a variety of climates. One option is placement of tailings under a water cover to reduce oxygen influx. However, for sites in Australia that lie in a cool temperate climate, the suitability of engineered water covers is not well documented. The Old Tailings Dam (OTD) located at the Savage River Mine, Tasmania provides a unique opportunity to evaluate water covers as a rehabilitation strategy. The OTD extends roughly 1500 m northeast to southwest and 170 m east to west at its widest point (Hassell, 2005). The northern end is submerged under water, as the OTD was constructed on a natural gradient. Therefore, up to 0.2 km² are permanently submerged under water (maximum depth of 10 m) with some seasonal variation. In contrast, the southern portion in contrast has been exposed since 1982. Tailings from 14 trenches and 11 sediment cores were collected across the OTD. Selected samples were subjected to geochemical testwork and also used in mineralogical studies whereby a modified sulphide alteration index was developed. Results revealed a complex, heterogeneous sediment architecture with ten facies observed. From this, four zones were defined (A, unsaturated and coarse-grained, B- unsaturated and fine-grained, C- intermediate saturation and D- saturated). Geochemically, Zone D was identified as the lowest risk, with Zones A to C classified as having a high to extreme acid forming nature, which concurred with bulk mineralogy evaluations showing relatively high quantities of unoxidised pyritic tailings (5 to 9 wt. %). Sulphide alteration index assessments allowed for oxidation to be observed on a micro-scale, and indicated that a minimum water cover depth of approximately 1 to 1.5 m would be sufficient to significantly retard oxidation. Ultimately, a phased rehabilitation strategy which requires geotechnical earthworks and installation of flow-through reactors in combination with a water cover should be developed to successfully reduce AMD from this site.

Keywords: acid mine drainage, rehabilitation, tailings, Australia, mineralogy

INTRODUCTION

Uncontrolled sulphide oxidation can lead to the generation of acid mine drainage (AMD) - characterised by metal-laden acid-sulphate waters (Jambor *et al.* 2003). Impacts of AMD on the environment can occur throughout the life of mine (LOM) with many examples published in the scientific literature; e.g., Harris *et al.* (2003), Edraki *et al.* (2005), Parbhakar-Fox *et al.* (2014) and Candeias *et al.* (2014). As lower grade ore deposits are now being exploited for a range of commodities (c.f. Mudd, 2007), the management challenges are ever increasing as a consequence of the increased volumes of waste (Lottermoser, 2010). In turn, the complexity of mine waste (i.e., tailings, waste rock) management increases, and remains the most significant environmental challenge facing the mining industry (Harries, 1997).

In order to prevent sulphide oxidation and AMD generation in tailings storage facilities, appropriate control strategies are needed. Widely used strategies aim to exclude one or more of the ingredients required for oxidation; e.g., water, iron, oxygen, bacteria and sulphide minerals. Lottermoser (2010) suggested that the most effective control on oxidation rate is specifically reducing oxygen availability. Generally, this is achieved by using wet or dry covers, with a summary given in the GARD Guide (2014). More technologically advanced and innovative strategies have recently been developed and involve induced hardpan formation, grouting or mineral surface treatments (e.g., Ahn *et al.* 2011; Quispe *et al.* 2013). Lottermoser (2010) concluded that no single technology is appropriate for all mine sites and in many cases a combination of technologies offer the best chance of success.

When dealing with historic/legacy tailings sites, a cost-effective management strategy is favoured as the funds available to the lease holder (typically the mine regulator; i.e., government) is likely limited. In such cases, the use of a water cover should be considered. Recent studies by Yanful *et al.* (2004), Bjelkevick (2005) and Kachhwal *et al.* (2010) demonstrate the use of shallow water covers in North America and Sweden. Such covers are effective as the maximum concentration of dissolved oxygen in water is greater than three orders of magnitude lower than that of the atmosphere, so AMD formation is significantly retarded (Lottermoser, 2010). However, a minimum water cover depth must first be established. Yanful *et al.* (2004) proposed two criteria to determine this: (i) hydrological forecasting of the probability of occurrence of a drought event; and (ii) the minimum depth necessary to prevent re-suspension of tailings. Generally, deep water covers are not favoured due to concerns regarding the engineering stability of the containment walls, with high associated capital expenditure. Published studies documenting the use of water covers in Tasmania are scarce. However, as this state experiences a cool temperate climate (~600 mm rainfall/year) it would be well-suited to this type of rehabilitation strategy (c.f. GARD guide, 2014).

