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Heap Leach Mining Solutions 2022**

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The conference has been organized in partnership with the Mackay School of Engineering and Earth Sciences, which has educated generations of renowned scientists and engineers and has a long and proud history at the University of Nevada, Reno.

Finally, we would like to thank all the delegates who attended the conference to exchange their valuable knowledge and expertise, thus contributing to the great success of the Fifth International Heap Leach Solutions Conference. We look forward to seeing you all again at the next conference in this series.

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Heap Leach Solutions 2022

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Chapter 1:

Project Development



Project Planning and Verification Using METSIM® Simulation Software at Brazilian Nickel's Piauí Nickel Project

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Abstract

Heap leach projects require extensive planning and data verification, from the preliminary scoping study through commissioning, including information management and proper testwork campaigns. This paper follows the progression of the Piauí Nickel Project (PNP) (*Piauí Niquel Metais*) located in Brazil, a nickel laterite heap leach that utilized METSIM® for these purposes. When Brazilian Nickel acquired the project, initial bottle roll tests for Ni, Co, Mg, Fe, Mn, and several other metals had been completed and showed high recoveries of both Ni and Co. The project then refurbished an existing pilot plant to create a large-scale demonstration plant. Demo plant-scale testwork was concluded on full-height heaps where 8,000 metric tonnes of ore was leached and the pregnant leach solutions were processed in a downstream plant and nickel and cobalt products were produced and sold.

To assist in designing a complex, counter-current leach cycle, the proposed heap geometry and available leach kinetics, along with the as-built demonstration plant, were used to build a dynamic METSIM® model aimed at consolidating and verifying all information. The model was built before the heaps and maintained throughout the test campaign, calibrated as the heap performances were measured over the life of the tests. Specific areas of interest included algorithms to simulate metal extractions for Ni, Co, Fe, Mg, and Mn and for overall acid consumption. Moreover, given the capabilities within METSIM® to model full operations in closed loop, from mine to product, the downstream processing plant was also included in the model, which proved crucial for investigating water and reagent balances. Impurity concentrations in solution streams were tracked and used to design critical processing facilities, such as the magnesium bleed and precipitation. Immediately following, the METSIM® model was then scaled to commercial production, including an annual mine plan and corresponding heap and downstream equipment sizing.

Introduction

Heap leach projects require extensive planning and data verification, from the preliminary scoping study through commissioning, including information management and proper testwork campaigns. The Piauí Nickel Project (PNP) located in Brazil, a nickel laterite heap leach, utilized METSIM® for these purposes, as well as to scale demonstration heap performance to commercial production.

Project planning and development

When Brazilian Nickel acquired the project in 2012/2013, initial modelling efforts were focused on the conceptual flowsheet (see Figure 1) from ROM ore receiving through nickel production, including heap and downstream precipitation plant operations and dynamic mass balances of solid and aqueous phases.

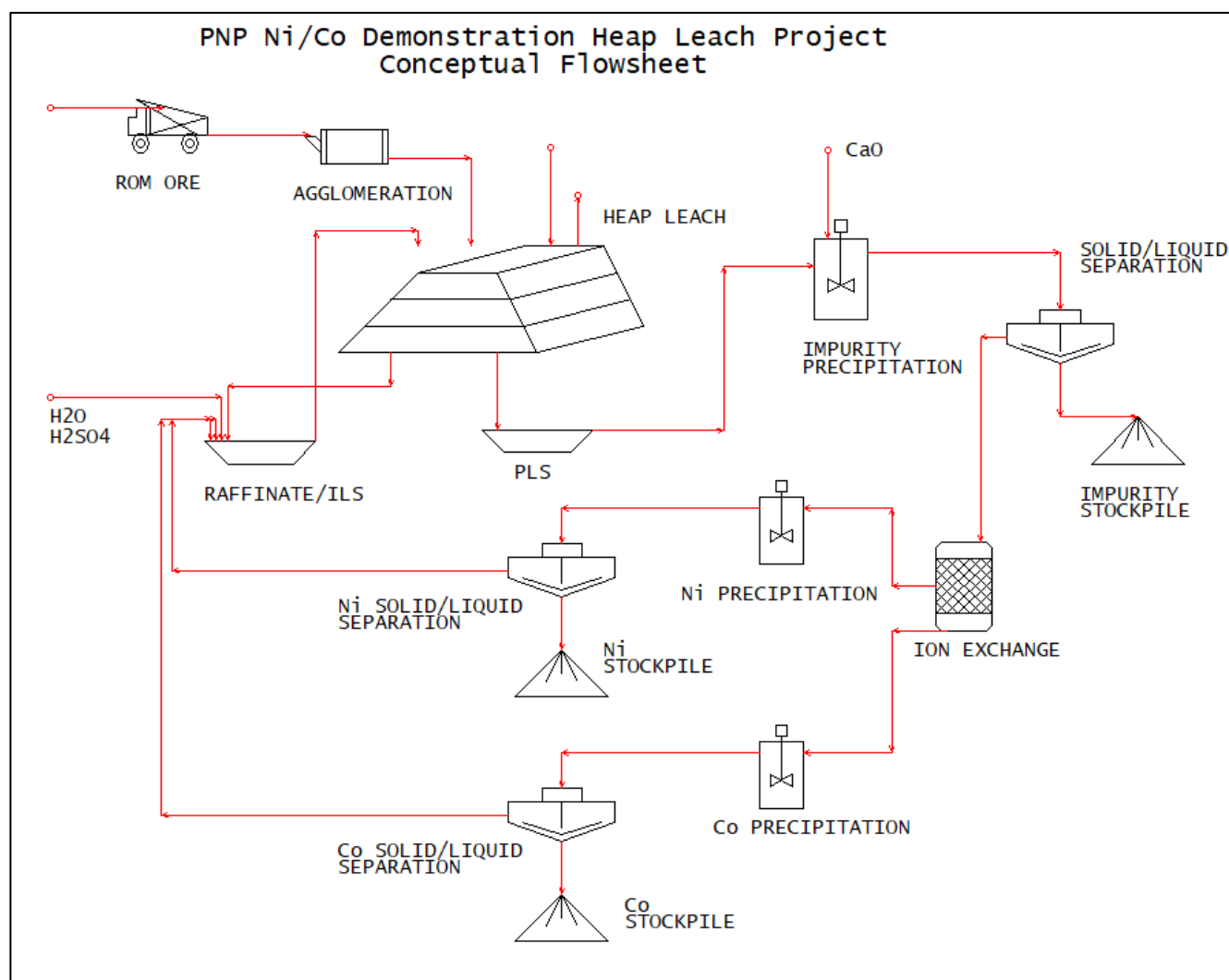


Figure 1: PNP demonstration heap project conceptual flowsheet

In parallel, initial bottle tests were conducted to measure extraction rates of Ni, Co, Mg, Fe, Mn, and several other metals. Tests showed high recoveries of both Ni and Co; these reaction kinetic coefficients would then serve as the basis for metal extraction rates in the heap. To assist in designing a complex, counter-current leach cycle, the proposed heap geometry and available leach kinetics, along with the as-built demonstration plant, were used to build a dynamic METSIM® model aimed at consolidating and verifying all information. This model would serve as the engineering plan for further project testwork.

The PNP team then refurbished an existing pilot plant to create the large-scale demonstration plant, including an 8,000 metric ton heap leach and all downstream processing facilities shown in Figure 1. Not depicted in the flowsheet is the solution neutralization process for the precipitation and removal of Mg from the raffinate and process water streams.

The demo plant was operated for approximately one year (365 days), during which time the METSIM® model was updated and calibrated with the measured results. Specific areas of interest included algorithms to simulate metal extractions for Ni, Co, Fe, Mg, and Mn and for overall acid consumption. Moreover, given the capabilities within METSIM® to model full operations in closed loop, from mine to product, the downstream processing plant was also included in the model, which proved crucial for investigating water and reagent balances. Impurity concentrations in solution streams were tracked and used to design critical processing facilities, such as the magnesium bleed and precipitation.

The demo heap was built and leached as three separate cells or modules, named Heap 0 (H0), Heap 1 (H1), and Heap 2 (H2). H0 was built and leached before H2 and H3 and was designated the “calibration heap” by the modelling team; i.e., all algorithms for metal extractions were calibrated against the measured ILS/PLS solution tenors from H0. Relationships between the metal head grades in the ROM ore and free acid available within the raffinate and heap inventory were developed and applied to the leach kinetics within the heap. The model iteration was one day, and each iteration evaluated these relationships and applied factors to each chemical reaction kinetic coefficient to adjust for the simulated conditions. Figure 2 shows the simulated PLS Ni tenor versus the actual PLS Ni tenor from H0 over the operating period of approximately 300 days.

Following this calibration stage for H0, the same algorithms were applied to H1 and H2 and the simulated PLS Ni tenors were compared to the measured results (Figures 3a and 3b). As demonstrated in these figures, the simulation results aligned extremely well, proving the validity of the developed algorithms and their ability to be used to calculate the metal leach kinetics.

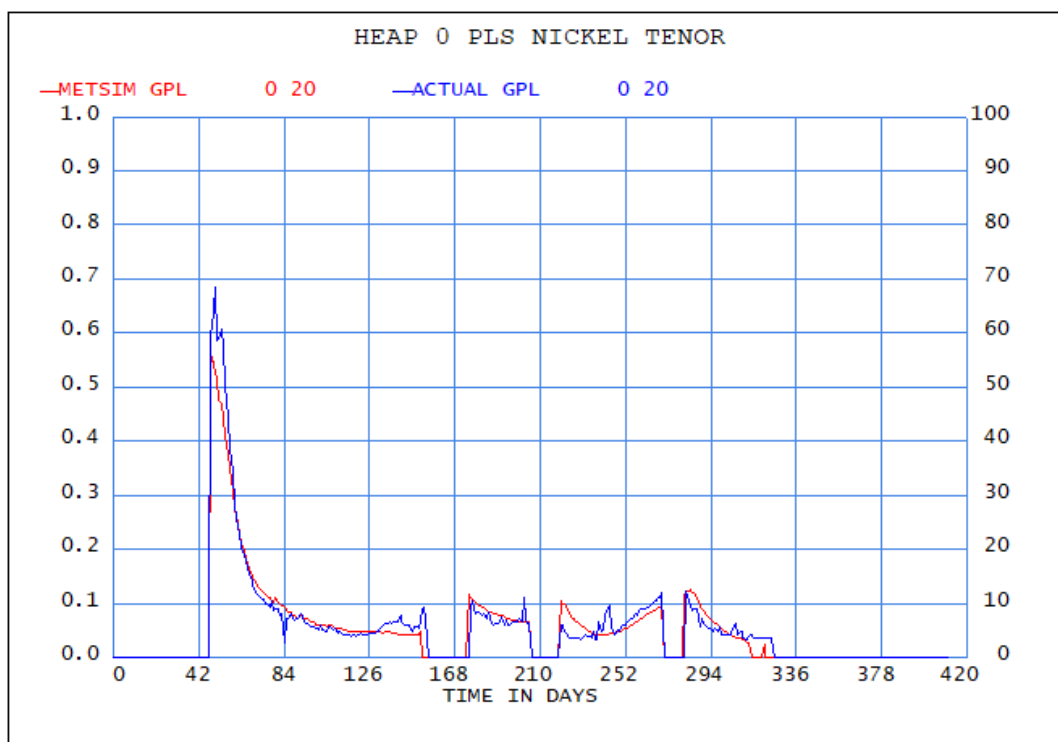


Figure 2: Simulated (red) vs. actual (blue) H0 Ni PLS solution tenor

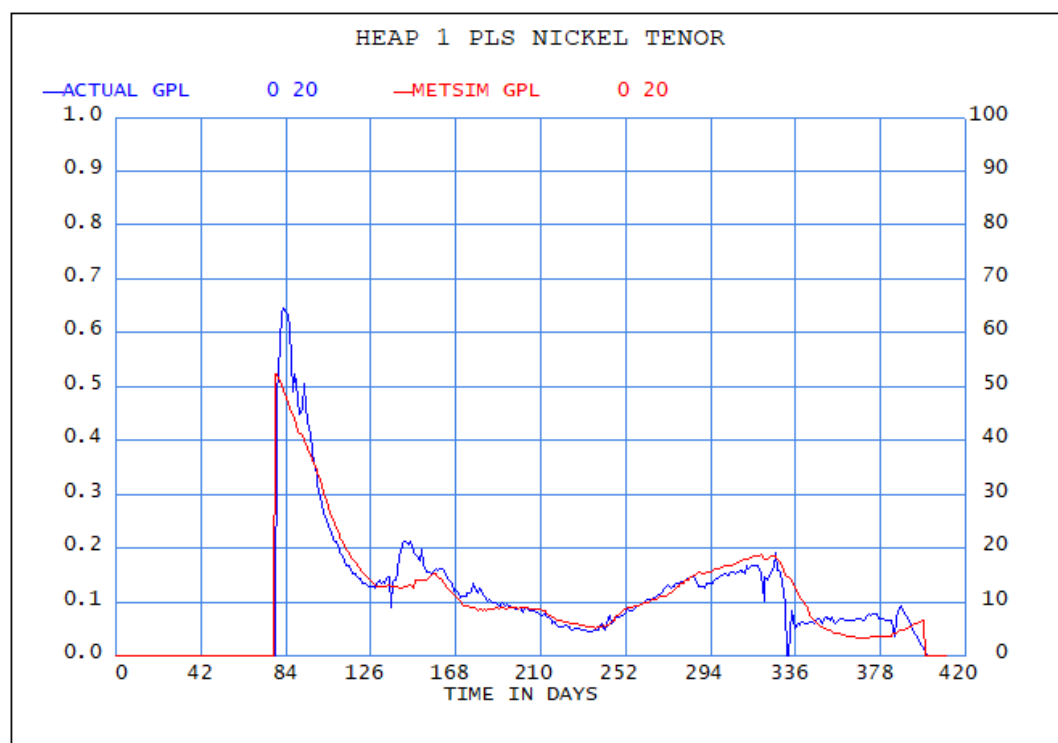


Figure 3a: Simulated (red) vs. actual (blue) H1 Ni PLS solution tenor

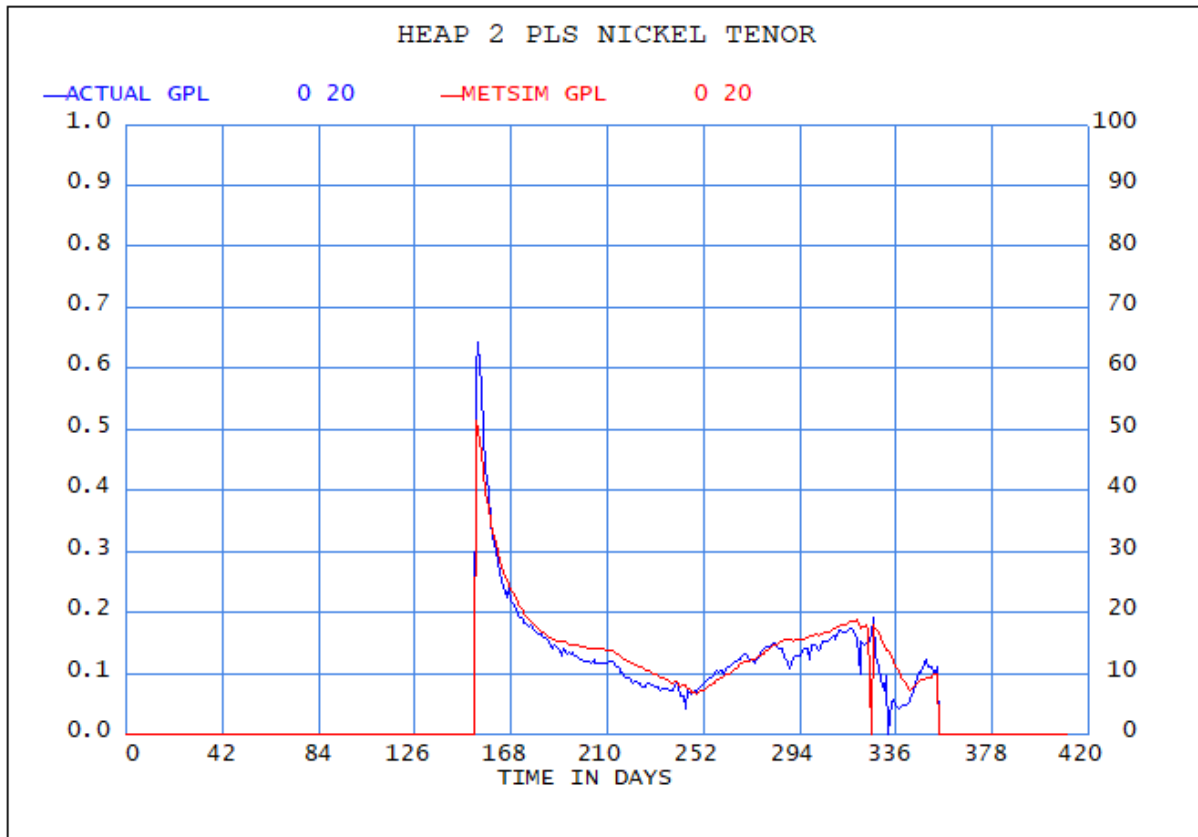


Figure 3b: Simulated (red) vs. actual (blue) H2 Ni PLS solution tenor

Immediately following, the METSIM® model was then scaled to commercial production, including an annual mine plan and corresponding heap and downstream equipment sizing. Static and dynamic mass balances for the commercial operation were developed and presented to local engineering companies, saving significant time and cost for the detailed engineering and design stages.

Moreover, with reliable production forecasts resulting from the scaled model, ore tonnes and grade plan variations could be run through the model to optimize not only the heap leach but also the mine operations.

Simulation verification

Since this abstract was originally submitted, the PNM heap leach project has begun commercial production of Ni and is closely tracking extraction rates within the heap's first leach module. Figure 4 compares the model-predicted Ni extraction rate versus the first 45 days of operation of the first leach module on the commercial heap leach.

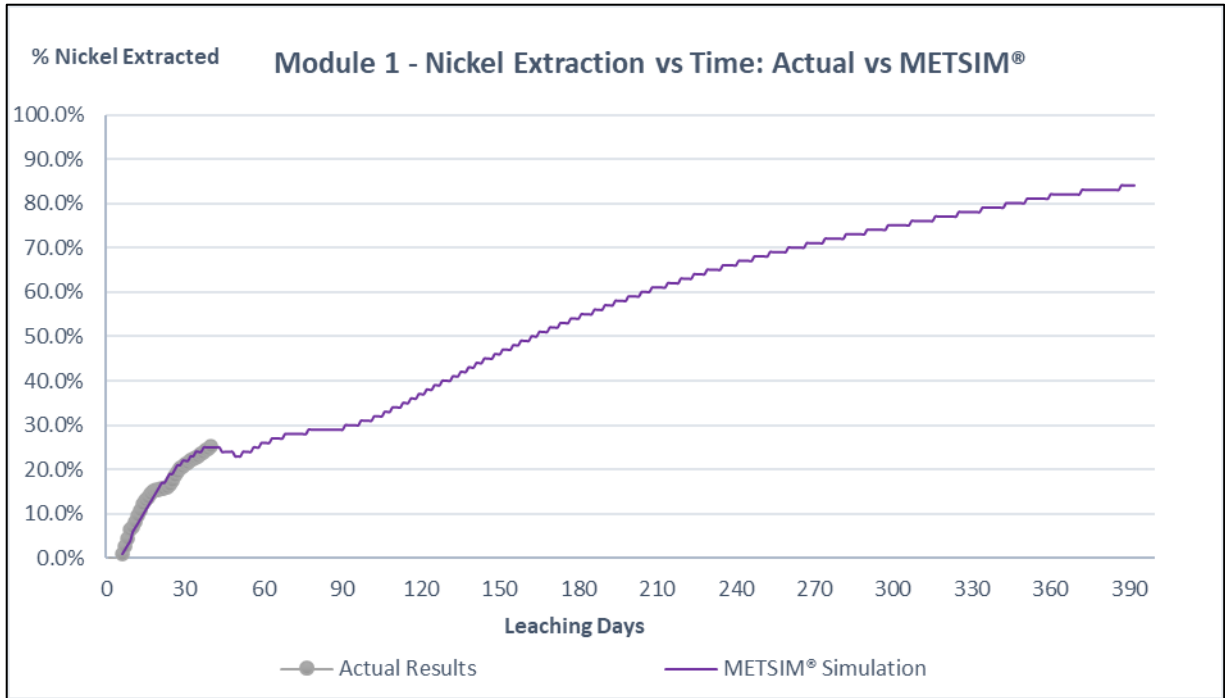


Figure 4: Simulated (purple) vs. actual (grey) module 1 Ni extraction rate

Conclusion

Though still in the early days of commercial production, initial operational results provide validation for the team's modeling efforts and the power of a well-calibrated simulation. In the future, additional heap leach project evaluations on similar ore bodies will become simpler, faster, and cheaper with the use of these METSIM® simulations.

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Metallurgical Considerations for Modeling the Recovery of Gold from Heap Leach Facilities

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Abstract

Significant efforts have been made in recent heap leach models to improve the accuracy and consistency of gold production forecasts. It starts with breaking the heap leach facility into manageable cells that can be customized based on the operators' level of resolution. The discretized model tracks numerous metallurgically relevant inputs such as ore type, tons and grade, particle size distribution, pad stacking and geometry, leach kinetics, and solution hydrodynamics. Frequently, these criteria are not well defined, or operations are unable to project changes to production based on changing site conditions. It is critical that the metallurgical testing programs are defined well enough upfront to provide a broader array of information, so that an operation is able to adjust based on their actual mining and process conditions. It is often the case that one or many of these steps are skipped, or assumed to be known based on prior experience, because the risk is assumed to be manageable and to keep budgets or timing deadlines intact.

A solid metallurgical test program for a heap leach operation begins with a comprehensive analytical campaign in the exploration stages to identify potential red flags such as the presence of clays, organic carbon, sulfide sulfur, or deleterious metals that may impact operational or closure designs. Small-scale, high throughput cyanide amenability shake leach programs are also important in understanding extractions and identifying problematic areas within a particular geology or ore body. These databases are common to see in one form or another from exploration teams and need to be evaluated on an iterative basis as the database is transferred between operational teams. A thorough database, which can start on RC drill samples, should move towards metallurgical core and include the following tests on samples identified above the assumed cut-off grade:

- geological logging (three-dimensional location, lithology, alteration);
- fire assay – gold and silver;

- cyanide amenability shake leach;
- total carbon and speciation;
- total sulfur and speciation; and
- mineralogy (quantitative XRD/XRF).

It is not necessary to complete these steps initially, but they should be completed as exploration continues and as metallurgical work commences.

After a large enough database has been built, a compositing campaign can then be undertaken to evaluate leach kinetics and potential gold extractions at multiple sizes under bench- and pilot-scale conditions. This should be an exercise involving both geological and metallurgical components to identify key drivers. Leach testing, such as the bottle roll or column leach, provides a basis for the kinetic response under various conditions. However, particularly for larger sizes, diffusion leach testing should be conducted to better understand the tortuosity requirements of a geological alteration as opposed to a pit or mine plan blend.

Throughout this process it is critical to understand the diffusion and deportment of gold. This stage can often be overlooked when leach extractions are favourably projected. In the event of a poorly performing sample, it becomes more difficult to pinpoint the reason for the performance. Often it is assumed to be due to many of the common gold leach pitfalls, such as silica encapsulation, preg-rob, or refractory ore; however, it is a small cost to understand and mitigate the overall risk.

After chemical properties are better understood, ore must then undergo degradation and compaction testing to project initial and future changes to geotechnical and hydrodynamic stability. These conditions are dependent on the overall operating size. Eventually a blasting and fragmentation study should be conducted for potential run-of-mine ores, or to determine a crushing schema for ore that has demonstrated a crush benefit to identify these potential operating size distributions.

Overall, the combination of the chemical and physical properties allows for an optimized operating size distribution based on the heap leach modeling and NPV analysis. It is also feasible, with a large enough database, to provide site-dependent proxy testing that can simplify on-site metallurgical laboratory testing and dictate operational control logic.

The accuracy of these inputs into the heap leach model starts with the metallurgical testing program, which should be evaluated on an iterative basis. The “industry-standard” testing that is utilized must be constantly questioned and tailored towards each site because conditions change, and testing may need to be modified accordingly. The results then directly correlate to the ability to forecast an accurate time-dependent production of gold.

Forte Dynamics (Dynamics) is a consulting and engineering firm based out of Northern Colorado, with a focus on open pit resource and reserve modeling, mine planning, blasting analysis, metallurgy,

process design, recovery modeling, detailed engineering, and EPCM services. In conjunction with Forte Analytical (Analytical), a mineral testing laboratory focusing on open pit mine and heap leach facility advancement, the two groups can elaborate on several case studies in which they have provided a broad scope review of greenfield exploration data or brownfield process debottlenecking needs to optimize heap leach operations. This in turn then provides operations with a site-specific and concise estimate of heap leach facility extraction rates, in-heap inventories, and production rates.

Case study 1

In the first case study, Dynamics and Analytical evaluated a clay issue within a deposit to provide the operator with operational conditions around solution percolation. The site had identified a region of high clay content, estimated at up to 35% passing 200M (74 μ m) material within the mining schedule, and had an agglomeration circuit in their process design. Forte assisted in developing a test program that consisted of compacted permeability testing of various blends of pits from the site.

After a review of the ore characteristics and permeability testing, it was determined that solution percolation could be maintained through a modified application rate and blending program to remove the agglomeration circuit from the design. Changes to the solution dynamics and overall process were easily incorporated into the Forte heap leach model.

Case study 2

The second case study comes from an internationally based brownfield site that did not fully understand the preg-rob nature of the ore they were stacking. Grade-control had been based on a visual identification of organic carbon, which was best described as “black is bad, everything else is good.” This has been a frequently noted issue with operations, and unfortunately is not unique to the case study in question.

However, not all organic carbons can be visually identified and not all organic carbons are made the same. It is common to see 0.1 to 0.2% organic carbon in samples, that do not present visually, which then can show extreme levels of “preg-rob”. As Dynamics and Analytical have experienced, the “industry-standard” preg-rob test is run differently by almost every laboratory and does not provide any interpretation on what, if any, levels of preg rob can be accepted. Preg rob tends to be less of an issue at coarser crush sizes; however, it must still be accounted for in the overall model.

Depending on the testing conducted, or the results produced, it is common to see a cut-off value of 10 to 20% preg rob for what should be considered ore, but these values do not truly identify the overall potential for gold loss and must be evaluated further. For this operator, Analytical was able to elaborate on the preg-rob extent of the material on-site, relative to the organic carbon content, and determine an overall grade of both organic carbon and total carbon that was operationally acceptable.

Case study 3

The final case study considered the progression of a detailed and thorough greenfield metallurgical program that was rescope at numerous steps along the way based on a collaboration between geological and metallurgical leads, and collaborations between Analytical and the site project team. The Gold Rock Project (GRP), outside of Eureka, Nevada was previously developed by Fiore Gold and recently purchased by the Calibre Mining Corporation (Calibre). The GRP 2020 metallurgical drilling program consisted of 15 PQ core holes, representing approximately 1,300 intervals. Each interval was rigorously logged for geological alteration and lithology, according to a four-digit code rock code, specific gravity, fire assay (in triplicate), cyanide amenability shake (in triplicate), and carbon and sulfur speciation. Ore grade samples were then sent for additional mineralogical XRD analysis.

After a statistical review of the interval-based data, Dynamics, Analytical, and the geological team from Calibre re-evaluated the logging to build a small interval composite program for bottle roll leach testing. Results from this stage then fed to a larger composite bottle roll leach, column leach and diffusion leach testing program. Specific refractory materials were identified and underwent additional evaluation including gold deportment to understand the refractory nature. While the result was a metallurgical extraction estimate, by four-digit rock code, which could then be incorporated into the Forté heap leach model, the test program also highlighted the need to modify particular testing procedures to accurately identify extraction rates. The database Analytical generated from the PQ core program was then expanded to the rest of the drilling database, and will provide the site with initial operating based on site-based grade control testing.

The end goal

Whether it is to optimize or improve upon current operations or to develop the necessary testing towards feasibility study, the end goal should always be a better understanding of a deposit to best model the production and make the appropriate NPV determinations. This logic is typically well used when determining process options of higher-grade milled deposits; however, heap leach operations are not often as well designed. Operations typically proceed based on assumptions and limited data, only to find that the assumptions do not hold up under operating circumstances. It is not inappropriate at any stage to step back and re-evaluate a process design based on new information to determine all of the production drivers. There is no definitive checklist that will work for every operation, as deposits vary widely, but the modeling provided by Dynamics and Analytical is focused on providing better answers to the operators before running in to situations such as case studies 1 and 2, among many others.

Bentonite-Polymer Composite Geosynthetic Clay Liners for Heap Leach Liners

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Abstract

Hydraulic conductivity of four commercially available bentonite-polymer composite (BPC) geosynthetic clay liners (GCLs) and one conventional sodium bentonite (NaB) GCL was evaluated using a synthetic copper heap leach solution. The BPCs were comprised of granular NaB dry-blended with granular polymer. Each BPC contained a unique proprietary polymer. The NaB GCL was comprised of the same bentonite as the BPCs. Tests were conducted under two different stress scenarios. In the low stress scenario, testing began at low initial confining stress (20 kPa) simulating a thin layer of ore on the heap leach pad, followed by a ramp up to 600 kPa. The high stress scenario began with a higher initial confining stress (200 kPa) simulating a 10-m-thick layer of ore rapidly placed on the pad, followed by the same ramp up to 600 kPa.

Permeation with the copper heap leach solution adversely affected the NaB GCL, with the hydraulic conductivity exceeding 10^{-9} m/s for all but one case (200 kPa stress under high initial stress condition). The BPC GCLs had much lower hydraulic conductivities ($< 9.1 \times 10^{-11}$ m/s), indicating that BPC GCLs can be more effective as liners for copper heap leach solutions than NaB GCLs. Hydraulic conductivity of the BPC GCLs varied with type of polymer and decreased modestly with increasing effective stress. The BPC GCLs permeated with higher initial stress had lower hydraulic conductivity compared to those initially permeated at low initial stress, indicating that high initial stress affords protection for BPC GCLs. Similar protection was not realized for the NaB GCL. Swell index (SI) and active water content (AWC) were evaluated as indices of hydraulic conductivity of the BPC GCLs. Neither index was an effective indicator of hydraulic conductivity.

Introduction

Geosynthetic clay liners (GCL) are used in lining systems for heap leach facilities to collect leach solution draining from the ore. Conventional GCLs are comprised of a thin layer of sodium (Na) bentonite granules

sandwiched between two geotextiles that are bonded together by needle punching or stitching. To achieve low hydraulic conductivity, the bentonite granules must swell sufficiently during hydration to fill the intergranular pores (Figure 1a) and seal off the needle punching fiber bundles. If swelling is insufficient, larger pores remain and a GCL will have high hydraulic conductivity (Figure 1b). The hydraulic conductivity of GCLs with Na bentonite can range six orders of magnitude or more depending on how much the bentonite granules swell (Jo et al., 2001; Kolstad et al., 2004; Tian et al., 2016; Chen et al., 2018).

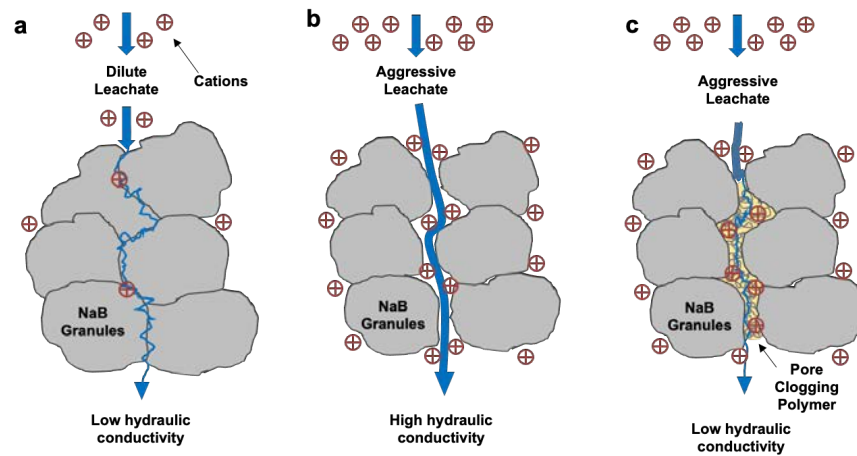


Figure 1: Swelling of Na bentonite granules in conventional GCL closes intergranular pores yield low hydraulic conductivity (a), suppressed swell of granules leaves by aggressive leachate intergranular pores open yielding high hydraulic conductivity (b), and polymer gel in BPC clogs intergranular pores yielding low hydraulic conductivity (c)

The swelling of bentonite is highly sensitive to the geochemistry of the solution to be contained. Solutions with higher ionic strength, a preponderance of polyvalent cations (charge of +2 or higher), or extreme pH, often referred to as “aggressive solutions,” tend to suppress bentonite swelling, resulting in a GCL with hydraulic conductivity that may be too high for the barrier application of interest (Jo et al., 2001; Kolstad et al., 2004; Scalia and Benson, 2016). In such cases, a GCL containing a bentonite-polymer composite (BPC) can be suitable (Scalia and Benson, 2016; Tian et al., 2019; Chen et al., 2019; Norris et al., 2022; Tan et al., 2022).

BPC GCLs contain a mixture of bentonite and polymer granules. When the BPC hydrates, the polymer granules form a gel that clogs open intergranular pores in the hydrated bentonite, resulting in narrow and tortuous flow paths and low hydraulic conductivity (Figure 1c). The low hydraulic conductivity persists provided the hydrated polymer remains in the intergranular pores and retains a gel structure. However, the rheology of polymer gels is sensitive to the geochemistry of the hydrating solution, and some gels are eluted from the intergranular pore space if their surface tension is altered, or the gel viscosity diminishes. If elution is significant, the hydraulic conductivity of a BPC GCL can increase significantly (Scalia and Benson, 2016; Tian et al., 2019; Norris et al., 2022).

In this study, four commercially available BPCs and one conventional Na bentonite (NaB) GCL were hydrated and permeated with a synthetic copper heap leach solution that had low pH and contained predominantly divalent cations. The NaB GCL and the BPCs were provided by Solmax Inc. The BPCs were comprised of granular Na bentonite dry-blended with granular polymer. Each BPC had a unique polymer. The NaB GCL was comprised of the same bentonite used in the BPCs. The polymers used in the BPC and the polymer loading are proprietary to the GCL manufacturer and were not divulged to the investigators. Tests were conducted under different loading conditions representing different overburden pressures in the heap leach facility. Findings from these tests are described in this paper.

Materials

Geosynthetic clay liners

Mock BPC GCLs were prepared by placing 5 kg/m² of BPC comprised of dry blended NaB and granular polymer between a nonwoven geotextile and a woven-nonwoven composite geotextile. All materials were provided by the manufacturer. The composite geotextile consists of a woven silt film geotextile bonded to a nonwoven geotextile, with the silt film geotextile internal to the GCL. The NaB GCL was provided by the same manufacturer and consisted of 4.8 kg/m² of granular bentonite sandwiched between the aforementioned geotextiles, which were bonded by needle punching.

Synthetic copper heap leach solution

A typical copper heap leach solution was identified using data on the geochemistry of five copper heap leach solutions reported by Ghazizadeh et al. (2018). The geochemistry of these solutions was described in terms of the two master variables affecting the hydraulic conductivity of NaB GCLs, ionic strength (I):

$$I = \sum_{i=1}^n C_i z_i^2 \quad (\text{Equation 1})$$

and RMD:

$$\text{RMD} = \frac{M_M}{\sqrt{M_D}} \quad (\text{Equation 2})$$

Where:

C_i is the concentration of the i^{th} ionic species in the solution,

z_i is the valence of the i^{th} species,

M_M is the total molarity of monovalent cations in solution, and

M_D is the total molarity of polyvalent cations in solution.

A recipe was created to represent the median concentration of the copper heap leach solutions reported in Ghazizadeh et al. (2018), with $I = 0.689$ M and $RMD = 0.020$ M^{0.5} (Table 1). The recipe was checked for charge balance and over-saturation using Visual MINTEQ. The pH was adjusted to 2.2 by titration with sulfuric acid. The final solution had an electrical conductivity (EC) of 0.2 S/m and an anion ratio (A_r = molar ratio of chloride to sulfate) of 0.59.

Table 1: Recipe for synthetic copper heap leach solution

Constituent	Concentration (g/L)
$Al_2(SO_4)_3 \cdot 18H_2O$	9.62
$CaCl_2$	1.46
$CuCl_2$	0.18
KCl	0.13
$MgCl_2 \cdot 6H_2O$	5.05
$MgSO_4$	8.77
$MnSO_4 \cdot H_2O$	1.16

Methods

Swell index and active water content

Swell index (SI) and active water content (AWC) tests were conducted on the NaB (SI only) and the BPC (SI and AWC) using the copper heap leach solution for hydration. Swell index tests were conducted in accordance with ASTM D5890 and the procedures in Tan et al. (2022). The NaB and BPCs were ground in a motorized grinder (Retsch RM 200, Haan, Germany) until all particles passed the No. 100 sieve and at least 65% passed the No. 200 sieve. Two grams of the ground and dry NaB or BPC were gradually added to a graduated cylinder containing 90 mL of copper heap leach solution. The cylinder was filled to 100 mL after the bentonite was added. The swell volume at 24 hours was recorded as the swell index.

Active water content (AWC) tests were conducted following the methods in Geng et al. (2022). BPCs were placed in 50-mL centrifuge tubes along with the copper heap leach solution at a 1:10 solid-to-liquid ratio. The tubes were tumbled end-over-end for 48 hours at 30 RPM, and then centrifuged at 10,000 RPM for 20 minutes in a Beckman Allegra 25R centrifuge (Beckman Coulter, Palo Alto, California) with a centripetal acceleration of 25,000 g. The solution separated from the BPC mixture during centrifugation was spiked with Rhodamine dye to identify the interface between the solution and the hydrated polymer gel and strongly bonded bentonite. Water content of the hydrated polymer gel and strongly bonded bentonite was measured at 105°C using ASTM D2216, and reported as active water content.

Hydraulic conductivity

The GCLs were hydrated and permeated with the synthetic heap leach solution in flexible-wall permeameters in general accordance with the procedures described in ASTM D6766. The test specimens had a diameter of 150-mm and were placed in the permeameters between two heavy nonwoven geotextiles used to spread flow uniformly across the surface of the specimen. NaB GCL specimens were trimmed from a GCL roll using a sharp razor knife, with deionized (DI) water applied around the edge to prevent loss of bentonite during trimming. Mock GCLs were prepared from the BPCs and geotextiles, as described previously. All GCL specimens were oriented in the permeameter with the nonwoven geotextile on the upstream side and a composite geotextile on the downstream side of the GCL. The woven slit-film geotextile was on the interior face of the downstream side of the GCL. To prevent sidewall leakage, a layer of paste was smeared around the periphery of the test specimen, and a thin wedge of paste was placed along the periphery of the downstream surface of the GCL before the specimen was sealed with the flexible membrane. The paste was prepared with NaB or BPC and DI water. O-rings were used to seal the membrane to the pedestals.

