Physical, Mechanical, and Metallurgical Properties of Pastefill Using Recycled Gold Tailings

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Abstract

Pastefill is used as an efficient, sustainable product to improve safety and stability of underground mines, while facilitating continued extraction of ore. Many mines have the advantage of using fresh tailings derived from the process plant to produce pastefill; however, this is not always possible if the mine and processing facility are not co-located, tailings quantities are insufficient, or the properties of the tailings are not suitable to make pastefill. Mines subject to these conditions are then required to look at alternative sources to procure pastefill, such as excavation of nearby tailings from historic mining operations.

This paper examines use of tailings from a historic gold mining tailings storage facility as pastefill for Kidd Mine. With the mine considering further expansion and deepening, the existing tailings source and two alternatives were critically assessed to determine a suitable source for continued pastefill production. Test work was completed and a thorough review of the tailings properties and their influence on pastefill strength and rheology was completed. The test work also reviewed the impact of temperature on the pastefill strength given the expected temperatures of the mine at depth. Further to the physical, mechanical, and metallurgical properties of the tailings sources, the economic implications and practical challenges associated with excavating tailings from each of the sites was evaluated to determine a suitable source to align with the potential mine expansion.

Introduction

Kidd Mine, located 24 km north of Timmins, Ontario, is the world's deepest base metal mine below sea level (Glencore Canada, 2024). The mine commissioned its existing pastefill plant in the fall of 2004, and it has since served as an exceptional example of best practice in pastefill. A unique feature of Kidd Mine operations is that the Kidd Metallurgical Site (Metsite) is located nearly 50 km away from the mine. With tailings produced this far from the mine, Kidd Operations-Glencore Canada Corporation (Glencore) decided to use an excavated source of gold mine tailings which did not contain sulphides and had low moisture content, facilitating easier transport. Initially, the paste recipe design included 55% sand in the paste blend, which had the benefit of easier excavation and transportation, shorter haul distance, and increased paste strength. (Lee and Pieterse, 2005). Sand is sourced from a local esker sand pit. During the first two years of operation, the mine used tailings from a different source, but then switched to the Site A tailings storage facility (TSF) in 2007 as this site is located nearer to the mine, reducing operating costs associated with trucking and limiting truck traffic in the City of Timmins. Site A and the esker sand pit are located 20 km and 15 km south of the mine, respectively (McGuinness and Bruneau, 2008).

After 55 years of mine production and with reserves declining, Glencore undertook a feasibility study (FS), referred to herein as the Mine 5 project, to investigate further deepening of the existing mine. Glencore, Paterson & Cooke, and WSP Canada Inc. formed the project team, tasked with developing the pastefill system FS, which included upgrades to the plant, extension of the underground distribution system (UDS), and evaluation of suitable sources for backfill materials (sand and tailings) necessary to fulfill the Mine 5 life-of-mine (LOM) requirements. The sourcing and evaluation of backfill materials was the focus of the study and this paper centers around the tailings characteristics, economic implications, and practical challenges associated with excavating tailings from the sites under consideration.

Paste Plant Process

The existing paste plant at Kidd Mine receives esker sand and excavated tailings, both of which are weighed on arrival to site and stockpiled separately in domes. A loader transfers the sand to an aggregate feed hopper, where it undergoes a final screening process prior to being conveyed to the plant and held in aggregate storage bins. A second loader transfers tailings to a separate hopper equipped with a live bottom feeder. The live bottom feeder discharges the tailings onto an inclined conveyor to a continuous mixer for repulping. Once repulped, the tailings are held in an agitated tank until a paste batch sequence is initiated. During a paste batch sequence, all inputs to the paste (esker sand, tailings, binder, makeup water) are diverted to weigh bins prior to entering the batch mixer to ensure that a reliable and repeatable paste recipe is achieved. From the batch mixer, the paste reports to a gob hopper, which feeds the underground borehole. Refer to Lee and Pieterse (2005) for the process flow sheet.

Paste Recipe

Defining the paste recipe is of utmost importance as it has a direct influence on the unconfined compressive strength (UCS), rheology, and economics of the paste-filling operation. When establishing the recipe, the material characteristics of each component are considered, along with the synergistic effect on rheology of the final paste product. Hydraulic analysis of the UDS must also be well understood such that a “rheology operating envelope” is established. Successful paste-filling operations can balance these parameters to maintain proper laminar full pipe flow characteristics to minimize pressure, water hammer and wear. The material blend (sand and tailings) at Kidd Mine targets a final PSD range of 25 to 30 % passing 20 μm. Currently, the site uses a 60/40 blend (esker sand/site A tailings) to achieve this target. The blend is operationally sound (in terms of paste hydraulics and UCS) and benefits from the higher proportion of sand, which has a lower cost per tonne compared to excavated tailings.

Tailings Requirements

As noted above, esker sand and tailings are used to produce paste backfill at Kidd Mine. Both materials are available from within reasonably short distances to the paste plant. At the outset of the project, the team established that the expected tailings demand until 2041 would be approx 5.5 Mt (3.6 Mm^3) based on the mine plan and paste plant mass balance.

Details of Selected Tailings Facilities

There are several historic tailings facilities in the Timmins, Ontario area where tailings could be sourced and repurposed for use in paste backfill. The authors of this paper have investigated the characteristics and available quantities of tailings in selected historic tailings facilities (Figure 1). Site A is located approximately 4 km northeast of Timmins, while Site B is located between Timmins and South Porcupine, south of Hwy 101. Sites A and B are owned by Newmont, and Site C is owned by Glencore in the Metsite Tailings Management Area (TMA). The distances between the three locations (Sites A, B, and C), and the mine site are approximately 20, 30, and 50 km, respectively.