At the Savage River Mine AMD is emanating from the OTD, where 38 million tonnes of pyritic tailings were deposited from 1967 to 1982. Consequently, the OTD accounts for 50% of Savage River Mine's AMD load (Williams *et al.* 2014). To manage this, the Tasmanian State Government established the Savage River Rehabilitation Project (SRRP) in co-operation with the mine operator (currently, Grange Resources). This study focussed on: (i) confirming the necessity for a water cover at the OTD, and (ii) establishing an appropriate minimum water depth, should this type of cover be used.

METHODOLOGY

Site description

The Savage River Mine is located on the Savage River 420 km from Hobart on the northwest coast of Tasmania (Figure 1). It consists of five open-cut workings exploiting a group of magnetite-rich lenses irregularly distributed within a series of highly metamorphosed rocks of marble, schist and metabasic rocks. Open-pit mining and processing has resulted in the formation of several large excavations, waste rock piles, and tailings dams (Hutchison and Brett, 2006; Kent, 2013). The concentrated magnetite is slurred by pipeline to Port Latta on Tasmania's northwest coast, before shipping to market. The presence of pyrite in the waste materials (i.e., tailings and waste rock) has

caused AMD to emanate from the mine operations, which is of primary concern to the SRRP (Kent, 2013).

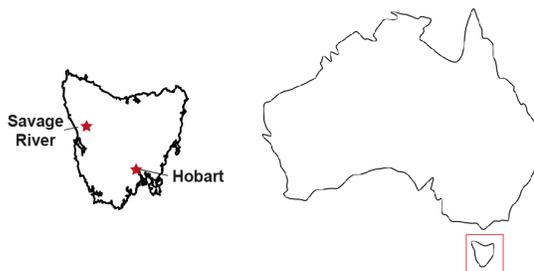


Figure 1 Location of the Savage River Iron Mine, Tasmania, Australia

Operations over the first 30 years of mine life caused environmental degradation to approximately 30 km of the Savage River, and downstream of the confluence with Main Creek exhibits the most severe degradation. This section was found in 1995 to have lost 90% of its invertebrate biodiversity and 99% of its invertebrate abundance. The OTD is located approximately 3 km from the township and 1.3 km from the processing plant. Pyritic tailings from Centre Pit South (CPS) and Centre Pit North (CPN) were deposited in the OTD (Kent, 2013). These were initially deposited behind the starter dam by end-of-pipe spilling into the southwest corner of the dam (Hassell, 2005). Subsequently, tailings were sprayed as slurry from the dam wall to allow it to be raised in lifts (Thompson and Brett, 1996). This deposition method has concentrated coarser, heavier and more permeable material against the dam wall (Kent, 2013). Consequently, the water table in the tailings has decreased proximal to the wall, resulting in extensive oxidation of the tailings in this area and AMD generation.

Sampling and sample preparation

In May 2014, an excavator was used to dig 12 trenches up to 1.5 m depth in the southern end of the OTD (Figure 2). At the northern end the tailings were saturated, so materials from locations 12, 13 and 24 were obtained by hand-dug trenches (c. 60 cm depth). From locations 14 to 23, a weighted piston corer was used, allowing the collection of samples below the northern pond under a varying water depth of 0.3 to 10 m. Samples to a maximum depth of 1.5 m were collected by the piston corer. All tailings samples were cooled and transported to the University of Tasmania (UTas). Cores were frozen to enable logging, and then oven-dried. Each sample ($n= 144$) was split in two, with one portion milled to $<63 \mu\text{m}$ for geochemical testwork, and the other used to prepare polished thin sections and grain mounts for mineralogical evaluations.