Cell pressure was applied using tap water in the permeameter plumbed to a pressure panel board employing compressed air and an air-over-water interface to provide the desired cell pressure. The influent was contained in a burette and the outflow was collected in a 50-mL polyethylene container. The influent burette and the effluent container were both vented in a manner that maintained atmospheric pressure while minimizing evaporation. No backpressure was used to preclude unintended geochemical alterations associated with elevated pressure (i.e., Le Chatelier principle). Flow was oriented in the downward direction to simulate the field scenario. The specimens were hydrated with the heap leach solution for 48 hours with the cell pressure and influent head applied and the effluent valve closed (no hydraulic gradient). Permeation commenced after 48 hours of hydration by slowly opening the effluent valve.

Tests were conducted under two different stress scenarios: low and high initial confining stress. In the low initial stress scenario, testing began at low initial confining stress (20 kPa) simulating a thin layer of ore on the heap leach pad, followed by a ramp up to 600 kPa in seven increments (20 kPa to 50, 100, 200, 300, 400, 500, and 600 kPa) representing increasing ore thickness. The high initial stress scenario began with a higher initial confining stress (200 kPa) simulating a 10-m-thick layer of ore rapidly placed on the pad, followed by the same ramp up to as much as 600 kPa.

The two scenarios were evaluated to determine if higher stress prior to contact with solution provides additional protection of the GCL from adverse chemical interactions. In all cases, tests were required to reach the hydraulic and chemical termination criteria in D6766 before the stress was incremented to the next level. Hydraulic equilibrium was defined as steady flow and inflow equal to outflow per the criteria in D6766. Chemical equilibrium was defined as EC of the effluent within 10% of the EC of the influent per

the EC criteria in D6766. Several of the tests are still being conducted. The most recent data at the time this paper was prepared are reported herein.

Results

A typical response of the NaB and BPC GCLs to permeation with the copper heap leach solution is shown in Figure 2 for low stress test (20 kPa) with the NaB and BPC GCLs. The hydraulic conductivity declines gradually as the bentonite swells (NaB GCL) and the polymer gel clogs pores (BPC GCL). An equilibrium hydraulic conductivity is reached in about 4 PVF for the NaB GCL and 2.0 PVF for the BPC GCL.

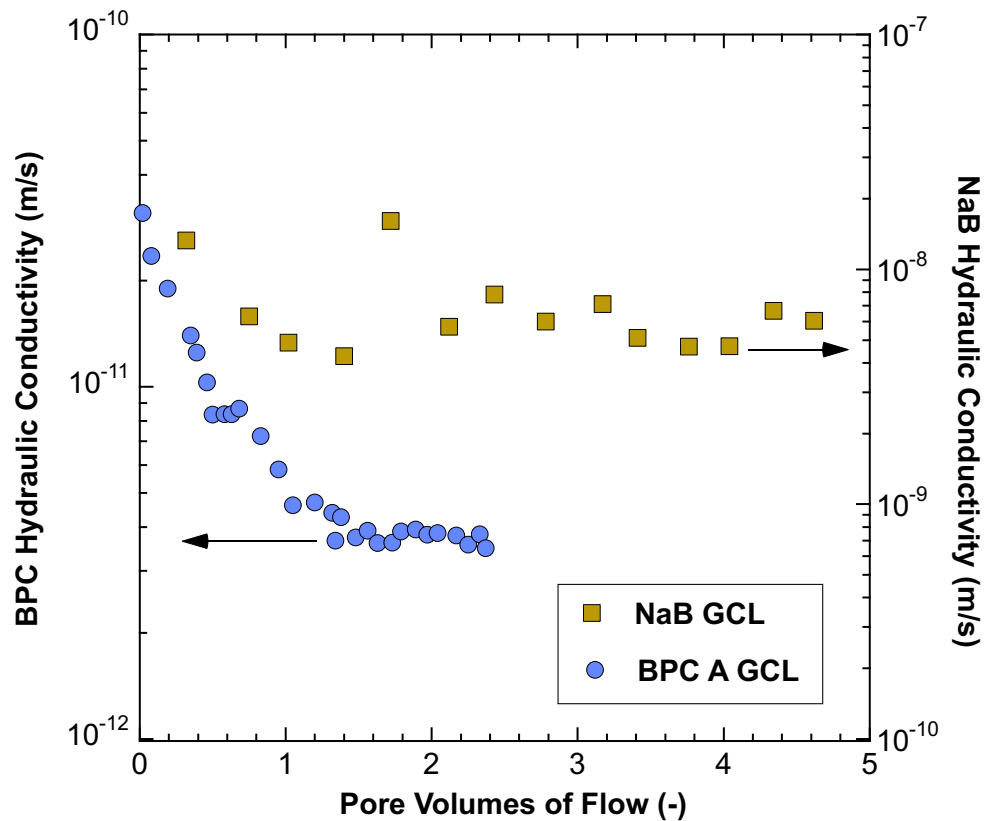


Figure 2: Typical response of NaB and BPC GCLs to permeation with copper heap leach solution. Tests conducted at 20 kPa effective stress with low initial stress

Hydraulic conductivities for the NaB and BPC GCLs are summarized in Table 2 and are shown as a function of effective stress in Figure 3. Hydraulic conductivities of the GCLs tested at low initial confining stress are shown in Figure 3 with solid symbols, whereas hydraulic conductivities of GCLs with high initial confining stress are shown with open symbols.

Table 2: Summary of hydraulic conductivity test conditions and outcomes

GCL	Initial stress (kPa)	Stress during permeation (kPa)	Pore vol. of flow	Test duration (d)	Hydraulic conductivity (m/s)
NaB	20	20	3.03	28.0	5.4×10^{-9}
		50	3.48	78.1	5.7×10^{-9}
		100	3.44	55.1	6.4×10^{-9}
		200	4.24	22.8	5.7×10^{-9}
		300	6.63	33.1	4.6×10^{-9}
		400	1.54	4.4	3.6×10^{-9}
		500	1.85	16.0	2.7×10^{-9}
		600	1.08	3.9	2.1×10^{-9}
BPC A	20	20	2.57	182.7	3.8×10^{-12}
		50	0.73	75.6	4.8×10^{-12}
		100	0.44	47.4	4.2×10^{-12}
		200	0.56	50.8	3.8×10^{-12}
		300	0.19	41.0	1.9×10^{-12}
		400	0.23	26.8	3.1×10^{-12}
		500	0.55	78.0	3.4×10^{-12}
		600	0.22	23.6	3.2×10^{-12}
BPC B	20	20	3.19	182.2	3.9×10^{-12}
		50	0.54	53.7	3.7×10^{-12}
		100	0.92	71.0	4.4×10^{-12}
		200	0.62	50.8	4.4×10^{-12}
		300	0.21	41.0	2.6×10^{-12}
		400	0.16	25.2	2.3×10^{-12}
		500	0.55	78.0	3.4×10^{-12}
		600	0.27	23.6	3.5×10^{-12}
BPC C	20	20	5.85	70.8	3.7×10^{-11}
		50	7.07	83.9	6.2×10^{-12}
		100	5.14	54.2	6.1×10^{-11}
		200	5.59	22.7	4.6×10^{-11}
		300	7.63	83.1	1.5×10^{-11}
		400	1.28	15.9	1.0×10^{-11}
		500	1.06	41.0	7.4×10^{-12}
		600	4.32	123.2	6.1×10^{-12}
BPC D	20	20	2.81	169.3	2.1×10^{-11}
		50	3.47	75.6	4.3×10^{-11}
		100	2.54	48.3	3.0×10^{-11}
		200	4.48	50.8	5.9×10^{-12}
		300	1.91	41.0	4.8×10^{-11}
		400	1.21	26.8	1.9×10^{-11}
		500	1.77	72.3	1.5×10^{-11}
		600	0.36	23.6	1.3×10^{-11}
NaB	200	200	4.41	171.2	6.7×10^{-10}
		300	6.42	68.1	2.0×10^{-9}
		400	0.79	4.4	1.8×10^{-9}
		500	2.24	17.0	3.2×10^{-9}
		600	1.41	3.9	2.5×10^{-9}

Table 2: Summary of hydraulic conductivity test conditions and outcomes (continued)

GCL	Initial stress (kPa)	Stress during permeation (kPa)	Pore vol. of flow	Test duration (d)	Hydraulic conductivity (m/s)
BPC A	200	200	0.71	240.4	8.1×10^{-13}
		300	0.19	141.0	7.3×10^{-13}
		400	0.07	26.8	8.3×10^{-13}
		500	0.14	103.8	6.2×10^{-13}
BPC B	200	200	0.75	240.4	1.0×10^{-12}
		300	0.33	148.0	1.3×10^{-12}
		400	0.29	113.6	1.1×10^{-12}
BPC C	200	200	0.95	240.4	2.4×10^{-12}
		300	0.69	151.2	2.4×10^{-12}
		400	0.15	26.8	1.9×10^{-12}
		500	0.44	103.6	1.8×10^{-12}
BPC D	200	200	1.26	240.4	2.5×10^{-12}
		300	0.86	151.2	3.3×10^{-12}
		400	0.20	26.8	2.6×10^{-12}
		500	0.70	103.8	2.5×10^{-12}

Hydraulic conductivity of NaB GCLs

The NaB GCL had hydraulic conductivity exceeding 10^{-9} m/s for all tests conducted with low initial stress, and for all but the first increment (200 kPa) for tests with high initial stress, indicating that the aggressive copper heap leach solution suppresses swelling of the NaB granules and closure of intergranular pores (Table 2, Figure 3). The high hydraulic conductivity of the NaB GCL to copper heap leach solution is consistent with the SI of the NaB, which was 9.1 mL/2g when hydrated in the copper heap leach solution and 34.9 mL/2g when hydrated in DI water. Hydraulic conductivities of the NaB GCL to the copper heap leach solution generally are too high for the GCL to be effective, indicating that liners for copper heap leach facilities require GCLs that are more chemically resistant to copper heap leach solutions.

Hydraulic conductivity of the NaB GCL tested at low initial stress increased slightly (1.2×) as the confining stress increased from the initial 20 kPa up to 50 kPa and then up to 100 kPa, which was unexpected. The hydraulic conductivity then decreased monotonically and gradually with increasing confining stress (Figure 3). The unexpected slight increase in hydraulic conductivity as the stress increased from 20 kPa suggests that the NaB GCL was not in geochemical equilibrium when the stress was increased, despite meeting the termination criteria in D6766. The increase in hydraulic conductivity at higher effective stress also corresponds to additional permeation, suggesting that slow rate-limited cation exchange between the solution and NaB may have been occurring, as reported in Jo et al. (2004; 2005; 2006). This effect was more significant for NaB GCLs with high initial stress, with the hydraulic conductivity increasing 4.7× as the effective stress was incremented upward from 200 kPa to 500 kPa, and only decreasing slightly as the

stress was increased from 500 to 600 kPa. Moreover, any benefits of the initial high effective stress were ameliorated as the stress increased, with the NaB GCLs with low and high initial stress having essentially the same hydraulic conductivity at 500 kPa and at 600 kPa.

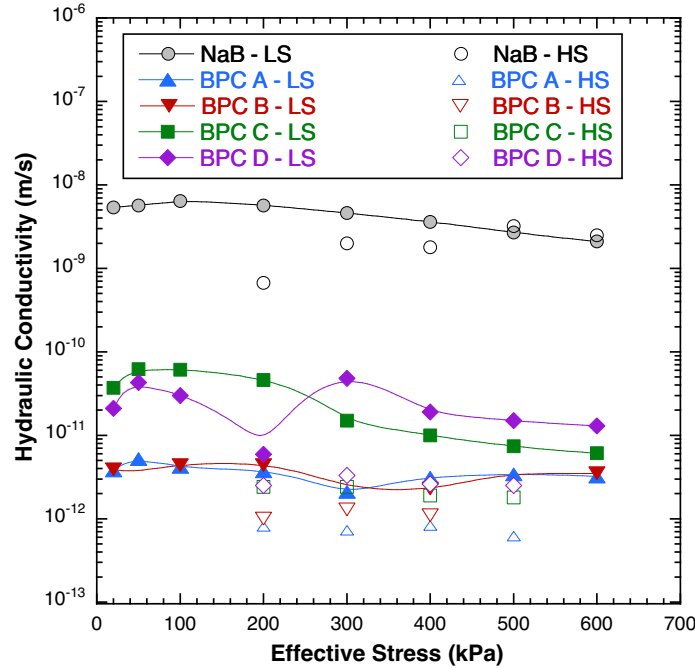


Figure 3: Hydraulic conductivity of NaB GCLs and BPC GCLs to copper heap leach solution as a function of effective stress for tests conducted at low (20 kPa) and high (200 kPa) initial stress

Hydraulic conductivity of BPC GCLs

Hydraulic conductivity of the BPC GCLs is also shown in Figure 3 as a function of effective stress for low and high initial stress. Hydraulic conductivities of the BPC GCLs with low initial confining stress are shown with solid symbols, whereas hydraulic conductivities of BPC GCLs with high initial confining stress are shown with open symbols. Much lower hydraulic conductivities (10^{-13} to 10^{-11} m/s) were obtained with the BPC GCLs than the NaB GCL (10^{-10} to 10^{-9} m/s), indicating that BPC GCLs can be more effective in liners for aggressive copper heap leach solutions.

For low initial confining stress, hydraulic conductivity of the BPC GCLs increases slightly ($< 2\times$) as the confining stress is increased from 20 kPa up to as much as 100 kPa, and then decreases nearly monotonically with increasing confining stress. As with the NaB GCLs, the unexpected increase in hydraulic conductivity between 20 kPa and 100 kPa suggests that the GCLs were not in geochemical equilibrium, despite meeting the termination criteria in D6766 before the stress was increased. As the stress is increased further, the hydraulic conductivity diminishes gradually, with a reduction of at most 10-fold from the highest to low hydraulic conductivity. This suggests that the pore space is only changing modestly

with increasing stress, and that the primary mechanisms contributing to low hydraulic conductivity (bentonite swelling and pore clogging by polymer) are established at the lowest stress.

BPC GCL D exhibited an unusual dip in hydraulic conductivity at 200 kPa, followed by an increase in hydraulic conductivity at the next increment in stress. This may indicate that the polymer is being re-distributed in the pore space when the stress is increased, that pores not filled with polymer at lower stress are closed as the stress increases, and/or polymer elution is occurring intermittently.

Effect of elevated initial stress

Hydraulic conductivities obtained from the low and high initial stress tests are compared in Figure 4. Hydraulic conductivities of the BPC GCLs with high initial stress were approximately 2 to 20-fold lower than BPC GCLs with low initial stress, with 5-fold lower being typical. All of the BPC GCLs with high initial stress had very low hydraulic conductivity, less than 3.3×10^{-12} m/s. The BPC GCLs with high initial stress also were nearly insensitive to the effective stress (Figure 3), with similar hydraulic conductivities obtained for stresses from 200 to 500 kPa. This suggests that larger pores are closed by the initial higher stress, providing some protection against higher hydraulic conductivities. Similar benefit was not realized for the NaB GCL, which had similar hydraulic conductivity for tests with low and high initial stress, except for the test at 200 kPa.

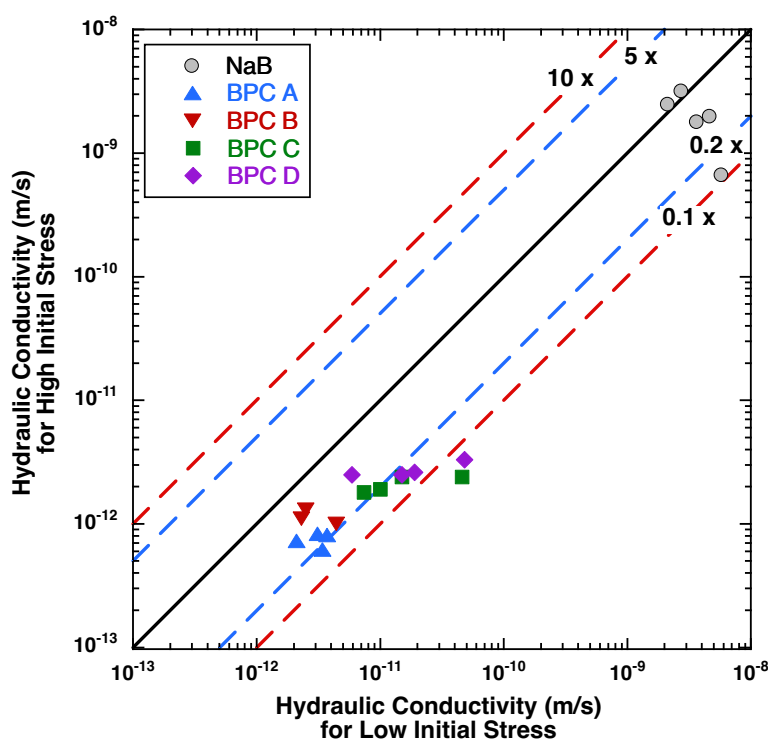


Figure 4: Hydraulic conductivity of NaB GCLs and BPC GCLs for tests conducted at high initial stress (200 kPa) vs. low initial stress (20 kPa)

Efficacy of swell index and active water content

Efficacy of SI and AWC as indicators of the hydraulic conductivity of the NaB and BPC GCLs to the copper heap leach solution is shown in Figure 5 using data from the low initial stress tests at 20 kPa along with data reported in Tan et al. (2022) (black dots). The data from the tests in this study are consistent with the data reported in Tan et al. (2022). The lowest hydraulic conductivities are realized for the highest SI and AWC, but neither index is effective in discriminating between less permeable and more permeable BPC GCLs. More study is needed to identify indicator tests that are effective predictors of the hydraulic conductivity of BPC GCLs.

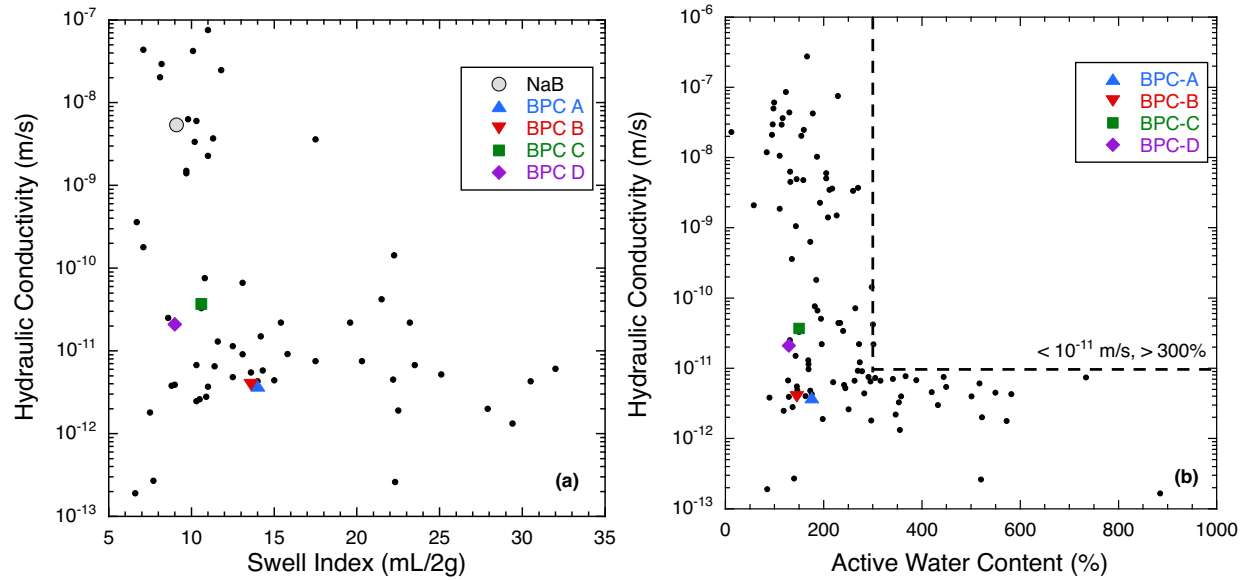


Figure 5: Hydraulic conductivity of GCLs vs. swell index (a) and active water content (b) for NaB or BPC hydrated in copper heap leach solution. Small solid black symbols from Tan et al. (2022)

Summary and conclusions

Hydraulic conductivities of geosynthetic clay liners (GCLs) containing commercially available bentonite-polymer composites (BPCs) or sodium bentonite (NaB) were evaluated using a synthetic copper heap leach solution as the permeant liquid. The BPCs were comprised of granular NaB dry-blended with granular polymer. Each GCL had a unique polymer, with the polymer used in each BPC and the polymer loading proprietary to the manufacturer. The NaB GCL was comprised of the same bentonite used in the BPCs.

The tests were conducted on NaB GCL specimens trimmed from a test roll provided by the manufacturer and on mock BPC GCL specimens prepared from the BPCs and geotextiles provided by the manufacturer. Tests were conducted under two different stress scenarios: low and high initial confining stress. In the low stress scenario, testing began at low initial confining stress (20 kPa) simulating a thin layer of ore on the heap leach pad, followed by a ramp up to as much 600 kPa in seven increments

representing increasing ore thickness. The higher stress scenario began with a higher initial confining stress (200 kPa) simulating a 10-m-thick layer of ore rapidly placed on the pad, followed by the same ramp up to as much as 600 kPa. The two scenarios were evaluated to determine if higher stress prior to contact with solution protects the GCL from adverse chemical interactions.

Permeation with the copper heap leach solution adversely affected the Na-bentonite GCL, with the hydraulic conductivity exceeding 10^{-9} m/s for all but one case (200 kPa stress under high initial stress condition). The BPC GCLs had much lower hydraulic conductivities ($< 9 \times 10^{-11}$ m/s), indicating that BPC GCLs can be more effective than NaB GCLs as liners for copper heap leach pads. The BPC GCLs permeated with higher initial stress had lower hydraulic conductivity at comparable stress compared to those initially permeated at low initial stress, indicating that high initial stress affords protection for BPC GCLs against excessive hydraulic conductivity. Similar protection was not realized for the NaB GCL. Hydraulic conductivity of the BPC GCLs varied between polymers, indicating that efficacy of a BPC GCL for containing leach solutions depends on the polymer employed. Hydraulic conductivity of the BPC GCLs decreased only modestly with increasing effective stress, indicating that the BPC GCLs had very small and tortuous pore spaces.

Swell index (SI) and active water content (AWC) were evaluated as indices of hydraulic conductivity of the BPC GCLs. Data from the tests in this study were consistent with SI and AWC data from past studies on BPC GCLs, but neither index was an effective indicator of hydraulic conductivity of the BPC GCLs. Additional study is needed to identify an appropriate index test for the hydraulic conductivity of BPC GCLs.

Acknowledgement

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At the Limit of Overliner Material Properties Applied for High-Capacity Heap Leach Projects

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Introduction

The overliner material in heap leach projects is very important and consumes a considerable part of the capital expenditure for these projects. Depending on the location, it is sometimes a major challenge to locate sufficient materials nearby. The overliner must have specific geotechnical properties to provide drainage and liner protection; a controlled maximum coarse size and sieve distribution are critical to avoid puncture failure in the liner system and provide efficient drainage.

This paper provides a summary of two recent studies of one of the biggest heap leach operations in Chile. This operation consisted of an expansion of the secondary leaching process to more than 168 million tons capacity, with the leached ore deposit reaching a maximum height of 120 m; plus a low-grade oxide dump leach expansion with a height of 80 m. This created a very high demand for overliner material in a location where natural sources are almost unavailable.

Methodology

The large-scale puncture test (ASTM D5514 modified, 2018) was developed with the collaboration of Geologic Associates in the USA. The objective was to study the influence of overliner materials and subgrade on a geomembrane by applying a normal load equivalent to the heap leach's maximum height.

The test configuration is shown in Figure 1a. Its specifications are as follows:

- As a variation on the ASTM standard where hydrostatic compression is applied, in this test the ore vertical load was simulated by a hydraulic pressure frame in a large-scale direct shear test box. In that way it was possible to generate the liner contact effect on both sides under the load, which is closer to the specific conditions in these kinds of projects.
- The test was carried out on a square hydraulic press of 12" × 12".
- Normal stress was applied at a speed of 70 kPa per minute for 48 hours, up to the maximum specified load. For this test, the maximum load was 2,400 kPa, which is equivalent to an ore height of 120 m.

- The overliner material was placed under the middle of the test box at 1.85 ton/m³ of dry density (corresponding to 95% maximum dry density) and a moisture content of 3.0%.
 - The subgrade was moisture conditioned to near optimum moisture, and compacted until no yielding was observed under pressure.
 - Coarse materials larger than 2" were in contact with the geomembrane during the test.
- (a) The textured side of the geomembrane was placed in contact with the subgrade material layer (Figure 1b).

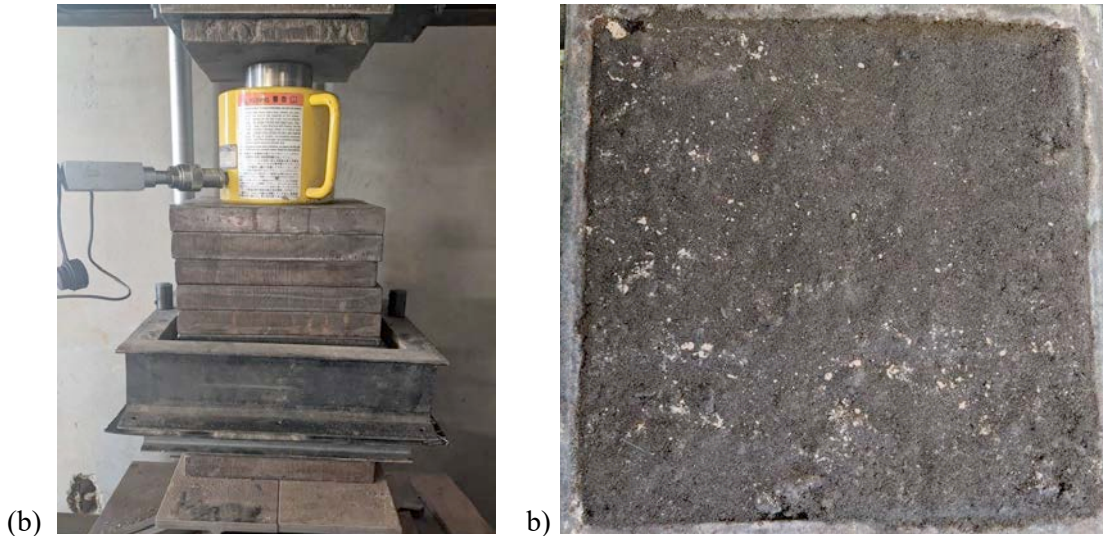


Figure 1: (a) Large-scale puncture test configuration; (b) Subgrade material

Increase in oversize material content

This analysis aims to verify the feasibility of using old leached crushed ore material as overliner on the geomembrane, by analyzing large-scale puncture tests for loads equivalent to up to 120 m on a 80 mils (2 mm) HDPE geomembrane, considering an increase of coarse size overliner material between 2" and 3". Previously, the project specified an overliner material with a maximum particle size of 1½", and coarse material over this size represented approximately 40% of the sources available, and a high cost of selection process.

Overliner alternative sieve analysis

The sample of the proposed material had 5% of coarse size (>2") and high fine content (16% under #200 sieve) according to the project specifications. Due to the major coarse content, the material may present a behaviour that could generate puncture failures in the geomembrane. In addition, it could allow the migration of fine particles from the leached ore during the irrigation process. Figure 2 shows the leached ore sample used as overliner in the large-scale puncture test and Figure 3 shows its grain-size curve.

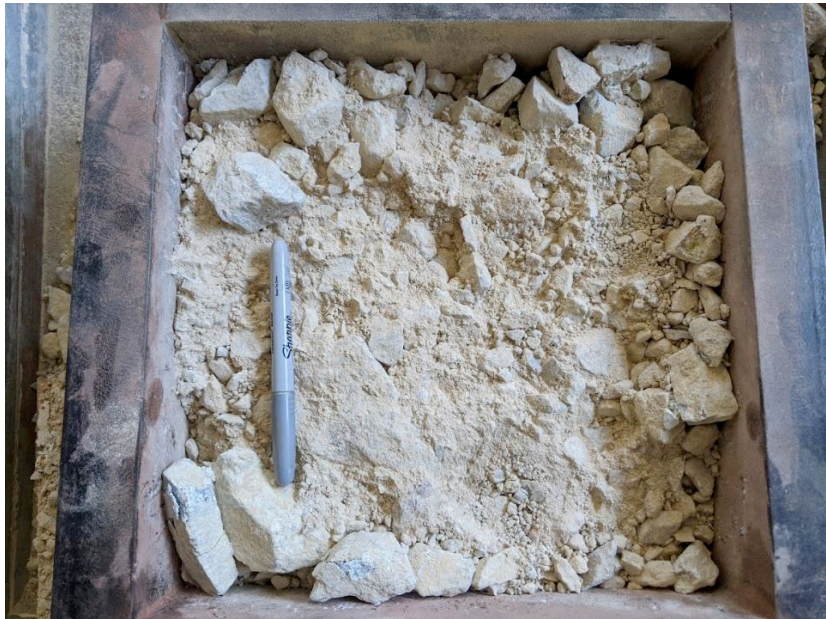


Figure 2: Overliner used in puncture test (leached ore sample)

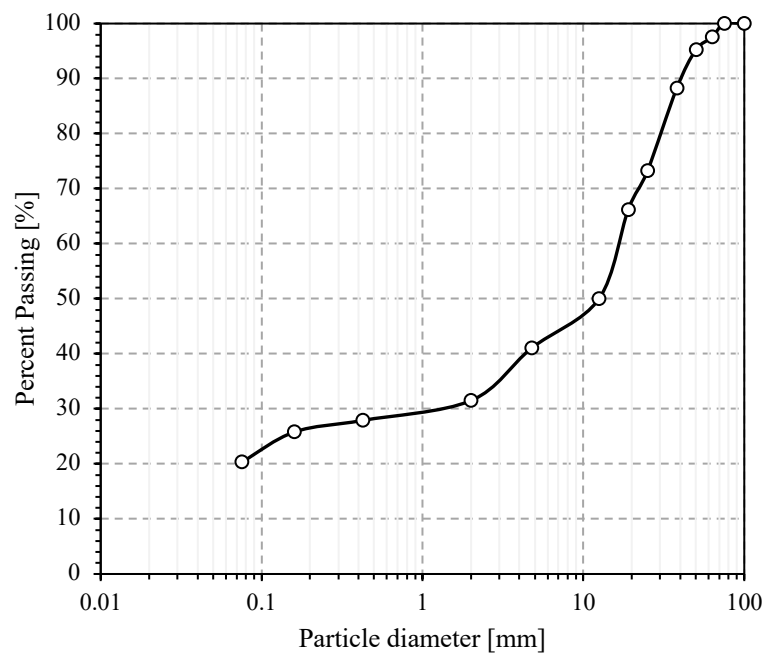


Figure 3: Overliner grain-size curve

Large-scale puncture test

Figure 4 shows the 2 mm HDPE geomembrane after the large-scale puncture test results, from which no puncture failures were observed in the geomembrane, showing only small yielding areas (plastic flow not recoverable after removal of the load). This is a normal response to the load level reached.



Figure 4: 2 mm HDPE geomembrane after puncture test

Puncture resistance vs. load increase

Because of the lack of additional approved areas to stack more secondary leached ore, the current operation looked to reuse abandoned phases of the deposit to load additional fresh ore. Based on this, the objective was to determine the maximum height that the current leached ore deposit can reach without affecting the integrity of the liner system and the drainage system due to the overload. The integrity of the liner material subjected to an increase in load was analyzed by the verification of the design safety factor and by performing different large-scale puncture tests using an overliner material of 2" maximum coarse size and normal stresses increments to up to 235 m ore height.

Puncture verification of existing geomembranes

The existing liner system of initial phases is composed of single-side textured HDPE geomembranes with thicknesses between 60 mils (1.5 mm), 80 mils (2.0 mm), and 100 mils (2.5 mm).

To determine the maximum height that the leached ore can be stacked, as well as for stability analysis, the geomembrane must comply with puncture failure resistance due to the additional load. For this analysis, it was assumed that stability would be assessed separately by the owner and was not included in this work. Similarly, the drainage pipes were analyzed separated and not included in this work, as the objective is related to the overliner characteristics and liner resistance.

Based on the project's design criteria, the safety factor is presented in Table 1.

Table 1: Puncture safety factor

Analysis	Safety factor
Verification for puncture due to ore load	0.5 – 3

To ensure that the ore load, the grain-size and particle shape of the drainage material over the geomembrane have no incidence, it is recommended that the safety factor be greater than 1.5.

Verification according to maximum permissible heights – conservative case

For the determination of normal stress, an estimate of the maximum ore loading heights (Hs) corresponding to each thickness of the geomembrane was made, iterating until the division of the allowable normal stress (P_{allow}) by the acting normal stress (P_{act}) results in a safety factor greater than 3.

To obtain the P_{allow} in each case, the methodology proposed by Godoy and Paredes (2011) was used, which modifies the methodology proposed by Koerner (2005), using information from specific testing applied to heap leach pads.

$$P_{allow} = \left(P_{GM} + 450 \frac{M}{H^2} \right) \left(\frac{1}{MF_S \times MF_{PD} \times MF_A} \right) \left(\frac{1}{RF_{CR} \times RF_{CBD}} \right)$$

Results for the verifications are presented in Tables 2, 3, and 4.

Table 2: Acting normal stress on 1.5 mm HDPE geomembrane

Variable	Value	Unit
γ	18	kN/m ³
HS	80	M
P_{act}	1,440	kPa
P_{allow}	4,381	kPa
SF	3.0	–

Table 3: Acting normal stress on 2.0 mm HDPE geomembrane

Variable	Value	Unit
γ	18	kN/m ³
Hs	90	M
P_{act}	1,620	kPa
P_{allow}	4,819	kPa
SF	3.0	–

Table 4: Acting normal stress on 2.5 mm HDPE geomembrane

Variable	Valor	Unidad
γ	18	kN/m ³
HS	95	M
P_{act}	1,710	kPa
P_{allow}	5,257	kPa
SF	3.1	–

Based on these results, the design safety factor is greater than 3, complying with what is specified in Table 1. According to this, for the most conservative case, the leached ore dump should not exceed 80 m, 90 m, and 95 m for 1.5 mm, 2.0 mm, and 2.5 mm HDPE geomembrane thicknesses, respectively; otherwise the liner would be subjected to acting normal stress that would exceed the maximum allowable limit, which could damage it.

Verification according to the current maximum heights

For the determination of the acting normal stress, maximum heights were considered, corresponding to each thickness of geomembrane according to the actual heights. The results are shown in Tables 5, 6, and 7.

Table 5: Acting normal stress on 1.5 mm HDPE geomembrane

Variable	Value	Unit
γ	18	kN/m ³
HS	116	M
P _{act}	2,088	kPa
P _{allow}	4,381	kPa
SF	2.1	—

Table 6: Acting normal stress on 2.0 mm HDPE geomembrane

Variable	Value	Unit
γ	18	kN/m ³
HS	127	M
P _{act}	2,286	kPa
P _{allow}	4,819	kPa
SF	2.1	—

Table 7: Acting normal stress on 2.5 mm HDPE geomembrane

Variable	Value	Unit
γ	18	kN/m ³
HS	116	M
P _{act}	2,088	kPa
P _{allow}	5,257	kPa
SF	2.5	—

With these values, safety factors of 2.1 and 2.5 were obtained, which are within the established design criteria.

Verification according to maximum proposed heights

As there is a requirement for an increase in capacity, puncture resistance was verified for an increase in height of approximately 25 m. For the determination of the acting normal stress, maximum heights of 140 m, 155 m, and 165 m were considered, corresponding to 1.5 mm, 2.0 mm, and 2.5 mm HDPE geomembrane thicknesses, respectively. The results are shown in Tables 8, 9, and 10.

Table 8: Acting normal stress on 1.5 mm HDPE geomembrane

Variable	Valor	Unidad
γ	18	kN/m ³
HS	140	m
P _{act}	2,520	kPa
P _{allow}	4,381	kPa
SF	1.7	–

Table 9: Acting normal stress on 2.0 mm HDPE geomembrane

Variable	Valor	Unidad
γ	18	kN/m ³
HS	155	m
P _{act}	2,790	kPa
P _{allow}	4,819	kPa
SF	1,7	–

Table 10: Acting normal stress on 2.5 mm HDPE geomembrane

Variable	Valor	Unidad
γ	18	kN/m ³
HS	165	m
P _{act}	2,970	kPa
P _{allow}	5,257	kPa
SF	1.8	–

With these values, safety factors of 1.7 and 1.8 were obtained, which are within the intermediate range (0.5 – 3); therefore, they meet the established criteria.

By obtaining values greater than 1.5, which is considered for the purposes of this document as a minimum recommended safety factor, it is empirically verified that the existing geomembranes could resist the increase in load and would not result in puncture failure.

While the design recommendation is a minimum safety factor of 3, both current and increased heights have lower safety factors. These are above 50% of the established range, so they could still be acceptable.

In any case, as the obtained safety factors are lower than initially recommended for $SF=3$, verification by laboratory tests was recommended.

Large-scale puncture test verification

Overliner sieve analysis

The same leached ore material as the previous test was used as overliner material, which complied with the grain-size specifications of the project. The sample's sieve analysis curve is shown in Figure 6.



Figure 5: Overliner material used in large-scale puncture test

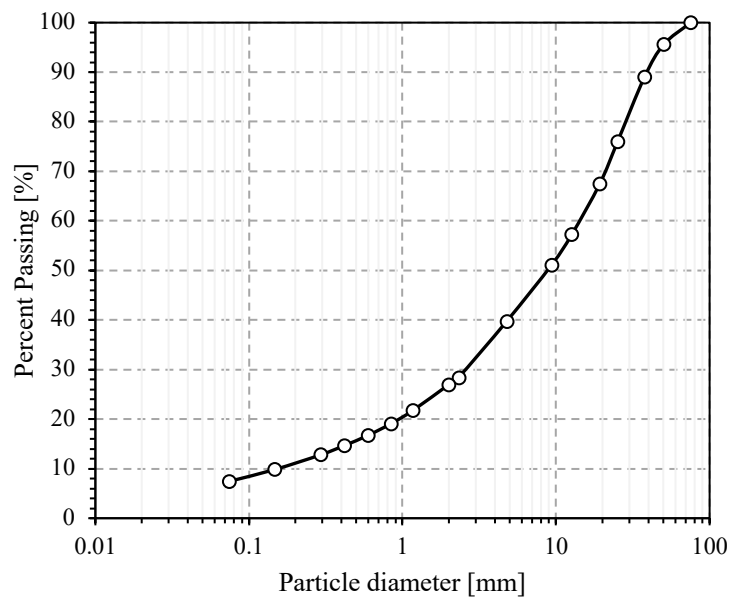


Figure 6: Overliner materials sieve analysis curve

Large-scale puncture test results

The large-scale puncture tests results are shown in Figure 7 and summarized in Table 11. These show that for heights lower than 180 m for 1.5 mm HDPE geomembranes and 185 m for 2.0 mm, the geomembranes developed between mild and moderate yield, in addition to not generating puncture failures of the material.