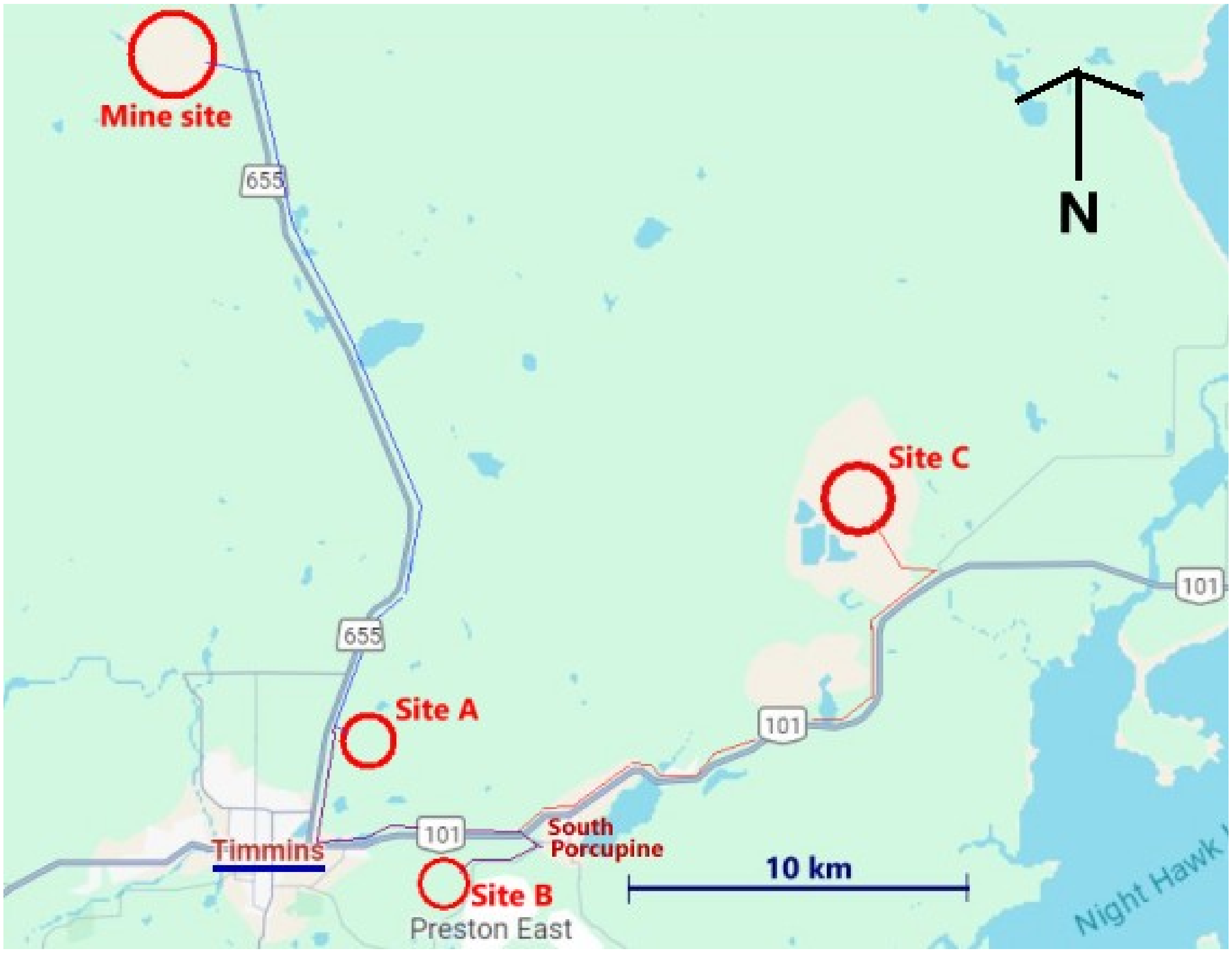

Figure 1. Location of the three tailings sites investigated and the mine site. Source: Google Maps.

Site A

Site A is located north of Hwy 101, about 20 km south of Kidd Mine. Between 1912 and 1989, about 21 Mt of tailings were deposited into five adjacent tailings containment areas within the facility, covering approximately 215 ha. Tailings were initially deposited to the south and southeast of the site in 1912. By 1947, construction of the northern and eastern perimeters had commenced and by 1981, all side-limits of the site were completed. Tailings deposition continued until 1989. Tailings were sub-aerially deposited to each cell by spigotting from a 150 m (500 ft) long peripheral wall section, sequentially closing and opening spigots after defined time periods to facilitate perimeter dam raising works. The hydraulic deposition of slurry tailings using rotational spigotting has produced a heterogeneous beach deposit. Typically, coarser tailings particles are deposited close to the point of discharge, and particles gradually become finer further from the point of discharge. The finest material (slime) settled in the decant-ponds. Further hydraulic particle sorting has occurred between spigotting points. Additional layering has taken place during the winter months, when the total tailings product was likely open-ended onto the slimes beach and may have reported directly to the decant pond. Starting in 2006, tailings were excavated from a portion of the facility in accordance with an initial tailings' recovery plan. Many boreholes, test pits, and cone penetration tests have been completed in the past within the facility to help with the design of internal drainage features and frequent review of the dams' slope stability.

Based on the results of historic in-situ and laboratory testing programs, the stratigraphy of the site can be summarized as follows (a typical soil profile across the center of the facility (east-west) is shown in Figure 2):

- **Tailings:** thickness ranges from 3 m (north) to 20 m (south) within the facility footprint
- **Trapped organics:** beneath the tailings deposit, a trapped layer of organics (peat) was encountered
- **Sand and Silty Sand:** native sand and silty sand deposit was encountered below the tailings and trapped organics, with thickness generally varying between 5–10 m, overlying bedrock