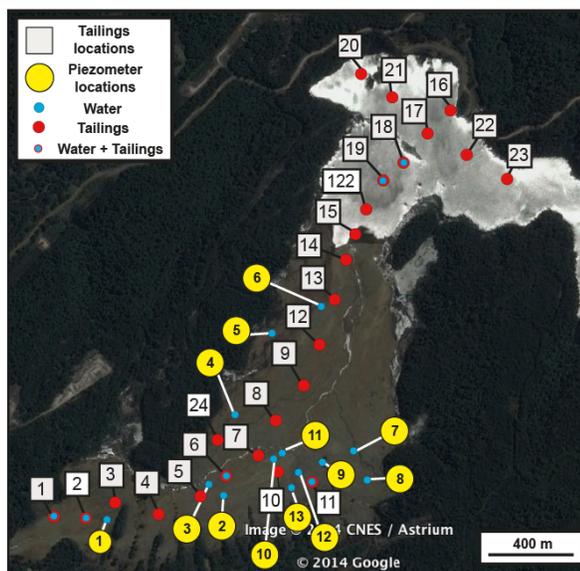


Figure 2 Plan view map of the Old Tailings Dam showing tailings sample locations (red; $n=24$), and water collection locations (blue). Established piezometer locations are shown in yellow circles (image from Google).

Geochemical testwork

Paste pH testing represents the most efficient manner by which to assess a sample's immediate acid forming characteristics, and was performed on all samples at UTas, according to ASTM D4972-01(2007) following the recommendations given in Noble *et al.* (2012). In addition, rapid and accurate measurement of total sulphur (wt. %) for calculation of maximum potential acidity (MPA) was performed on all samples using an Eltra C-S 2000 instrument at UTas. A sub-set of samples ($n=55$) was chosen for multi-addition NAG pH (UTas) and acid neutralising capacity (ANC) testing (Australian Laboratory Services, Brisbane, Australia). Pore water chemistry was assessed on 20 samples collected from established piezometers (Figure 2), the base of trenches, and the frozen waters in cores. These were analysed by solution ICPMS methods at UTas. Six samples were selected from across the OTD for laser-ablation ICPMS (LA-ICPMS) testwork to provide an indication of the pyrite chemistry. Appropriate sample blanks and standards and certified reference materials were used throughout all analyses, with the standard deviation for each data set calculated as <0.5 .

Mineralogical evaluations

To determine their mineralogy, tailings samples were subjected to XRD analysis ($n=55$). A benchtop Bruker D2 Phaser X-ray diffractometer (XRD) instrument with a Co X-ray source was used to perform these analyses at UTas. Each sample was ground in an agate mortar and pestle, loaded into the sample holder and placed into the machine chamber. XRD analysis was performed for 1 hour at an operating voltage of 30 kV and 10 mA using a cobalt X-ray tube. A 0.6 mm (0.3°) fixed divergence slit, 2.5° soller slit and an iron-filter were also used. Each scan ranged from 5 to 120° (2θ) with a 0.02° step size and a measurement time of 0.6 seconds per step. Mineral phases were identified using the Bruker DIFFRAC.EVA software package with the PDF-2 (2012 release) powder diffraction file mineral database. Mineral abundances were semi-quantified by Rietveld analysis using TOPAS pattern analysis software. In general, bulk mineralogy is considered unable to confidently identify the presence of trace sulphide oxidation products (i.e., < 1 wt. %), so a modified sulphide alteration index (SAI) following Blowes and Jambor (1990) was established. Criteria for this SAI (Table 1) were established following the examination of select grain-mounted samples ($n=10$) using a Hitachi SU-70 field emission scanning electron microscope (UTas, Central Science Laboratory). Thin sections ($n= 55$) prepared from tailings samples were evaluated by reflected light

microscopy, with 20 pyrite grains ranked by this criteria and the values averaged to give a final SAI value.

Table 1 Modified sulphide alteration index (SAI) defined and used in this study.