On the other hand, tests 4617I and 4617J show major yielding for the 2.0 mm and 1.5 mm HDPE geomembranes when reaching equivalent heights of 235 m and 225 m respectively, which is expected due to the considerable increase in the applied normal load.

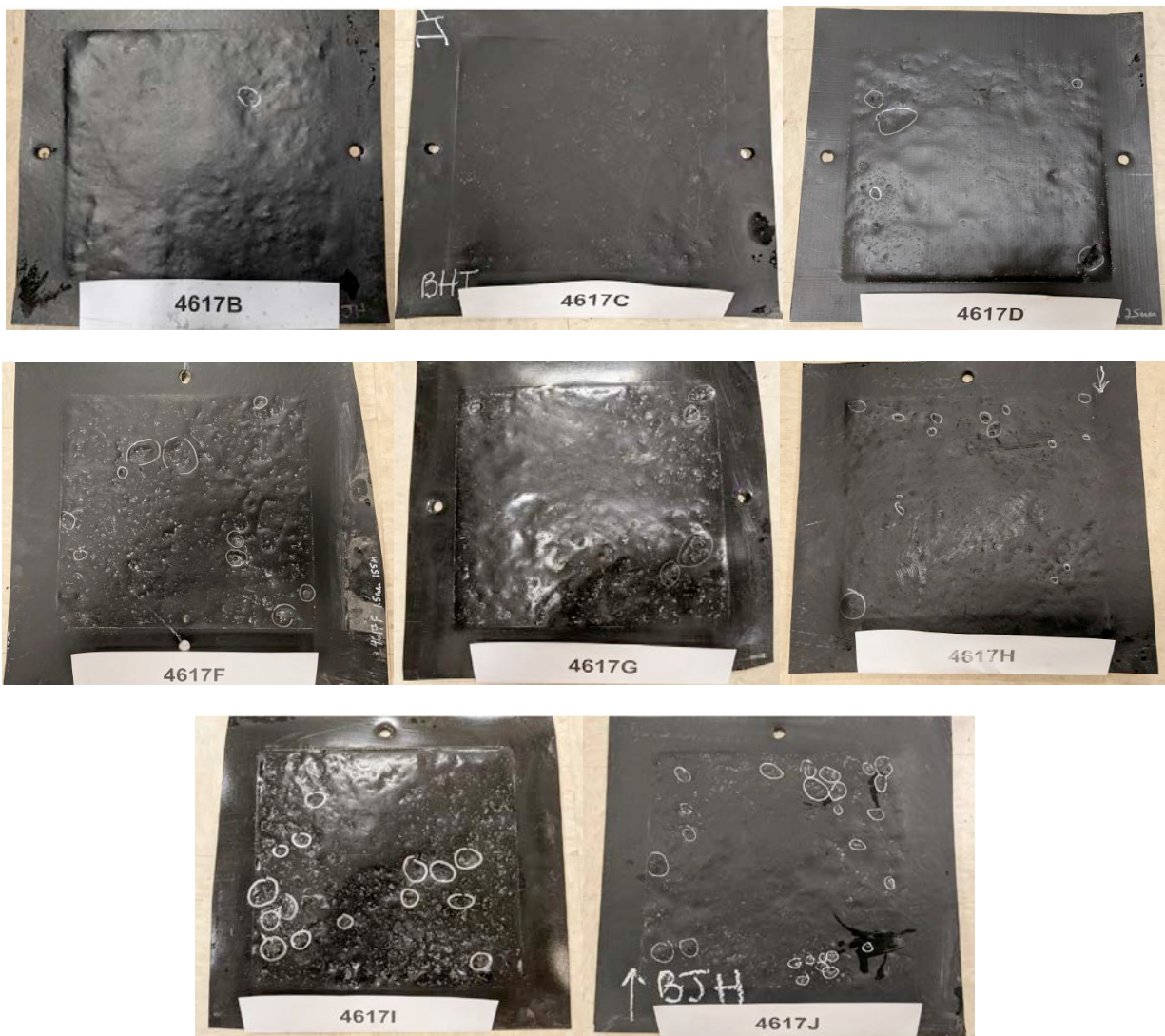


Figure 7: HDPE Geomembranes after large-scale puncture test

Table 11: Large-scale puncture tests results

Test ID	HDPE Geomembrane thickness (mm)	Equivalent height (m)	Yielding	Puncture
4617B	1.5	140	Mild	Not observed
4617C	2.0	155	Mild	Not observed
4617D	2.5	165	Mild	Not observed
4617F	1.5	155	Moderate	Not observed
4617G	2.0	185	Mild	Not observed
4617H	1.5	180	Moderate	Not observed
4617I	2.0	235	Major	Not observed
4617J	1.5	225	Major	Not observed

Conclusion

The current study assessed the feasibility of using granular materials with a maximum particle size greater than 2" as overliner, as well as the viability of increasing the vertical load to which the liner is subjected. It is important to consider that these analyses were based solely on the integrity of the liner system under higher vertical loads, and did not evaluate the specific variables of each project that may affect its overall slopes stability.

Regarding the increase in the coarse material content in the overliner, the puncture test results verified that despite the large coarse size (2" to 3"), the leached ore used as overliner did not generate punctures in the geomembrane. There was minor yielding in the contact side that generated an observable amount of plastic flow, which does not recover after removal of the pressure.

Therefore, it is concluded that the overliner material proposed with a maximum size of 3" at an equivalent vertical load of 120 m would not generate punctures in the geomembrane currently considered in the dump leach project, since the portion of 3" coarse size does not exceed 5% retained on a 2" sieve; the material must be well graded with sand and gravels content, and fine fraction under #200 sieve with maximum of 4%, to avoid migration of fine particles from the ore during the leaching process.

As for the vertical load increase analysis, the theoretical verification of the normal stress applied by the dump leach on the geomembrane, considering a safety factor >3 (conservative case), determined that the maximum allowable height for the dump leach would correspond to 80 m, 90 m, and 95 m for HDPE geomembrane thicknesses of 1.5 mm, 2.0 mm, and 2.5 mm, respectively, so that the geomembrane does not experience acting normal stress that can produce puncture failures.

The puncture tests results showed that, for heights below 185 m for the 2.0 mm and 180 m for 1.5 mm geomembranes, respectively, there was a mild to moderate yielding, and puncture failure was not caused.

On the other hand, tests equivalent to load heights of 235 m and 225 m showed a major yielding for 2.0 mm and 1.5 mm geomembranes respectively. From these results it is determined that although there is no puncture failure observed in the geomembranes under these height increases, they are probably very close to puncture failure with an increase in load or yielding over time.

It is important to note that the original project was developed 20 years ago, while the geomembranes tested for this study were of current manufacture, so it could be concluded that current geomembrane material developments are delivering a better performance.

To conclude, heap leach projects need to be assessed as independent cases, since overliner materials are obtained from natural soils and rock with high variability in geotechnical properties. However, when natural sources are not available, alternatives such as waste rock and leached ore can be used as proven options. These are easier to access and have a lower cost.

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New Projects: Review of Lone Star Leach Pad Design, Construction, and Operation

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Abstract

The Freeport-McMoRan Safford mine in Southeastern Arizona has become one of the company's highest producing copper mines in North America. The Lone Star (LS) resource was identified in 1886 and drilled and developed from the 1940s onwards. Feasibility studies were completed in the late 1980s, with mining permitted and operations initiated in late 2017. The mining of the deposit takes place more than four miles from the crushing facility, and required construction of new infrastructure, including extending ore conveyors, and building a new leach pad. The ore has low calcite and approximately 70% chrysocolla, making it a highly desirable expansion opportunity. The first Lone Star ore was delivered to a new leach pad on February 14, 2020.

The Lone Star pad is the newest leach stockpile constructed within Freeport's operations. The pad was designed with considerations and learnings from all previous leach pad projects within Freeport, using the best technologies available. The pad is 1.75 miles (2.8 km) long and a half mile (0.8 km) wide.

This paper summarizes the project development and operation, including lessons learned during construction, start-up, and operation. Also, specific examples of good practice that have been applied to the new pad will be discussed. This includes important Key Performance Indicators (KPIs) that are tracked and reviewed regularly to optimize pad performance. KPIs are discussed in the context of their impact to upstream and downstream performance of Safford copper production processes.

Safford uses remote monitoring of each module on the leach pad, including measurements of flow and pressure. Flushing is automated and monitored remotely to help ensure effectiveness.

Pad design, lift height, module layout, and piping are reviewed, along with over-liner neutralization, instrumentation, and monitoring. Also reviewed are flow transition between pads and predicting the solution balance and its associated challenges, and production predictions.



Figure 1: Aerial view of the Lone Star pit and leach pad

Introduction

The Lone Star resource was identified over 140 years ago. An aerial view of the site is shown in Figure 1. The deposit was drilled and developed beginning in the 1940s. The deposit is predominantly oxide, with

some mill grade sulfide at deeper levels. Feasibility and design of the pit, roads, and leach pad facility were ongoing from the late 1980s through 2016, with mining commencing in the pit in late 2017. The leach pad construction began in June 2018, and was completed in June 2020. The first ore was delivered to the new leach pad in February 2020.

Design and construction

The Lone Star pad was designed using input from learnings derived from crushed leach pad operations throughout FMI in conjunction with technical analysis from experienced FMI engineers and the engineer of record (EOR).

The design needed to safely optimize copper recovery, while maintaining a geotechnically stable stockpile. This involved testing of construction materials for inclusion in the pad.

A considerable amount of learnings came from the construction and operation of the original Safford Phase 1 pad. This includes initial construction, stacking practices, operating practices, and compaction reduction strategies.

The Lone Star pad has a proposed height of 350 feet constructed in three phases, with potential capacity of up to 1.6 billion tons of crushed leach ore. The liner is comprised of low permeability fill material overlaid with an LDPE geomembrane. The LDPE was protected by 2 feet of fine over-liner fill and 2 feet of run of mine (ROM) material to complete the base of the heap leach pad. Crushed ore is fed to the prepared leach pad using a conveyor/stacker system, which includes overland conveyors, portable grass-hoppers, and automatic stackers. Most of the pre-existing portable equipment used for the original Safford pad was re-used. Phase 1 of the Lone Star leach facility is designed to have 10 lifts. Each lift is intended to be 18 to 20 feet.

The first lift was designed to include the 4 feet of over-liner material, so the crush ore stacked was 16 feet. The ROM/crushed leach over-liner was neutralized during stacking and leaching the first lift of new ore. Column testing had shown this was feasible without extreme acid consumption.

Solution is applied to the ore and carried through a solution collection system to the LS PLS tank. The solution collection system was installed within the over-liner and immediately above the geomembrane liner. Lateral collection pipes of 4" and 6" diameter, perforated, corrugated, polyethylene were laid across the pad at 20 feet spacing with a slope of approximately 2%. These route flow to larger perforated HDPE header pipes up to 24" in diameter. These pipes were laid in collection trenches running lengthwise in the pad (see Figure 2). Solution flows by gravity to a collection sump (LS sump) and then on to the LS tank. Solution can then be pumped to the solvent extraction/electrowinning(SX/EW) plant for processing. The solution can also be returned to the pad for additional cycles of leaching or, in stormwater runoff conditions, it may overflow into a process solution pond.

If the upset condition is extensive the process solution pond may overflow into the non-stormwater impoundment. The collection ponds were designed to contain a 100-year rain event. Additional collection pipes were installed to the sump in preparation for the future Phase 2 and 3 expansions of the pad. The collection sump has 3 × 36" underdrain collection pipes that convey flow to the PLS tank. Both the PLS tank and the sump are designed to handle stormwater flow surges, and if necessary, overflow to the process solution pond (a diagram of the collection piping is shown in Figure 2).

The climate is generally arid in this area. Seismic activity is not a significant factor in the region. The only significant surface water control issue to consider was a means to accommodate surface water drainage during seasonal rainstorms. A stormwater collection channel was designed to route stormwater from the upstream areas around the leach pad.

The leach pad and piping corridors, conveyor corridors, etc., are all on liner. Sumps/ponds have leak detection systems. Safford is a zero-discharge facility, so all process solutions stay within the mine site.

The pad has a minimum slope of 1% north to south, with slopes west to east of up to 2%. The final side-slope angle for the pad is 3.33:1 west; all others are 2.6:1. Individual lift angles are 1.3:1 (1.4:1 after settlement).

Initially ore delivered to the leach pad was a combination of ore from the San Juan pit and the Lone Star pit, while mining was expanding in Lone Star.

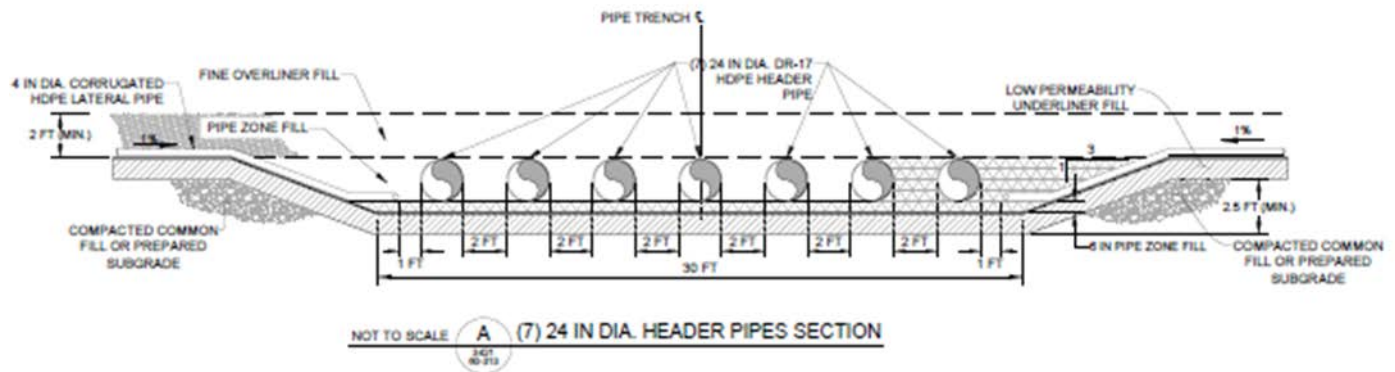


Figure 2: Solution collection diagram for the Lone Star leach pad

Operation – stacking

The ore is crushed to a P80 of 0.58" and then agglomerated with sulfuric acid and water as part of the acid cure process. Ore is then conveyed to the leach pad after agglomeration through a series of conveyors. The pad is stacked east to west by retreat stacking, so that the ore does not get compacted. Compacting the ore reduces permeability and leach recovery and can cause geotechnical concerns.

No equipment with a ground pressure of 5 psi or above is allowed on the pad, except the stacker. Compaction controls will be covered in more detail in a later section of this paper.

In preparation for stacking after each leach cycle is completed, the area is stripped of piping and dripline. The surface is slightly graded and then ripped with a low ground pressure (LGP) track dozer. Residual drilling, stability assurance, and any other necessary activities take place during the stripped period before the next lift is stacked. See Figure 3, which shows the Lone Star pad with stacking in progress.



Figure 3: Active stacking of the Lone Star leach pad

Operation – leaching

Phase 1 of the Lone Star pad operates a nominal flow of 36,000 gallons per minute (gpm) of raffinate to the pad.

Recipe

- The days under leach (DUL) target is 150 days.
- The application rate is 0.0022 gpm/ft².
- The P80 target is 0.58".
- Lift heights are 20 feet. The optimum lift height was determined by column testing.

The sulfuric acid concentration is targeted at 13.5 gpl. This enables operations to apply enough acid in the first cycle to recover the optimum amount of copper and align with estimated recovery curves. Lone Star ore has lower acid needs from gangue minerals than the prior pits mined at Safford, but has a much higher-grade ore, so the material is still a high acid consumer.

Our leachable area is separated into cells going south to north, which are in turn broken out into modules (see Figure 4). Lift 1 had 18 cells and 11 modules, but as the pad area decreases with additional lifts, the number of cells/modules shrinks.

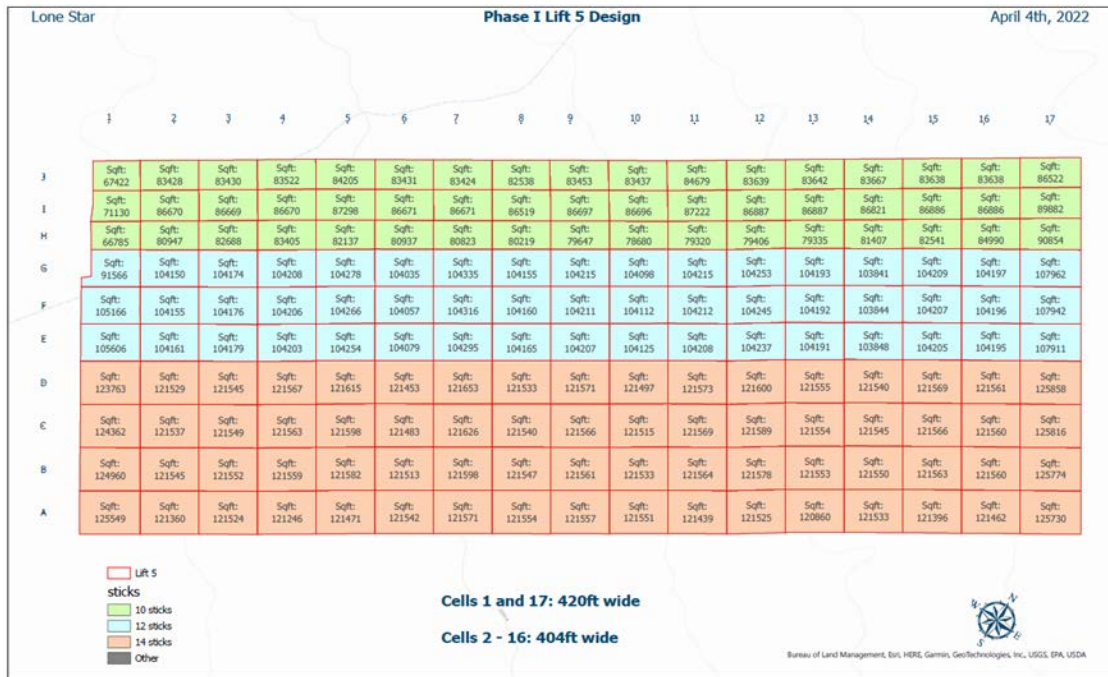


Figure 4: Lone Star leach pad layout

Raffinate piping is installed with 12" HDPE main feed lines to each cell. Then, 4" PVC pipe is used to connect to the modules. Dripline is installed to apply the raffinate solution. Module and cell widths are optimized based on previous flow testing and operation to ensure even flow across the area. Modules are typically 420 feet by 300 feet. Dripline feeds from north to south. Module flow feeds from the center in both directions. Solution is also applied to the side slopes.

Controls and monitoring

Each module has a feed side instrumentation skid, which controls the flow to the area and measures flow and pressure. These are battery-operated with radio communication.

Flushing the system occurs on a regular frequency and is essential to prevent drip line plugging from upstream solids or organic from the SXEW plant. On the non-feed side of the module there are two

automatic flush valves on each end of the line, mounted on skids. These are also battery-operated with solar panels, and controlled by radio remotely to maintain an optimum flushing cycle. See the typical module layout in Figure 5.

The company has a team of water balancers who adjust skid flows as needed to stay within the intended design parameters. A team of maintenance personnel address and repair issues with both types of skids, as needed.

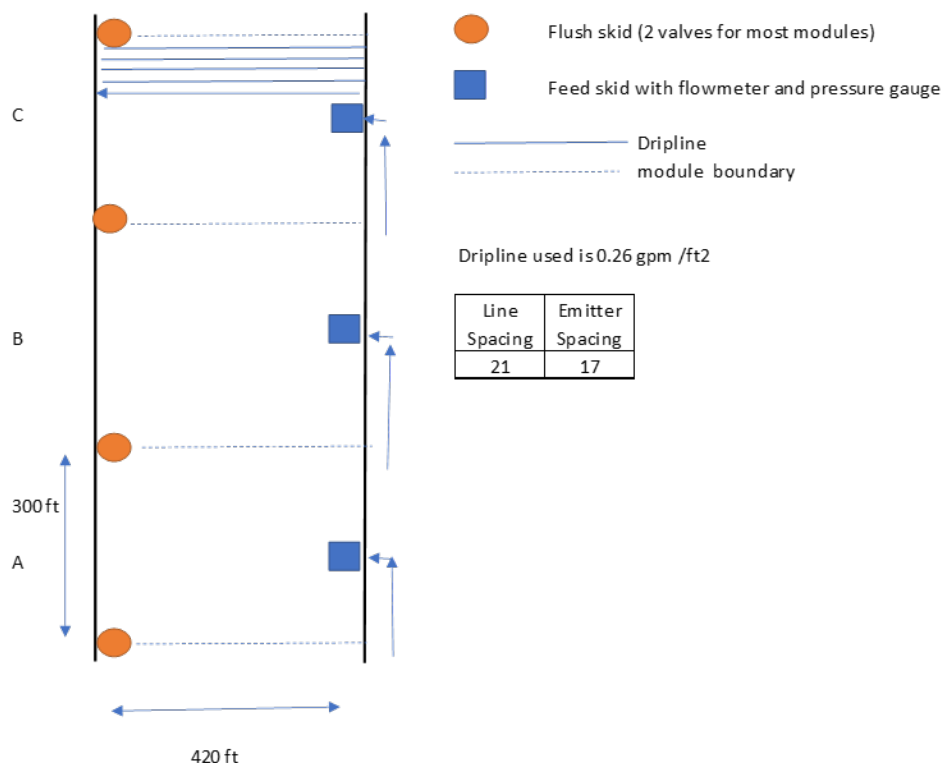


Figure 5: Flow and pressure skid layout

The solution application rate and flushing quality is monitored daily and during each shift, using many tools. A flow target is generated each day for each module. This is application rate in gpm/ft² multiplied by its surface area. The water balancers target this flow and adjust the skids in the field. Tablets are used with applications that are customized so that the real time flows coming from data in the PI database can be seen. Daily reports are generated to show the previous day's performance, and given a score as a percentage. At the same time, a geotechnical report is generated showing anything that is unacceptable. Geotechnical KPIs will not be reviewed in this paper for reasons of brevity.

To prevent preferential pathways or channelling in the stockpile, flow is stepped up in stages as new ore is placed under leach.

An example of a typical report is shown in Figure 6.

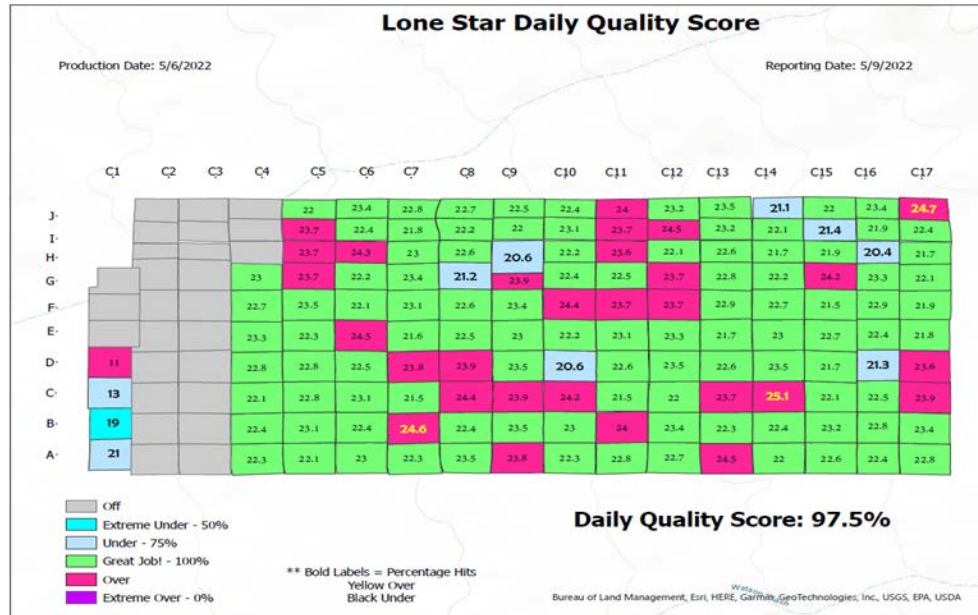


Figure 6: Typical daily quality score

In addition to application rate monitoring, flushing per module is tracked and reported daily.

An extensive drone program is utilized to observe issues on the pad, including slopes.

On the original Safford pad extra steps were learned to prevent any geotechnical issues. Equipment access to the leach pad is controlled to prevent compaction. Compaction of the new or pre-leached ore affects permeability, which affects pad porosity, flow distribution, and copper recovery.

Porosity can result in solution build-up within the pad, which can be a slope-stability concern.

Generally, no equipment is allowed on the pad with a ground pressure over 5 psi. A permit system has been established for unusual situations where no alternatives are possible. Previously leached areas are dried for three days before allowing any equipment on the area. No equipment is allowed on freshly stacked ore. Drain lines are installed around the perimeter during stacking to drain solution, to prevent solution build up. Vertical wick drains are installed throughout the pad to assist in solution percolation. The application rate is monitored closely to identify modules that exceed geotechnical KPIs, so that they can be corrected in a timely manner.

Performance

Copper recovery is monitored using mass balance accounting and recovery drilling. Copper grades, mineralogy, acid consumption/addition, P80, and many other parameters are tracked to the shift level and exact pad location using GIS, and by sampling the ore using a belt-cut sampler. The belt-cut sampler is located between fine crushing and the agglomerator. The unit automatically takes a sample every two hours from the crushed material as it passes to the agglomerator. After the leaching cycle, copper recovery is

calculated by drilling some locations and comparing that to the original stacked data. Typically, three holes are drilled in each location to get an average recovery. To date, performance has been in alignment with expectations.

Geotechnical monitoring

The stability of the heap is monitored using piezometers, which monitor pore pressure (which indicates solution level), survey monuments, and GPS units to detect displacement, shape arrays monitor subsurface movement and also InSAR, which compares satellite images to monitor displacement. These controls are monitored daily and reports generated.

Challenges/learnings

The first issue encountered was while establishing the first lift of crushed material, as the stacker began having lateral stability issues. This was due to the ROM being too coarse in some areas, resulting in the stacker tracks becoming misaligned and tripping out on safety factors. This was remedied by removing the larger rocks with skid steers and having 24-hour ground observers to assist in identifying potential issues.

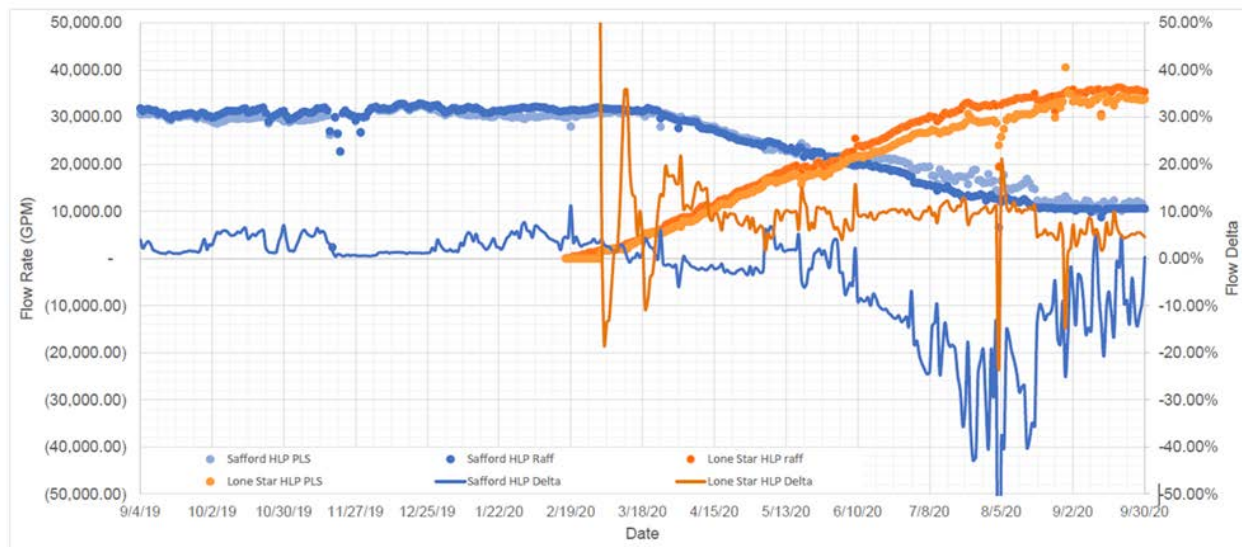


Figure 7: Flow transition between leach pads

See Figure 7 for charts showing flow transition. Transitioning flow from an existing pad can be a challenge. A deep stockpile will drain slowly, and there will be a period of time when the new pad is taking a small amount of solution but reporting quickly. Water management needs to be planned with sufficient storage for the existing flow to continue to drain, while adding increasing flow to the new area. Upfront

planning was completed, and excess solution ponds were utilized during the transition and operations gradually decreased the old pad flow. Diesel pumps were also added in line to help with temporary pumping.

The leach pad was started in February, which was ahead of monsoon season in the southwest. Startup timing should be a consideration to adequately manage additional solution from storms. Fresh water make up should also be a consideration, and additional sources should be identified for the desired flows if needed.

The possibility of turbidity from the new area needs to be considered, and associated storage needs should be available. During initial operation, Lone Star did not generate any significant turbidity.

Acid consumption and down-stream effects should be considered. For an oxide copper deposit, the PLS was expected to have low acid initially. Column testing had given an indication of what to expect, and generally the results followed expectations. However, the PLS free acid increased more dramatically than anticipated, so low SX extraction efficiencies were seen for a short period.

Engineered raffinate piping designs did not quite meet the total flow needs when stacking the north end of the first lift. Engineering had divided the flow to feed an equal number of cells in the original design. Flow was preferentially flowing to the lower cells on the pad due to the effect of hydraulics, as the flow travels downhill for the first few lifts. This was remedied by moving the northern line tie-in point to feed cells further north on the pad, leaving the other two feed lines to feed more cells.

A significant flow discrepancy between the individual flowmeters on the modules (differential pressure meters) and the main line (magnetic) meters was also identified. Collaboration with the vendor was completed to help change the parameters in each DP meter to correct for the difference. Now the flow meters match closely.

Summary

By using collaborative resources for engineering and planning, resulting in controlled procedures for operation and construction, there are no production or geotechnical concerns. In conclusion, the Lone Star leach pad is operating as expected and producing the forecast copper, which is maximizing tank-house capacity.

Statistical Functions for the Development of Heap Leach Pads Cost Estimation

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Abstract

In Peru, mining facilities and especially heap leach pad construction for metal production such as gold, silver, and copper is mainly carried out in the Andes region. There, heap leach pad construction faces many adverse and aggressive environments including weather, physical and topographic conditions, high altitude, and others, which have impacts on construction cost, both for the start-up of operations and for the expansion of the following phases of the leach pad. The construction cost represents a major part of a heap leach pad project and directly impacts the mining project's viability.

The purpose of this paper is to provide a useful tool for a construction cost estimation of a heap leach pad project under certain conditions. Such an estimation is necessary during viability evaluation and decision-making stages in investment projects of this type.

For the development of the cost estimation tool, an analysis using the database of projects executed in Peru during the last ten years was carried out. This was used to generate cost functions of the main heap leach pad construction activities such as topsoil removal, earthworks, underdrain system, liner system, solution collection system and overliner, which are the most significant costs in the construction of these types of facilities. Furthermore, the paper includes the cost function development of the main complementary facilities for start-up of operations, such as the PLS pond, the emergency pond, and the underdrain pond.

This tool will help to quickly estimate the cost of a heap leach pad construction with minimum effort, which is very useful in the early stages of an economic evaluation; as a result, this tool can help to expedite decision-making in mining investment projects. In addition, with the objective of evaluating the application of this tool, simulations were carried out with information from projects already built to verify that the costs obtained are within an acceptable margin compared with the actual construction costs.

Finally, this tool is provided in this paper and the authors hope to serve as a reference for designers, contractors, and professionals associated with this market for the cost estimation of future heap leach pad projects.

Introduction

Heap leach pad construction in Peru, mainly for gold and silver recovery, has increased significantly over the last ten years as a result of the low cost of construction and operation associated with the exploitation of low-grade ore deposits. Different works are required for the construction of heap leach pads, such as earthworks to reach the foundation and grading surfaces, construction of the underdrain system, installation of the liner system and leachate solution collection system, and finally the placement of overliner on the lined area, onto which ROM, crushed or agglomerated ore, and overliner are usually placed as protection and/or drainage materials, as appropriate.

Currently, there are no documented records of the costs associated with the construction of heap leach pads in the Andean regions of Peru that would serve as a basis for assessing the feasibility of this type of projects in such regions. Construction costs are often determined based on previous projects in a pre-bidding process; in many cases, without considering the specific characteristics of the area where the project is to be located. Therefore, in view of the aforementioned limitation and the importance of estimated construction costs in the initial stages of the evaluation of a mining project – which is important and necessary information for mining companies, consultants and government agencies – we concluded that it is of major importance to have a statistical tool, such as a cost-based function, that allows one to estimate construction costs at a conceptual level with greater accuracy and in accordance with the surrounding environment.

This paper outlines the methodology, the collection and processing of data, and the results obtained in terms of statistical functions associated with the construction costs of heap leach pads. To obtain the cost functions, this research focused on the analysis of the main activities and/or associated systems such as the foundation, surface grading, underdrain system, liner system, solution collection system, and overliner.

Objective

The objective of this paper is to introduce a statistical methodology that allows us to obtain a reliable estimation of the costs of the various components of a heap leach pad in Andean regions, so that it can be used as a tool for a more accurate estimation of conceptual costs. In addition, statistical analysis (correlation, control charts, etc.) will be used to determine the equations or intervals that will allow the determination of the estimated a priori costs of the components of a heap leach pad system.

Research methodology

The proposed analyses are intended to draw inferences from population values (proportions, means) based on a sample. For a finite population, the sample size is determined using:

$$n = \frac{N \times Z_{\alpha}^2 \times p \times q}{d^2 \times (N - 1) + Z_{\alpha}^2 \times p \times q}$$

Where:

N: total population size;

Z_α: 1.96² for a 95% level of confidence;

p: expected proportion;

q=1-p;

d: precision.

In this case, p=5%, q=1-0.05=0.95 and d=3%. In order to estimate a proportion, the following must be identified:

- The level of confidence (1-α). The preset level of confidence yields a coefficient (Z). α=1.96 for 95% level of confidence and α=2.58 for 99% level of confidence.
- The desired precision of the study.
- An approximate value of the parameter to be measured (in this case, a proportion, p). It may be obtained from existing literature or previous pilot studies. If such information is not available, a p=0.5 (50%) will be used.

We considered 21 projects at the detailed engineering level, executed between 2015 and 2022; therefore, the sample size is as follows:

$$n = \frac{21 \times 1.96^2 \times 0.05 \times 0.95}{0.03^2 \times (21 - 1) + 1.96^2 \times 0.05 \times 0.95}$$

$$n = 19$$

Therefore, only 19 engineering studies will need to be evaluated for a 95% level of confidence with a precision of 3%. This methodology involves three steps: 1) data collection and compilation; 2) data processing in parallel to collection; and 3) statistical analysis.

Data collection

Data was collected from the Anddes database, which contains a digitalized version of the projects. From the information on heap leach pad project costs from various mines, a total of 21 projects were selected.

It should be noted that the costs of the investment projects correspond only to direct costs. Local taxes, profits, general expenses, supervision, CQA, among others, were not included to avoid the fluctuation

inherent to these items and obtain closed cost function data, except for the statistical analysis of the total CAPEX of the project. Variables were identified for each of the components of the heap leach pad projects after being identified in the Anddes database.

Database

The 21 projects located in Peru are depicted in Figure 1, where the highest incidence is observed in 2019 and 2020.

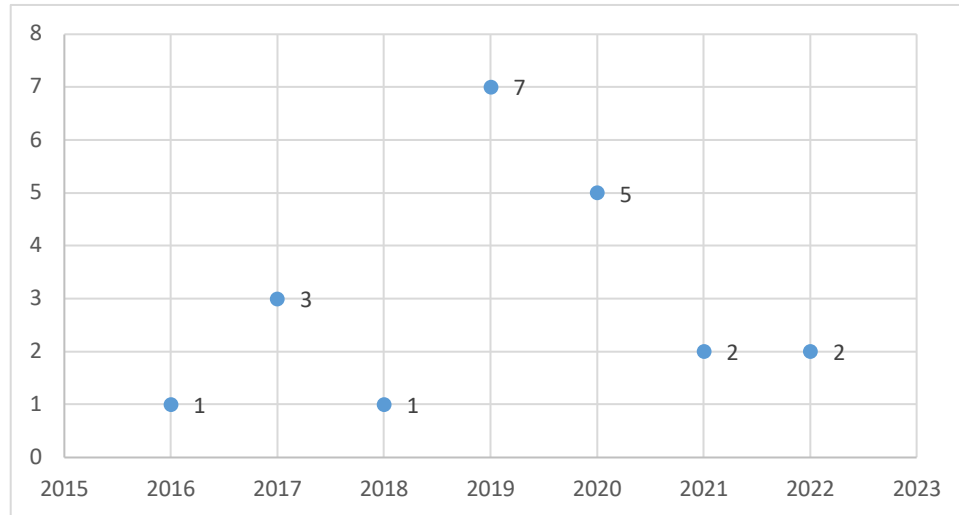


Figure 1: Number of projects timeline

Data processing

Each of the main characteristics of each system of heap leach pad projects was identified. The systems considered were: foundation, surface grading, underdrain system, liner system, solution collection system, and overliner (Wills and Finch, 2016). For further information and understanding of the activities and systems associated with heap leach pad construction, a summary of each of them is presented below.

Foundation

The foundation refers to the surface where the heap leach pad will be founded and is estimated based on geotechnical field and laboratory investigations. Costs associated with this activity include clearing and grubbing, and cutting of topsoil and unsuitable materials until the depth of competent foundation is reached.

Surface grading

Surface grading refers to the final surface where the heap leach pad liner system will be installed. A gradient should be configured towards a point of concurrence for drainage and discharge of the leachate solution to the process ponds.

Costs associated with this activity include cut and fill works to achieve the levels set in the leach pad design.

Underdrain system

Its function is to capture groundwater flows originating within the limits of the heap leach pad and divert them underneath the liner system to the underdrain ponds located downstream. The underdrain system is composed of main and secondary drains installed in the lower areas of the pad, consisting of trenches with dual-wall HDPE perforated pipes at the bottom, covered with drainage gravel and nonwoven geotextile on top as protection to prevent fine-particles migration.