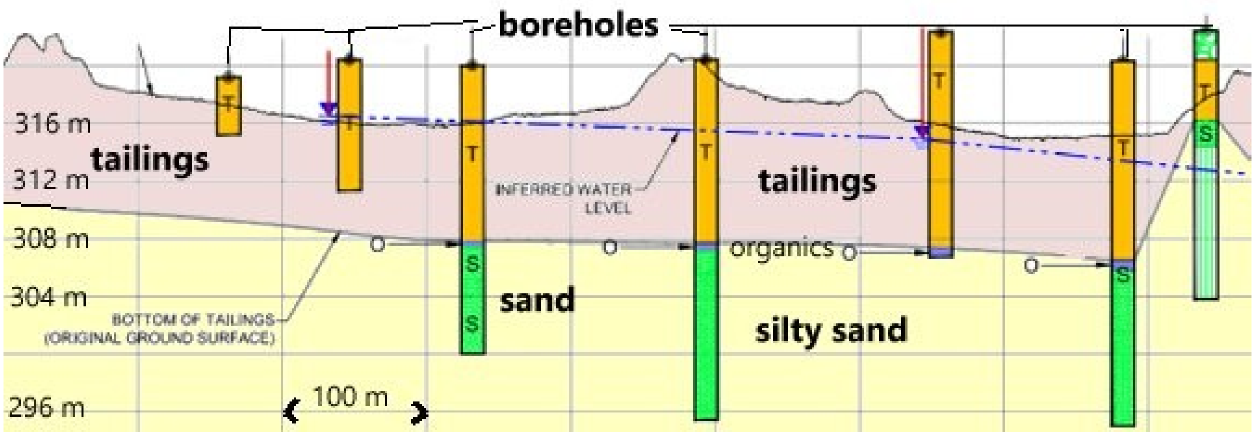

Figure 2. Typical soil profile across Site A.

The average specific density of the tailing particles, based on testing, was determined to be 2.8 t/m^3. A relevant dry density value of 1.54 t/m^3 was considered for the estimation of dry tailings tonnage available for extraction from the tailing facility, based on a solid content of 55% by volume.

Site B

Site B is about 5 km east of Timmins, bounded by a local road to the east, an abandoned railway line to the north, a lake to the west and another tailings facility to the south. Historically, the perimeter dams of site B were raised with the upstream construction method, using tailings and waste rock for construction. Within the facility, a decant structure regulated the pond in the central area of the tailings mass and discharged the effluent via a decant pipe through one of the perimeter dams. Since the late 1960s, the facility has undergone various phases of rehabilitation; the decant system has been decommissioned and replaced with an open spillway over the north dam. Presently, almost all historically deposited tailings outside of Site B are either below the current water level of the lake or are covered with marsh vegetation.

Site C

The majority of Site C is in Hoyle township to the northeast of Timmins, Ontario. The site started operation in 1966. Currently, the Metsite has a concentrator that processes copper and zinc ore from the mine, and a TMA. The Metsite TMA occupies approximately 1300 ha and includes:

- active tailings deposition area (approx 524 ha),
- low-density sludge water treatment system (ie, network of ponds)
- sludge storage cells
- seepage collection ditches and ponds
- additional support infrastructure (tailings thickener, pipelines, lime stations, pump stations, etc.)

In 1973, operations implemented the Robinsky thickened tailings deposition method to replace conventional slurry deposition. Flotation tailings are pumped to the tailings thickener, increasing the solid content to approx 60% before deposition in the TMA. The high-density tailings underflow is discharged near the center of the active deposition area. This deposition method has formed a cone-shaped deposit with an average slope of approx 2%. At the time of writing, from the perimeter dyke to the center of the cone, the tailings surface rises by approx 32 meters.

Since 1966, approx 139 M tonnes of tailings have been deposited within the TMA. In addition to copper and zinc tailings, other waste was deposited within the TMA, such as nickel and gold tailings, jarosite, pyrite concentrate, waste rock, granulated slag, and low-density sludge. Considering the sulphide-containing properties of copper and zinc ore from the mine, Site C tailings are potentially acid generating (PAG). The absence of a supernatant pond covering the tailings surface requires a constant staged deposition throughout the year to keep the surface saturated with fresh tailings to minimize acid generation. Water from runoff and seepage is collected in ditches and is treated with lime to elevate pH and precipitate suspended metals in the water treatment ponds. Carbon dioxide is added to adjust pH and meet regulatory standards before discharging water to the environment.

Material Extraction and Testing Requirements

Sampling campaign

An important factor when assessing the suitability of the tailings sources is to map out the full cross-sectional area of the pits/ponds at varying depths to define the material source potential over the LOM. Benchmarking practices were used as an initial assessment to understand the viability of the materials at each proposed material source. For Sites A and B, two types of samples were obtained:

(1) A defined sample using a hollow-stem auger and split spoon at each metered interval in accordance with ASTM D-156. Figure 3 shows the drill rig used during the winter sample campaign.

(2) Bulk samples obtained from the material sampled at each metered interval.

The split spoon soil sampling method allows for a defined sample at each interval, with minimal contamination from the previous depth/level of tailings. This permits accurate sampling at each metered interval across several locations so that characterization (ie, particle size distribution, solids specific density, pH) could be assessed within the tailings area. For Sites A and B, the samples were obtained at every meter below the surface until native soil was encountered at various depths, generally between 4–16 m below grade. This was indicated by a layer of black/brownish organics (peat), followed by the native soil layers. For Site C, the boreholes were advanced to a depth of 5 m, as the historic tailings below that depth were expected to contain unfavorable materials for pastefill use.

Since the split spoon sampler is limited in the quantity of material it can obtain, bulk sampling was also completed to fulfill the tailings requirements for the pastefill portion of the test work. For Sites A and B, an auger drill rig was used to obtain the samples. As the auger drilled down, auger “cuttings” were collected at each metered interval, ensuring that no native soils/sands were captured. Due to drill rig mechanical issues, an alternative bulk sampling method was used at Site C, which involved digging test pits. The first 0.5 m of material was excavated to break through the frost layer and remove snow from the sample. The excavator then dug down from 0.5–1.5 m, allowing collection of a sample.

Figure 3. Drill rig at Site A in early February 2022.

Material source characterization

The pH of the source used in a pastefill recipe can provide indication of the mineralogical composition of the materials, while changes in pH can impact the rheology and long-term strength development of the pastefill. The pH of each split spoon sample was measured to provide indicative data that, coupled with the PSD benchmarking values, were used initially to identify viability of the potential source materials. The following sections summarize these data.