Score (/10)	Features
0-1	Only a few grains of pyrite are weakly altered along edges and fracturing is minor; >95% of the grains have sharp, fresh margins.
2-4	Pyrite grains have rims emerging and fracturing is minor.
5-6	Pyrite grains have >narrow alteration rims with fracturing becoming predominant.
7-8	Appearance of secondary minerals, increase in abundance of fracturing and subsequent altering.
9	Pyrite <30% pyrite remaining, dominated by alteration around rims and fractures.
10	Only traces of pyrite remain.

RESULTS AND DISCUSSION

Tailings facies

Tailings within the OTD demonstrate a high degree of heterogeneity with ten facies identified, and four distinct zones defined: A - unsaturated coarse-grained (locations 1 to 3); B - unsaturated fine-grained (locations 4 to 6); C - intermediate (saturated/unsaturated; locations 7 to 14, 24); and D - saturated (locations 15 to 23). Oxidation of sulphide minerals in the tailings was indicated by a change in colour from grey to yellow, orange or red. The vertical extent of oxidation into the tailings varied throughout the OTD. For example, in Zone A, (proximal to the discharge) water percolation occurs due to the coarse nature of these tailings, therefore this zone is extensively oxidised (Figure 3a). In general, the oxidised hardpan layer thickness decreases towards the northern pond. Thin oxidised layers observed at depth (e.g., c. 10- 50 cm; Figure 3b) in trenches represent former hardpan surfaces, which have subsequently been covered with tailings. Some subsurface layers, particularly coarse sulphide sand layers, appeared to be undergoing some degree of oxidation (Figure 3c). This is likely caused by vertical fissures (occasionally observed) allowing water and oxygen ingress through the buried tailings. Re-sedimentation of hardpan material in the shallow pond area (e.g., Figure 3d) added complexity in determining the minimum depth of water cover required, as there may be a combined protective effect of both hardpan and water.

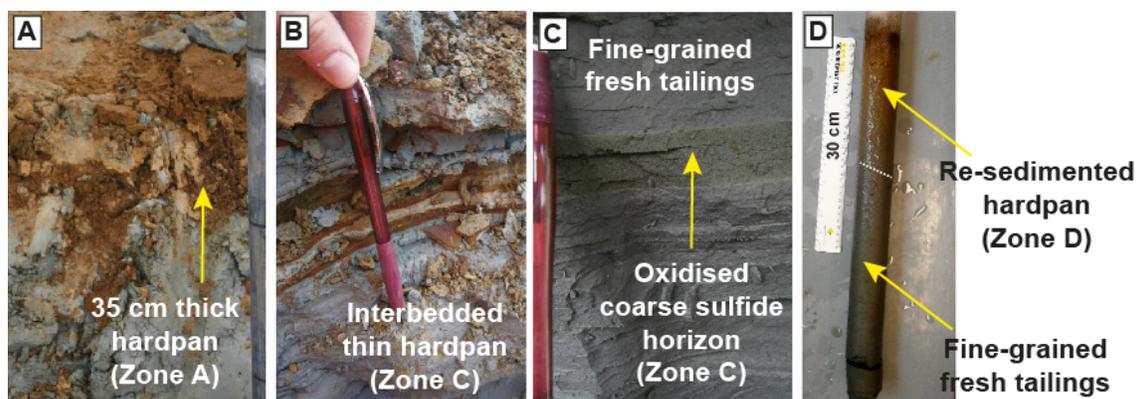


Figure 3 Examples of tailings oxidation observed in the OTD. A: Thick oxidised hardpan (30 to 35 cm depth) prevalent in Zone A; B: Thin hardpan layers interbedded within fresh tailings packages at depth (c.1 m; Zones B and C); C: Coarse oxidised sulphide horizon observed in Zones B and C; D: Re-sedimented hardpan seen in Zone D.