Costs associated with this system include excavation for trench construction, installation of pipes and geosynthetics, placement of drainage material.

Liner system

Its function is to serve as an impermeable barrier to prevent leakage of the leachate solution into the ground. It consists typically of a 300 mm thick low-permeability soil layer or geosynthetic clay liner (GCL) as secondary liner, and a geomembrane as primary liner.

Costs associated with this system include earthworks for the placement of the low-permeability soil (or installation of the GCL) and the installation of the geomembrane.

Solution collection system

Its function is to collect and quickly convey the leachate solution, accumulated within the bottom of the leach pad, to the operating pond. The solution collection system is composed of primary and secondary dual-wall HDPE perforated pipes installed in the lower areas of the pad on the geomembrane, which are covered with drainage gravel or leached ore to ease solution conveyance.

Costs associated with this system include the installation of pipes and placement of drainage material.

Overliner

The overliner corresponds to the layer of granular material, with a thickness varying between 0.50 to 0.80 m, designed to protect the geomembrane and/or drain the leachate solution. Such material must be durable and must be placed on top of the geomembrane and collection pipes. When used as protection, the overliner is intended to reduce damage and rupture of the geomembrane due to ore discharge and the operation of the ore transport and spreading equipment; when the overliner is used as drainage, relatively high permeability coefficients are required to facilitate the collection of the leachate solution at the bottom of the heap.

Costs associated with this system include the placement of granular material.

Statistical analysis

Following the methodology described above, data was extracted and organized in specific and detailed tables, sorted by each characteristic of each of the heap leach pad systems described above. The methodology used to determine the cost functions was based on the statistical analysis of the information obtained from the records of the heap leach pad characteristics (21 projects). In Appendix A of this chapter, a table showing the characteristics of each heap leach pad project used in this study is presented.

Data analysis

A first estimation of the standard deviation (σ) and the average updated cost or average updated unit cost, as appropriate, was performed to detect extreme records in the data sample that might distort the results. Considering that according to Chebyshev's Theorem, at least 75% of the data in any set of observations lies within at most two standard deviations from the mean, and that about 95% of the data from data sets that have a normal distribution lies within less than two standard deviations from the mean, the range "average $\pm 2\sigma$ " (Aguilar, 2021) was set as the limit. All data outside this range were removed from the sample.

Determination of the mathematical model

Upon discarding the sample data that exceeded the limit range set (average $\pm 2\sigma$), the standard deviation and the final average were calculated. The latter represents the Average Unit Cost of the construction component; thus, the cost function equation is as follows:

$$Y = K * X \text{ (constant function)}$$

Where:

Y: total cost of the system

K: average unit cost of the system

X: quantity of the system (m, m², m³, etc.)

Upon obtaining the data sample from the first analysis, a table was drawn. The first column contained the data regarding the system characteristics (capacity or dimension, height, volume, pipe length, etc.), which correspond to the independent variable (X). The second column contained the information regarding the updated cost/updated unit cost, which corresponds to the dependent variable (Y). For example: capacity vs. CAPEX, cut volume vs. cost, foundation area vs. cost, underdrain pipes length vs. cost, etc.

Statistical software Microsoft Excel 2021 was used to determine the mathematical models for cost functions that depend on a single variable (Valentin, 2022), for which purpose the data obtained from the data processing was input. Thus, after verifying their statistical validity, functions were obtained for each group of data according to each system of the leach pad; therefore, the cost function obtained has the highest correlation coefficient among the various charts that could be obtained using Excel. In addition, charts with

a correlation coefficient (R^2) lower than 0.7 were discarded based on the fact that the data set has a medium or strong level of dispersion. An example is shown in Figure 2.

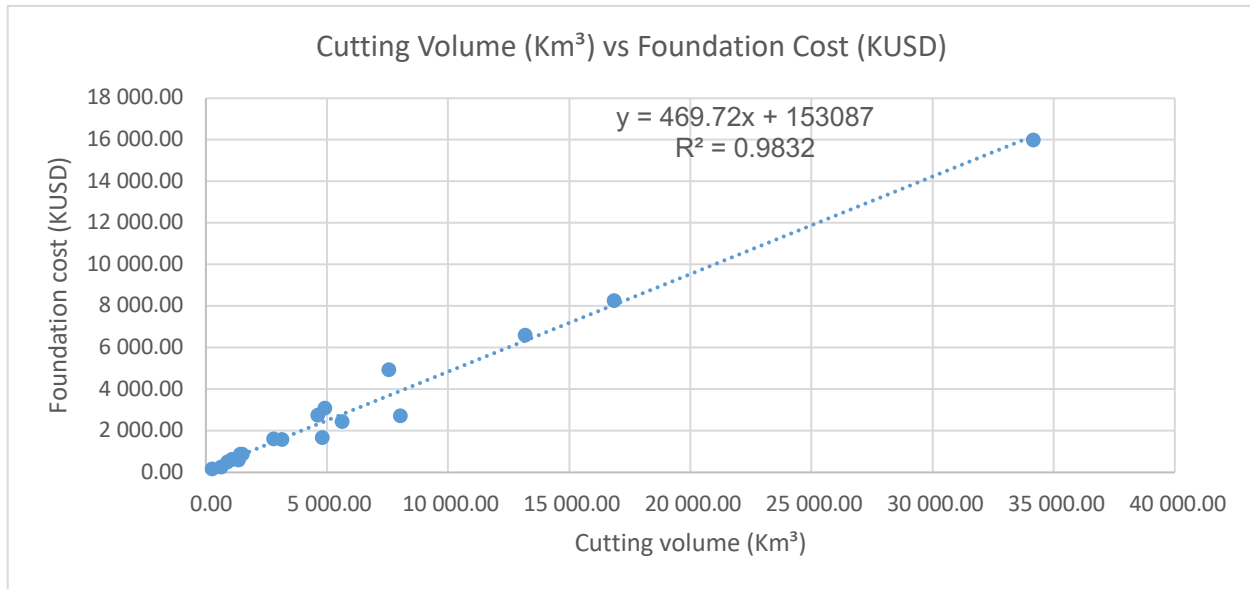


Figure 2: Example of one variable cost function

Similarly, GMDH Shell 3 software (Koshulko, 2022) was used to establish a correlation between more than two variables, where a number of variables were input and iterated to obtain an equation for several variables with a good correlation according to parametric estimation standards (Hastak, 2015). Functions with a correlation coefficient (R^2) lower than 0.7 were also discarded. An example is shown in Figure 3.

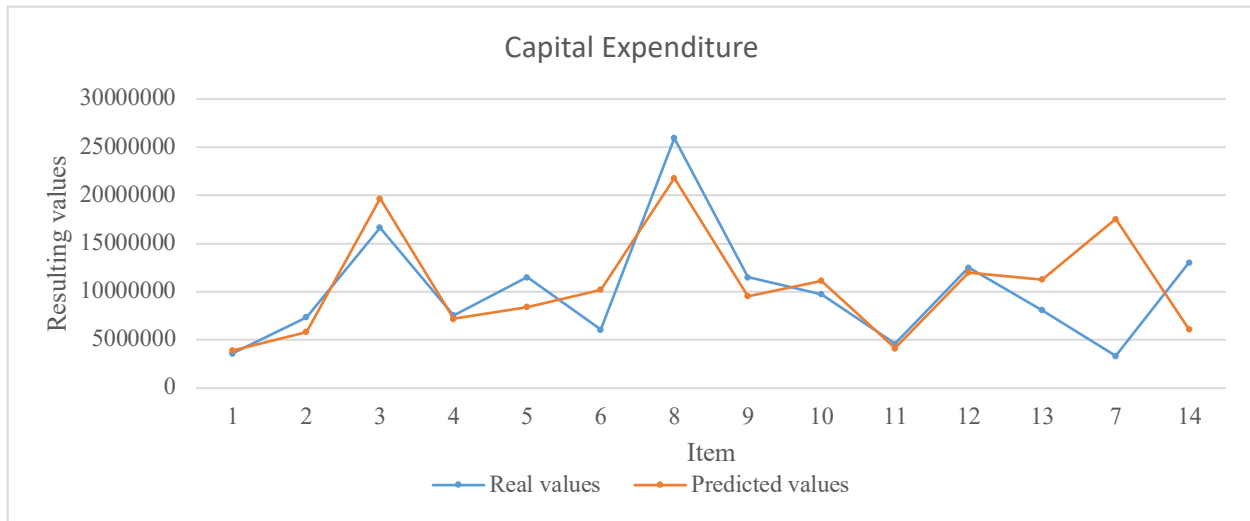


Figure 3: Example of many variables cost function

Results

After obtaining the representative values, regression was determined to obtain the cost function formula for each component found in the projects analyzed, resulting in the cost functions shown in Table 1. From the data analyzed, 16 cost functions with an adequate correlation were determined, as shown in Table 1.

Table 1: Cost functions

Item	Components of the system	Parameter	Unit	Cost function	Type of function	R ²
1	FOUNDATION					
1.1	Cost of foundation cut	Volume, V	m ³	$4.6972V + 153087$	Linear	0.98
1.2	Cost of foundation cut	Area, A	m ²	$C = -9E-15A^4 + 5E-09A^3 - 0.0009A^2 + 48.709A + 460428$ $-7.46719e-09 + 730476 * A * 6.69275e-07 + A * 0.511111$	Polynomial	0.75
1.2	Volume for foundation	Site area, A _{site} Average D _f , D _f Area of foundation, A _f	m ²	$A = -44133.5 + A_{site} * B * 4.61247e-06$ $B = 316330 + D_f * C * 0.0950892$ $C = 305429 - ((58280 + D_f^2 * 28859.5 + A_f * 0.46544)) * ((251827 + D_f^2 * 29571)) * 6.60125e-07 + ((158280 + D_f^2 * 28859.5 + A_f * 0.46544))^2 * 9.91017e-07$	Polynomial	0.95
2	GRADING					
2.1	Cost of cut	Volume, V _{cut}	m ³	$7.93V_{cut} - 402886.26$	Linear	0.93
2.2	Cost of fill	Volume, V _{fill}	m ³	$5.37V_{fill} + 891945$	Linear	0.81
3	UNDERDRAIN					
3.1	Total cost of piping	Pipe length, L	m	$0.0002L^2 + 29.74L - 32325.70$	Quadratic	0.89
3.2	Cost of installation	Pipe length, L	m	$0.0007L^2 + 3.9923L - 1528.7$	Quadratic	0.92
3.3	Cost of procurement	Pipe length, L	m	$-0.0005L^2 + 25.754L - 30797$	Quadratic	0.93
3	COLLECTION					
3.1	Number of pipes	Max. slope area of 10%, A _r	m ²	$1.29E-06A_r^2 + 0.057A_r + 6222.55$	Quadratic	0.88
3.3	Total cost of piping	Pipe length, L	m	$-0.0002L^2 + 31.391L - 16740$	Polynomial	0.67
4	LINER					
4.1	Total cost of liner	Lined area, A _{lined} Area greater than 25%(GCL), A ₂₅	m ²	$229265 + A_{25} * 11.654 + A_{lined} * 7.10$	Polynomial	0.86
5	OVERLINER					
5.1	Cost of overliner placement	Volume, V	m ³	$y = 2E-05V^2 + 2.2454V + 498514$	Polynomial	0.93

Item	Components of the system	Parameter	Unit	Cost function	Type of function	R ²
5.2	Cost of overliner placement	Lined area, Alined Area greater than 25%, A25 Area greater than 40%, A40	m ²	$\text{Alined} \cdot 3.11 - 13.8 \cdot \text{A40} + \text{A25} \cdot 23.11 + 1093981.3$	Polynomial	0.59
6	CAPEX			$Y1 = 1.74215e-07 - \text{Asite} \cdot 2.8135e-12 + \text{"A"} \cdot 1$ $\text{"A"} = 1.3689e+06 + \text{"B"} \cdot 1.11307 - \text{"D"} \cdot 0.244371$		
6.1	CAPEX	Site area, Asite Lined area, Alined Average Df, Df Site area > 40%, Asite40		$\text{"B"} = -4.04774e+06 + \text{Df} \cdot 2.16951e+06 + \text{"C"} \cdot 0.994604$ $\text{"C"} = 2.02666e+06 - \text{Asite40} \cdot 69.7453 + \text{Alined} \cdot 80.2128$ $\text{"D"} = 1.92584e+06 + \text{Asite} \cdot 20.7416 + \text{"E"} \cdot 0.387097$ $\text{"E"} = 3.14215e+06 + \text{Asite} \cdot 33.8416 - 27855 - \text{V} \cdot 9826.86 + \text{A} \cdot 1.52949$	Polynomial	0.82
6.2	Site area	Volume, V Gorge length, L Crest area, Acrest		$\text{A} = -29609.4 + \text{L} \cdot 60.9164 + \text{B} \cdot 0.967463$ $\text{B} = 100803 + \text{V} \cdot 18908.7 - \text{Acrest} \cdot 13893.3$	Polynomial	0.85

Conclusions and recommendations

The cost functions obtained provide an order of magnitude estimate of the potential costs associated with the construction of a heap leach pad project. Said functions have a good correlation and were obtained using a single variable. Nonetheless, several variables can be considered, such as the classification of areas according to a set range of slope gradients to obtain a function to determine a rough estimate of the cost.

The costs associated with the most frequent activities performed during the construction of a heap leach pad such as foundation, grading, underdrain, solution collection, liner, and overliner were obtained in this research.

The development of cost functions is applicable for Class 4 and 5 estimates in accordance with the cost-estimate classification system established by AACE International (Bredehoeft et al., 2019). It constitutes a baseline to achieve accuracy in cost estimates ranging from -50% to +100% variance.

It is recommended to use the largest possible source of data to broaden cost functions, not only for the location of a new mine or its expansion on typical Andean areas of Peru, but also considering the

complexities and unique characteristics of each region in the world, such as Latin America and North America, among others.

It is recommended to look for different variables, as well as the various correlations existing between them, using methodologies focused on data analysis such as neural networks.

Although cost functions presented in this paper are limited, it is advisable to conduct research on this subject to serve as tools in the mining industry and provide benefits from the conceptual engineering level, and thereby save time and resources during the engineering stages of the project.

Acknowledgements

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Appendix A – Database

PROJECT	1110.10.05	1110.10.10	1110.10.14	1110.10.18	1112.10.21	1112.10.32	1112.10.48	1112.10.61	1112.10.65	1113.10.27	1113.10.39	1116.10.17 3A	1116.10.17 3B	1116.10.17 4	1116.10.17 5	1408.10.07	1408.10.26	1408.10.32	1112.10.87
Expansion or New	EXP	EXP	EXP	NEW	EXP	EXP	EXP	NEW	EXP	EXP	EXP	EXP	EXP	EXP	EXP	NEW	NEW	EXP	EXP
Capacity (Mt)	1.8	0.624	13.52	6.75	10.83	22.4	22.78	41.26	18.32	30.86	9.71	21.24	6.87	15.18	12.21	11.05	10.88	44.97	5.87
Assumed density (t/m ³)	1.6	1.6	1.6	1.8	1.7	1.7	1.7	1.7	1.7	1.68	1.68	1.58	1.58	1.58	1.58	1.7	1.7	1.78	1.7
Design volume (Mm ³)	1.123	0.39	8.45	3.75	6.37	13.18	13.4	24.27	10.78	18.37	5.78	13.44	4.35	9.61	7.73	6.5	6.4	25.23	3.45
Crest area (Ha)	2.9	1.35	3.08	2.6	2.05	2.24	10.54	9.45	9.3	3.46	2.02	4.99	2.39	2.64	0.98	5.59	6.375	36.17	3.6
Global slope	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Maximum height (Total)	32	23	75	70	95	97	118	127	69	96	130	100	56	79	94	80	64	120	92
Year	2014	2014	2017	2018	2014	2015	2017	2017	2017	2016	2017	2016	2016	2016	2016	2016	2018	2019	2022
Site area (TN)	62630	57608	270710	219100	185290	367076	201210	216100	197800	297402	173769	298487	83038	286028	299467	347565	411623	732780	259380
Slope less than 10%	4430	14171	12650	26400	45900	81103	23660	23400	26400	98064	43849	114370	67399	30708	88921	33450	8112	37154	31381
Slope 10-40%	48670	28057	144260	91200	47540	97641	106700	91200	88200	146362	110975	119566	6715	181993	109546	230395	69621	192472	96928
Slope greater than 40%	9530	15380	113800	101500	91850	188332	70850	101500	83200	52976	18945	64551	8924	73327	101000	83720	333890	503154	131071
Length	360	360	660	610	725	1040	1517	1380	864	273	310	330	110	460	795	708	650	659	520
Greater height	3488	3560	3435	3419	3425	3467	3410	3407	3400	4155	4149	4382	0	4283	4411	2895	2555	2660	3401
Lower height	3433	3515	3346	3333	3341	3306	1833.5	3295	3305	4128	4117	4366	0	4225	4293	2740	2445	2587.5	3285
Height variation	55	45	89	86	84	161	1576.5	112	95	27	32	16	0	58	118	155	110	72.5	116
Slope	15.7	12	13.5	14.2	11.5	15.5	12.55	8.1	10.89	9.7	10.31	5	4.5	12.4	14.83	22	17	22	22.75
Foundation area (2D)	52660	27740	1123760	186080	129550	180600	22080	234222	11200	245400	117480	148300	4120	229550	249600	284100	284100	453100	191900
Average Df (m)	1.3	2.5	5	8.2	2	1.9	2.9	9.9	5.8	3	1.2	0.7	0.9	0.9	0.4	1.5	1.5	1.3	2.3
Grading area	51210	46920	246500	197140	197100	315100	157350	197020	197020	258059	154631	254230	72050	252817	272869	313151	322356	597787	225372
Slope less than 10%	21100	12550	20500	43840	43850	52950	24650	43830	43830	131228	71689	105500	47523	21503	90763	143183	141991	91000	62550
Slope 10-40%	17500	24050	126900	62400	62400	137200	95500	62355	62355	88334	56647	106128	11030	176630	80723	66085	54933	164116	70423
Slope greater than 40%	12610	10320	99100	90900	90850	124950	37200	90835	90840	38497	26295	42602	13497	54684	101383	103883	125432	342671	92399
Grading area (no access)	41460	35255	228450	149520	142440	259870	135070	100748	149520	256313	126097	209033	56169	219772	250424	233760	227097	494603	162930
Slope less than 10%	19620	10890	18700	34920	34920	45400	19670	34920	34920	130425	61994	95487	44153	15871	88698	121024	117883	69092	46224
Slope 10-40%	15740	22290	125950	60400	60420	121710	90700	60408	60400	89123	49988	97722	6543	176201	76712	61547	46607	164972	67270
Slope greater than 40%	6100	2075	83800	54200	47100	92760	24700	5420	54200	36765	14115	15824	5473	27700	85014	51189	62607	260539	49436
Perimeter road	590	575	1168	1120	920	1785	573	750	884	1960	1150	2162	825	1580	2021	2053	1958	1653	1336
Perimeter length	590	575	1168	1120	920	1785	573	750	884	1960	1150	2162	825	1580	2021	2053	1958	1653	1336
Track width	4	3.5	3	3	6	6	6	5.5	5	8	8	6	6	6	3	6.5	4.5	8.75	5.6
Ver	1.2	1.2	4.1	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2/1.7	1/1.5	1.2	1.2	1.2	1.2	1.2	1.2
Slope less than 10%	305	575	385	900	270	834	380	450	414	1960	1150	1350	715	715	1390	1676	1770	3030	1186
Slope 10 to 20%	200	0	280	0	585	661	666	0	470	0	0	360	0	545	245	250	0	220	150
Slope greater than 20%	85	0	503	220	65	290	100	300	0	0	0	452	110	320	386	127	188	56	0
Cutting in foundation (m ³)	62832	313560	1316260	1685123	278410	802890	563200	3414679	134200	754930	149050	108460	25685	490886	88627	479200	462165	143529	468710
Cutting in grading and access road (m ³)	38137	98376	379753	151096	497420	725780	258555	311366	64130	541640	264000	169400	55176	152845	135080	1642800	3045405	790352	712250

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PROJECT		1110.10.05	1110.10.10	1110.10.14	1110.10.18	1112.10.21	1112.10.32	1112.10.48	1112.10.61	1112.10.65	1113.10.27	1113.10.39	1116.10.17 3A	1116.10.17 3B	1116.10.17 4	1116.10.17 5	1408.10.07	1408.10.26	1408.10.32	1112.10.87
Fill in grading and access (m³)		147477	384216	285340	494802	28292	291489	51205	1870440	66220	20020	22495	407000	163042	363682	109780	1393400	1990725	344080	736780
Overliner fill (m³)		21978	30982	96250	458759.4	63146	141086	104060	151800	57200	517781	207350	113520	54087	115159	113872	123640	247203	376189	106700
Solution collection pipes (m)		3586	6312	12038.4	10150.8	3395	6270	5201	8173	5173	22517	9790	13532.4	6514.2	11530.05	10930.5	16907	16463	6963	3837
Underdrain pipes (m)		1353	2256	6314	10269.6	2206	4243.5	2482	3472	2435	12507	7887	1518.3	0	3912.3	4265.1	3392.4	2017	2329	3478
GCL (m²)		8326	0	70070	21450	12202	26013	0	3300	0	281435	108350	0	0	0	140259	0	15750	69584	12650
Geocomposite (m²)		0	0	70070	0	12202	26013	0	3300	0	80190	17600	0	0	0	88809	0	14374.5	61688	0
LLDPE 2.0 mm geomembrane (m²)		58742	83028	266200	163152	0	79695	0	0	0	0	0	0	0	0	0	128700	0	0	0
LLDPE 1.5 mm geomembrane (m²)		0	0	0	0	145815	225170	94160	262900	95260	281435	141350	225330	108024	231682.5	270259.5	141900	232890	215881	180400
Non-woven geotextile of 270 g/m² (m²)		0	0	0	0	0	0	0	1650	0	201245	90750	0	0	0	40425	0	35458.5	60900	0
Height variation	Cut in foundation (USD)	278019.5	1596020.4	6610814.1	8270753.37	1614778	2738666.7	2464154.88	15999190.9	588368.88	4934900.85	891899.8	645741.8	157848.9	3081159.62	516711.25	1690640	2759342.85	870876	2380743.2
	Cut in grading and access (USD)	435905.91	500733.84	1948199.99	744903.28	3735624.2	2866831	1331558.25	1560347.14	330269.5	6587163	4403520	880708.4	294479.13	2226633.2	764063.74	8993256	25783461.1	7805568.46	3447535.3
	Fill in grading and access (USD)	1936368.28	3958405.08	2705452.31	4835022.72	214812.4	1620863.2	411176.15	4517317.2	838844.6	307907.6	271564.15	4715961.8	2518457.92	3426592.84	1198911.12	7118133	12584927	3787709.64	4951266.1
	Fill in overliner (USD)	287032.68	404624.92	1257025	5381476.72	317624.38	709662.58	814789.8	1152162	447876	7922049.3	2965105	1624471.2	773984.97	1624893.49	1446288.58	707977.6	2116298.8	3212654.06	990176
	Solution collection pipes (USD)	10816.3	19603.32	158137.91	63046.1	6727.3	13771.56	156430.61	27953.71	187718.62	323040.08	251218	132838.272	63529.578	304404.293	208334.763	71867.73	136408.27	33841.04	21974.62
	Underdrain pipes (USD)	4079.57	6311.88	38834.88	189265.96	3494.8	10196.13	20391.76	9478.8	19192.28	157670.48	90756.82	15587.145	0	42253.092	48955.095	8508.456	14662.99	6311.06	14707.36
	GCL (USD)	6993.84	0	58858.8	15658.5	5124.84	146193.06	0	1584	0	233591.05	89930.5	0	0	0	166908.21	0	7560	33400.32	21758
	Geocomposite (USD)	0	0	58858.8	0	21841.58	79339.65	0	1584	0	66557.7	14608	0	0	0	105682.71	0	6899.76	29610.24	0
	LLDPE 2.0 mm geomembrane (USD)	255527.7	361171.8	1157970	220255.2	0	398475	0	0	0	0	0	0	0	0	0	173745	0	0	0
	LLDPE 1.5 mm geomembrane (USD)	0	0	0	0	772819.5	1193401	127116	354915	128601	650114.85	326518.5	520512.3	249535.44	535186.575	624299.445	191565	314401.5	291439.35	243540
	Non-woven geotextile of 270 g/m² (USD)	0	0	0	0	0	0	0	676.5	0	599710.1	270435	0	0	0	120466.5	0	0	22533	0
	Non-woven geotextile of 540 g/m² (USD)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	563.955	0	0
	Geomembrane (USD)	255527.7	361171.8	1157970	220255.2	772819.5	1591876	127116	354915	128601	650114.85	326518.5	520512.3	249535.44	535186.575	624299.445	365310	314401.5	291439.35	243540
	Liner (USD)	262521.54	361171.8	1275687.6	235913.7	799785.92	1817408.71	127116	358759.5	128601	1549973.7	701492	520512.3	249535.44	535186.575	1017356.87	365310	329425.215	376982.91	265298
SUPPLY	Solution collection pipes (USD)	18154.95	55398.3	349743.11	122966.45	35265.3	82216.2	129752.84	97127.8	182854.97	353133.2	274054	120095.282	42481.1331	150043.45	182019.647	142435.92	313871.2	178042	73221.2
	Underdrain pipes (USD)	15090.57	20867.16	124585.72	210916.53	14502	65857.05	44787.7	49693.5	44384.7	193787	156521.2	28013.1145	0	33778.41	56724.199	29194	60032	14515.2	53984.7
	GCL (USD)	50372.3	0	423923.5	106177.5	70527.56	150355.14	0	21417	0	1547892.5	595925	0	0	0	848566.95	0	77962.5	398023.6	77924
	Geocomposite (USD)	0	0	348948.6	0	87244.3	185992.95	0	23595	0	400950	88000	0	0	0	488449.5	0	77478.555	644641.5	0
	LLDPE 2.0 mm geomembrane (USD)	308395.5	435897	1397550	538401.6	0	255024	0	0	0	0	0	0	0	0	0	474259.5	0	0	0
	LLDPE 1.5 mm geomembrane (USD)	0	0	0	0	641586	990748	567596.48	1301355	574227.28	1407175	706750	1053417.75	505012.2	1083116.75	1263463.16	702405	1088760.75	1116104.3	747577.6
	Non-woven geotextile of 270 g/m² (USD)	0	0	0	0	0	0	0	1996.5	0	251556.25	113437.5	0	0	0	48914.25	0	0	70339.5	0
	Non-woven geotextile of 540 g/m² (USD)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	3177.405	0	0
	Geomembrane (USD)	308395.5	435897	1397550	538401.6	641586	1245772	567596.48	1301355	574227.28	1407175	706750	1053417.75	505012.2	1083116.75	1263463.16	1176664.5	1088760.75	1116104.3	747577.6

STATISTICAL FUNCTIONS FOR THE DEVELOPMENT OF HEAP LEACH PADS COST ESTIMATION

	Liner (USD)	358767.8	435897	2170422.1	644579.1	799357.86	1582120.09	567596.48	1348363.5	574227.28	3607573.75	1504112.5	1053417.75	505012.2	1083116.75	2649393.86	1176664.5	1247379.21	2229108.9	825501.6
	Cut in foundation (USD)	278019.5	1596020.4	6610814.1	8270753.37	1614778	2738666.7	2464154.88	15999190.9	588368.88	4934900.85	891899.8	645741.8	157848.9	3081159.62	516711.25	1690640	2759342.85	870876	2380743.2
	Cut in grading and access (USD)	435905.91	500733.84	1948199.99	744903.28	3735624.2	2866831	1331558.25	1560347.14	330269.5	6587163	4403520	880708.4	294479.13	2226633.2	764063.74	8993256	25783461.1	7805568.46	3447535.3
	Fill in grading and access (USD)	1936368.28	3958405.08	2705452.31	4835022.72	214812.4	1620863.2	411176.15	4517317.2	838844.6	307907.6	271564.15	4715961.8	2518457.92	3426592.84	1198911.12	7118133	12584927	3787709.64	4951266.1
	Fill in overliner (USD)	287032.68	404624.92	1257025	5381476.72	317624.38	709662.58	814789.8	1152162	447876	7922049.3	2965105	1624471.2	773984.97	1624893.49	1446288.58	707977.6	2116298.8	3212654.06	990176
	Solution collection pipes (USD)	28971.25	75001.62	507881.02	186012.55	41992.6	95987.76	286183.45	125081.51	370573.59	676173.28	525272	252933.554	106010.711	454447.743	390354.41	214303.65	450279.47	211883.04	95195.82
	Underdrain pipes (USD)	19170.14	27179.04	163420.6	400182.49	17996.8	76053.18	65179.46	59172.3	63576.98	351457.48	247278.02	43600.2595	0	76031.502	105679.294	37702.456	74694.99	20826.26	68692.06
	GCL (USD)	57366.14	0	482782.3	121836	75652.4	296548.2	0	23001	0	1781483.55	685855.5	0	0	0	1015475.16	0	85522.5	431423.92	99682
	Geocomposite (USD)	0	0	407807.4	0	109085.88	265332.6	0	25179	0	467507.7	102608	0	0	0	594132.21	0	84378.315	674251.74	0
TOTAL	LLDPE 2.0 mm geomembrane (USD)	563923.2	797068.8	2555520	758656.8	0	653499	0	0	0	0	0	0	0	0	0	648004.5	0	0	0
	LLDPE 1.5 mm geomembrane (USD)	0	0	0	0	1414405.5	2184149	694712.48	1656270	702828.28	2057289.85	1033268.5	1573930.05	754547.64	1618303.33	1887762.61	893970	1403162.25	1407543.65	991117.6
	Non-woven geotextile of 270 g/m² (USD)	0	0	0	0	0	0	0	2673	0	851266.35	383872.5	0	0	0	169380.75	0	0	92872.5	0
	Geomembrane (USD)	563923.2	797068.8	2555520	758656.8	1414405.5	2837648	694712.48	1656270	702828.28	2057289.85	1033268.5	1573930.05	754547.64	1618303.33	1887762.61	1541974.5	1403162.25	1407543.65	991117.6
	Liner (USD)	621289.34	797068.8	3446109.7	880492.8	1599143.78	3399528.8	694712.48	1707123	702828.28	5157547.45	2205604.5	1573930.05	754547.64	1618303.33	3666750.73	1541974.5	1576804.43	2606091.81	1090799.6
	CAPEX (USD)	3606757.1	7359033.7	16638902.7	20698843.9	7541972.16	11507593.2	6067754.47	25120394	3342337.83	25937199	11510243.5	9737347.06	4605329.27	12508061.7	8088759.12	20303987.2	45345808.6	18515609.3	13024408.1
	CAPEX (KUSD)/CAPACITY(Mt)	2003.75394	11793.32324	1230.68807	3066.4954	696.396321	513.73184	266.363234	608.831654	182.442021	840.479552	1185.40098	458.443835	670.353606	823.982985	662.470035	1837.46491	4167.81329	411.732472	2218.80887
	CAPEX (KUSD)/VOLUME (MM³)	3211.71603	18869.31718	1969.10091	5519.69171	1183.98307	873.110259	452.817498	1035.0389	310.049891	1411.93244	1991.3916	724.50499	1058.69638	1301.5673	1046.41127	3123.69034	7085.2826	733.872742	3775.19075
	Earthworks and grading (m³)	185614	482592	665093	645898	525712	1017269	309760	2181806	130350	561660	286495	576400	218218	516527	244860	3036200	5036130	1134432	1449030
	Earthworks and grading (USD)	2372274.19	4459138.92	4653652.3	5579926	3950436.6	4487694.2	1742734.4	6077664.34	1169114.1	6895070.6	4675084.15	5596670.2	2812937.05	5653226.04	1962974.86	16111389	38368388.1	11593278.1	8398801.4

Chapter 2: Operations



The Effects of Haul Roads on Ore Compaction and Wettability of Leach Pads

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Abstract

Ore stacked on leach pads can be transported by an automated conveyance system, or it can be transported manually with haulage equipment. For large haulage equipment, it is necessary to design and maintain wide roads for two-way traffic to pass safely and sometimes turn around. Road widths can exceed 30 m for a kilometer or more across the leach pad, and will be used by haul trucks, graders, dozers, water trucks, and light-duty vehicles. The cyclical and vibrational compaction of the ore from the high traffic loads can significantly impact how the ore will respond to wetting, leaching, and drainage.

To investigate the impact of haul roads across leach pads, we conducted geophysical surveys with electrical resistivity to map the bulk electrical properties of the ore where current and historical haul roads were placed. From the electrical properties, we can make inferences about hydrogeological parameters such as saturation to determine reagent mobility through the leach pad.

In our investigations, we found that haul roads permanently located through the heap life can have significant long-term effects on how the solution percolates, such as creating large swaths of unleachable ore from the top of the pad to the liner. The unleachable ore further divides the flow and drainage of the solution through the pad. Temporary haul roads also have an effect, but their damage is less severe.

Introduction

The efficiency of haulage equipment, in terms of minimal routing, safety, construction, and maintenance, is a necessary component in haul road design and many have used these variables in optimizing for costs and safety in leach pad operation (Baek and Choi, 2017; Chowdhury et al., 2021). Short, wide, and flat routes are desirable for increased tire life, and this may mean that roads traverse leach pads, not only for dumping ore but for general movement through the mine. Given that leach pad materials are not ideally suited for roads, as the bearing capacity is generally lower than those comprising hard rock as the base,

subbase, and subgrade in the mine pit (Tannant and Regensburg, 2001). In addition, the constant traffic, vibration, and heavy loads can potentially cause compaction deep within the pad. Compacted and dense material will have lower permeability (Guzman et al., 2013), which affects flow within the leach pad (McBride et al., 2017).

There are few methods for observing flow within porous media to understand compaction issues resulting from haulage equipment. Vibrating-wire piezometers measure changes in pore pressure related to wetting and drying but do not provide a means for tracking the fluid arriving at the piezometer (e.g., Ayala et al., 2013; Reyes and van Zyl, 2015). Geophysical methods that measure variability in electrical properties are well suited to observe broad changes in saturation and fluid flow across leach pads (e.g. Rucker et al., 2009a). For example, the direct-current electrical resistivity and time-domain electromagnetic methods can measure the ore's electrical resistivity (or its reciprocal, electrical conductivity) to determine if the material is generally wet or dry with low resistivity associated with wetter materials. Compacted ore with low porosity and low moisture content will typically have higher resistivity relative to the surrounding material.

This paper shows examples of electrical resistivity geophysical data across leach pads with known haul roads. In some examples, the haul roads were permanent, where the haulage equipment traversed the same geographic location as the leach pad was constructed. In others, the haul road was temporary and either covered with ore or still in use during acquisition of the electrical resistivity data. In each case, high resistivity features were coincident with the road's location and low resistivity could be seen on either side of the road. From a hydrogeological perspective, it appears that the roads can create a divide through the leach pad and segregate leachate from flowing uniformly. From a metallurgical perspective, the compacted areas left behind from the roads represent a significant quantity of material that is underleached or unleached. Our recommendation to minimize impacts from haulage equipment is to ensure adequate deep ripping of roads when new ore is placed and verify that historical roads are not impacting the wettability of the ore.

Haul roads across leach pads

For a truck-dump operation, haul roads are necessary to place ore for leaching. Given their size, the roads are easily viewable in aerial photographs from services such as Google Earth. Figure 1 shows several leach pads over time with haul roads highlighted in yellow. The top row of Figure 1 shows a series of photographs of a run-of-mine gold leach pad between 2011 and 2016 with 15 meter lifts. Some of the roads tend to persist over the duration of the construction, while others are temporary. The middle row, with aerial photos from 2011 to 2017, shows a gold mine of crushed ore. The access to the heap stayed practically at the same location throughout this period. The bottom row is a copper crushed ore leach pad from 2010 to 2018. A couple of main roads remain for the duration, while others are temporary.



Figure 1: Aerial photos from Google Earth. Haul roads are outlined in yellow
Top row – gold mine with temporary haul roads from 2011 to 2016;
middle row – gold mine with permanent haul road from 2011 to 2017;
bottom row – copper mine with temporary haul roads from 2010 – 2018

Haul roads cover a significant portion of standard mining operations, meaning that the relative volume of the potential area and ore tonnage that haul roads may impact can be quite large. Haul roads across leach pads are designed to have two-way traffic for the efficient dumping of ore, and the recommendations for the road width are shown in Figure 2. Considering that a particular haul truck has a width of X , the total road width (W) for a given number of lanes (L) is (Tannant and Regensburg, 2001):

$$W = (1.5L + 0.5)X \quad (\text{Equation 1})$$

For a CAT 797 with a width of 9.15 m, the total two-lane road width can be 32 m. If we make additional assumptions that the ore density increases to 2,000 kg/m³ at a depth of 15 m (the haul road has been in place during the placement of several lifts), the impacted tonnage of ore is approximately one million metric tonnes per kilometer of haul road or approximately 2% of a lift comprising an area of 200 hectares. This relative mass of ore will have low permeability with likely little chance of leaching metal from the stock. To demonstrate the effects of compaction on haul roads and their impact on wetting the ore beneath the roads, we used electrical resistivity geophysics to image the spatial distribution of electrical resistivity, and we then inferred the degree to which the ore wetted relative to surrounding material.

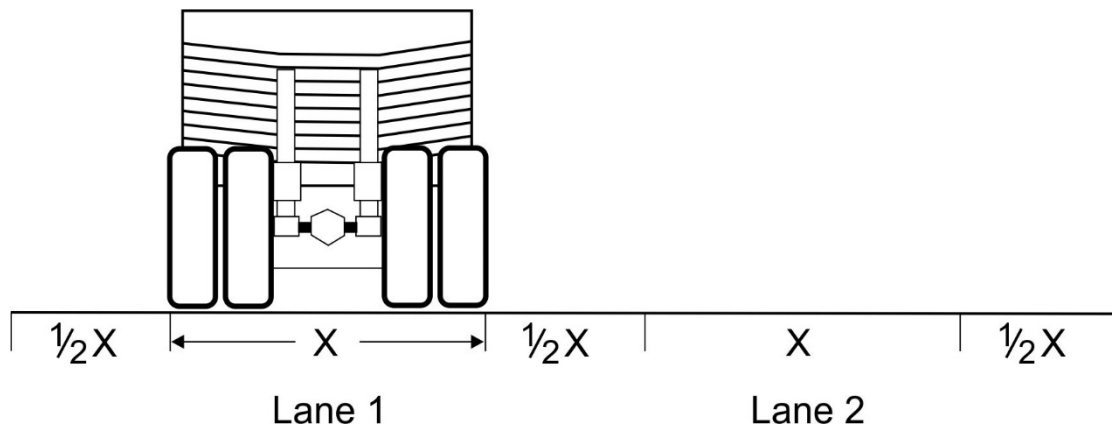


**Figure 2: Haul road design for accommodating road widths
(adapted from Tannant and Regensburg, 2001)**

Description of electrical resistivity geophysics

Electrical Resistivity Tomography (ERT) comprises both field data and numerical models. To acquire ERT data, DC electrical current is transmitted through the ground via stainless steel electrodes. The electrodes are driven into the ground at regular intervals and connected to a resistivity meter through a cable. Electricity flows by way of an electrolytic conduction mechanism. The resistivity meter controls which pair of electrodes are to pass current at any given time. At other electrodes along the line, the receiving voltage is measured, and data for a full profile of data is acquired by sequencing between many possible electrical current transmitting electrode pairs and voltage receiving pairs. The specific sequencing of pairs is called an array; the Schlumberger and Alt3 Wenner arrays (Cubbage et al., 2017) were primarily used to characterize the haul roads. A more thorough description of electrical resistivity methods can be found in Loke et al. (2013).