Site A

Many boreholes were drilled across the tailings source, with numerous samples obtained. Figure provides an overview of the typical tailings sample obtained from Site A in wet and dry conditions. The pH of the split spoon samples ranged from approx 6–7.5. The mineralogy of the tailings consisted mainly of inert silicate and carbonate minerals from a cement chemistry perspective. However, the results reported trace amounts of sulfide minerals (pyrite, pyrrhotite), which in sufficient quantities can negatively impact the strength development of cemented backfill.

The minimum and maximum void ratio values were also tested in the laboratory. Based on test results, the average solid content by volume in the tailings samples at Site A varied between 42% (loose) and 64% (dense). The PSD results for the split spoon samples are summarized in Figure. According to the test results, the tailings samples were comprised predominantly of silt (particle size 2–75µm), with less than 10% clay sized particles (< 2 µm). The percentage of sand sized tailings particles by dry mass generally varied between 5–30%; however, somewhat coarser tailings with slightly more than 30% sand sized fraction was also intercepted in several boreholes. SPT N-values, recorded during the sampling program indicated very loose to compact conditions within the saturated tailings mass.

Figure 4. Wet (L) and dried (R) bulk tailings sample from Site A.

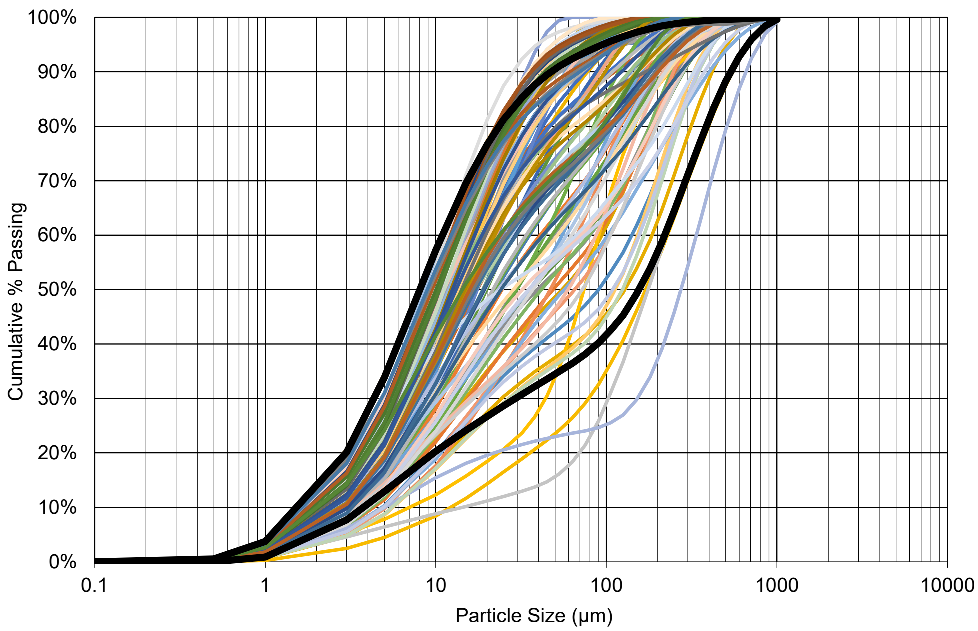

Figure 5. Site A split spoon samples PSD comparison.

Site B
For Site B, 11 boreholes were drilled and sampled. The drilling, sampling, and laboratory test campaign followed the same methodology as Site A. Figure 6 provides an overview of the typical tailings sample obtained from Site B in wet and dry conditions. The pH of the split spoon samples ranged from approximately 6–7.5, and the mineralogy of the tailings also consisted mainly of inert silicate and carbonate minerals, with trace amounts of sulfide minerals (pyrite and pyrrhotite). The minimum and maximum void ratio values were tested in the laboratory for the tailings from Site B. Based on test results, the average solid content by volume in the tailings varied between 44% (loose) and 60% (dense). A relevant dry density value of 1.54 t/m^3 was determined. Tailings samples were comprised predominantly of silt (particles size 2–75µm), with less than 10% clay sized particles (< 2 µm), as with Site A. However, the percentage of sand sized tailings particles by dry mass was generally < 20%. Variation in particle size distribution in a typical borehole at Site B is shown in Figure.

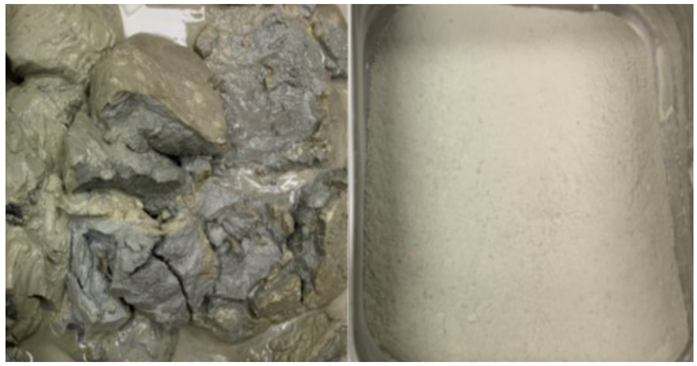

Figure 6. Wet (L) and dried (R) bulk tailings sample from Site B.