Tailings geochemistry

The assessment of lag-time to AMD and therefore risk (from high to low) was determined using the net acid generation (NAG) pH vs. paste pH classification scheme (Figure 4; Weber *et al.* 2006). The majority of samples from Zones A (unsaturated, coarse-grained), B (unsaturated, fine-grained) and C (intermediate zone) were classified as intermediate-high risk with acid formation rate predicted as rapid to immediate. This is new evidence that these tailings are severely acid forming at depth and require rehabilitation, as previously this has only been inferred from water chemistry and assessments of samples taken from the upper 30 cm (c.f. Hassell, 2005). The majority of Zone D samples fall into potentially acid forming (PAF)-low risk, inferring a potential relationship between permanent saturation and higher paste pH values. A net acid producing potential (NAPP) vs. NAG pH classification plot showed that the majority of OTD samples are PAF, with a NAPP range from -2.5 to 797 kg H₂SO₄/t (average: 187 kg H₂SO₄/t; median; 160 Kg H₂SO₄/t). Acid neutralising capacity (ANC) values were in the range from 0 to 41 kg H₂SO₄/t, with an average value of 15 kg H₂SO₄/t. Despite the presence of magnesite in the primary ore-mineral assemblage at Savage River, it was not reported by XRD (i.e., only trace amounts below the detection limit of 1% present). Therefore, ANC is likely sourced directly from chlorite, an effective silicate neutraliser (c.f. Craw, 2000; Jambor *et al.* 2002; 2007). Additionally, Hutchison and Brett (2006) and Gerson *et al.* (2014) have documented the use of the chlorite-calcite schist/amphibolite for AMD control at Savage River, from which both calcite and chlorite are thought to contribute to the net-neutralising potential.

Chemical analyses of pyrite (LA-ICPMS; *n*= 180) in six samples collected across the OTD indicated that generally the pyrite is enriched in Co and Ni (Figures 5a and 5c). Copper, which is elevated in the waters draining the site (c.f. Hassell, 2005), is also relatively high in these samples (average 620 ppm), with the highest concentrations measured in Zone A (Figure 5b). Zinc was generally low (average 10 ppm): however, its concentration steadily increased towards Zone D (Figure 5d), indicating that it has leached from Zone A; i.e., susceptible to meteoric water influx and under low pH conditions (average pore water pH: 2.7).

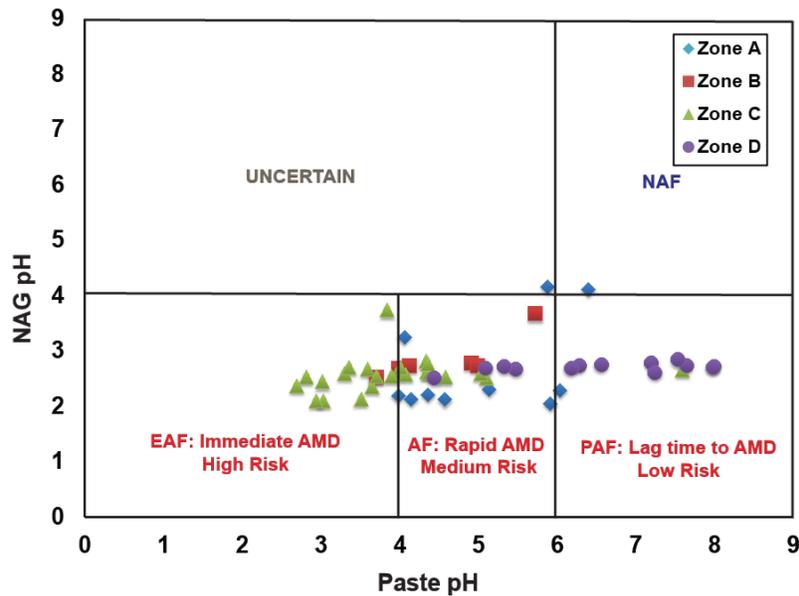


Figure 4 NAG pH vs. paste pH risk classification plot (after Weber et al. 2006) for tailings samples collected across the OTD (samples grouped by zones: A- unsaturated and coarse-grained; B- unsaturated, C- intermediate saturation and D- saturated). Abbreviations: EAF, extremely acid forming; AF, acid forming; PAF, potentially acid forming; NAF, non-acid forming.