Automated inverse methods applied through numerical models convert the measured current and voltage data to an estimate of the electrical resistivity structure (Rucker and Glaser, 2015). The inverse method relies on nonlinear optimization, requiring an iterative procedure to march towards a solution. Its objective is to minimize the difference between the modeled and measured data, usually in a weighted least-squares sense. The objective function has been updated many times over the years to also include other terms, such as smooth model constraints (i.e., a smooth model based on minimizing the second spatial derivative of the resistivity) and known boundaries such as those posed by liners beneath a leach pad.

Results of electrical resistivity profiling

Leach pad 1

The first electrical resistivity example is shown in Figure 3. The line was collected on an 80 m deep gold heap with crushed ore using a 6 m electrode spacing and the Schlumberger array. The haul road was in a permanent location throughout the heap's construction and operation, and it is unlikely that any surface preparations were conducted to remove and prepare the area for leaching. The figure shows a range of resistivity values, with the highest values associated with the under-liner that collects the pregnant leach solution at the bottom of the pad. Because the liner is an electrically insulating boundary, its resistivity values are greater than 10,000 ohm-meters (or 4 as a logarithmically transformed value). The lowest resistivity values are shown to be associated with areas of active leaching in the left and right halves of the figure. Between the active leach areas is a vertical, high resistivity dry area that coincides with the haul road. The road is about 35 m wide.

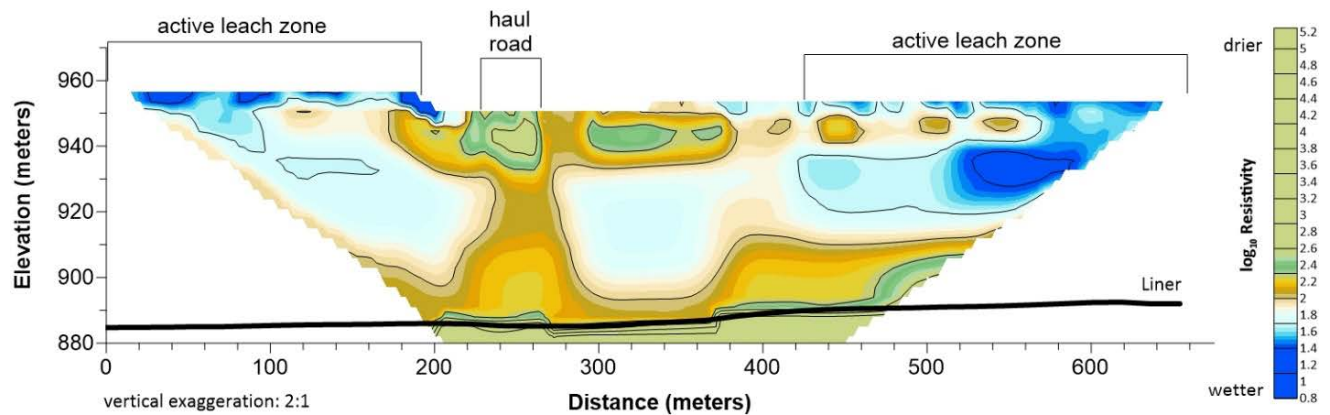


Figure 3: Electrical resistivity acquired across a crushed gold heap with a permanently placed haul road

When considering the shape of the leach pad, we estimate that four million metric tonnes of material have been impacted by haul roads, where little to no leaching can occur. Historical documents showed a mean gold grade of 0.3 g/t with 75% recoverability. It is estimated that up to 27,000 ounces are trapped in

the volume of material affected by the haul road when extrapolating these numbers across the affected haulage area. This is a significant amount, but likely represents the high end of the estimate.

Leach pad 2

In Figure 4, we show an electrical resistivity section acquired across a ROM gold heap roughly 60 m deep. The data were acquired with a Schlumberger array with 6 m electrode spacing. Across the top of the section, we have identified two types of roads: haul roads and light-duty (LD) roads. There is high resistivity material that is likely compacted beneath each of the four roads. This is contrasted with material actively being leached shown as low resistivity. Moving from left to right, the first LD road shows a small footprint as expected, with a limited depth of impact at 16 m. The road is temporary, in existence for a couple of lifts. Being a haul road, it does not have active irrigation that could contribute to high resistivity values.

The next haul road to the right is relatively narrow, accommodating two lanes of traffic with a width of 28 m. Its impact depth is noticeably greater at 25 m. The second haul road to the right is extremely wide at 75 m and likely accommodates several lanes. The road was created shortly after construction began. We consider this a semi-permanent haul road that is unlikely to be removed in the future. The last LD road is between the large haul road and the edge of the heap. Similar to the first LD road, its impact depth is relatively shallow.

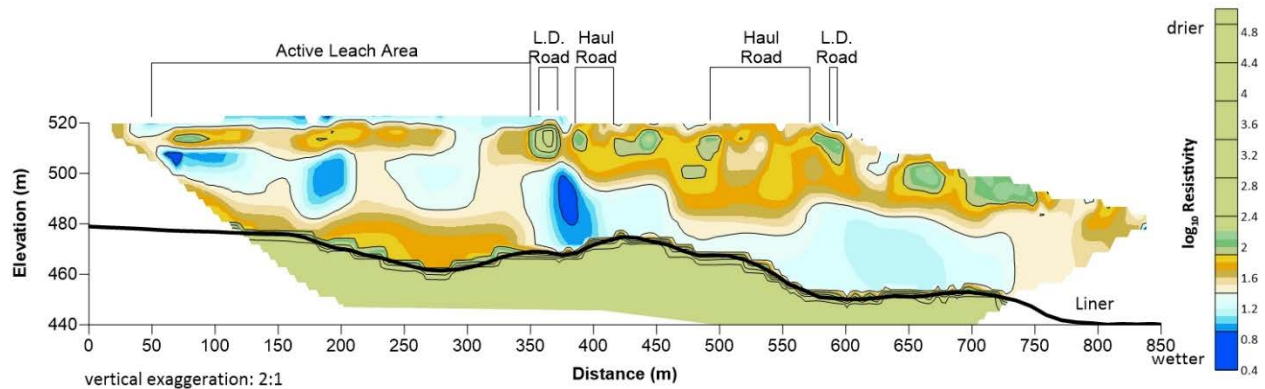


Figure 4: Electrical resistivity acquired across a ROM gold heap with a semi-permanently placed haul road, temporary haul road, and light-duty roads

Performing a similar exercise on the potential gold inventory remaining in the pad beneath haul roads, the smaller of the two haul roads is estimated to have affected a mass of 840,000 tonnes at an estimated stacked grade of 0.16 g/t at 50% recoverability, equating to nearly 2,000 oz. The larger haul road shows an approximate mass affected by haulage at 3.3 million metric tonnes, equating to 8,200 oz. Together, both roads represent a significant inventory for the mine that should be considered for some type of secondary recovery operation that can overcome the mechanical strength of the compacted materials. This would include either physical removal or high-pressure injection (e.g. Rucker et al., 2009b).

Leach pad 3

In the third example, we investigated a copper leach pad with crushed ore placed atop native bedrock without a liner. Figure 5 shows the electrical resistivity line placed over the top of the heap where a semi-permanent haul road was buried under a new lift and irrigated with leachate. The data were acquired with a 6 m electrode spacing using the Alt3 Wenner array (Cubbage et al., 2017). The location of the haul road is obvious in the electrical resistivity data as the highly resistive haul road material bisects two areas of the leach pad with conductive material. The buried haul road has rough dimensions of 70 m width and 50 m height, extending for 200 m in length across the pad. With an acid soluble copper grade of 0.2%, this equates to nearly 2.8 million kilograms of copper lost to extraction from surface leaching.

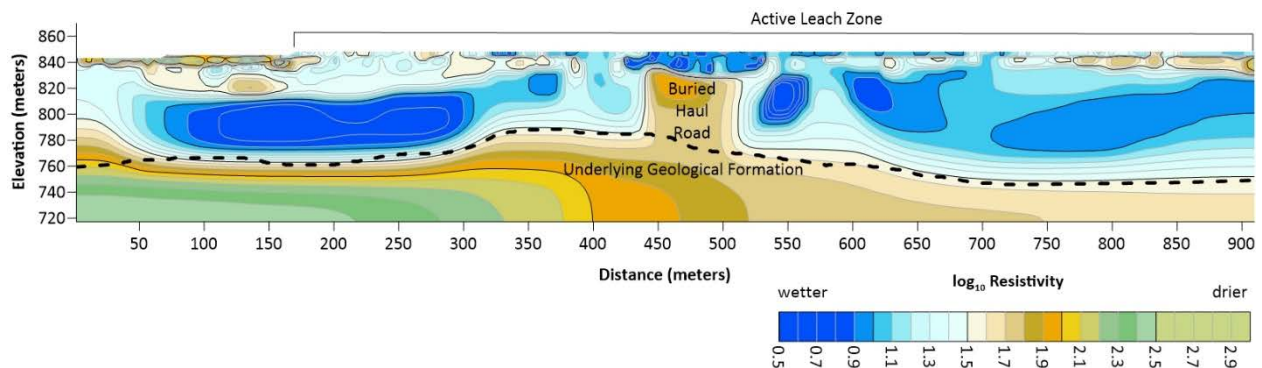


Figure 5: Electrical resistivity acquired across a crushed copper ore heap with a buried historical haul road

Conclusion

Haul roads across leach pads were investigated with electrical resistivity geophysics to determine the impact of compaction on the efficacy of surface irrigation. Haul roads can be large to accommodate wide haulage equipment, and if not routinely moved can have impacts extending deeply within a leach pad. The effects of haulage equipment were obvious in the electrical resistivity data because their signatures were of consistently higher resistivity relative to the surrounding material. The high resistivity is hypothesized to result from compacted ore and lower saturation. When multiplying the tonnage of ore that is impacted by compaction by the specific grades across several leach pads, we estimate that millions of dollars' worth of metal inventory is left behind.

Based on our findings, we recommend that electrical resistivity geophysics be conducted routinely over haul roads to determine the significance of their impacts to the operation of the leach pad. The method can also test the effectiveness of existing ripping operations by acquiring data before and after haul road removal. Lastly, if the haul road has made deep impacts into the leach pad, below what is feasible for

ripping, then material removal or injection may be the only methods available for inventory drawdown (e.g. Rucker et al., 2017).

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Wireless Heap Leach Monitoring and Control

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Abstract

Heap Leach solution extraction is a cost-efficient process for recovering metals in various ores, including copper, gold, and uranium. While the concept is easy to understand, until now it has not been possible to execute flow control and monitoring over the entire operation. Traditionally a single flow meter has been used to measure flow to the pad, while monitoring of the distribution of the solution on the pad and through the pad has been lacking. This means that accurate solution distribution on the pad requires labour-intensive operations to regulate valves and check for leaks and pipe breaks, and assumes no “ponding” within the pad during daylight hours. Instrumentation capable of withstanding the rigours on the pads exposed to weather, sun, and corrosive solutions, and able to communicate process variables reliably and accurately, has been missing.

Using WirelessHART instrumentation, Emerson’s complete solution has been able to measure and control the flow of extractive solutions to the pad, the distribution on the pad, pore pressure and temperature in the pad, and flows off the pad. Activation of valves wirelessly, and monitoring and controlling vertical wells and air blowers have all been accomplished with wireless instrumentation. WirelessHART has been used for communication to create a mesh network capable of accurately measuring and transmitting process variables and diagnostics. Hence, the operation of leach pads can now be controlled and monitored 24/7, making leach pads safer.

Beyond the obvious productivity and reporting advantages, the ability to monitor and control around the clock improves health and safety. It also bolsters the environmental stewardship that is so necessary in today’s world, where social license is required to operate. Having knowledge of all aspects of a leach pad’s operations in real time brings a mine operations into the 21st century.

Introduction

Heap leaching is an efficient process for recovering metals in various ores including copper, gold, and uranium. Leach pad instrumentation with WirelessHART pressure and flow meters has made a difference with more efficient operations, and has improved safety and health. It has safeguarded the environment by enabling the measurement of the actual usage and distribution of raffinate, reduced time spent on the pad

by employees, and enabled operations to be alerted to hazardous situations. This paper will focus on copper heap leaching, but the lessons learned are transmissible to other leaching metals.

While the concept of heap leaching is easy to understand, until now it has not been possible to execute flow control and monitoring over the entire operation. Traditionally, wired flow meters have been used to measure solution flow to the pad and flow from the pad, with everything in between left to guesswork and computer models. Distribution on the pad and other process variables have been unknown, or extrapolated from just a few measurement points. Accurate flow and pressure distribution on the pad required labour-intensive operations to regulate pressure, adjust valves and flows, and note the pressure or flow rate on a leach pad that could contain many square miles of surface area. Checking for leaks, broken pipes, and plugged pipes can only be conducted during daylight hours; even then, it exposes employees to safety, health, and environmental risks. Until now we have not had instruments capable of withstanding the harsh environment on the pad, exposed to weather, sun, and corrosive solutions, yet able to reliably communicate process variables accurately. With WirelessHART, flow and pressure in a single device transmits process variables every minute.

Heap leaching

Heap leaching is defined as the use of a solution to extract desired metals from an ore that has been crushed to a degree and stacked. Various solutions have been used, depending on the metal being recovered, with sulfuric acid the choice for extracting copper from copper ore.

Copper heap leaching uses a weak sulfuric acid (raffinate) to free copper from the surrounding ore in the leach pad. The solution is pumped to the top of the pad, and then spread over the surface using sprinklers (wobblers) or drip lines. From the surface the raffinate percolates down through the pad, picking up copper along the way. Finally it exits the pad at the bottom, and then pools in ponds. The copper-laden solution is now pregnant leach solution (PLS).

The PLS has made its way through the pad and picked up dissolved copper. Next the PLS is concentrated and enriched in the solvent extraction process. In this process, the PLS is mixed with an organic solution, which pulls the copper out of the PLS. The PLS is then recirculated back to the leach pad as raffinate. The next process is the reverse of the first. The organic solution is mixed with a weak electrolyte, and the copper is pulled into the weak electrolyte, creating a concentrated copper solution, which is now termed rich electrolyte. This is conveyed to the tank house for electro winning.

With the introduction of WirelessHART flow meters and pressure transmitters to the pad distribution system, many benefits can be realized, including more efficient operations, improved safety and health, and enhanced environmental stewardship.

Efficient operations

The ability to know what the flow rates are throughout the pad, and the pressures that they are operating at, allows for improved management and more efficient operations. Some ores may require more solution due to acid-neutralizing minerals, and the flows can then be directed accordingly. Some areas may need flows reduced due to ponding, and again that can be corrected and monitored. Over time, a historian can trend the application rates and areas and then compare them to expected yield. When yields do not match forecasts, it is possible to review flows and pressure over any time period, even back to when they were first installed. This can help to identify errors in applications. Conversely, when no issue is found, it becomes evident that it is necessary to look elsewhere for the shortfall. Without data to work from, forecasting can be little more than an educated guess.

Efficiencies are further enhanced if flow and pressure are known; then plugged lines can be spotted and corrective action can be taken long before suboptimal leaching occurs. Conversely, if enough solution has reached the area to meet targets, then plugged lines may be allowed until fresh ore is stacked.

Safety and health

Typically, the distribution system can only be evaluated for integrity and correct flow and pressure, and then manually adjusted, during the day when employees are on the pad. Once the sun sets or inclement weather arrives, the pad is unmanned and without monitors. This is where WirelessHART instrumentation can fill in the gap and provide monitoring of essential parameters like flow and pressure. By relaying pressure and flow to operations, the instruments can detect problems like a ruptured pipe, which could lead to destabilizing events (erosion, pad stability, etc.). Quickly identifying the issue can enable the initiation of a process to stop flows and minimize damage.

Adding instrumentation to the pad means that fewer working hours will be required for routine inspections. Looking for pipe breaks, plugged drip emitter walkdowns, ponding, or dry areas requires more employees on the pad, which means required PPE and exposure to the elements and other hazards. Sun and heat, or storms and lightning, are hazards that can impact a crew on the pad year-round. Similarly, uneven pad surfaces can result in tripping hazards and injuries, and lost time. Lastly, due to the ore mineralization, hazardous gas may be present. This needs to be monitored, but exposing crews to even low levels of gas should be avoided.

Environmental hazards

Pipes, from small drip lines to larger distribution lines, crisscross a typical leach pad and can pose an environmental risk – from small leaks to catastrophic failures. Leaks from broken lines can run off the pad

and off the property. This can result in fines and expensive clean-up work. If the operation is relying on reports from human observers, a large line that ruptures may not be dealt with quickly enough to prevent damage to the pad, or to prevent runoff from going into national waters. Picking up errors quickly with instrumentation enables a faster response and therefore ensures less damage.

Perhaps one of the most serious leach pad events is liquefaction of a portion of the pad, which can result in damage, lost equipment, injuries, and even death. Liquefaction is the sudden loss of cohesion of solid ground, which turns into flowing mud due to the pad being saturated. Liquefaction can happen suddenly and without warning, endangering everyone on the pad and in the vicinity. The change can be a result of high levels of solution, or lower levels of solution in the pad if triggered by an earthquake. The time and effort to repair the pad and get back into production means lost revenue and redirected resources.

WirelessHART

The mining operations that have installed WirelessHART instrumentation have benefited from better operational visibility, fewer crews exposed to hazards, and better environmental outcomes. WirelessHART is similar to the wired HART version that has been in use in the industry since the 1980s. There are an estimated 40 million wired HART devices installed worldwide, in industries as diverse as chemicals and refining to food and beverage, pharmaceuticals, and oil fields. Because of this, crews are familiar with how HART operates. The Wireless version is an extension of this already known protocol.

Emerson has developed over 65 different WirelessHART instruments to monitor all areas including flow meters, pressure transmitters, temperature transmitters, discrete switches, and more. The WirelessHART network uses the 2.4 GHz ISM frequency, and is an open protocol that is managed by the FieldComm Group. All WirelessHART devices have robust, multi-tiered, always-on security, which is provided through advanced standards-based encryption, authentication, verification, key management, and anti-jamming techniques. Security is built-in and cannot be switched off.

The field device that is the center piece for leach pad flow and pressure measurement is the Emerson 3051SMV WirelessHART differential pressure (DP) flow meter. Using advanced electronics to measure the differential pressure across an orifice plate or Annubar, flow is calculated and then transmitted. There is also a second sensor in the unit that measures static pressure. These two variables, flow and pressure, are transmitted back to a Gateway, where it is received and then sent directly to the mining operations process control. The 3051SMV flow meter is a unique instrument, and when mounted to the 405C Conditioning Orifice Plate presents a capability unmatched by any other wireless instrument. The 405C conditioning orifice plate features a patented design that enables accurate measurement with reduced pipe straight run. This orifice plate is compact and easy-to-install, and features self-centering mechanisms for optimal

accuracy. This combination of the 3051SMV and 405C then gives a 1 – 2 punch that is unmatched in industry; flow and pressure delivered via WirelessHART



Figure 1



Figure 2

Summary

What then have been the results? To date, several mines in the USA have installed multiple flow meter/pressure transmitter units on their pads, turning what was a data desert with no information into a data avalanche. Starting twelve years ago, these mines have increased their recoveries, reduced negative events related to safety, health, and the environment, and improved water management and process transparency.

Before the introduction of WirelessHART instrumentation on leach pads little data was available to evaluate strategies and leaching campaigns. WirelessHART has added thousands of data points daily and, over time, millions of data points, allowing timely evaluation of adherence to leaching plans. This process transparency is only possible with timely and accurate data, from which conclusions can be drawn.

The ability to control application rates and distribution and get that data back is central to process transparency. The WirelessHART networks have the security and reliability to deliver process data with integrity. This results in higher recovery rates, reduced adverse events in safety, health, and environment, and better water management.

Ore Permeability and Blending Evaluation to Optimize Leach Performance

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Abstract

Efficient metal recovery from heap leaching requires good ore permeability. Depending on the ore type, consolidation under increasing heap heights may significantly decrease ore permeability, thereby causing increased leach times, incomplete recovery, and reduction of the economic value of the process. Physical, chemical, and hydraulic property laboratory screening tests were conducted on potassic and hydrothermally altered (argillic) ore samples from Round Mountain Gold Corporation's Gold Hill deposit in order to determine the solution permeability of the ore samples under increasing ore consolidation and controlling factors (e.g., particle size distribution and clay types).

Results were applied to evaluate the suitability of each ore type for standard heap leach operations and, for low permeability ores, identify appropriate ore blend ratios to maintain sufficient permeability. Correlation analyses indicated that "PSD Indicator" values may be used to predict permeability and blending requirements. Leach pad performance indicates that the test program properly defined blending ratios and successfully mitigated poor permeability ores.

Introduction

The Round Mountain Gold Corporation's (RMGC) Round Mountain mine is located in south-central Nevada approximately 55 miles north of Tonopah (Figure 1). In 2011 RMGC expanded the operation with the addition of a satellite deposit about two miles to the north, known as Gold Hill. The Gold Hill resource is a quartz-adularia low-sulfidation hydrothermal deposit hosted in volcanic ash-fall tuffs. Mineralization is fine grain (<50 microns) with electrum, silver sulfides, and sulfosalts in veins and veinlets. Whereas the unoxidized portions of the deposit have low precious metal recovery, the upper portions of the deposit have been oxidized, rendering the ores in these areas amenable to conventional cyanide heap leach methods.

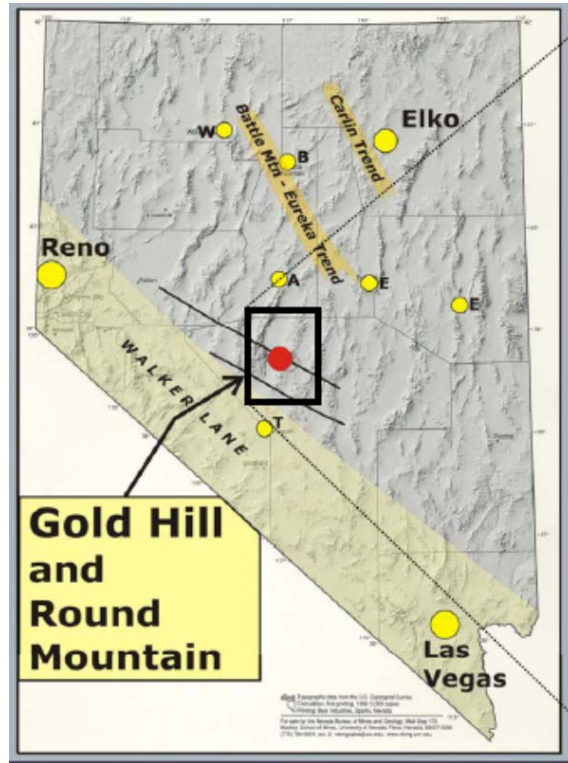


Figure 1: Location of Round Mountain Gold Corporation in Nevada

Gold Hill leach pad loading began in August of 2012 and conventional run-of-mine (ROM) heap leach cyanidation is used to process the ore. Almost immediately, percolation problems were encountered on the leach pad at the design irrigation rate of 0.003 gpm/ft². To prevent ponding, the application rate had to be reduced by half, which slowed the recovery rates and resulted in production not meeting budget.

In addition, the lift height was reduced from 50 ft to 25 ft to increase the gold production rate as much as possible. The reduction in irrigation rates and lift heights caused the mining rate to be cut in half, essentially doubling the mine life. Several studies were initiated to characterize the permeability problem and find solutions, including the completion of physical, chemical, and hydraulic property laboratory tests on the ROM ore to determine the solution permeability of the ore samples and blended ore samples to evaluate their suitability for standard heap leach operations.

Methods

Laboratory tests

ROM samples were selected by the owner and shipped to a third-party testing laboratory located in Tucson, Arizona, for testing. The ore samples originated from a single lithologic unit; however, they were categorized under three alteration ore types: argillic (A), advanced argillic (AA), and siliceous potassic (SP). Three non-leached ROM samples from each alteration ore type were tested to capture the expected

variability within the ore types, in addition to three ore blends to evaluate blending strategies for low permeability ores. Table 1 lists the tests and standard methods that were conducted on the ROM ore samples. The A and AA samples were also blended at different ratios to determine whether hydraulic properties could be improved via blending; blend samples underwent hydraulic testing only.

Physical property screening tests included:

- Particle size distribution (PSD) of ore samples to determine the material size fractions.
- Particle density to determine the mineral density distribution.
- Atterberg limits to estimate soil plasticity.

Chemical property screening tests included:

- XRD analysis to determine the clay mineral composition of the ore samples.
- XRF analysis to determine the relative contents of major elements in the ore samples.

Hydraulic property screening tests were comprised of saturated hydraulic conductivity (K_{sat}) testing in 12-inch diameter by 12-inch-high dual-wall permeameters (Figure 2) over a range of bulk densities representative of various heap heights from the top of the heap to approximately 80 ft below the heap surface to estimate changes in K_{sat} under varying levels of overburden pressure (i.e., heap heights). The dual wall permeameter provides a more even distribution of pressure to the ore compared to the vertical pressures applied in a rigid wall (uniaxial) consolidation permeability test. The permeameters were packed with samples with a maximum particle size of 2.0 inches (51 mm).

Table 1: Laboratory tests and test methods

Test Type	Test method	Standard
Physical	Particle size distribution	ASTM D422–63, C136-03
	Particle density	ASTM D854-02
	Atterberg limits	ASTM D4318
Chemical	X-ray diffraction	MOSA Part 5, Chapter 4
	X-ray fluorescence	MOSA Part 5, Chapter 14
Hydraulic	Consolidation permeability (K_{sat})	ASTM D5084-03

(Sources: ASTM, 2009; MOSA, 2008)

Criteria for desirable ore permeability characteristics were established to include:

- A measured K_{sat} greater than 100 times the owners; forecast irrigation rate of 0.003 gpm/ft² (2.0×10^{-4} cm/s) for estimated heap heights up to approximately 80 feet. A safety factor of 100 is applied to account for variations in ore, irrigation rate and frequency, precipitation, and topographical variation in the heap.

- Absence of significant amounts of swelling (i.e., montmorillonite, smectite) type clays.
- Lack of significant ore consolidation under pressures mimicking increasing heap heights.

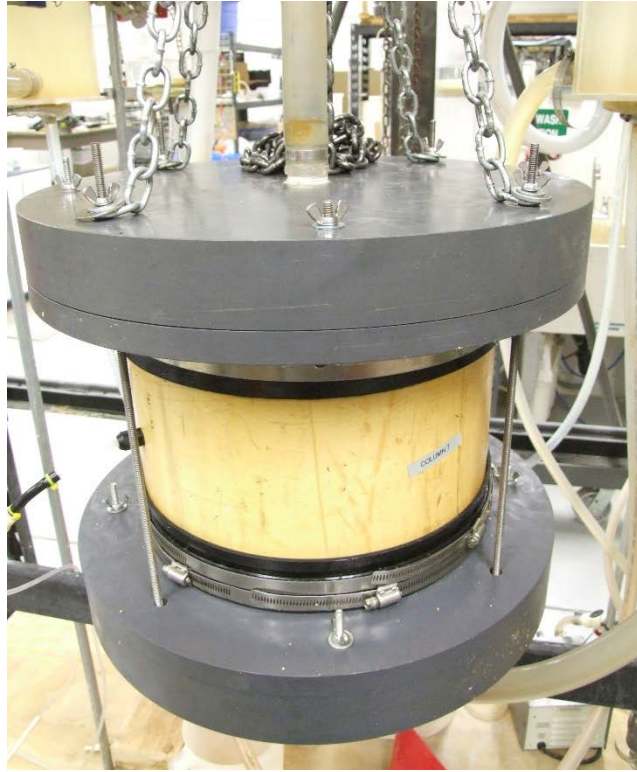


Figure 2: Dual-wall permeameter

Correlation analysis

Consolidation permeability testing of ROM ore material provides the best means of characterizing sample solution K_{sat} , but these tests require specialized equipment and highly trained staff, and therefore are expensive and time consuming. To determine whether lower cost physical property tests could be used to estimate K_{sat} , correlation analyses were conducted between the hydraulic property and physical and chemical property test results to determine parameters that may be used to help characterize permeability.

Correlation analyses were conducted by using the best fit functions (power or exponential) to the relationships between K_{sat} values at estimated heap heights of 20 ft (near the bottom of a first lift) and the following sample parameters:

- percent clay
- percent passing the #100 mesh
- PSD indicator
- clay activity from Atterberg limits testing
- total sample percent smectite from the XRD results.

Results

Laboratory tests

Particle size distribution

Table 2 provides the percent: gravel (larger than the #4 mesh [4.75 mm]); passing #100 mesh (0.15 mm); silt and clay (passing the #200 mesh [0.075 mm]), and clay (<0.002 mm) for the nine ore samples. Table 2 also provides the estimated PSD for the blended samples calculated from the weighted average PSD of the ore material comprising the blend samples. AA ore material was the finest grained sample with 14.7 to 28.4 percent of material classified as silt or finer and only 6.0 to 38.0 percent of the material classified as gravel. SP ore material was the coarsest ore sample, with 7.9 to 8.4 percent of the material classified as silt or finer and 71.0 to 78.0 percent of the material classified as gravel. In terms of both gravel and silt and clay content ranges, the A ore material was in between the AA and SP samples.

Table 2: Percent gravel, passing #100 mesh, silt and clay, and clay and PSD indicator

Sample	Gravel (>4.75 mm, %)	Passing #100 Mesh (0.15 mm, %)	Silt and clay (<0.075 mm, %)	Clay (<0.002 mm, %)	PSD indicator (% Gravel / % Passing #100 mesh)
SP-1	71	9.1	8.3	4.1	7.8
SP-2	78	9.1	8.4	3.8	8.6
SP-3	72	8.8	7.9	2.5	8.1
SP average	73.7	9	8.2	3.5	8.2
A-1	43	9	8	3.2	4.8
A-2	52	12.5	11.3	5.1	4.2
A-3	58	10.9	9.9	4.4	5.3
A average	51	10.8	9.8	4.2	4.7
AA-1	38	16	14.7	7.2	2.4
AA-2	6	32.8	28.4	12.1	0.2
AA-3	17	22.6	20	6.8	0.8
AA average	20.3	23.8	21.1	8.7	0.9
75% A-1/25% AA-2	33.8	14.9	13.1	5.4	2.3
75% A-1/25% AA-1	41.8	10.7	9.7	4.2	3.9
75% A-2+A-3/25% AA-3	45.5	14.4	13.0	5.3	3.2

Of the blend samples, sample 75% A-1/25% AA-1 had the least amount of material classified as silt or finer (9.7%), while samples 75% A-2+A-3/25% AA-3 and 75% A-1/25% AA-2 had nearly the same amount of silt and clay material (approximately 13%).

The calculated PSD indicator values are presented in Table 2. The PSD indicator is the ratio of the percentage of gravel (>4.75 mm) to material passing the #100 mesh (0.15 mm), and can be used as a

potential index of ore permeability with the logic that gravel-sized particles influence macropore solution flow and finer-sized particles influence material consolidation and matrix pore solution flow. The PSD Indicator ranged from 0.2 to 2.4 for the AA ore samples, to 7.8 to 8.6 for the SP ore samples, with intermediate values of 4.2 to 5.3 for the A samples. Blend sample PSD indicator values ranged from 2.3 to 3.9.

Particle density and Atterberg limits

Particle density and Atterberg limit results for the ten ore samples are presented in Table 3. Particle density values ranged from 2.66 g/cm³ for the A-3 sample to 2.80 g/cm³ for the SP-2 sample. The SP ore samples had the highest particle densities, whereas the A and AA samples were similar.

The clay activity is calculated as the plasticity index divided by the percent clay in the Atterberg limits test sample. It provides an indication of the tendency of the ore to shrink and swell with changes in water content. Clay activity greater than 1.25 indicates soils with high shrink and swell tendencies (i.e., smectite type clay). Based on Atterberg test results, all samples had low shrink and swell tendencies.

Table 3: Particle density and Atterberg limits test results

Sample ID	Particle density (g/cm ³)	Liquid limit (LL)	Plastic limit (PL)	Plasticity index (PI)	Clay activity	Clay type
SP-1	2.72	44	21	23	0.65	Inactive
SP-2	2.8	37	22	15	0.43	Inactive
SP-3	2.73	27	23	4	0.19	Inactive
A-1	2.68	39	24	15	0.57	Inactive
A-2	2.68	41	22	19	0.59	Inactive
A-3	2.66	44	23	21	0.67	Inactive
AA-1	2.67	45	25	20	0.55	Inactive
AA-2	2.69	50	30	20	0.73	Inactive
AA-3	2.67	44	29	15	0.67	Inactive

XRD clay speciation

XRD analysis results (Table 4) indicate that swelling type smectite clays, which can greatly reduce hydraulic conductivity in small amounts, were present in largest quantities in samples AA-1 (4%) and AA-2 (8%). All other samples had 2% or less smectite clays. The low shrink-swell potential of samples AA-1 and AA-2 (Table 3) may be due to the large gravel fraction present in the ore material. In general, Atterberg limits calculated clay activity increased with percent smectite type clays present in the ore material.

Table 4: Clay fraction X-ray diffraction results

Sample ID	Clay (<0.002 mm)	Percent of total sample			
		Illite	Kaolinite	Smectite	Palygorskite
SP-1	4.1	1	1	2	ND
SP-2	3.8	2	2	<1	ND
SP-3	2.5	1	1	<1	ND
A-1	3.2	<1	<1	2	ND
A-2	5.1	2	2	1	ND
A-3	4.4	1	1	2	ND
AA-1	7.2	1	2	4	ND
AA-2	12.1	2	2	8	ND
AA-3	6.8	3	2	2	1

Consolidation permeability

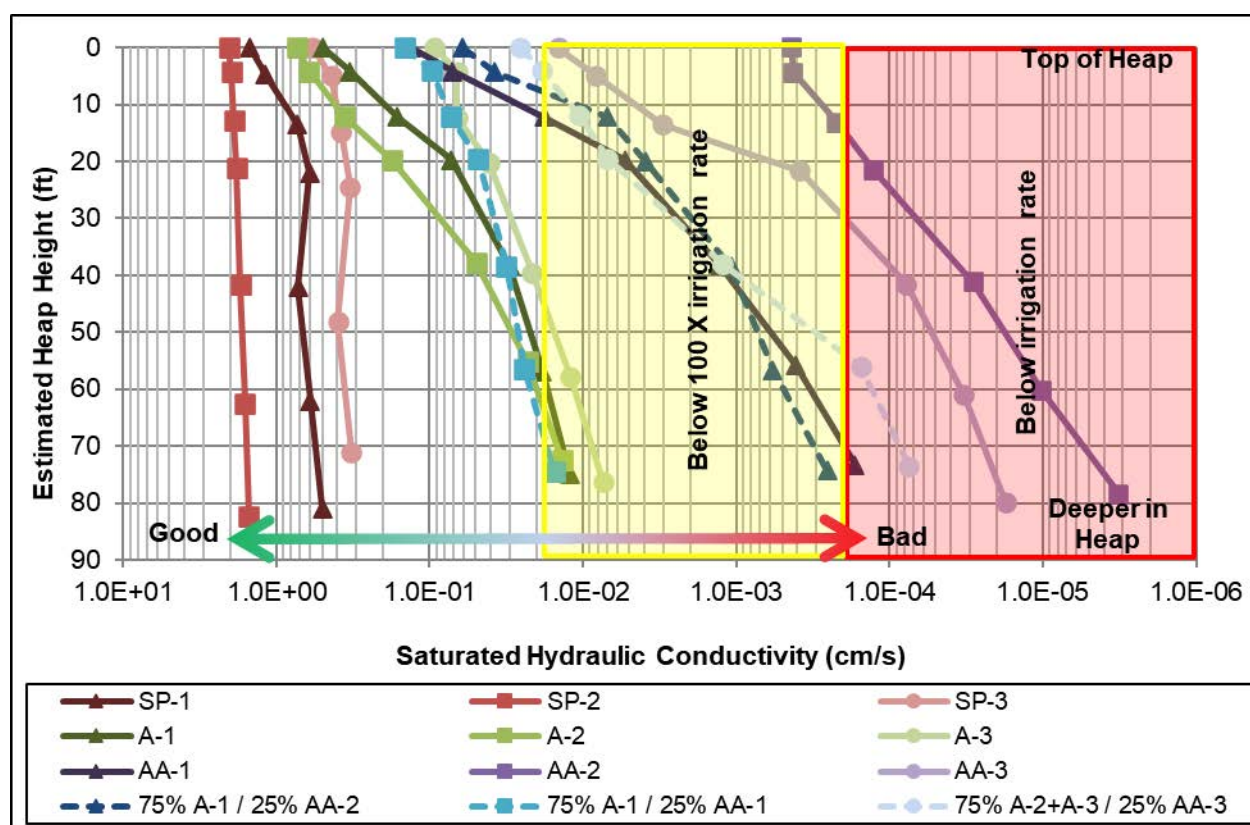


Figure 3: Consolidation permeameter measured K_{sat}

Consolidation permeameter measured K_{sat} values, reported as a function of calculated heap height, are presented in Figure 3. Measured K_{sat} values for the A ore samples decreased by approximately one to 1.5 orders of magnitude as simulated heap heights increased from zero to approximately 75 feet. AA ore

measured K_{sat} decreased two to three orders of magnitude as the simulated heap height increased from zero to approximately 80 feet. In contrast, measured K_{sat} for the SP material decreased only slightly as heap height increased, reflecting the coarseness of these samples and relatively small amount of fines fraction. K_{sat} trends for blend samples 75% A-1/25% AA-2 and 75% A-2+A-3/25% AA-3 decreased by approximately 2 orders of magnitude from zero to 75 feet while blend sample 75% A-1/25% AA-1 decreased by only one order of magnitude from zero to 75 feet.