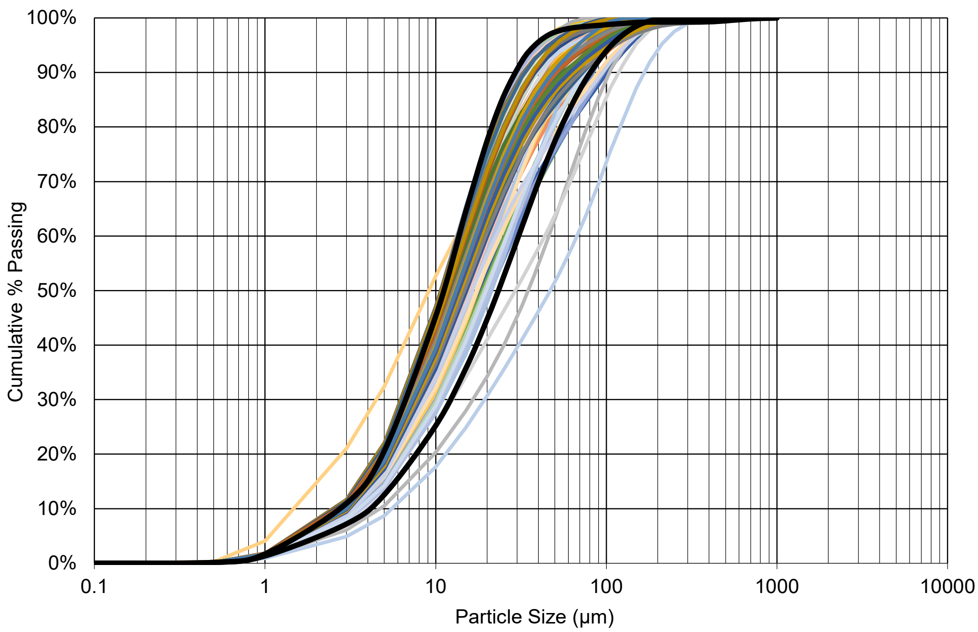

Figure 7. Site B split spoon samples PSD comparison.

Site C

To assess the suitability of TMA tailings for paste backfill, 17 boreholes and 14 test pits were used for sampling within the active deposition area. In five selected boreholes, soil sampling was performed within each stratum at intervals not greater than 1 m to collect representative tailings samples for mineralogy testing. The pH of the Site C tailings was notably lower than that of the other sites and ranged from 4.5–7. As at other sites, the mineralogy of Site C tailings consisted mostly of inert silicate and carbonate minerals; however, there was increased content of pyrite and pyrrhotite noted, as well as the presence of siderite. Kidd Operations staff also cautioned that jarosite had been historically co-deposited in the tailings stream. According to the test results, the tailings samples were comprised predominantly of silt (particles size 2–75µm), with < 15% clay sized particles (< 2 µm). Variation in particle size distribution in a typical borehole at Site C is shown in Figure 8.

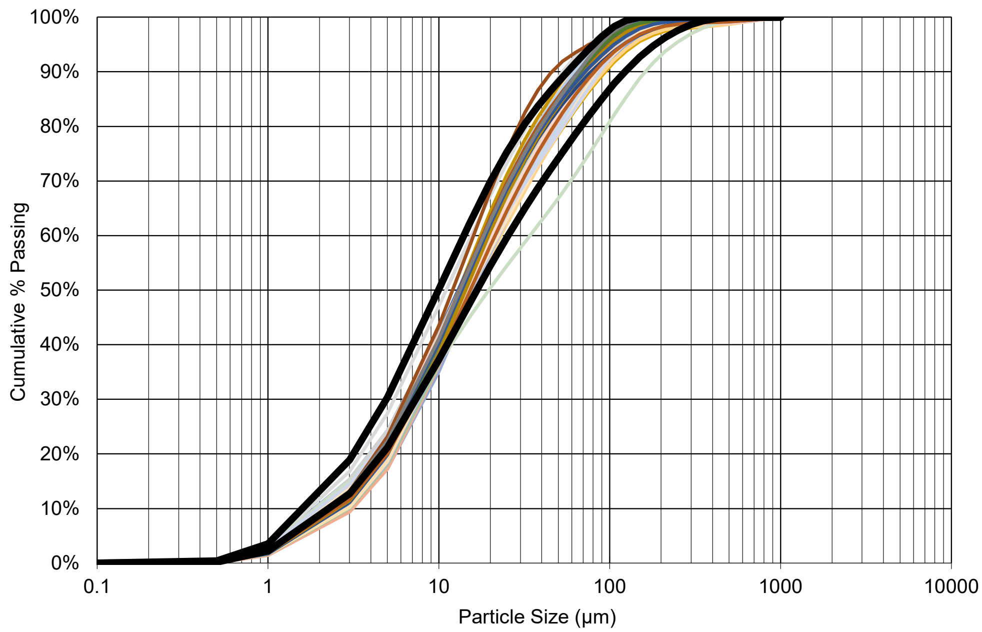

Figure 8. Site C split spoon samples PSD comparison.

Bulk sample characterization
Additional pH and PSD testing was completed on each of the bulk samples. Once it was confirmed that the bulk samples were within acceptable benchmarking limits (ie, pH >6), and that the material contained enough fines to maintain laminar flow (20% > 20 µm), project test work progressed. Theoretical blend calculations were used to confirm the blending ratio of esker sand to source tailings required to achieve a target PSD of 25–30% passing 20 µm. For each of the tailings sites, a blend of 60% esker sand to 40% tailings satisfied this target.

Rheology
Once the blending strategy was established, flow characteristics of each paste blend were evaluated using static yield stress rheology tests. For each blend, a yield stress vs paste solids concentration was established over a wide range of values to confirm laminar flow for various operating scenarios. Given the depths of the planned mining expansion and increased retention time in the paste line, rheology was also completed on each of the pastefill recipes using a set retarding admixture to understand the effect the admixture could have on the flow properties of the paste. A nominal admixture dosage of 40 l/tonne by mass of binder (admixture/binder) was used, and rheology results are summarized in Figure 9.