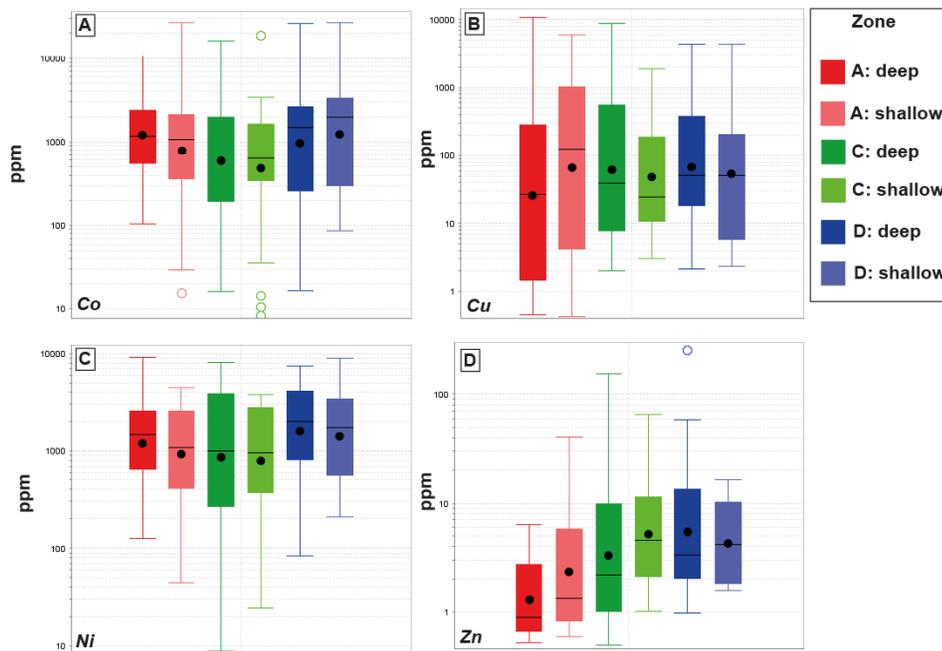


Figure 5 Tukey geochemical plots of select potentially deleterious elements (ppm) contained in pyrite (n=6) from the OTD: Co (A), Cu (B), Ni (C) and Zn (D). (NB/ black dots indicate average, solid black line the median, solid colour box ends indicate the interquartile range, whiskers include outliers (indicated by open circles and triangles) in the recalculated interquartile range).

Chemical analyses of tailings pore water showed high sulphate concentrations (max. 12,900 mg/L; Zone C) in the unsaturated and intermediate saturated zones, indicating active AMD formation. In Zone D concentrations below ANZECC (2000) drinking water guideline (DWG) values (500 mg/L) were measured (average 157 mg/L), confirming that the established natural water cover is effective at retarding AMD formation. Furthermore, average pH values of 4.5 were measured in this Zone. Across the OTD As, Cd, Ni, Sb, Pb and U were below ANZECC (2000) DWG values. Occasional exceedances of Al (max. 180 mg/L), Cu (max. 35 mg/L), Se (max. 0.1 mg/L) and Zn (max. 4.6 mg/L) relative to ANZECC (2000) DWG values were identified, predominantly in Zones B and C, but not in Zone D. In general, Fe was high across the OTD, with six locations reporting concentrations >1000 mg/L; however, no ANZECC (2000) DWG value is available for comparison. This is also the case for Co, with average concentrations of 5 mg/L measured.

Tailings mineralogy

The bulk mineralogy of the tailings across the OTD was dominated by albite, actinolite, antigorite, quartz, tremolite, talc and chlorite, as shown in Figure 6. In all zones the content of pyrite increased with depth, confirming the presence of fresh tailings at depth. In the upper most samples from Zone A and B trenches (0 to 30 cm), secondary minerals goethite, ferrihydrite, hematite and jarosite were observed, which are common hardpan minerals. The presence of goethite and jarosite indicated the latest stages of sulphide oxidation (c.f. Jambor, 2003). The highest jarosite concentrations were observed in Zone C (intermediate saturation), which concurs with the environment of its formation (c.f. Dutrizac and Jambor, 2000). In Zone D there was a notable decrease in the quantity of these secondary minerals, confirming that these materials are relatively fresh. Additionally, there is also a lower pyrite content, likely related to the distance from the discharge; although on average the grain size was finer. The bulk mineralogy confirms that the established water cover is effective in retarding oxidation, with very minor (< 2 wt. %) secondary minerals developed. Where these were observed (i.e., locations 15 and 18; Figure 2) it indicated re-sedimentation of hardpan materials from Zone C.