The red box in Figure 3 indicates K_{sat} values lower than the forecast irrigation rate of 0.003 gpm/ft² (2.0×10^{-4} cm/s); the yellow box indicates K_{sat} values lower than the 100X safety factor (0.3 gpm/ft², 2.0×10^{-2} cm/s), but greater than the forecast irrigation rate. The consolidation permeability results can be summarized as follows:

- All of the SP samples had greater K_{sat} values than 100× the forecast irrigation rate for all heap heights evaluated. These data indicate the SP material is sufficiently permeable under all anticipated heap conditions.
- The A samples A-1 and A-2 had K_{sat} values above the 100× safety factor at simulated heap heights less than 60 feet; sample A-3 K_{sat} was above the 100× safety factor at heap heights less than 40 feet. These data indicate the A material is sufficiently permeable under conditions of two to three 25-foot lifts.
- The AA samples AA-2 and AA-3 had K_{sat} values below the 100× safety factor for all heap heights tested and below the forecast irrigation rate at heap heights greater than 13 and 30 feet, respectively. Sample AA-1 K_{sat} was below the 100× safety factor at simulated heap heights above 12 feet, and below the forecast irrigation rate at 70 feet of estimated heap height. These results indicate that all AA materials are likely to experience problems with permeability at the forecast irrigation rate under first lift conditions and that solution percolation and irrigation rates should be carefully monitored for the AA ore materials.
- The 75% A-1/25% AA-2 blend sample had K_{sat} values above the 100× safety factor at simulated heap heights less than 60 feet and K_{sat} values above the forecast irrigation rate at all heap heights up to 75 feet. Blend samples 75% A-1/25% AA-1 and 75% A-2+A-3/25% AA-3 had K_{sat} values below the 100× safety factor at simulated heap heights exceeding approximately 5 feet. These results indicate that blending at a proportion of 75% A material and 25% AA-1 material results in ore that is sufficiently permeable under conditions of two lifts. Blending at a proportion of 75% A material and 25% AA-2 or AA-3 material results in an ore that is likely to experience permeability problems at the forecast irrigation rate under first lift conditions.

Correlation analysis

The parameter with the highest correlation to K_{sat} results was the PSD Indicator ($R^2 = 0.83$) (Figure 4). A strong correlation between K_{sat} and percent passing the #100 mesh, percent clay, percent smectite, and clay activity was not observed, producing R^2 values of 0.57 or less. The correlation analyses indicate that the PSD indicator may be used as an indicator of sample permeability. For example, from these relationships a ROM ore with a PSD indicator less than 4 may experience permeability problems under first lift conditions.

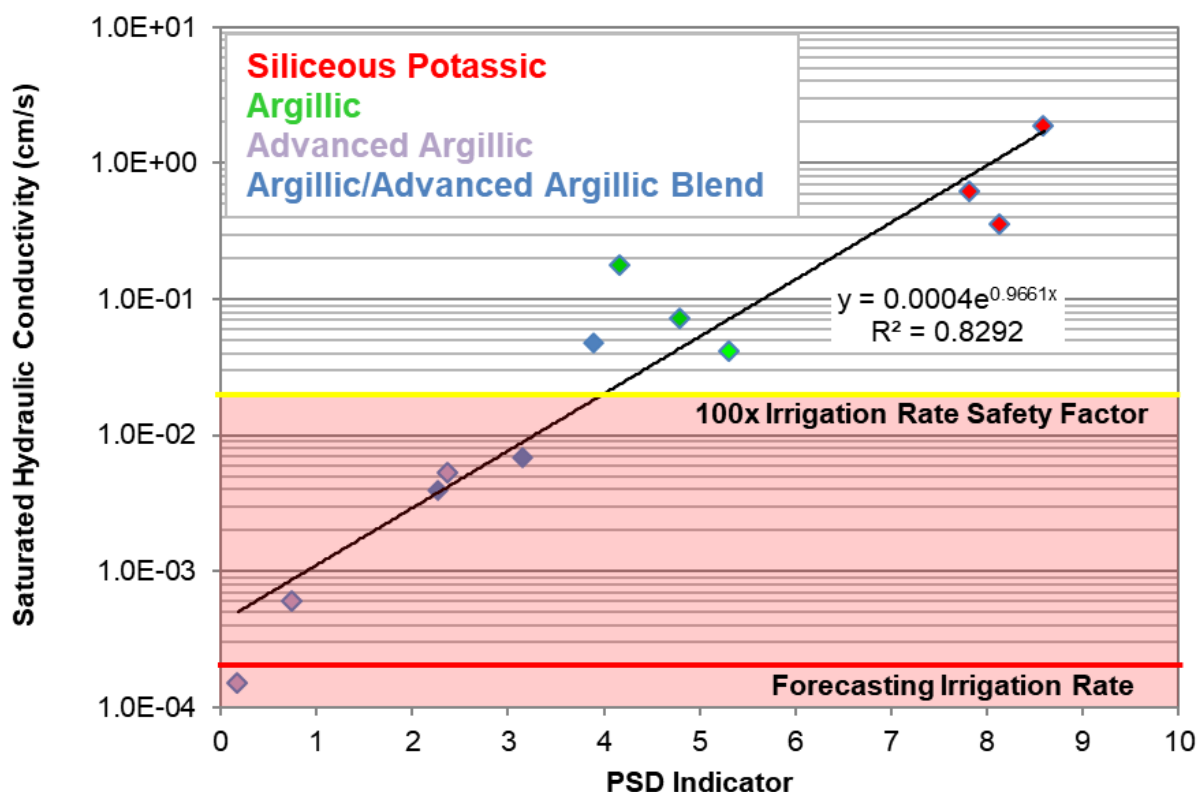


Figure 4: Correlation between PSD indicator and K_{sat} at an estimated heap height of 20 feet

Operational implementation

The leach pad recovery model was adjusted based on operating and permeability test results. Figure 5 shows the budgeted and actual monthly gold ounces produced from the leach pad from October 2012 through June 2022. Actual gold production was consistently less than the budgeted gold production through 2014. Integration of the test work and recalibration of the recovery model in 2015 resulted in the forecast model being able to better forecast gold production, in addition to allowing for identification of problem ore types and to optimize ore blends.

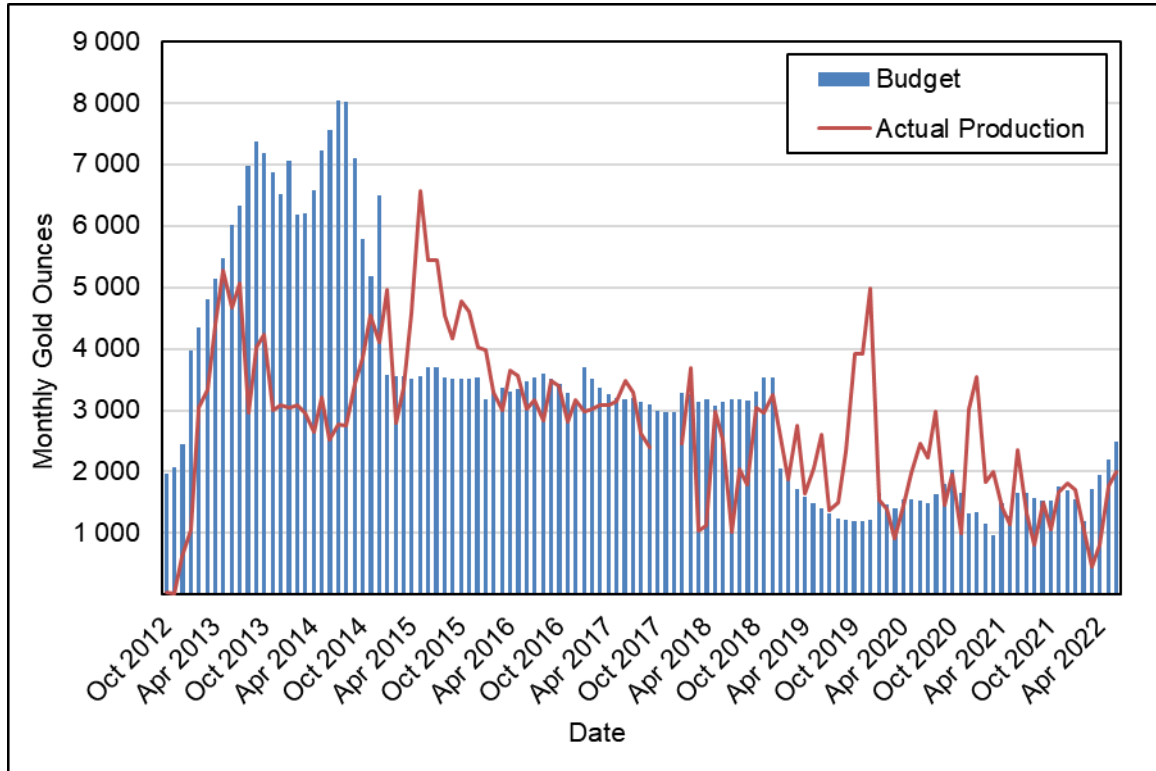


Figure 5: Budgeted and actual gold produced

Conclusion

SP ROM samples had the best permeability, showed minor consolidation under increasing pressure, and exceeded the $100 \times K_{sat}$ value safety factor up to simulated 70- to 80-foot heap heights. The A ROM samples and blend sample 75% A-1/25% AA-1 had intermediate permeability performance, exceeding $100 \times$ safety factor up to a minimum of 20 feet of estimated heap height and exceeding the forecast irrigation rate for all heap heights tested. AA ROM samples and blend samples 75% A-1/25% AA-2 and 75% A-2+A-3/25% AA-3 were below the $100 \times$ safety factor under first lift conditions (less than 20 feet).

Correlation analyses performed to assess relationships between measured K_{sat} and various physical parameters indicated that the PSD indicator values showed the strongest correlations to the measured K_{sat} values and may be used as an indicator of sample permeability. Using the K_{sat} versus PSD indicator relationship, ore blend compositions with a PSD indicator value greater than 4 should be targeted for the blend to be sufficiently permeable under one lift conditions.

Results from this study indicate that well designed hydraulic property testing can simulate operational conditions. The laboratory test data can be used to evaluate the effect of different ores on heap permeability, optimum heap height and irrigation rate to minimize effect of consolidation, and appropriate ore blend

ratios. Furthermore, the use of physical property data correlated to permeability results can allow for mapping of ore body permeability.

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Geotechnical Evaluation of Properties and Liquefaction Potential Using CPTu for Heap and ROM Leaching Facilities

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Abstract

In the mining industry, the evaluation of geotechnical properties and the liquefaction potential has been a challenging task for large leaching operations over heap or ROM materials. The technique of using CPTu measurements has become a useful and practical means for evaluating the geotechnical conditions within the soil mass, considering the material variability and the fact that the ore may acquire more complicated geotechnical properties due to its fines content.

In northern Chile, large mine operations have begun confronting the need for deeper and more reliable geotechnical assessments of their leaching operations due to the height that facilities have reached after several years of leaching process, as well as the rigorous permit requirements and the risk potential for operations, installations, and environment.

Geotechnos was required to conduct a Geotechnical Field Campaign to assess the geotechnical properties on a heap and ROM leaching operation where facilities reached up to a height of one hundred metres. The main purpose was to evaluate the liquefaction potential and determine the existence of a perched water table; which along with determining ore resistance properties allows for the evaluation of liquefaction risk.

The field campaign executed CPTu tests in 19 selected points with penetration depths of 25 to 30 m on the heap facility and in 22 selected points with penetration depths between 15 to 30 m on the ROM facility. The tests included records for pore pressure, down-hole at selected depth, and sampling of selected materials for moisture content. The total length of executed tests was 486 m for the heap facility and 373 m in the ROM facility. All work was done in a period of 2½ months.

The location of the testing points was discussed with the operation and the Review Board of Experts with a view to assessing the most likely portions of ore that might experience difficulties or require solutions.

For the interpretation of data several correlations were used to define the stratigraphic profile according to the standard penetration test (SPT) interpretations referred to as SBT (soil behaviour type). The interpretation allows the geotechnical evaluation of ore variability and consistency of the material profile, showing a more homogeneous material in heap ore and more heterogeneous material in ROM. The data interpretation permitted the determination of the following geotechnical parameters: Ksbt (permeability based on SBT), N60, Young's Modulus, Relative Density, Friction Angle, Elastic or Contraction Modulus, Shear Modulus, Undrain shear resistance, Overconsolidation Ratio, and Unit Weight.

Introduction

Safety in heap leaching operations has been an enduring preoccupation of geotechnical engineers, who have focused on providing the best possible understanding of the behaviour of the material subject to leaching under different physical and environmental conditions. It has been a challenge to provide a characterization of materials that adjusts to the reality of each operation, due to the great variability over time of the materials of the original mineral matrix, caused by the high exploitation tonnages and large geological areas involved in the process, and also the difficulty of ensuring a homogeneous operation in terms of wetting the heap.

These recurring issues in all leaching operations must be carefully focused upon. In each operation, the size of the heaps and the variation of the type of ore over time present new requirements that must be addressed in order to develop reliable geotechnical studies of stability and ensure operational continuity in the recovery of metals.

Moreover, long years of exploitation of surface minerals, generally associated with oxides, have resulted in operations moving to the exploitation of a deeper mineral matrix, with different geotechnical qualities and a greater exploitation of sulfide ore that has been processed for leaching. In addition, the grades of copper contained in the mineral bodies have decreased, so the heap leaching units are increasingly larger; therefore, obtaining representative parameters is a great challenge.

Techniques to characterize a large mass of soil include a wide variety of invasive and non-invasive methods, comprising a combination of classical surveys with sampling and tests of specimens in the laboratory and geophysics, which have been widely validated for the development of studies in stacks due to their reasonable cost. The incorporation of drilling is also considered an alternative to calibrate geophysical results.

For geotechnical studies to ascertain the properties of soils and assign parameters to the integral mass of soil, the varieties of penetration test have excellent and proven results when the soils present granulometric characteristics of a type of fine gravel or smaller particle size and have relatively homogeneous environmental and geotechnical conditions. However, the limitations in granulometric terms restrict the widespread use of the method due to high cost, and the high probability of not achieving favourable results when applied in heterogeneous soils with sizes of particles with a high content of gravel or blocks. This is because there is a probability of encountering a particle that blocks the penetration path, causing the test to stop. In particular, measurements with DCPT, CPT or CPTu methodology have been widely used and with very good results in tailings sands and, moreover, there is a wide variety of formulations and calibration to adjust the results to this type of soil.

As geotechnical specialists, we have sought ways to adjust techniques or derive results in different media and applications in engineering projects for projects involving large soil masses, such as tailings dams, leach pads, and dumps. In consideration of the particle size and homogeneous characteristics of the leached material, Geotechnos began to provide services by performing measurements using the DCPT technique in ripios dumps from dynamic heaps. After several application campaigns with good results, it was proposed by a mining operation to apply the CPTu technique to a high-rise leach pad and on a ROM dump leach operation, to assess whether the results would be consistent and reliable for use in geotechnical stability and deformation studies and to evaluate the liquefaction potential of heap and dump.

The following sections present a summary of the application of CPTu measurements in a crushed ore leach pad and a ROM ore dump. The technique is complemented with geophysical measurements and laboratory tests that were integrated to calibrate the measurement results and confirm if they are consistent to be used in stability and deformation studies. Finally we highlight the advantages and disadvantages of the application of the method in heaps and dumps and present recommendations for improvements in the application in these kinds of facilities.

Technical framework

The evaluation of the liquefaction potential is expressed as a safety factor against the occurrence of liquefaction. If the cyclic stress caused in the ground by a seismic event CSR (Cyclic Shear Stress Ratio) is greater than the cyclic resistance of the ground, CRR (Cyclic Resistance Ratio), liquefaction may occur.

The factor of safety is determined by the following equation:

a) $F.S. = CRR/CRS$

b) The following figure shows a summary of the Boulanger and Idriss (2014) method applied.

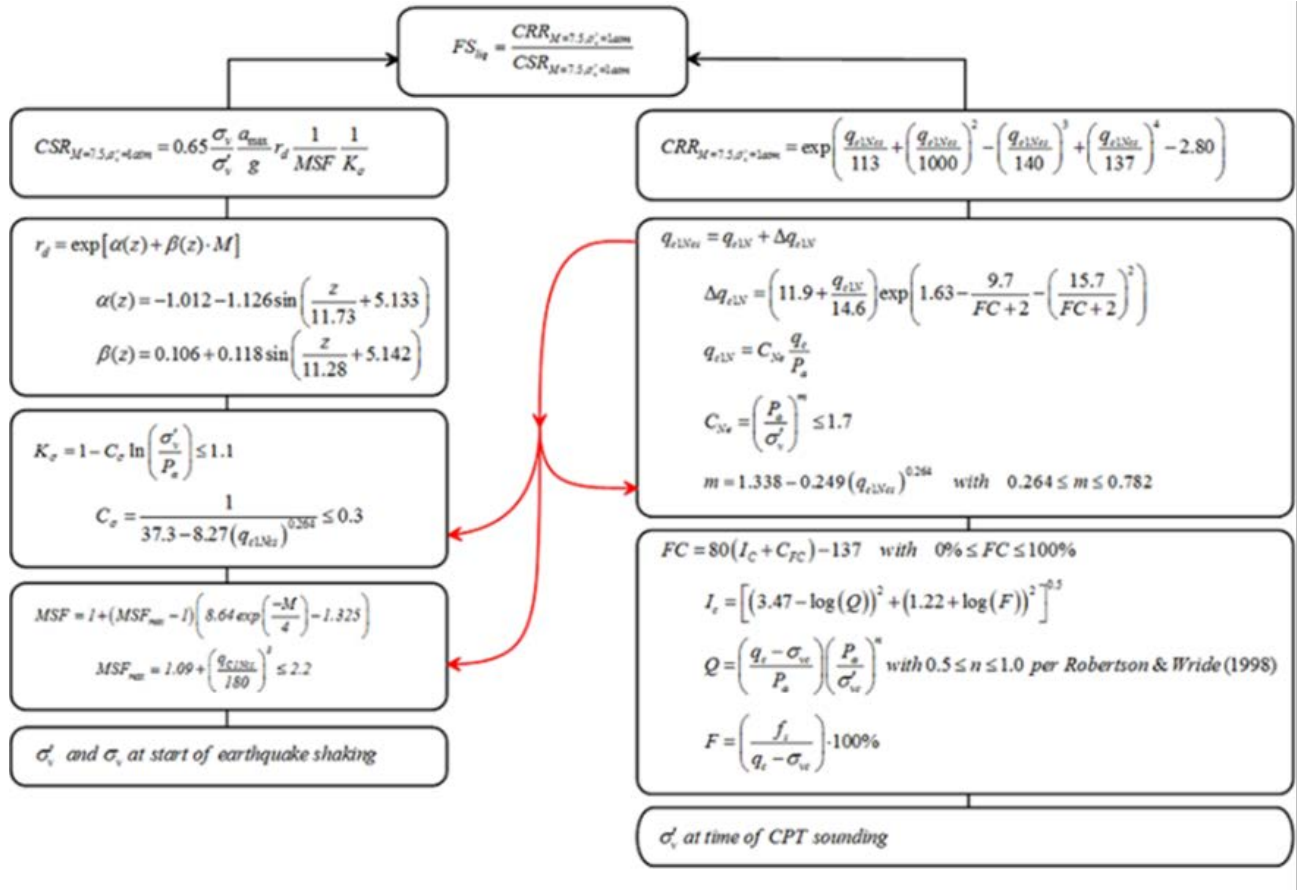


Figure 1: Summary of the Boulanger and Idriss (2014) method applied

CPTu application in heap and leaching dump

Geotechnos was retained to conduct a study to define the liquefaction potential of the heap through the execution of CPTu tests, measurements of pore pressure dissipation and down-hole tests, to measure shear wave velocities (Vs). As part of the work, the taking of samples, laboratory tests, determination of the stratigraphic soil profile and geotechnical parameters have also been included.

Field work

It was planned to execute 19 CPTu drillings located in defined sectors on the surface of the heap with a total of 485.75 linear meters executed. In Figure 2 below, the location of the CPTu tests over the heap are shown in red.

GEOTECHNICAL EVALUATION OF PROPERTIES AND LIQUEFACTION POTENTIAL USING CPTu FOR HEAP AND ROM LEACHING FACILITIES



Figure 2: Location of CPTu tests in heap

For reference, the data output from part of the tests is presented in Figure 3, which includes the total tip resistance, shaft resistance, and pore pressure measured for the CPTu tests.

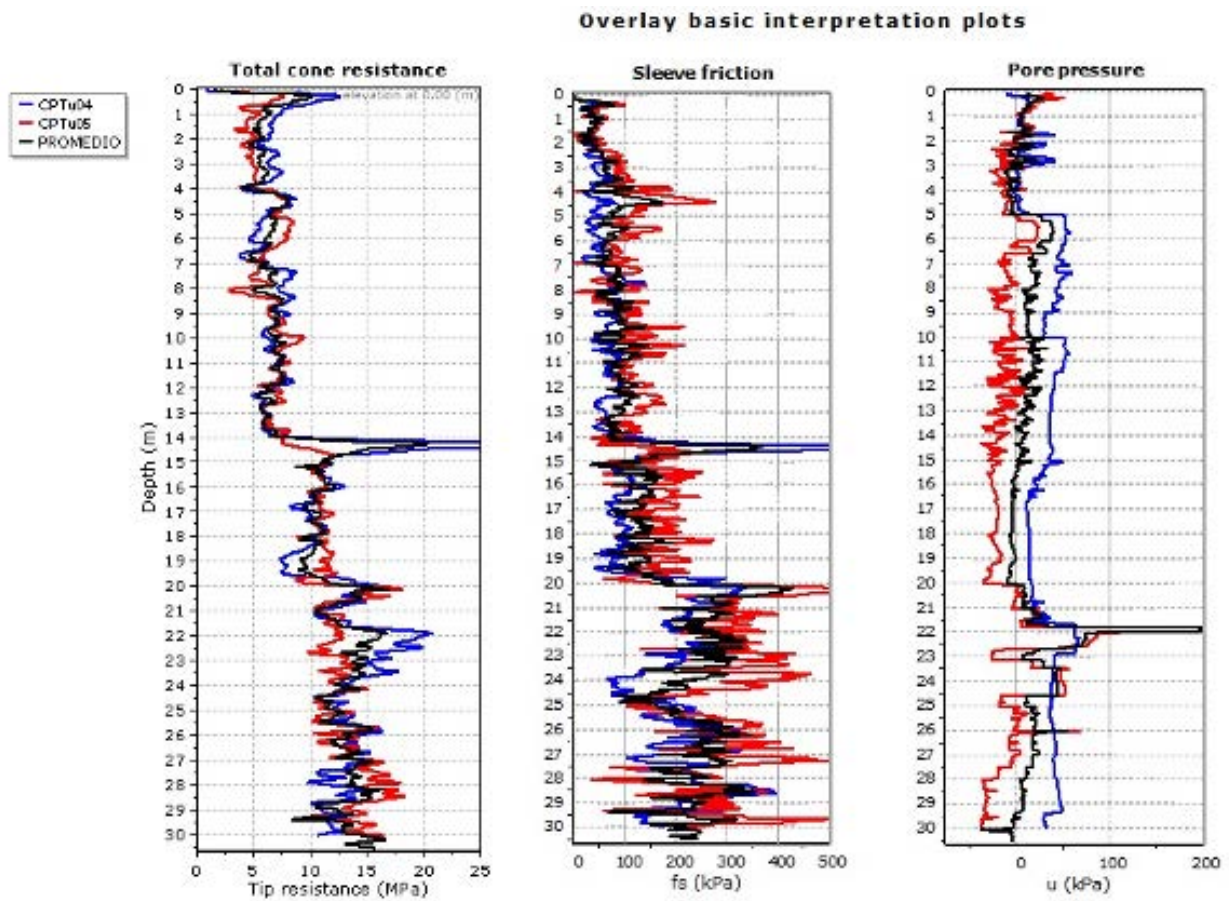


Figure 3: Summary data output of CPTu assays

In addition, a total of 10 down-hole tests were conducted, considering the ratio of 1 D-H test for every two prospections.

Next, in Figure 4, a record of part of the results obtained from the test for a particular CPTu test is presented.

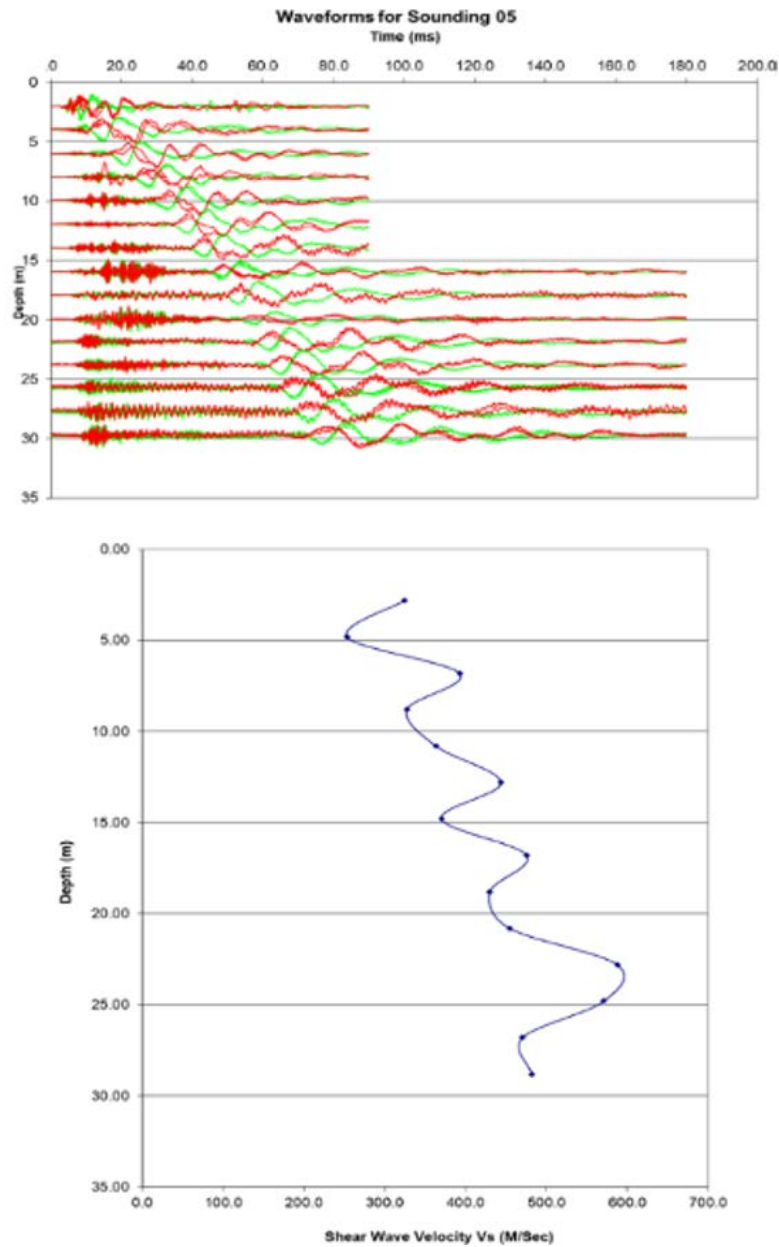


Figure 4: (Top) Shear wave log; (bottom) wave velocities in depth, CPTu

As part of the measurement requirement, it is necessary to determine the degree of saturation of the ore mass in the heap/dump, for which dissipation tests were conducted at different depths at each CPTu test point. The log presents the dissipation of pore pressure (kPa) as a function of a given amount of time (s). The analysis depths correspond to 5, 10, 15, 20, 25, and 30 m, depending on each drilling.

Next, Figure 5 presents a record of the results obtained from the dissipation test in CPTu drilling.

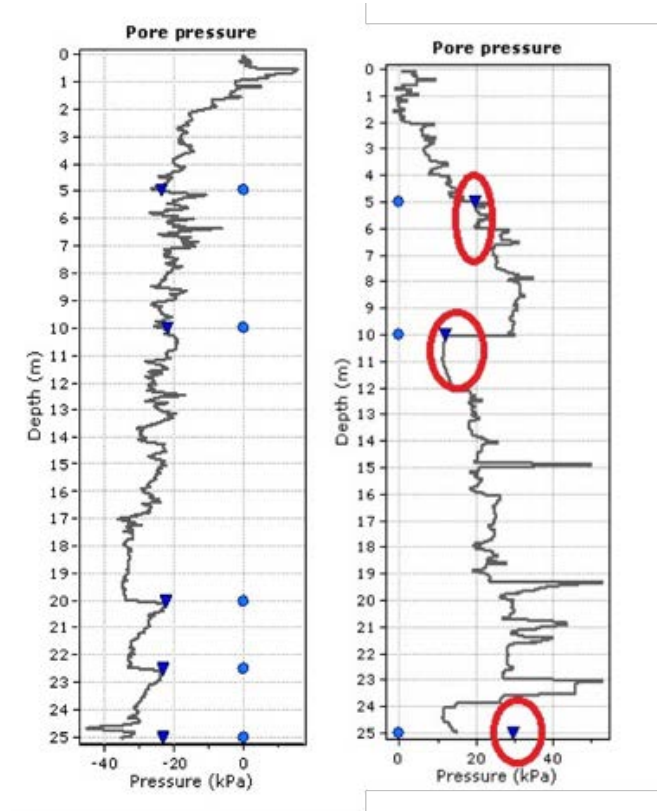


Figure 5: Dissipation tests in CPTu drilling

Pore pressure dissipation test results show zones with negative pore pressures and zones with positive pore pressures close to zero (see Figure 5, left). However, in some dissipation tests possible saturated zones are observed (see Figure 5, right).

The results of the test indicated the level of saturation of the material from 5.0 m deep, and the value of the pore pressures at depth; in this way it is possible to indicate the presence of a water table inside the heap. In some points values close to 0 (equilibrium) of pore pressures were obtained, and in others values were slightly negative and positive.

Sampling

To calibrate the stratigraphic interpretation of the CPTu measurements and the results of dissipation tests, obtaining undisturbed samples was considered to determine their USCS classification and moisture content

in the laboratory. The samples were obtained at depths of 5 and 10 meters and although they had high humidity, they were not saturated, confirming that there was no groundwater at those depths. Figure 6 shows the condition of the samples obtained at the point of a CPTu test.



Figure 6: Condition of sample at 5 m depth (top), and sample at 10 m depth (bottom)

The samples were sent to the laboratory for the preparation of the following tests:

- Determination of granulometry (8.102.1 MC Vol 8, December 2003).
- Liquid limit determination (NCh 1517/1 Of. 1979).
- Plastic limit determination (NCh 1517/2 Of. 1979).
- Density of solid particles under sieve No. 4 (NCh 1532. Of. 1980);
- Moisture content (NCh. 1515 Of. 1979)

Stratigraphic profile

It should be noted that the CPTu cannot provide precise information on the physical characteristics of the soil, such as grain size distribution, but it can provide a guide to the mechanical characteristics of the ground

and the behavior in-situ in the immediate area of the drilling. From the above, the estimation of the soil type based on the CPT tests is referred to as SBT (soil behavior type).

Robertson proposed the first graphs (SBT) using the basic parameters of cone resistance (q_t) and friction ratio (R_f). When penetration resistance and shaft resistance increase in depth, the CPT data requires normalization for overburden stress. Finally, the full normalized plots include an additional plot, based on the normalized pore pressure parameter (B_q).

From the above, it was possible to represent the results in stratigraphic profiles; Figure 7 presents the estimation of the profiles.

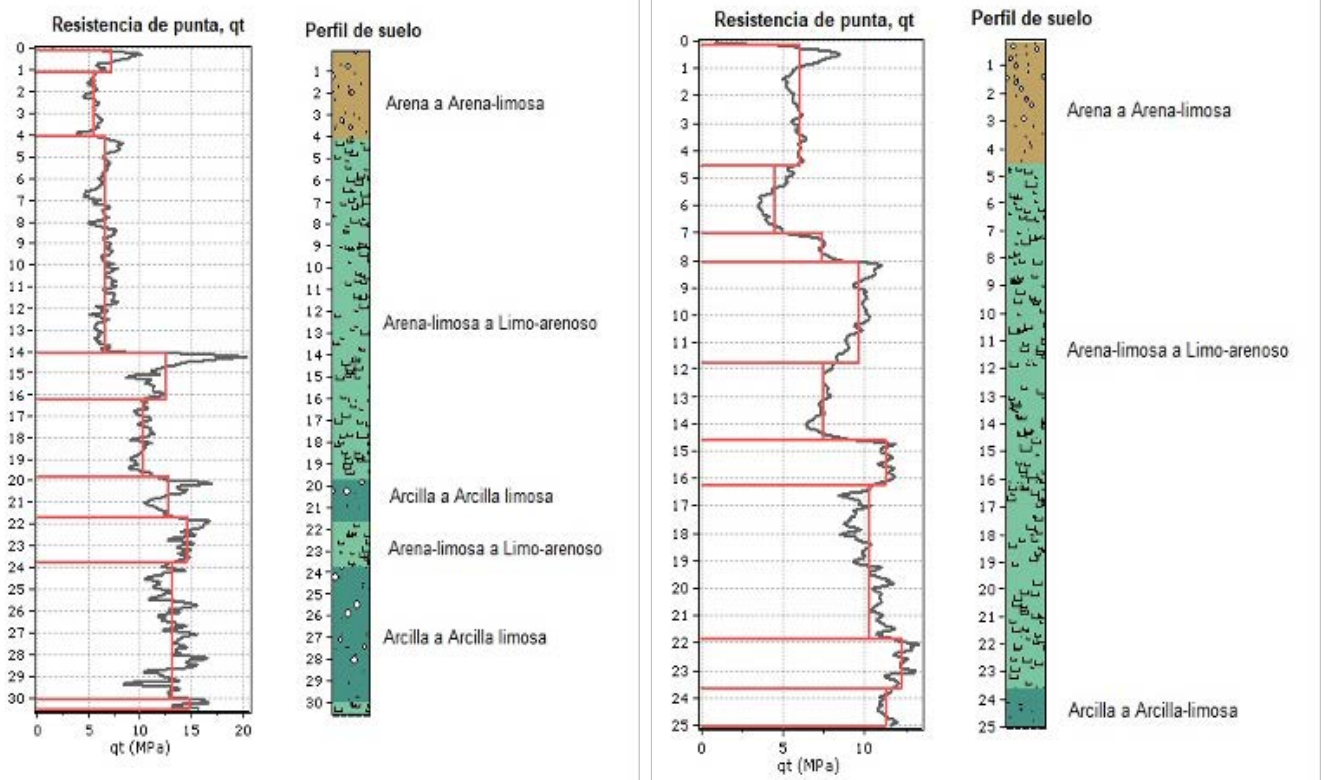


Figure 7: Stratigraphic profile for different CPTu tests

According to Figure 7, it is possible to observe that the profile varies from sand to silty sand in the superficial strata to soils with a higher content of fines (silt and clay) at depth, which is considered consistent with the expected soil profile.

Definition of geotechnical parameters

Together with the stratigraphic profile, it is possible to estimate the profile by layers of geotechnical parameters in depth. In Figure 8, the geotechnical parameters for a particular CPTu test are presented.

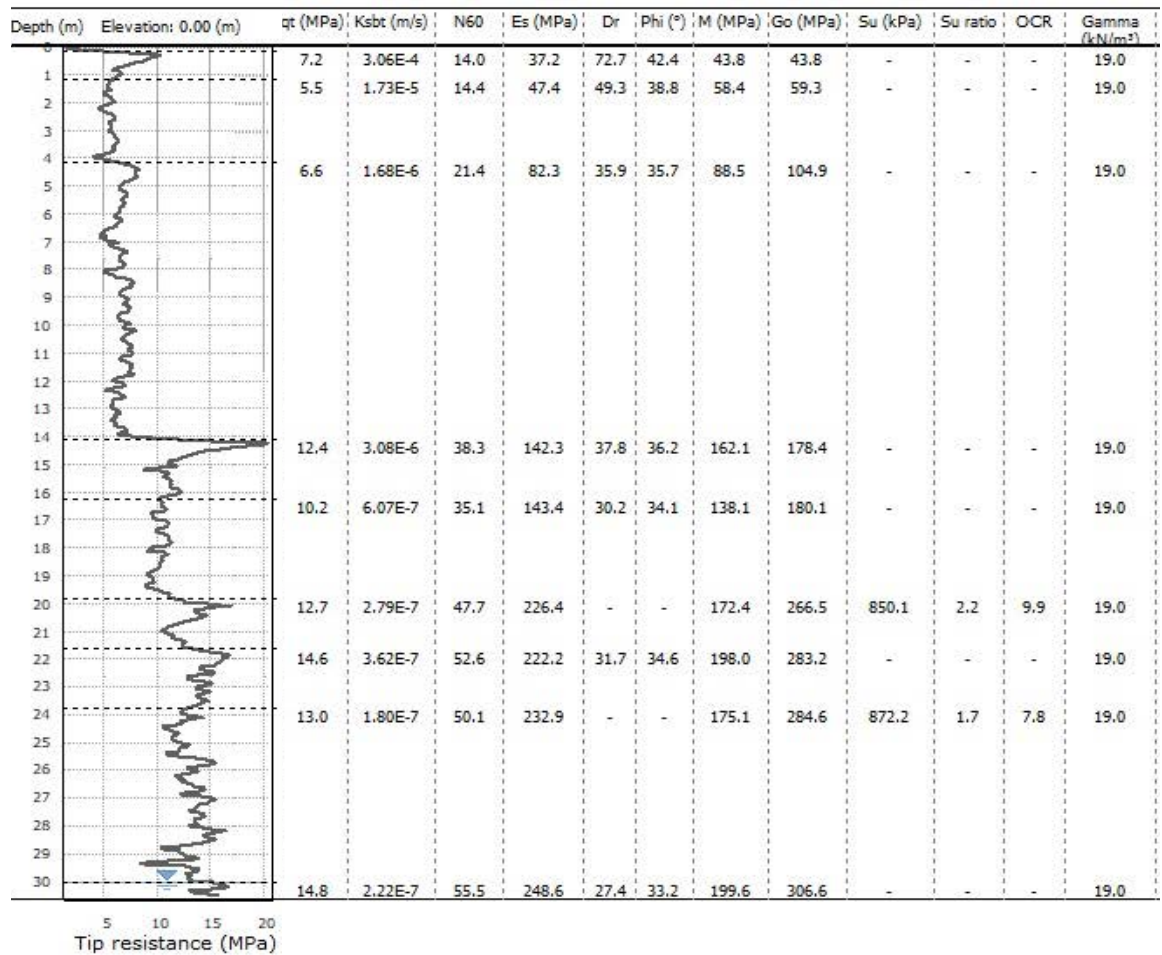


Figure 8: Geotechnical parameters deduced from the CPTu test

Liquefaction analysis procedure

The evaluation of the liquefaction potential was conducted using the analysis method proposed by Boulanger and Idriss (2014) and through the measurements of CPTu drilling every 5 cm of depth, the seismic stress (CSR) and the resistance of the material (CRR) were evaluated.

Seismic stress (CSR) is evaluated using the simplified method initially proposed by Seed and Idriss (1967), in which the PGA value is of vital importance for the analysis.

The cyclic resistance (CRR) of the material is evaluated using the measured values of resistance of the soil through the CPTu drilling.

For the liquefaction analysis, the following moment magnitude of the earthquake was assumed:

- Mw: 8.0 (momentum magnitude of the earthquake).