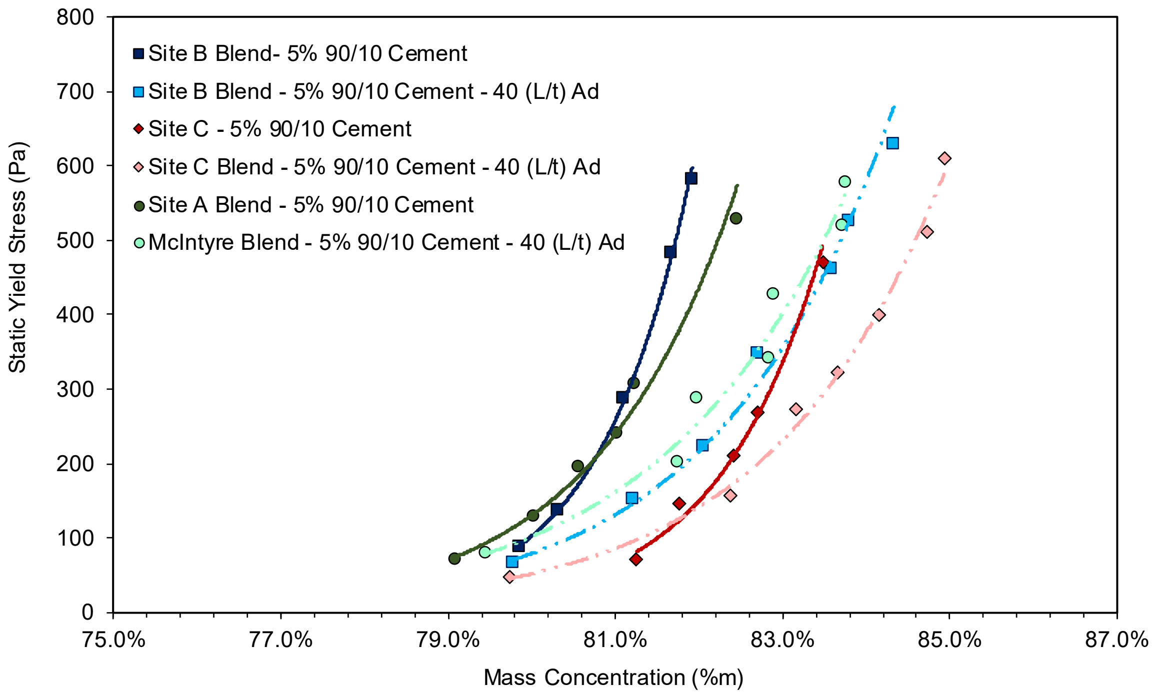

Figure 9. Static yield stress vs mass concentration summary results.

Vicat cement setting test work

As the depth of the mine and the expected paste retention time in the UDS increased, it became important to quantify the reaction time required by operations staff should an upset condition occur (ie, timeline for paste to set if blockage occurred). To gain appreciation for the set time of the pastefill, a modified Vicat needle penetration test was used. A sample of the Site A paste (60% esker sand, 40% tailings, 5% Terraflow 90/10 cement) with and without admixture was prepared for the following two scenarios:

(1) *A simulated worst-case scenario where there is a blockage without the material travelling in the line (ie, no cement shearing or minimal shear).* The sample was mixed only enough to disperse the cement thoroughly into the pastefill, creating a homogenous mixture. The sample was then put into a container and was not mixed or disturbed, other than during the Vicat needle penetration.

(2) *A simulated nominal operating scenario where the material was sheared for an expected retention time in the Mine 5 UDS (approx 55 mins), and then a blockage occurs.* The material was homogenized, and then mixed at lower intensity for 55 minutes, after which the material was placed in a container and was not mixed or disturbed, other than during the Vicat test process.

An admixture dosage of 40 L/tonne by weight of cement was used for the tests. Test results indicate that without the admixture, the baseline paste samples require approximately 20–24 hrs to begin setting, and that the set time could be effectively delayed with an appropriate admixture dosage. These results should be considered as index tests, as actual experience and *in situ* instrumentation has shown that the baseline paste with 4.5% binder will nominally begin to set in approximately 8 hrs within an open stope (Figure 10). Note that this test was completed at room temperature, while paste is expected to be approx 37°C at the end of the pipeline, which could explain the shorter set time.

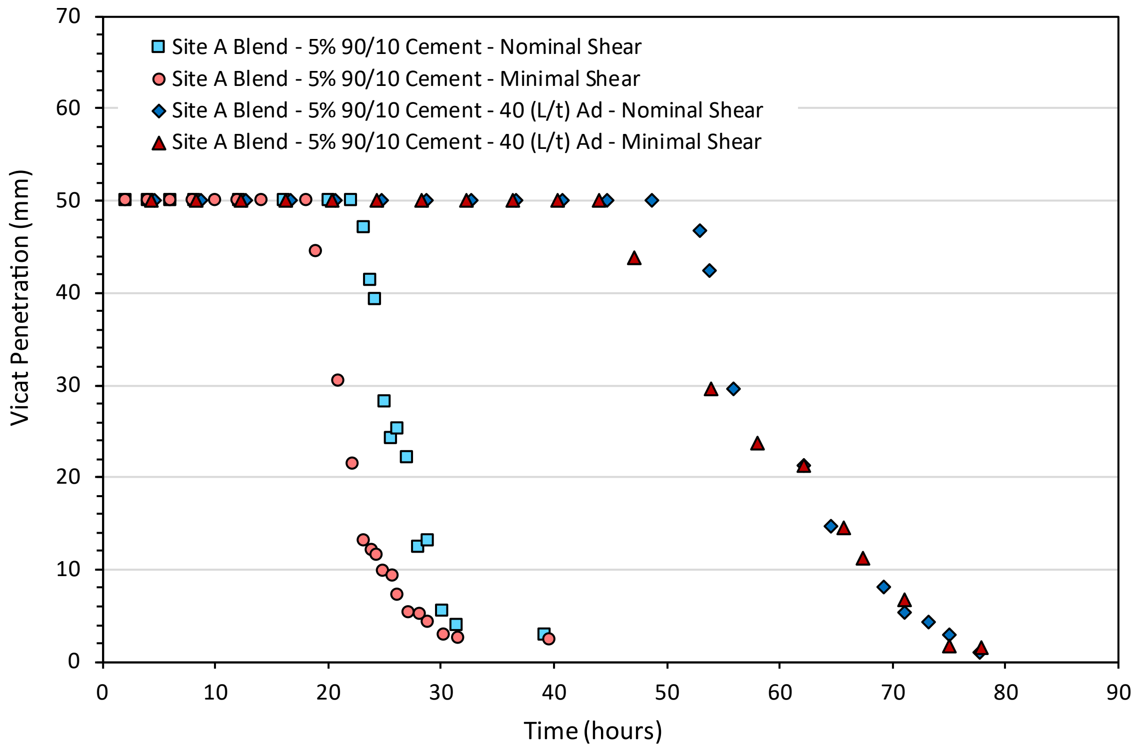

Figure 10. Vicat penetration vs time.