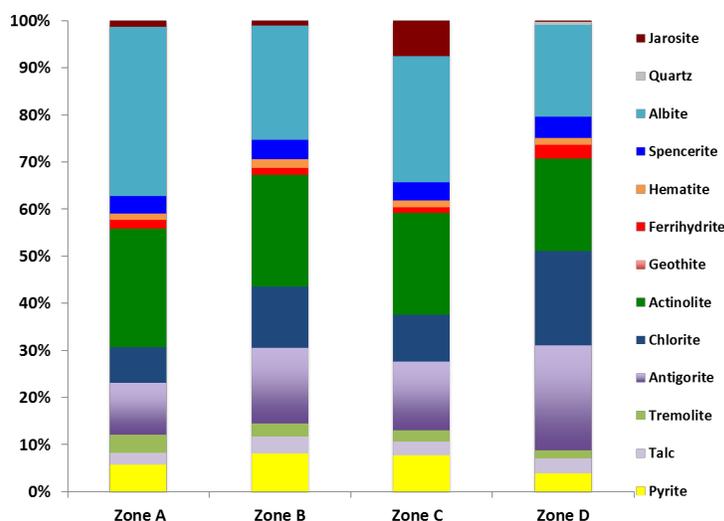


Figure 6 Summary of bulk mineralogy (measured by XRD semi-quantitative, modal abundance as a % shown) for Zones (A to D).

Sulphide alteration index (SAI) values calculated across the OTD are shown in Figure 7. The highest SAI values were measured in Zone A (6/10) in the upper horizons (c. 30 cm; Figure 8a). The lowest readings (1/10) were recorded in Zone D at depth (150 cm, under 430 cm water cover; Figure 8b). These results concur with the tailings geochemistry. From these values, the minimum depth of an effective water cover for retarding AMD formation is identified as 1 to 1.5 m.

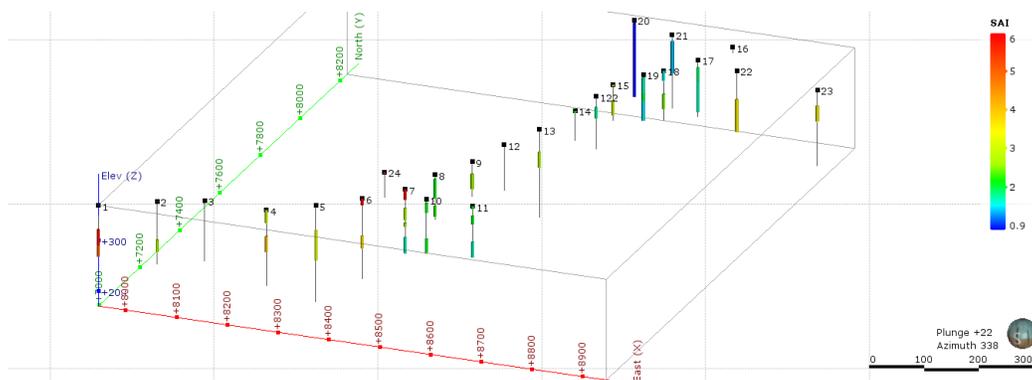


Figure 7 3D schematic diagram of the OTD showing sample locations and sulphide alteration index (SAI) values. Zones A to D are indicated (by thick black lines) and the northern pond is shown in Zone D.