Regarding the maximum acceleration on the surface, a sensitivity analysis was conducted considering the following three cases of maximum acceleration on the surface (PGA):

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- 0.3g.
- 0.4g.
- 0.5g.

These three cases consider three return periods (100, 475, and 975 years).

In a first analysis, therefore, it is considered that the material is completely saturated ($S = 100\%$, conservative case) and that the equilibrium pore pressures are equal to 0 throughout the depth analyzed. This scenario corresponds to one with a lower probability of occurrence, but with a higher risk for the operation.

Results with $S = 100\%$ and pore pressure $u = 0$, in depth

a) $PGA = 0,3 \text{ g}$:

With the average values of measurements of tip, shaft, and pore pressure of CPTu drillings executed in the area, the results shown in Figure 9 below are obtained for $PGA = 0.3 \text{ g}$.

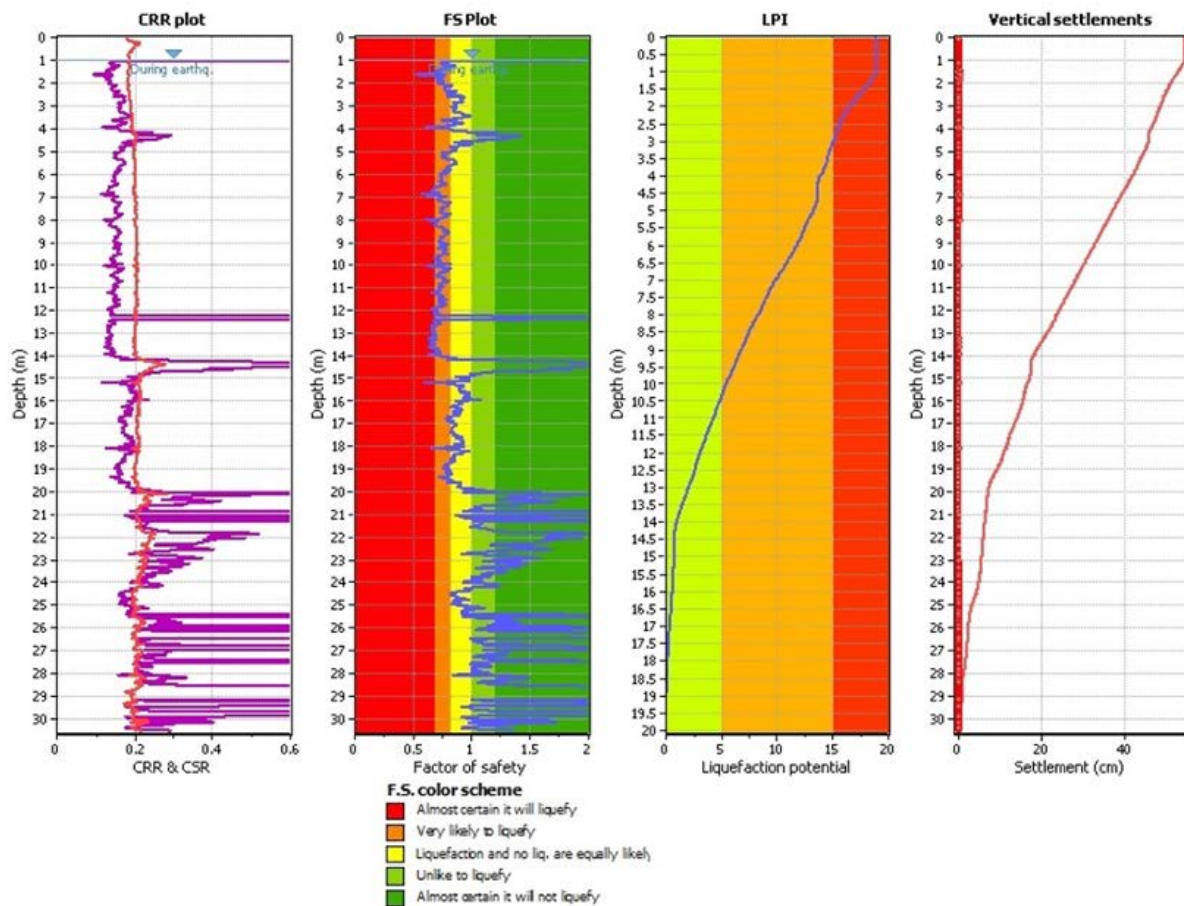


Figure 9: CRR&CSR; F.S; liquefaction and settlement potential, $PGA = 0.3 \text{ g}$

b) $PGA = 0.4g$:

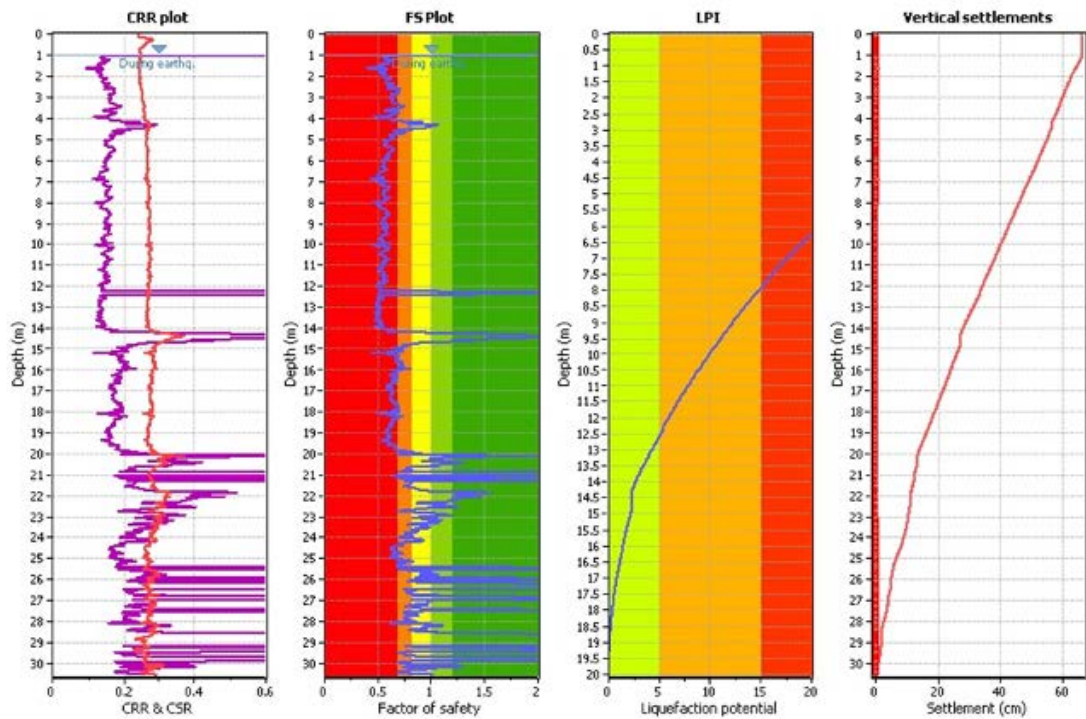


Figure 10: CRR&CSR; F.S; liquefaction and settlement potential, $PGA = 0.4 g$

c) $PGA = 0.5g$:

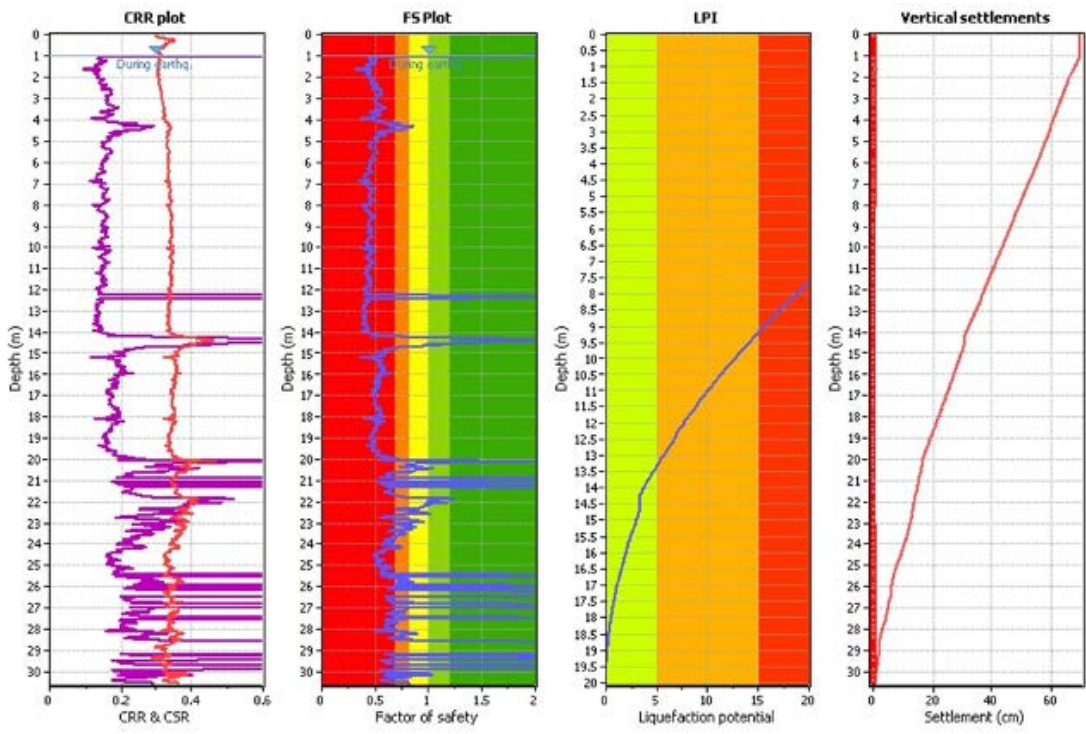


Figure 11: CRR&CSR; F.S; Liquefaction and settlement potential, $PGA = 0.5 g$

In summary, the liquefaction analysis in the heap shows that there is a risk of liquefaction down to at least 15 to 25 m depth, depending on the sector. The factor of safety tends to increase with depth because the lower layers have had a longer time to consolidate and are subject to greater confinement.

This scenario corresponds to a more unfavourable situation due to a condition where there is a saturation level of 100% and pore pressure = 0, and it corresponds to a less probable case, but with greater risk.

Influence of the saturation degree of the material in the heap

The results of the estimation of the degree of saturation in depth are shown in Figure 12. In general, it can be seen that the samples obtained during the prospecting are not saturated. This coincides with the appreciation in the field when obtaining the samples.

There is no clear trend as to whether the saturation goes up or down with depth; this depends on the location observed and probably when the measurement is made.

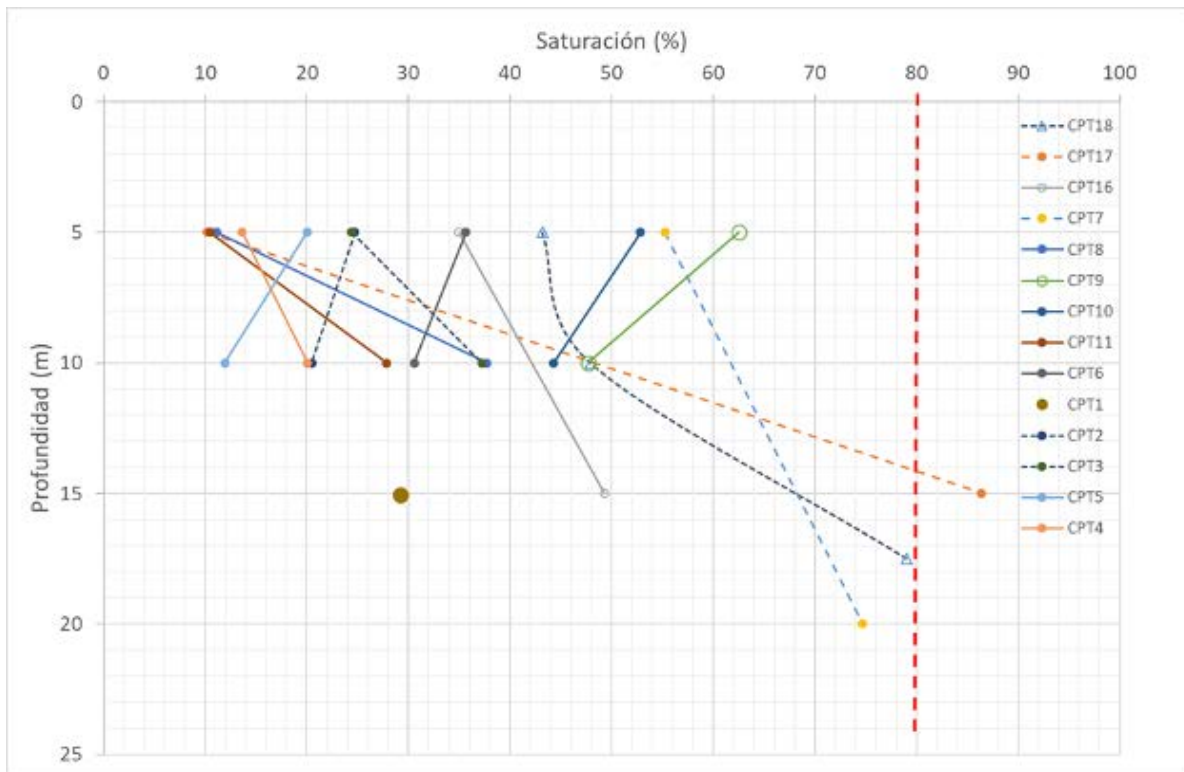


Figure 12: Degree of saturation deduced

Figure 12 shows that the maximum saturation found is 86%. Of the latter, only one sample is above the 80% observed in the CPTu-14H probe, the rest of the samples remain below 80%.

It is important to mention that it has been found that resistance to liquefaction rises rapidly for degrees of saturation below 98%, and that below 90% it is very difficult to achieve this phenomenon in practice (Yang et al., 2004; Eseller-Bayat et al., 2013).

Based on the foregoing and considering conservatively as 80% the saturation limit below which liquefaction would not be generated, then it can be considered that there is, at the time the campaign is conducted, a low probability that a liquefaction phenomenon will be generated in the heap. It is estimated that for closure the situation would be similar with a partially saturated material, which would improve over time due to the drainage that would occur inside the heap.

CPTu over dump leach

The same methodology was intended for application over a dump leach at the same operation. Since scope, objectives, and methodology were the same as previously described, no further details will be included in this section.

The application of the method was conducted with difficulty since there was much blocking experienced in the penetration phase, and in some cases the penetration did not reach a depth of more than 1 or 2 m. In some measurements it was necessary to repeat the execution at a nearby point. In the case of the dump leach, the effort was much higher than in the case of the heap leach, and results and interpretation are to be used with discretion, considering that a much more detailed evaluation by geotechnical experts and operations records will indicate whether CPTu may proceed, since technical and economic efforts are high, and the probability of encountering a good deal of blocking due to boulders and oversize material might make it impossible to proceed.

Conclusions

Conclusion 1

It is deduced that the tested areas of the heap consist mostly of silty sand on the surface, sandy-silt in the center, and clay materials at the bottom of the heap. These types of materials are consistent with what is expected based on mine activity.

Conclusion 2

According to the results of CPTu soundings, dissipation tests, and data obtained from piezometers, no hydrostatic pore pressure (as water table) is detected in the heap. Readings from CPTu soundings do not agree with the presence of an actual phreatic surface. This is attributed to the fact that heap leaching is an operation that consists of material being deposited in layers with high moisture content. The degree of saturation of the material in the heap varies due to changes in moisture content during deposition,

evaporation, and consolidation of this earth structure. To determine the degree of saturation with higher confidence, it was necessary to complement the readings from CPTu, with soil samples obtained at different locations and depths. Variations in degree of saturation were found in this study; some zones were found to be close to complete saturation and others had a degree of saturation as low as 10 to 20%.

Conclusion 3

The liquefaction potential of the heap was evaluated for the most unfavourable condition ($S = 100\%$, conservative case). A conservative analysis shows that there is a risk of liquefaction down to at least 15 to 25 meters depth, depending on the analyzed area; pore pressures are considered equal to 0 throughout the analyzed depth to calculate effective stresses.

Conclusion 4

A sampling program was carried out to analyze the influence of the degree of saturation in the heap. It was observed that, in general, soil samples obtained during the in-situ testing program, except for one point ($S = 86\%$), are not close to saturation, which coincides with observations made in the field when obtaining the samples. There is no clear trend as to whether the saturation increases or decreases with depth, this depends on the location observed and probably the time of the year when the measurements are made.

Conclusion 5

Conservatively assuming a saturation threshold of $S = 80\%$, below which liquefaction would not occur, it can be deduced that there is a low probability of triggering a liquefaction phenomenon in the heap at the time the campaign was conducted.

Conclusion 6

In terms of the CPTu application, this testing is considered to be adequate for its use in heap leaching and the results from this execution are considered to be consistent with what was expected geotechnically. The value of the gathered information is of very high importance and the use of CPTs soundings is considered a suitable alternative for geotechnical characterization for heap leach facilities.

Conclusion 7

On the other hand, the difficulties encountered for the execution over the dump leach leads to the recommendation that even though the method should not be totally discarded, the geotechnical conditions must be evaluated in advance and if there are very heterogeneous materials, the cost and effort should be reviewed before proceeding.

Acknowledgements

For the execution of the field campaign, data collecting and recommendations, the authors wish to acknowledge the support of LMMG, who provided all equipment and technicians with the expertise in CPTu operation and data collecting following technical practice and standards for the corresponding confidence on all post analysis that lead to proper analysis and conclusions.

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Process Update on Dynamic Heap Bioleaching of a Black Schist Ore

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Abstract

The Terrafame multi-metal mine, located in Sotkamo, Finland, utilizes heap bioleaching to extract metals from black schist ore. The ore leached at the mine is a complex sulfide ore with pyrrhotite and pyrite as the main sulfides (5–15 w-% each). In addition, the ore contains 5–10 w-% graphite. The valuable metals in the ore deposit, Ni, Zn, Cu, and Co, are in pentlandite/pyrrhotite, sphalerite, chalcopyrite, and pyrite, respectively. The heap leaching process is divided into two steps: dynamic primary leaching and multi-lift secondary leaching. The primary leaching stage containing a total of 24 to 28 million tonnes of ore takes approximately 15 months. If the primary leaching process is seen to be continuing well after 15 months, the leaching time is lengthened by extending the leaching pad. Extensions of the primary leach pads were started with a 150 m area for two of the four pads in 2021. Construction of new secondary leaching areas will also commence in future. Explorations in the Kolmisoppi mine reserve are ongoing, and mining will extend to Kolmisoppi in the future.

Operation of the dynamic primary pads started in 2008, and the reclaiming of the ore from the primary to the secondary pads commenced in 2010. A review of the operation was presented at the Heap Leach Mining Solutions 2016 Conference in Lima, Peru, by Antti Arpalahti. The seventh round of operation on the primary leaching pads will start in the summer of 2022. In 2021, 28,582 tonnes of nickel and 54,353 tonnes of zinc were produced, and the net sales were EUR 378.4 million. The production of nickel and cobalt sulfates for the battery industry started in 2021, and is increasing. In 2021, the production of battery chemicals made up 7% of the total sales. Planning for the uranium recovery plant update and start-up is ongoing.

Increased aeration capacity in the primary heaps has been one of the contributors to the enhanced leaching. Extra air feed into the heaps accelerates the leaching reactions and increases heat formation and evaporation. Therefore, primary heaps contribute significantly to control of the water balance. The solution volume in the mine area has settled in the targeted level. Since 2015, the solution balance has been also

controlled with a discharge pipeline. This paper provides an update on the current process developments at Terrafame mine, ongoing production and operation ramp-up, and future production plans, including battery chemical production and the recovery of uranium.

Introduction

Terrafame mine (previously called Talvivaara mine) is located in Sotkamo, Eastern Finland. Geological mapping of the mine area was started in the early 1900s by the Geological Survey of Finland (GTK), and several Finnish companies carried out explorations at the site in the 1950s and 1960s.

After metallurgical sampling of the deposits and investigation of the exploration and research data, heap bioleaching was chosen as the most economical option to extract the metals from the ore.

Operation of the dynamic primary pads started in 2008, and reclaiming of the ore from primary to secondary pads commenced in 2010. Since 2015, the mining operation has been in a new commissioning phase. The operation ramp-up has been successful, and the seventh round of operation on the primary leaching pads will start in the summer of 2022.

Extra air feed into the heaps accelerates the leaching reactions and increases heat formation and evaporation. Therefore, primary heaps contribute significantly to the water balance control. The solution volume in the mine area has settled at the targeted level. Since 2015, the solution balance has been also controlled with a discharge pipeline.

Terrafame process update

Mineral resources

The mineral resources of Terrafame Oy consist of two Ni-, Zn-, Co-, and Cu-bearing deposits, Kuusilampi and Kolmisoppi (Figure 1). Currently the mining takes place at Kuusilampi, in an open pit mine.

Ni-Zn-Co-Cu mineralizations occur within the Kainuu Schist Belt in Central Finland. The schist belt consists mainly of quartzites, mica schist, and black schist. The schist belt lies unconformably on the Archaean basement gneiss complex.

The Kuusilampi orebody has overall dimensions of approximately 2,500 m × 100 to 400 m, and the Kolmisoppi has dimensions of approximately 4,000 m × 50 to 300 m. The extensions in the depth are open under the level of 300–500 m.

Previous mineral resource and ore reserve estimates were published in 2020. Since then, Terrafame Oy has drilled around 24 km of diamond core drilling and a 0.7 km geotechnical diamond core at the Kuusilampi and Kolmisoppi deposit.

As of 2022, the mineral resources amount to 1,453 million tonnes of ore at 0.26 %Ni, 0.54 %Zn, 0.019 %Co and 0.15 %Cu.

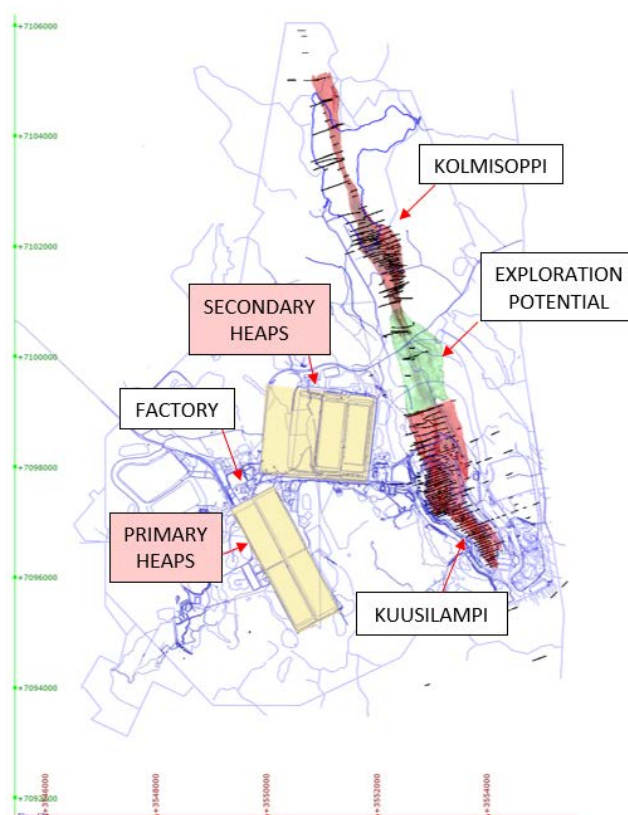


Figure 1: Overview of Terrafame's industrial area, highlighting the location of Terrafame's mineral resources (Kuusilampi and Kolmisoppi) and primary and secondary leaching pads

Primary heap leaching area

Extension of the primary leaching residence time

It has been observed that primary leaching process performance continues at a good level after the original 18-month residence time at the primary leaching pad. In 2020 it was decided to increase said residence time by extending the primary leaching pads towards the south (greater area and same mining rate, leading to extended residence time in leaching). The primary leach pads were extended in length during 2021 by 150 m. The ore tonnage in the primary area has since increased to about 1,5 million tonnes, and the residence time (under normal mining rates) by approximately 30 additional days, prior to the start of reclaiming operations (Figure 2).

With these actions Terrafame strengthens the share of the primary leaching production compared to total production (primary plus secondary), thus leading the operation towards more efficient production of nickel and cobalt sulfates, and zinc sulfide. In addition, copper production benefits from the increased leaching time in the primary phase.

As a next step, planning of additional extensions of primary heaps is ongoing, and it is anticipated that detailed planning will begin to gain momentum during 2022.

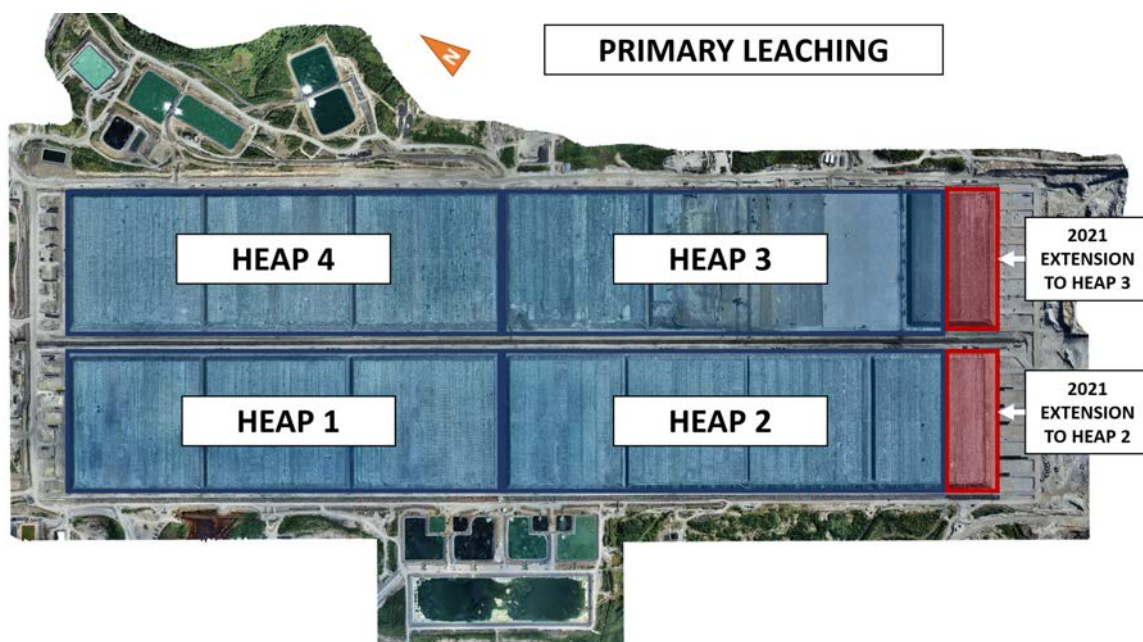


Figure 2: Overview of the primary leaching stage.
The extension of the leaching pad during 2021 is highlighted in red

Heap height increment

At the beginning of this heap leach operation, the stacking height of primary heaps was 8 m. Continuous improvements addressing aeration capacity and efficiency have led to the possibility of increasing the stacking height, and therefore the tonnage over leaching pads and leaching residence time. Currently, the heaps at the primary stage are stacked to a height of 9.5 m, which is the maximum of the stacking equipment capabilities. The stacking height increase has led to an additional 1 million-tonne ore/heap, resulting in approximately 80 days additional leaching time in the primary leaching phase.

Aeration development

Aeration of the heaps has been continuously improved since the beginning of the project. Research done in Terrafame (Arpalahti, 2019) supported the idea of adding aeration capacity in the heaps. The increased heap height led also to a new aeration concept. With the new construction method, the aeration pipelines are installed at three different heights (1, 3.5, and 6 m).

Installation

Under normal operating conditions, the aeration distribution array is constructed continuously as the stacking of the new heap progresses. Aeration lines are installed over the heap and dropped over the stacking

front to three different heights set by preconstructed rest-benches. The lines are then connected to the main aeration lines and fed by blowers located at both sides of the heaps. Stacking resumes and the lines are effectively buried within the heap.

The aeration capacity can be adjusted at mainline level for each blower. In addition, the air flow can be further trimmed for each individual air line, by operating the single-line valves located at the connection to mainline. This makes it possible to operate the air supply with extreme precision, improving air supply efficiency, heap temperature control, work safety, and ergonomics.

In a different scenario, some heaps have all their blowers located on one side of the heap (instead of blowing air from both sides). In this case, Terrafame has been testing a modified design for aeration supply. This different model ensures that about 16% of the total air blown to the heap is directed straight to the opposite side, bypassing the proximal end of the heap. This different approach minimizes the risks of reduced air supply to the far end of the heap (furthest from the source of air), if there are any air pressure drops over time. This conceptual model is presented in Figure 3.

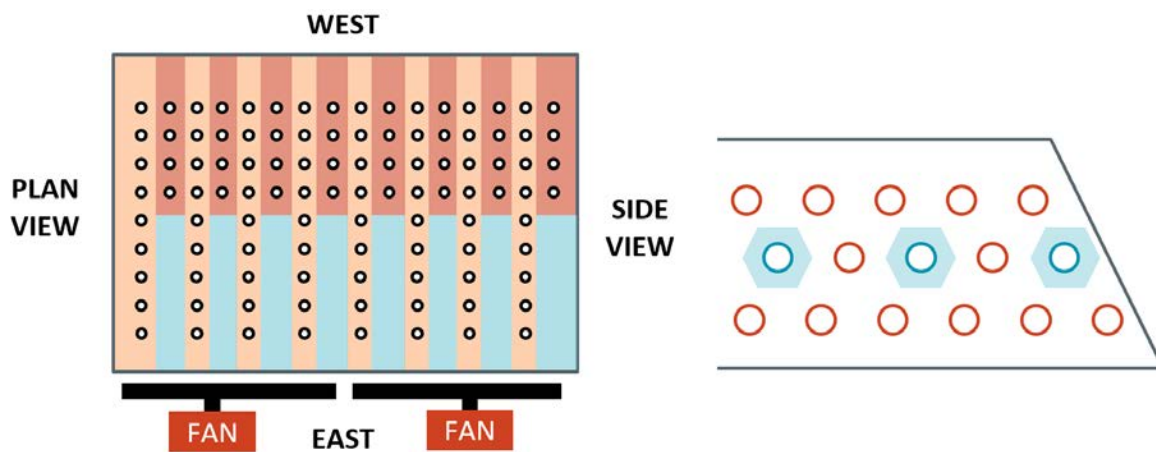


Figure 3: Special case, heap with all blowers on one side of the heap, different aeration setup, where part of the air lines supply air straight to the western half of the heap

Extra air

Over the last four years, Terrafame has focused on boosting aeration capacities for each primary heap. Whereas the aeration supply network (Figure 4) is currently close to an optimal level in terms of cost-efficiency, it has been identified as a developmental potential, with the increase in absolute air volumes blown within the heaps.

As of now and in the past, increasing aeration capacity in the primary heaps has been one of the contributors to enhanced leaching performance and acceleration of yield gains. The additional air volume fed into the heaps is seen to accelerate the leaching reactions, and increase heat formation and evaporation. The aeration capacity achieved for the primary heaps as of 2022 is +100% of the original capacity.

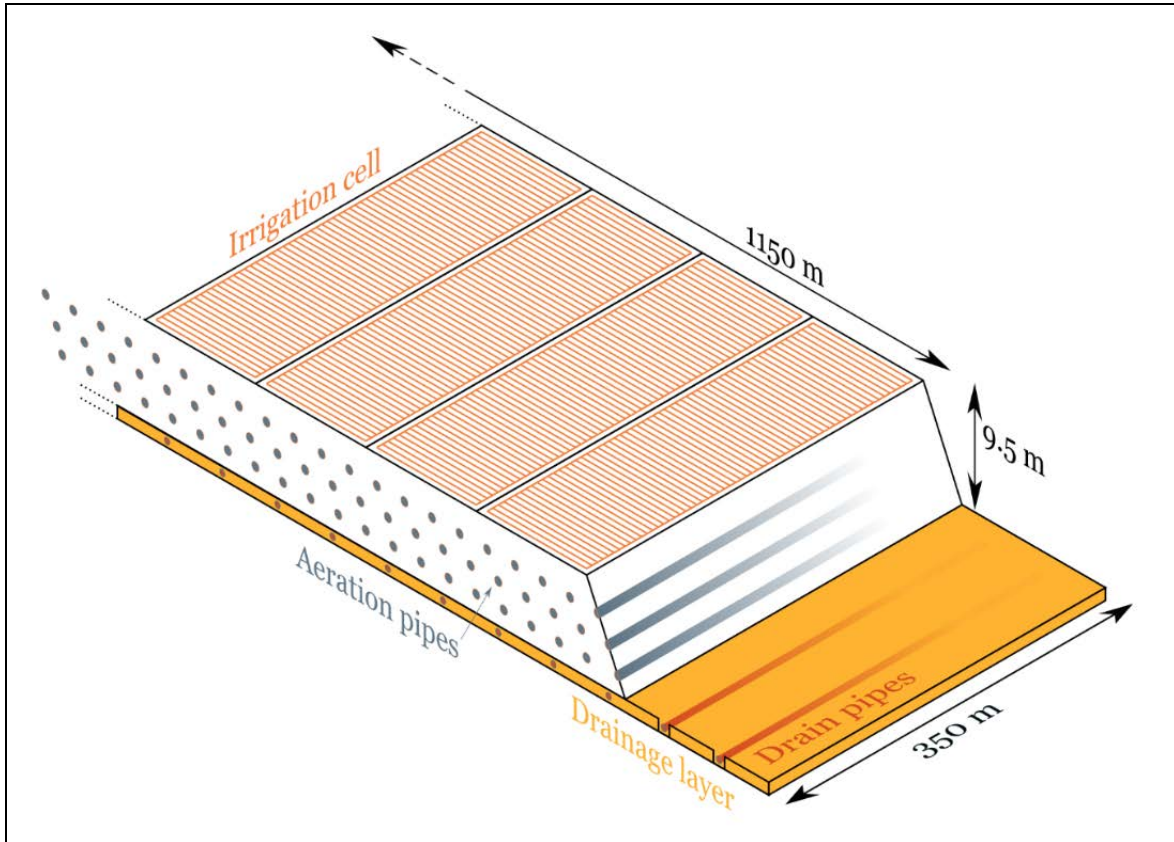


Figure 4: Primary heap installation concept, depicting irrigation cells, aeration lines, and drainage layer

Secondary leaching

Secondary heaps, pad 1

Secondary heaps pad 1 has been in use as a multi-lift heap leach operation since 2010. The secondary leaching stage hosts partially leached ore reclaimed from the primary heap leaching pad after the primary leaching stage ends. The rate of stacking at the secondary leaching pad is linked to the mine-to-heap production rate. Secondary heaps pad 1 contains four separate secondary heaps. Heaps 1 and 2 have already stacked to the level of five lifts, while heaps 3 and 4 contain four and three lifts, respectively.

The current life-of-pad plan aims to achieve nine lifts for secondary heaps pad 1 (Figure 5), which achieves the maximum height permitted by the regional environmental authorities. Progressive reduction of the top of the heap surface, irrigation pumping capabilities, or geotechnical stability of pad have not been deemed to be limiting factors for the number of lifts to be constructed at secondary heaps pad 1.

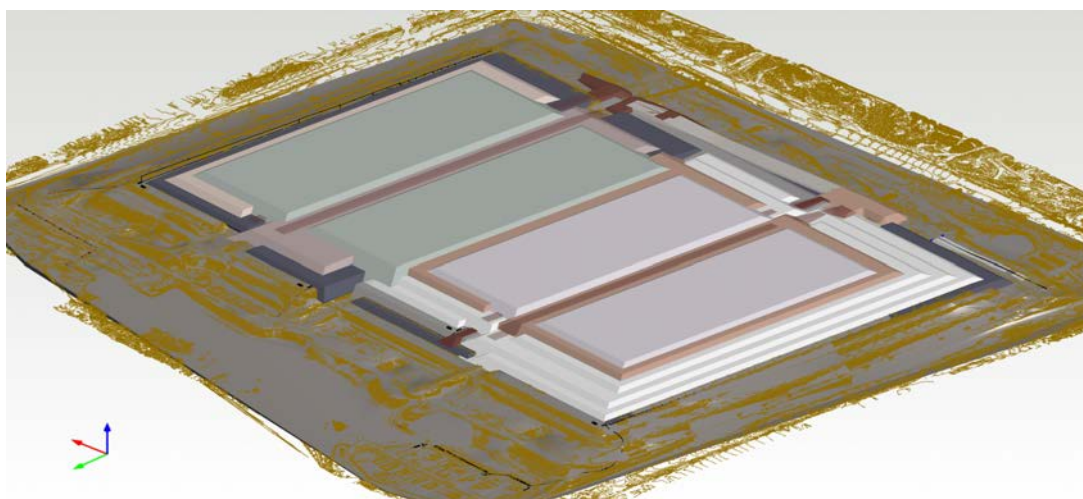


Figure 5: Life-of-pad stacking plan

Secondary heaps, pad 2

Since 2019, Terrafame has been actively designing the future secondary heaps pad 2. This second area expands Terrafame's secondary leaching stage and will be of comparable size to secondary heaps pad 1 (Figure 6). Differences in logistics and operability between both secondary leaching areas are caused by process and technology developments achieved during this last decade.

Construction of the new secondary heaps pad 2 starts during 2022. The new pad is expected to be commissioned for production during 2026, when the stacking of the first heap and first lift will start.



Figure 6: Planned location for secondary heaps pad 2

Stability of multi-lift leaching pads

Geotechnical stability control and monitoring of the multi-lift heap environment have attracted increased attention at Terrafame for the last two years. As of now, there is no evidence of imminent stability risks. The different failure mechanisms inherent to a multi-lift structure, and observed world-wide in comparable operations, have been understood by Terrafame, but not yet observed. Internal and external partners agree that a solid stability monitoring program and in-house expertise buildup are necessary, as Terrafame's multi-lift pads grow in height.

Besides daily field inspections, our means for monitoring multi-lift heap movements are currently focused on medium-term deformations, and the method used is Interferometric Synthetic Aperture Radar analyses (InSAR and derivate interpretations). This method evaluates differences in satellite-bound radar imagery with a visitation frequency of every six days. The method works well during the warmer half of the year, and millimetric movements can be identified if the movement persists in time or accelerates.

During the winter months the coherence of radar data is reduced significantly, which limits the ability to monitor the heap stability with InSAR. To improve reflectivity and data coherence over the winter months, Terrafame is currently installing physical corner reflectors at several key locations. This will improve the consistency of InSAR analysis, as well as reactivity if a continuous lift movement develops.

Challenges

As an integral part of any operation of this scale and complexity, there are inherent challenges with different natures and origins, which must be addressed organically. Terrafame is no different in that regard, and expert and production teams work towards improving process and cost efficiency. In this section, we present two examples of operational challenges that the company addresses continuously.

Gypsum

Calcium-containing minerals are leached and the production of gypsum is continuous. Gypsum precipitation in the irrigation pipeline is significant, and the leaching operation demands intensive monitoring and maintenance of the irrigation equipment. Each year the consumption of the irrigation dripline is around 40 million km. The development of crushing and washing processes for the used driplines started in year 2018, and currently most of the dripline (~70%) is recycled.