Unconfined compressive strength test work

Each pastefill recipe is unique to a given mine with the strength development of the material dependent on the chemistry of the constituents used (sand, tailings, process water, binder, etc.). With each of the source materials differing chemically, baseline UCS testing was required for each pastefill blend to quantify the required binder consumption to meet Glencore strength targets. Mining to the depths required of the Mine 5 project presented challenges that required careful consideration. Introducing a new chemical such as an admixture can impact the strength development of the backfill. Additionally, the *in situ* paste temperatures were expected to be approx 37°C at the time of placement, with the potential to reach > 60°C due to the heat generated during hydration, shown by the results of previous test work.

Given that the mine is currently in operation, binder selection test work was not required. Rather, the UCS test campaign was designed to:

- understand the strength development of the pastefill using each of the material sources to highlight considerations during the engineering design and operating expenditure (OPEX) costing
- investigate the impact of using binder set retarding admixture on the short and long-term strength development of the pastefill
- understand the influence of high *in situ* cure temperatures on strength development of the pastefill

A suite of UCS testing was completed using cylinder load rates (ASTM D2166) to quantify the binder consumption required for each of the scenarios presented above. As part of the study, ground water and surface water samples from the sites were sent for chemical analysis. The results detected elevated levels of boron in one of the groundwater samples collected from Site A. Boron is known to delay setting of binder and there was concern that the reported levels (approx 300,000 ppb) could impact the strength development of the pastefill. From a cement hydration perspective, the industry standard is to report metal concentrations by weight of cement (mg metal/mg cement) to understand the potential effect on cement setting. The above noted concentration of boron corresponds to a nominal value of 1.5% by weight of cement (accounting for supplementary cementitious material portion of the cement blend).

All source materials were able to achieve the mine's strength targets for the baseline pastefill blends. Each of the blends performed similarly, without a clear advantage in terms of binder consumption. Long-term strength degradation due to mineralogical composition was not observed. The results of the UCS testing with the three paste blends with admixture varied between sample sets, suggesting UCS test work with specific tailings is always required when considering an additive.

The following conclusions were noted based on the UCS test work results:

- For Sites A and C, the 7 day strength development of the paste was only slightly impacted by the addition of the admixture. However, Site B UCS cylinders with admixture experienced set delay and developed approximately 10–15% of the strength that the baseline samples did.
- All three sites experienced an increase in strength at the 28,180, and 365 day cure when compared to the baseline samples. Again, each site varied, and performed differently based on the chemical constituents of the tailing's sources.
- The increased cure temperature UCS cylinders all experienced similar strengths to those completed at room temperature; strength slightly increased in some cases and slightly decreased in others. This may be attributed to experimental error between test sets. The Site C paste blend at 5% binder was the only exception to this. The strength for this mix at the 7 day cure time approximately doubled in strength.
- The boron sample UCS mixes all performed similarly in strength, with no meaningful set delay at boron concentrations between 0.05–0.50% boron by weight of cement. However, large decreases in strength were noted at concentrations > 1% boron by weight of cement. This was noted for consideration when developing the mass balances and paste recipes for the project.

Challenges of Excavating Historic Tailings

Although being a sustainable solution for managing mine waste, excavation of historic tailings for use in pastefill can pose several challenges. The following challenges were encountered to a greater or lesser degree at the three sites.

Water management / erosion control

Approximately 96% of the surface water by Site A flows east to the Porcupine River. There are two surface water collection ponds within the facility: an upper settling pond, and a lower sedimentation control pond. Controlling the collection of surface water and its discharge represent major challenges in the facility, primarily due to substantial surface erosion of the exposed tailings during extraction. A combination of windrowing and ditching was adopted within the facility to minimize erosion and transportation of suspended solids to the discharge points. Construction of several check dams in the internal drainage ditches were recommended and are in place. As a result of efficient erosion control, concentrations of total suspended solids are below the Certificate of Approval monthly average discharge criteria for the decant tower, located at the upper settling pond, and at the spillway of the lower sedimentation control pond. At Site B however, no sedimentation control pond is present within the facility, although open water is periodically observed above grade in areas of low depressions where small ponds are seasonally formed due to accumulation of precipitation and runoff. All excess water from snowmelt and rainfall is drained from the TSF through a single spillway at the north end of the facility. Due to increased risk of sedimentation from to excavation activities, a settling pond dedicated to runoff from the TSF would be required within the TSF, which would further minimize the quantity of available tailings for pastefill.

Tailings composition

Tailings are not always suitable for backfill due to their composition. For instance, high concentrations of pyrite, pyrrhotite, and siderite can have a long-term deteriorating effect on the binder in cemented fills, compromising the stability of an underground mine. Thorough testing must be conducted prior to selecting a site as a backfill source. Considering the nature of ore deposits processed at Site C, the test work demonstrated an elevated presence of pyrite and pyrrhotite. On the other hand, gold tailings from Sites A and B were relatively neutral, containing similar and substantially lower concentrations of sulphides. One area at Site A contained boron, known for retarding the setting of the binder, which is critical information to consider for mine planners, particularly with respect to the implications to barricade loading during pours.

Tailings geotechnical stability

Excavation of historic tailings also creates stability risks within the facility. Excavations may lead to slumping, settling, or liquefaction, imposing safety and handling challenges. Therefore, a thorough geotechnical investigation is required before developing an excavation plan. At Site A, the stability of a section of the perimeter dam had to be evaluated in greater detail and excavation of tailings from the dam and adjacent to the dam will require a site-specific excavation and monitoring plan. To facilitate safe tailings extraction, several vibrating wire piezometers have been installed and will be monitored throughout the planned excavation process.