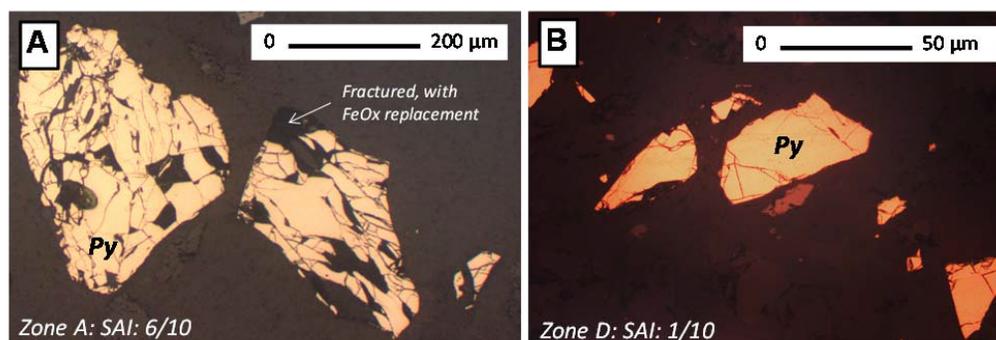


Figure 8 Photomicrographs (reflected light) of pyrite. A: Grains from Zone A (30 cm depth) showing extensive oxidation with a sulphide alteration index (SAI) score of 6/10 given; B: Grains from Zone D (45 to 75 cm depth, under; 100 cm of water) are comparatively smaller, and much less altered with a SAI score of 1/10 given, indicating this is an appropriate water cover depth.

CONCLUSION

The presence of a natural water cover in the northern portion of the OTD at the Savage River Mine provided the unique opportunity to evaluate the application of a total water cover as a cost-effective rehabilitation strategy. The OTD currently contributes 50% to the total AMD load from the Savage River Mine. As the OTD tailings are historic (managed by the SRRP), knowledge of the tailings geochemical and mineralogical characteristics at depth were limited, and therefore the nature of potentially reactive pyritic tailings remaining in the OTD unknown. This study has penetrated the OTD to depths of 1 to 1.7 m, revealing complex sediment architecture, with ten facies identified and four zones (A to D) defined based on grain size and degree of saturation. Geochemical results indicated that these tailings remain acid forming, with Zones A to C (unsaturated coarse-grained, unsaturated fine-grained, and intermediate, respectively) classified as high risk. Tailings in Zone D (saturated) were less reactive, and classified as low-risk. The sulphide alteration index (SAI) mineralogical assessment allowed for the oxidation front to be better approximated, and confirmed the natural water cover as effective for retarding oxidation. Furthermore, the results indicate that a minimum water cover depth of 1 to 1.5 m is required across the entire OTD. Predictions of re-suspension by wind or wave action have yet to be performed; however, observations of re-sedimented hardpan in the surface of core samples from Zone D show that at 1.5 m such sediments are not seen. Potentially, this more conservative depth of 1.5 m should be used.

Ultimately, a phased tailings rehabilitation management strategy which utilises a water cover will prove the best solution at the OTD. Phase one should focus on the removal of the upper most > 0.5

m of tailings (i.e., oxidised portion) in Zone A, followed by clay capping. This will leave this zone with characteristics similar to the remainder of the unsaturated portion of the dam (i.e., Zones B and C), allowing them to be managed together. Phase two involves the extension of the northern pond area to saturate tailings in Zones A (after clay-capping), B and C to 1.5 m depth in order to reduce sulphide oxidation rate. However, this should only be undertaken after geotechnical work to raise the dam walls has been completed. Phase three requires installation of flow-through reactors to neutralise any AMD emanating from the OTD. At least two alkaline flow through reactors should be installed, one at the north slot monitoring station where seepage is occurring, and the other at the former southwest spigot point in case the water in the OTD back flows as it is being filled. Phase four involves establishing a long-term monitoring programme, which focuses on identifying any additional seepage points, measuring water quality to test how effective the water cover is, measure water level fluctuations, and identifying if there has been any re-suspension of tailings. In several years, a follow up mineralogical study should be performed to evaluate the performance of the water cover by measuring the degree of oxidation using the SAI.

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