Environmental impact

Excavation of historic tailings can expose potentially harmful substances that may be released to the environment through run-off water, such as heavy metals, acid, or other contaminants. Proper contaminant management measures and monitoring plans must be in place to minimize the impact on the surrounding ecosystem. For instance, at Sites A and B, geochemical characterization of tailings showed elevated concentrations of arsenic, boron, cobalt, iron, and nickel that potentially could be released into the environment during excavation. At Site C, the excavation of historic tailings could trigger an accelerated rate of acid generation, a greater concentration of suspended heavy metals and selenium, and water treatment system overloading.

Tailings dewatering

Historic tailings may be saturated with water, which can complicate or make excavation impractical. An effective water management plan to lower the overall phreatic surface of the facility must be developed. Tailings dewatering was the major challenge for Sites A and B. Considering the extra milling required for the processing of gold ore, both sites have fine, water-retaining tailings, and high phreatic surfaces. Conceptual excavation plans included a system of water collection ponds and drainage channels to lower phreatic surfaces within the facilities. Furthermore, the transportation plan included extra time for stockpiled tailings to drain before loading onto trucks to prevent material from sticking.

Regulatory compliance

Regulatory approvals are often lengthy and stringent processes. The excavation plan must be thoroughly reviewed by the engineer of record, the environmental team, site management, and other relevant stakeholders before submitting a request to regulatory bodies for a permit. In some cases, public consultation is required. The primary aspects of an excavation plan that are scrutinized by the regulatory bodies are geotechnical stability, geochemical stability, and the impact on the surrounding ecosystems. Even without significant challenges, acquiring a permit may take several years. Thus, proactive planning, testing, and design of the excavation program is necessary. For Site A, conceptual design amendments to the existing excavation plans and geotechnical/geochemical assessments were reviewed by all relevant stakeholders, and submitted to initiate the pre-consultation stage of the approval process. The estimated duration of the review and approval was three years. Considering Site B is a closed tailings facility, the expected time for regulatory approval was estimated to be significantly longer.

CAPEX and OPEX

Excavation of historic tailings for backfilling is often CAPEX and OPEX intensive, considering challenging site conditions, safety measures, transportation, and environmental compliance (including provisions for closure). Being an active tailings deposition area and the furthest located site from the mine, Site C had the highest CAPEX and OPEX. The first cost driver was the complexity of the excavation. Site C required constant saturation of the tailings surface to minimize acid generation. Thus, the excavation plan required a staged approach and additional infrastructure such as roads, screening, and storage and loading pads, which are difficult to implement on an active tailings site. Second, the greater transportation distance and the resulting larger number of trucks needed to meet the backfilling requirements drove up the costs.

Site B would require investment to establish an appropriate excavation and water management plan, considering that the tailings are saturated, and the fine PSD retains pore water. This site is also located further than Site A from the mine, increasing OPEX related to trucking and introducing more truck traffic within the City of Timmins. Further to the above, Kidd Mine would need to take ownership of the site closure, and approx C$15M allowance was included for this in sustaining CAPEX. Site A remained the lowest cost solution when considering both CAPEX and OPEX.

Conclusion

The three sites were critically assessed in the interest of identifying which tailings source was best suited for use in the feasibility studies to meet the demand for Mine 5 pastefill. The assessment considered inputs from the test work, geotechnical reports, volume assessments for Sites A and B TSFs, conceptual tailings excavation plans for all tailings sites, and a tailings transportation analysis. The tailings from each of the three sites generally met the required characteristics for pastefill, demonstrating acceptable flowability and strengths in both short- and long-term scenarios. A comparison of the opportunities and risks associated with each of the three sites is included in Table 1.

Table 1. Opportunities and risks of tailings sites.

Site	Opportunities	Risks
Site A	▪ Adequate tailings quantities for LOM. ▪ Material properties suitable. ▪ Current tailings source (lower risk) ▪ Met UCS requirements. ▪ Favourable location (lower OPEX). ▪ No community impacts.	▪ Continued excavation on site requires careful planning. ▪ The tailings will be high in moisture content requiring time to drain pore water. ▪ Presence of boron in pore water may cause delays in strength gain. ▪ Environmental compliance approval (ECA) amendment required for continuous on-site water management.
Site B	▪ Material properties suitable. ▪ Met UCS requirements.	▪ Inadequate tailings quantity for LOM ▪ Cost prohibitive as it introduces new closure costs/liabilities. ▪ Saturated tailings expect challenging excavation and additional time to drain porewater. ▪ Lengthier transportation distance. ▪ Moderate community impact from trucking. ▪ New contractual agreements and permits required for tailings excavation.

Site C	▪ Adequate tailings quantities for LOM. ▪ Met UCS requirements. ▪ Rehabilitation costs less than opening new site due to shared footprint with existing TMA (restoration already planned). ▪ Glencore owns the site; streamlined permitting process to excavate.	▪ Material properties suitable; however, risk of mineralogy (pyrite, siderite) causing strength issues or increased binder demand. ▪ Transportation distances considerably longer (higher OPEX). ▪ Higher CAPEX for site excavation and complex logistics for excavation (active TMA). ▪ Concern of inadequate coverage on acid generating tailings, leading to higher run-off water treatment costs. ▪ Most significant community impact due to trucking through Porcupine and Timmins.

Based on these findings, it was clear to all stakeholders that Site A was the most appropriate source of tailings to continue forward with for the Mine 5 project feasibility study. The site offered adequate tailings reserves, achieved required UCS, mitigated risk from the current source of tailings, resided closest to the mine (lower OPEX), eliminated the need for new contractual agreements, and had limited community and environmental impact. There were no critical risks identified; however, the excavation would require careful planning and implementation such that the required quantities of tailings are extracted safely from the chosen site.

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