Clogging is also observed in the whole solution circulation equipment, including pumps and valves. Research and development work is ongoing to resolve the challenges of both process and maintenance operations.

Water balance

As described by Arpalahti (2016), the mine is located in a region where the yearly rainfall/evaporation is

net positive. However, the continuous production enables good processing conditions in the heap leaching, which generates enough heat to evaporate the extra water. The annual change in the precipitation amount significantly affects the mine solution balance. In 2016–2018, the solution balance was negative, enabling the company to decrease the volume of stored solutions in the area. In the years 2019–2021 the balance has been positive, forcing the company to use extra capacity in the solution purification processes. During recent years, about 8 million m³ of purified water has been released annually into nature. Water management is currently well organized, and the amount of contaminated water in the area is considerably less than in earlier years.

Battery chemicals

Terrafame decided to invest in a battery chemicals plant in 2018. Construction started in 2020 and the facilities were ready in June 2021.

Nickel and ammonium sulfate production started in August 2021, and cobalt sulfate production in February 2022.

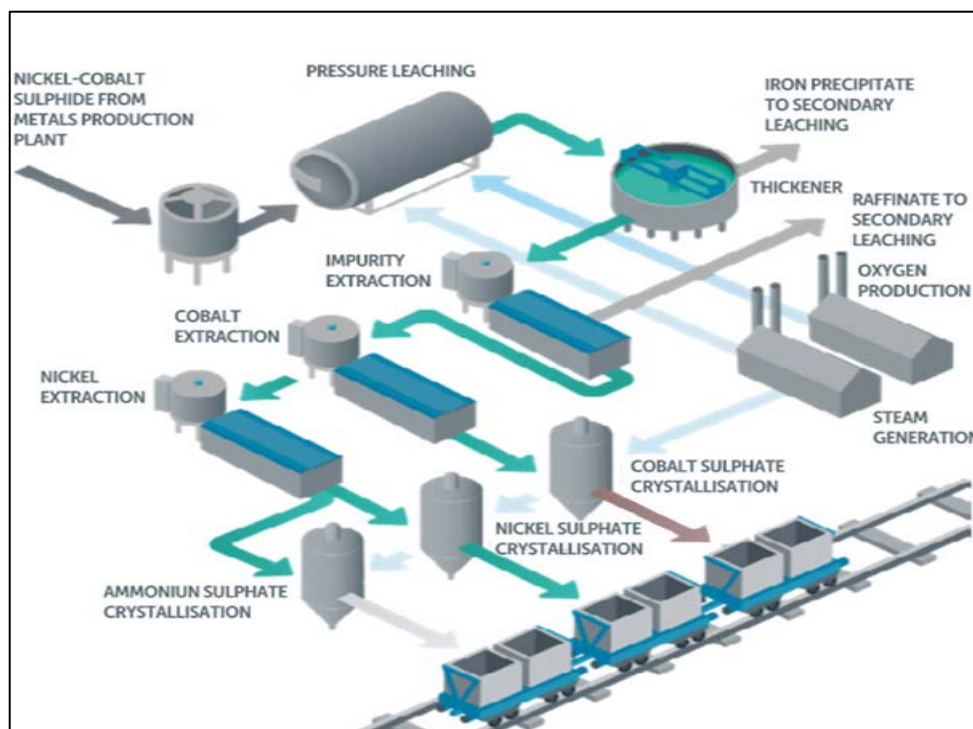


Figure 7: Battery chemicals process

The production capacity of nickel sulfate in the spring of 2022 was approximately 25% of the maximum capacity. Ramp-up of the production to the targeted capacity of NiSO₄ 170,000 tonnes per year and CoSO₄ 7,400 tonnes per year is ongoing.

Uranium recovery

A uranium plant was built in 2011 by the previous owner of the mine. The plant has not started operating yet. However, equipment checking, updating, and planning for start-up is ongoing, and the start-up schedule will be set up in 2022.

Production statistics

Production of metals in 2021 remained at the same level as in the three previous years. Nickel production was 28,582 (28,740) tonnes, representing a decrease of 0.6% year-on-year. Zinc production was 54,353 (55,100) tonnes, which was a decrease of 1.4% year-on-year. The production ramp-up of the battery chemicals plant began in June 2021.

In 2021, net sales of the battery chemicals business were EUR 27.1 million and net sales of the metal intermediates business were EUR 351.3 million. In 2020, total net sales came from the metal intermediates business.

Conclusions

Since taking over the operation of Talvivaara mine in 2015, Terrafame has continuously increased the production volumes of both nickel and zinc. Currently, production is close to the planned capacity. The building of a battery chemicals plant to produce chemicals for the continuously growing electric car industry has been successful, and the production ramp-up is ongoing. Simultaneously with the successful operation ramp-up, excess water at the mine site has decreased from approximately 10 million m³ to 3 million m³. This is because it has been possible to pump large quantities of water from, for example, the open mine pit to the bioleaching process, in order to make up for some of the liquid volume evaporated from the heaps. Between 2016 and 2018, evaporation from the bioleaching pads was higher than precipitation from rain or snowfall, producing a negative water balance. Continuous process development is essential, as the enhanced bioleaching process has also caused new operational challenges, such as gypsum precipitation.

In the near future, Terrafame plans to increase the primary leaching residence time to enhance the leaching process and improve the efficient use of the mine's mineral reserves by recovering natural uranium from the mine's process solution. The mine has been granted permission to start operating an extraction-based uranium recovery plant. An investment decision on the plant will be made in 2022.

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Geotechnical Engineering: The Observational Method Applied to Monitoring of Slopes during Operational Activities for Heap Leach Pad Facilities

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Abstract

The soil and rock materials within heap leach facilities (HLF) are heterogeneous, and precise assignment of their engineering geotechnical material properties is challenging during the design phase prior to operations. As such, it is critical to develop a comprehensive, robust “Observational Method” program using various methods to monitor the continually changing conditions of the HLF during operation activities. Prior to significant movements, there are typically measurable deformations, pore pressure changes, or other observable phenomena that provide indications of potential slope instability or HLF damage. Therefore, ongoing routine engineering oversight and investigations are incorporated into an Observational Method program to enable engineers to detect these continually changing conditions due to changes to the engineering geotechnical material properties (i.e., permeability, shear strength, material behaviour) during operations that may lead to slope instability.

To minimize this risk of slope instability, the Observational Method is used to verify that the construction and operation of the HLF is continually managed as discussed with the Engineer of Record (EOR) and/or within the intent of the engineering parameters from the design criteria. The design criteria are determined based on completing a thorough geotechnical investigation with surface and subsurface collection sampling, as well as laboratory testing programs, during the initial design stage and construction phases.

In addition, as part of this management, the geotechnical and operational teams in conjunction with the EOR use a series of overlapping collection methods for data sources, which generally consist of, but are not limited to, geotechnical investigations (sample collection and laboratory tests), visual inspections, pore pressure monitoring, and deformation monitoring. Monitoring should be performed on a regular schedule (i.e., hourly, daily, weekly, monthly, etc.) dependent on the data collection method, even if slope instability is not indicated, so that baseline data are collected and early slope instability data are detected.

Methods for sources of data include regular visual inspections by the geotechnical and operational teams, pore pressure measurements from piezometers, and deformation measurements using equipment such as, but not limited to, Global Positioning System (GPS) units or survey prisms, inclinometers, or shape arrays, and/or Interferometric Synthetic Aperture Radar (InSAR), Light Detection and Ranging (LiDAR), or radar.

The geotechnical and operational teams, in conjunction with the EOR, utilize these data to develop critical slope stability models regularly spaced around the perimeter of the HLF, or critical locations as discovered during the Observational Method. Afterwards, slope stability model results are used to identify data gaps along each critical section, which are then circulated back into the Observational Method monitoring program. In addition, there may be a need to collect in-situ data via cone penetration tests (CPT) or drilling to collect samples for laboratory testing and/or install additional instrumentation, such as piezometers or GPS units, to provide more information to incorporate within the slope stability models and evaluation of the results.

Furthermore, slope stability model results and field observations are used to establish trigger levels to allow for proactive slope stability observational monitoring and provide for timely responses to changes measured by instrumentation (i.e., piezometers and deformation devices). Lastly, this interactive system is utilized to develop mitigation measures (i.e., changes with irrigation areas, irrigation rates, drainage systems, buttresses, etc.) to maintain the local, intermediate, and global slope stability of the HLF.

Introduction

The Observational Method was best described in the 1969 Rankine Lecture presented by Ralph Peck (Peck, 1969), regarding its use for the stability evaluation of a facility during construction where reliable factors of safety may not be readily quantified due to changing conditions in the field. A permanent HLF is under continual construction during the facility life as new ore is loaded onto the top of the pad, and irrigation rates as well as areas under irrigation vary and continually change across each lift. It is unlikely that a detailed design and engineering program will be able to accurately predict the engineering geotechnical material properties of the entire ore body prior to the start of mining, or estimate the conditions under which the facility will be operated during the life of mine. A comprehensive, robust observational monitoring program is required to provide sufficiently reliable data to evaluate the performance of the HLF during the life of mine with stacking and irrigation processes.

During the life of the HLF, these visual inspections and collected data (i.e., laboratory, instruments, etc.) as part of the Observational Method are utilized to adjust and reset the acceptance criteria originally determined during the design, and reassessed as necessary through the continual use of the Observational Method. As the amount of data collected from the monitoring program increases, the acceptance criteria for HLF performance may become more critical to the operation of the facility than the factor of safety for

slope stability. Controlling seeps, erosion, and/or preventing large rates of deformation become the main focus of the Observational Method, using trigger levels as the focus for proper and safe operation of the HLF on a daily basis.

Geotechnical investigations

Initial geotechnical investigations during the design stage focus primarily on evaluating the geotechnical foundation conditions of the HLF and the geomembrane liner system. It is unusual to obtain fully representative samples of the ore material at this stage, and these are primarily sourced from core samples or metallurgical column test samples. This limited number of samples, which also tend to be limited spatially within the ore body, likely does not reflect the full variability of the engineering geotechnical material properties of the ore as stacked and irrigated during the life of the facility.

Periodic geotechnical field and laboratory investigations should be completed during the operational life of the HLF to allow for the collection and evaluation for the potential range of engineering geotechnical material properties for the ore. The investigation should include the collection of relatively undisturbed samples representative of ore materials stacked within the heap, as well as identify any areas of the facility that may be saturated, or nearly saturated.

In addition, the investigation should include laboratory tests that generally consist of, but are not limited to, particle size distribution, Atterberg Limits, triaxial, direct shear, etc. The collection and laboratory tests of these representative samples are particularly important as engineering geotechnical material properties will change over time due to the process of decrepitation, which is the mechanical and geochemical breakdown of ore during ore processing and leaching. Decrepitation is of particular concern when acid solutions are used as part of the leaching process where there is a high degree of geochemical breakdown of the ore.

An additional tool that can be used to develop engineering geotechnical material properties is the cone penetration tests (CPTs), if the HLF has a crush size to allow for pushing of the cone. CPTs can provide data on material behaviour types (Robertson, 2016), shear strength, pore pressure conditions, and other index properties of the ore. Periodic CPTs can also allow a geotechnical engineer to track if material behaviour changes over time (i.e., soils change from contractive to dilative, or sand-like to clay-like behaviour).

These behaviour types, as shown in Figure 1, are also useful to identify zones of the HLF that should be sampled over time, installation of piezometers, and identify potential areas of concern for further slope stability analyses.

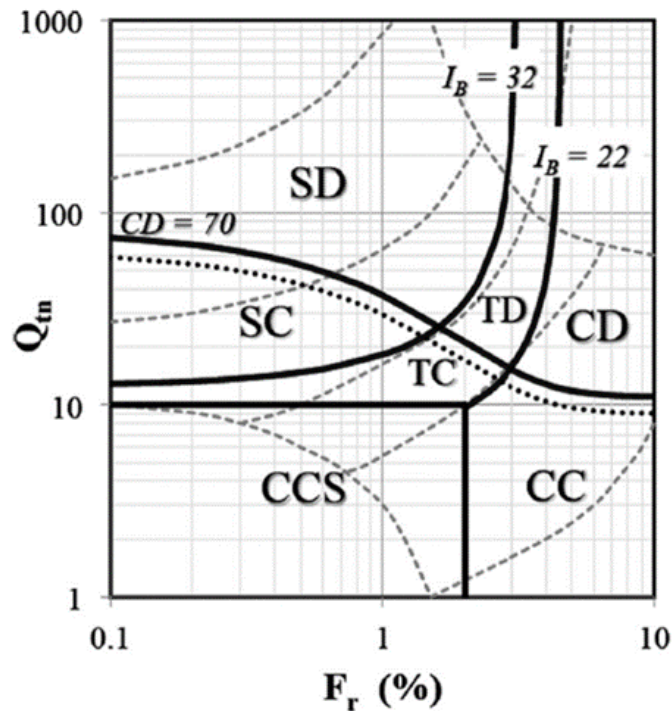


Figure 1: Robertson (2016) normalized CPT behaviour type (SBT_n) chart

Visual monitoring

Visual monitoring provides direct observations of the HLF performance and allows operators to evaluate whether conditions have changed as compared to the expected conditions defined per the initial design criteria. For example, seepage reporting to the slope face could be a direct indication for the buildup of pore pressure within the HLF. A majority of designs for HLFs assume a free draining ore, and any pore pressure within the heap would constitute an abnormal condition. In addition, identification of cracking, bulging, or scarp development could be an early indication of slope movement.

The operation teams of the HLF should make periodic (i.e., weekly) visual inspections of the HLF to evaluate if any unusual or abnormal conditions exist. Unusual or abnormal observations include, but are not limited to the following:

- surface erosion
- gullies
- cracking
- scarps
- displacement
- bulging

- settlement/subsidence
- sinkholes and depressions
- piping (internal erosion)
- sloughing and slumping
- ponding on benches or near perimeter crest
- ore outside of the footprint of the pad
- seepage exiting the pad face
- vandalism or wildlife damage
- other abnormal conditions.

Regular visual inspections are the best way to identify potential slope instability or conditions that could lead to slope instability, as the majority of the surface of the HLF can be inspected. Monitoring using instrumentation can be limited as they are typically a single point in space and may not be currently installed in an area of concern. However, the use of instrumentation may also indicate specific areas that the operational team should investigate as part of the visual inspections. Furthermore, the visual inspections and the monitoring using instrumentation may dictate locations for geotechnical investigations.

Pore pressure monitoring

Piezometers are the primary method for measuring water levels or pore pressure within the HLF. Water levels are typically measured using open standpipe piezometers with a short section of screened casing that is sealed at the top of the screened interval to prevent water higher within the heap from infiltrating along the casing and reporting to the screened interval. Multiple open standpipes can be installed at a single location to provide information on the gradient of solution flow or determine if the water within the HLF is static or perched.

Pore pressure is measured using either vibrating wire piezometers or pressure transducers that are buried within a borehole so that they have direct contact with the ore and pore water. This allows for the immediate measurement of pore pressure changes where water levels measured in open standpipe piezometers will take time to fill up the piezometer casing through time in response to changes in pore pressure.

Figure 2 illustrates a piezometer time history plot where the measured pore pressure (orange line) is nearly at a trigger level (green line) and is actively being managed by pulsing irrigation to maintain an adequate factor of safety with respect to slope stability.

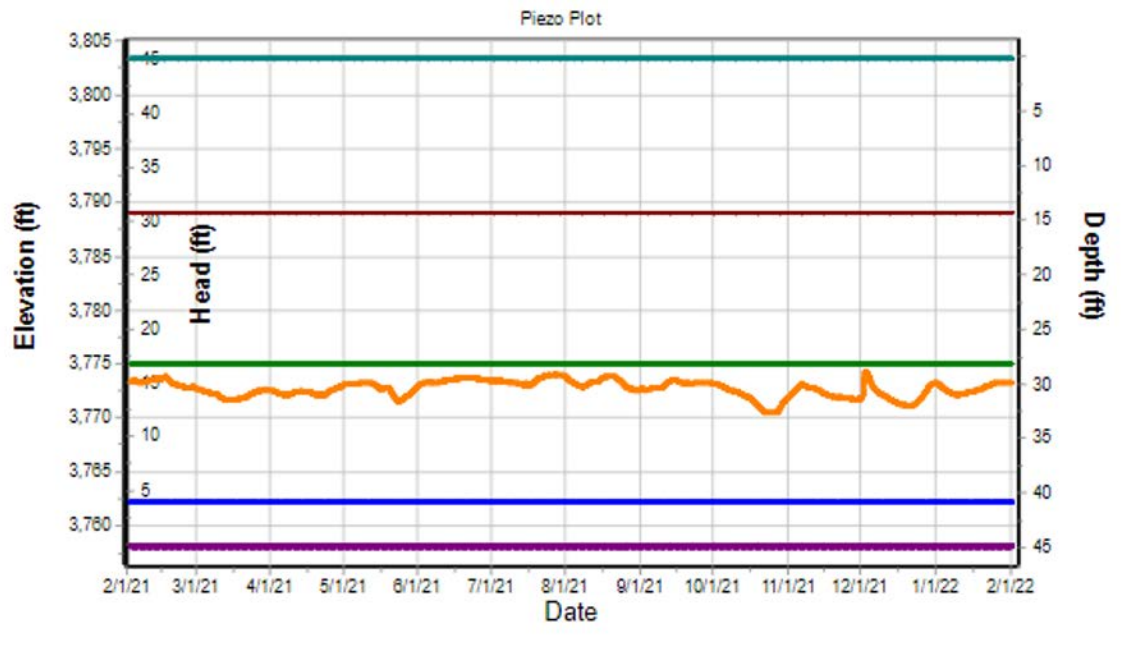


Figure 2: Piezometer time history plot

A companion method for evaluating pore pressure conditions within a HLF is the use of geophysical methods to identify areas of potential saturation using electrical resistivity measurements. This method allows for measurements taken along cross-sections of the HLF that can span across the entire height or length of the heap. Multiple cross-sections can be used to obtain an estimate of saturated areas, which can then be targeted during subsequent geotechnical investigations to confirm the presence of saturated conditions or identify areas for the installation of piezometers to measure the pore pressure.

Deformation monitoring

The intent of deformation monitoring is to observe the HLF and check that the performance and behaviour are as expected with the design intent so that the facility remains safe and operationally efficient. As a benefit, it also provides early warning of slope instability along the HLF. Remote deformation monitoring requires a variety of methods to develop a wholistic view of the response of the HLF to continuous stacking and irrigation of the ore. Each method of measurement provides a different scale of view, accuracy, frequency of measurement, and requirements for interpretation. They can also be used to corroborate the measurements from other instruments.

Any of the below methods have limitations with respect to accuracy of measurement that are often site-dependent, and it is critical that more than one method of deformation monitoring is used to verify measurements. For instance, InSAR can provide deformation measurements with sub-inch accuracy across areas of several square miles as long as the viewing direction of the satellite is oriented correctly with the

HLF. Furthermore, it may not always be obvious that the measurements are accurate and another method such as survey prism or GPS units should be used to confirm measurements in areas of abnormal deformations.

GPS units

Deformation measurements are collected by placing a GPS unit at a single fixed location, similar to a survey prism, and can be placed on a survey rod that can be moved to take measurements at multiple locations as required. The GPS unit simultaneously communicates with a nearby base station that corrects for any induced error by the satellite system providing the signal.

Stationary GPS units are permanent installations at a single location that continuously take measurements of the given location. Each unit takes numerous readings every few minutes and averages those readings every few minutes or hours to provide a more accurate location of the GPS unit. Readings can be automatically uploaded to a network server for review by the operation teams.

Survey prisms

Prisms are reflective mirrors attached to a frame that is anchored to the ground to allow for measurement of surface movement of the HLF. A total station instrument uses a laser that reflects off the prism's mirror and measures the distance between the total station and the prism. Measurements taken over time allow for estimates of the direction and rate of deformation. The total station requires relocation to different parts of the facility to obtain line-of-sight for each prism due to the size and shape of each facility.

Typically, a site uses a permanent total station location to provide more consistent measurements, reduce the error between measurements, and reduce the time required to collect measurements. However, this arrangement may require installation of several permanent total stations to monitor an HLF.

LiDAR

Fixed LiDAR stations are similar to total stations except they do not require a prism as a reflector. The energy projected by the LiDAR instrument is sufficient to obtain readings from a wide range of surfaces. LiDAR is also used on mobile equipment, typically truck mounted or unmanned aerial vehicle (UAV)/drone mounted.

This allows for rapid measurements across a large area, provided there is adequate line-of-site between the mobile equipment and the area under survey. UAVs provide the greatest opportunity for making large LiDAR surveys.

InSAR

InSAR satellite imagery is used to evaluate the deformation of HLFs across large areas. Data is collected

as a satellite passes over the mine site in either a descending or ascending flight path. Each flight path has a different orientation with respect to each facility and takes measurements. The accuracy varies per facility due to the relative orientation of the facility to the flight path. Each mine has areas without any measurements taken when the relative position of the area does not have a direct line-of-sight to the satellite flight path.

Radar

Radar measurements are made in the same fashion as InSAR except from a land-based position, which allows for greater accuracy and frequency of measurements. Fix monuments are not required as readings are taken from any point on the ground within the line-of-sight of the radar instrument. The rapid measurements taken can provide highly detailed maps of deformation across large areas and typically are used for monitoring active slope failures within open-pit mines. This allows for the rapid identification of movement and quick response to those movements.

The evaluation of the movement rate is primarily dependent on the time required to receive the data from the instrument and processing time for the quantity of data collected. Radar is significantly limited to measuring small incremental movements. However, the number of measurements taken often offsets this limitation by comparing snapshots of measurements through time to develop total displacement measurements and an average displacement rate as well as an inverse velocity.

Inclinometers

Inclinometers are used to measure the tilt of a structure and the occurrence of shear planes, typically below ground surface, using a casing with aligned grooves and a probe installed within the grooves. Subsequent measurements allow for a calculation of the differential tilt as well as an estimate of the rate and direction of deformation.

Measurements can be taken manually with a probe being lowered and raised in each pair of grooves, a permanent installation using a string of probes locked to the grooves, or a shape array installed within the casing but not within the set of grooves. The manual probe and fixed probe in the grooves are limited to measuring deformation in the orientation of the grooves, which requires an interpretation of the data to estimate the direction of movement.

The shape array allows for measurement of deformation in any direction. Figure 3 provides an example inclinometer plot with measured displacement indicating the potential development of one or more shear planes.

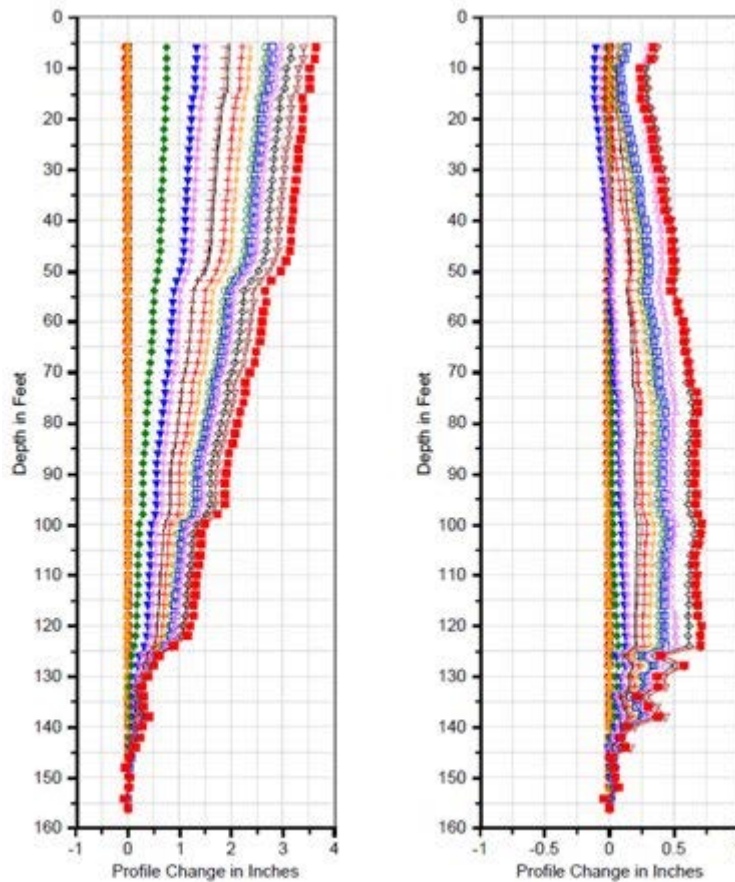


Figure 3: Example inclinometer plot indicating potential shear plane development

Inclinometers are typically installed into a layer of soil or rock that is estimated not to move as part of the general deformation for the HLF monitored to allow for valuations of total deformation. However, this is problematic for geomembrane-lined HLFs where the inclinometer cannot anchor unless anchored through the geomembrane liner. These instruments can still be useful to evaluate mid-slope deformation, but cannot be used for total deformation estimates if there is any movement along the geomembrane liner surface.

Slope stability modeling

The data collected from the Observational Method monitoring program should be used to develop, and over time, refine slope stability models along critical sections to define factors of safety under existing conditions and evaluate pore pressure conditions for maintaining acceptable factors of safety. The slope stability models should include ore material characterization developed from the geotechnical investigation (including laboratory tests and CPT results), pore pressure conditions, and slope geometry of the existing HLF as well as expected geometry during the near future. Potential failure modes can be evaluated with the

model utilizing data obtained from the various methods. Deformation monitoring might indicate zones of high deformation or the development of a failure plane within the HLF. Trigger levels for pore pressure monitoring should be developed for pore pressure conditions to allow the operating team to evaluate whether changes of pore pressure will lead to unsafe slope stability factors of safety.

Figure 4 provides an example slope stability model with material behaviour types developed from CPT and a network of piezometers used for development of pore pressure conditions.

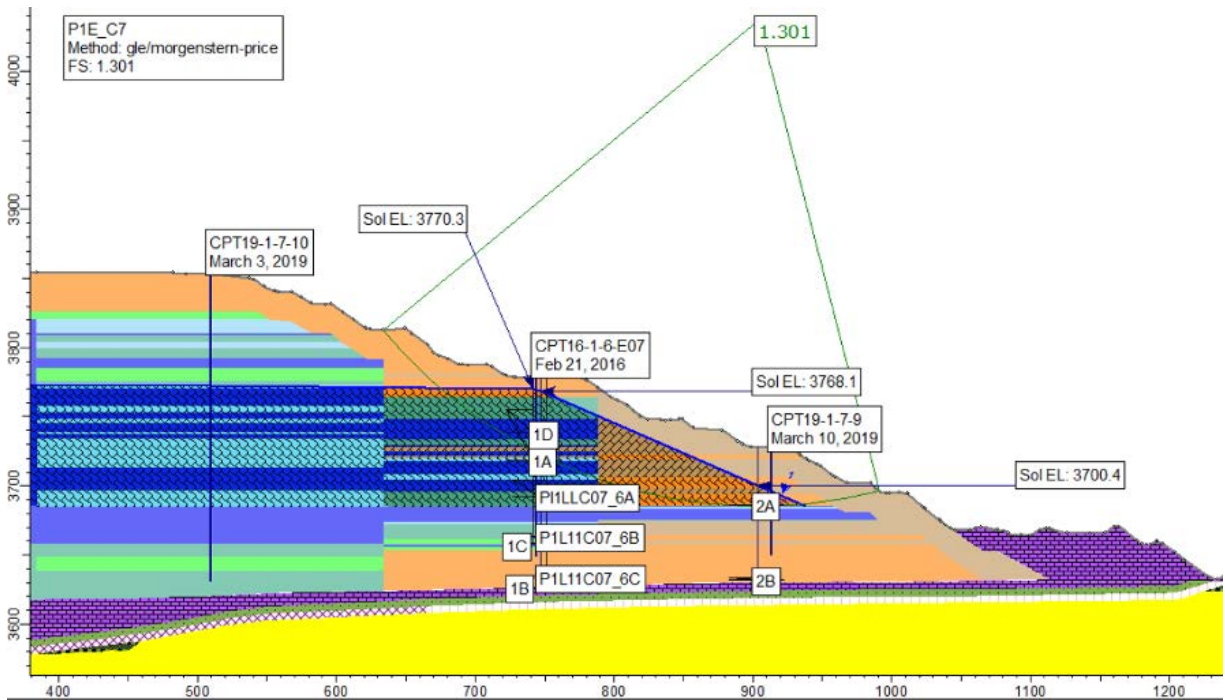


Figure 4: Example slope stability model

During the model development, each critical section should be evaluated to determine if there is sufficient data to accurately represent the subsurface conditions for both material behaviour types and for pore pressure conditions. This evaluation can be used to identify future geotechnical investigation locations for collecting samples to perform laboratory tests or CPTs, as well as where piezometers should be installed to provide additional detail about the pore pressure conditions.

The results of the slope stability models can also be used to evaluate and design mitigation measures if minimum factors of safety are not reached. If piezometers rapidly react to changes in irrigation, then irrigation rates could be reduced to decrease pore pressures (refer to Figure 1) along a critical stability section. If seepage at the slope face is identified as the critical condition for failure, then drainage systems (i.e., French drains, horizontal drains, wells, collection pipes, etc.) could be installed to control the seepage. If conditions continue to deteriorate, then a buttress could be designed to improve factors of safety for

current conditions and anticipated future conditions to maintain slope stability during the remaining life of the HLF.

Acceptance criteria

The results of slope stability models, data collected from instrumentation (i.e., piezometers, deformation, etc.), InSAR, and visual monitoring are used to generate acceptance criteria (identified through key performance indicators) that are likely to govern daily performance and maintenance of the HLF facility more than by factors of safety alone. For example, seepage flow rates, erosion, and height of saturated ore within the slope face will dictate local slope stability that might not be captured by assumed critical stability cross-section locations.

As another example, deformation rates that exceed previously measured rates may indicate a zone of instability. These changes may occur during time frames that do not allow for updating slope stability cross-sections or locations with insufficient data that factors of safety cannot reliably be calculated.

Conclusions

Operational variability among ore engineering geotechnical material properties and operating conditions (i.e. stacking rates, irrigation rates and timing, rates of decrepitation, etc.) over time will likely require re-evaluations of design assumptions through the life of a HLF. The Observational Method provides a framework for the geotechnical investigation associated with the collection of data that can be used to verify initial design criteria or develop new design criteria based on actual field performance of a HLF.

The key to the Observational Method is having sufficient data to justify a need to change the initial design criteria that was used for the design of the HLF. The Observational Method is utilized to possibly adjust and reset these acceptance criteria originally determined during the design and reassessed as necessary through the continual use of the Observational Method. The data also needs to be verifiable from different sources as there are always inherent errors in measurements and observations.

Therefore, a comprehensive, robust Observational Method program having more than one method and source for evaluating pore pressure conditions (seepage mapping from visual monitoring, piezometers for direct pore pressure measurements, geophysical methods for area of extent measurements and pore pressure distribution based on CPT) and/or deformation (visual monitoring for unusual cracking, GPS units for displacement measurement, and InSAR for area of extent measurement) is critical to verifying field conditions, performance and slope stability models for the continual monitoring of the HLF to maintain slope stability.

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See the Metal You Left Behind: In-Heap Sensors for Monitoring Fluid Flow

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Abstract

Through analysis of data collected from wireless sensors embedded inside a heap leach pile at an operating gold mine, we demonstrated there was 9.6% more recoverable gold available to be leached. Further, we identified an additional 0.7% recoverable gold on the edges of the leach pad. We estimate process operating cost savings of 13% can be realized by reduction of the reagents and energy used to leach metals.

The geotechnical stability of a heap dictates that the flow of solutions through it should occur under unsaturated conditions. The two key concerns are adequate flow and uniform flow through the heap. Adequate flow is necessary for the heap to be leached in an economic time, while a uniform flow is needed to allow all the ore to be thoroughly leached. If the solution does not flow evenly throughout the heap and instead flows preferentially through distinct paths, portions of the heap will not receive sufficient contact with the solution and will remain dormant.

Through the review of independent scientific literature and our own column testing, we have confirmed that moisture variation by 25% affects gold recovery by up to 69%. We also have observed that moisture within a heap is not uniform and varies by up to 50%.

Scanimetrics has developed a wireless hardware and data analysis software solution for assessing the leaching effectiveness of a heap leach operation and predicting yield. The system measures moisture and temperature of ore at the surface or inside a heap leach pile in real time using wireless sensors. The data is collected from the sensor network to a cloud server, analyzed and made available through a web application. The heap leach data are used to increase productivity, lower operating costs, improve worker safety, and lower the cost of regulatory compliance. In addition, operators can be notified electronically when critical heap leach parameters are outside their normal operating range.

Introduction

The goal of this work was to improve the leaching process and, in doing so, increase the yield of this process by producing more precious metals.

This was the first of several undertakings to evaluate leach processing to improve yield, through characterizing and modeling key heap dynamics. This work focused on conductivity, considered as one of the most important heap variables, and its impact on yield. The hypothesis is that, by characterizing moisture dynamics within the heap, we can correlate it to the irrigation process to achieve better moisture uniformity and yield.

The project shall explore the following heap dynamics:

- heap conductivity profile;
- heap conductivity curve;
- heap density profile.

The work characterized these heap moisture dynamics, in the context of the existing profiles for density and mining process. The premise is that better management of the irrigation system, through sprinkler head placement and flow rates, should result in higher yields of the leaching process. From this data we hope to model:

- heap irrigation effectiveness;
- heap geotechnical integrity;
- and their relationship to heap density profile.

This work does not intend to make recommendations regarding other mining process such as crushing and stacking; that may be the focus of future work. Nevertheless, we monitored the density since it may seriously affect heap conductivity.

The scope of work included installing sensing equipment and wireless data logging devices called “Motes” (see drawings and pictures in Appendix A for more details). Data from these sensors were collected and stored in the Motes and then transmitted via a wireless connection to an Internet Gateway. The Internet Gateway uploaded the data to the “Scanimetrics’ Cloud,” which reports to a web application. The Internet Gateway communicated to the Scanimetrics’ Cloud via an Internet connection. A local Wi-Fi network facilitated communication from the Internet Gateway to an internet access point at the mine site.

The sensors and Motes were placed in an array of commercial grade urethane piping and covered as the heap was built. The array’s design was to protect the moisture sensors from damage, either by the pressure of surrounding ore, or the movement of ore as the lift was built.

Each Mote can be programmed to an appropriate sample rate or for a specific time interval. The Motes were programmed to sample continuously once per minute for this application.

This work required router/repeater devices to improve communications between the Motes and the Gateway. The Motes generally need to be placed within 10 to 20 meters (m) of the Internet Gateway to communicate directly with it.

Methods

Column testing

Scanimetrix provided the necessary supplies for installing sensors into columns at the mine site laboratory to measure and evaluate the moisture vs. gold recovery of the columns at different moisture content profiles.

Two columns were used, 6 feet high by 6 inches in diameter, using an automatic drip system with variable drip rate of 4 to 16.1 L/hr/m². The leaching solution consisted of NaCN and was adjusted for pH to reduce the production of HCN. A solution collection tank was used for each column.

Ore was uniformly mixed, and large particles greater than 1 inch were removed. The weight of the ore was recorded before drying and then the ore samples were dried using a kiln at the mine laboratory. A gold assay was performed on a sample of the ore. The weight of the dry ore was recorded. The initial moisture level was recorded using a moisture sensor.

The ore was placed into each column, with a moisture sensor near the bottom and a second sensor near the surface. The depth of the sensors relative to the top of the ore was recorded. Drip rates were set by the mine laboratory for each column and dripping was initiated.

The laboratory maintained a daily record of the volume of solution collected from each column and collected a sample for gold assay.

At the end of the test period the column drips were stopped, the ore sample weight for each column was recorded, the ore was dried, and the dry ore weight was recorded.

In-situ heap testing

Scanimetrix provided the necessary supplies for the installation and did the system validation checks once the sensors were assembled and embedded into the heap. Scanimetrix assisted the mine operator in integrating the sensor array to connect to the mine Internet.

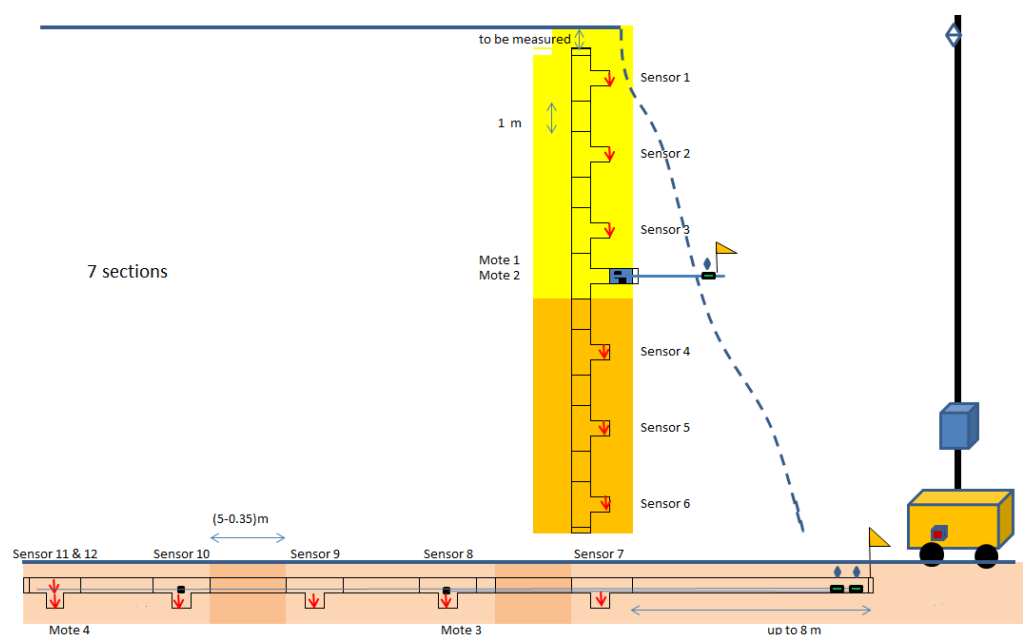
A web portal was provided to allow the processed data and reports from the Scanimetrix' Cloud to be accessed from any computer through a secure Internet login.

The sensor array was comprised of 15 moisture sensors, six Motes, three routers and one Gateway, with Wi-Fi interface to the mine Internet.

The mine collaborator indicated that cellular service is unreliable at the mine site; therefore, we deployed a Wi-Fi service in the Internet Gateway and link to the site-wide Wi-Fi network, extending it to the heap location using a point-to-point link. In addition, a site survey was performed to establish the optimum line of site placement of the Internet Gateway and routers.

The leach pad is approximately 5 m in height, and the bench slope face was estimated to be about 45 degrees at the sensor's vertical position. The top sensor (Sensor V1 in Figure 1) is approximately 2 – 3 m inside the perimeter of the leach pad.

Regarding the vertical column of sensors, the sensor located at position V1 is approximately 0.3 m below the pad's surface. The vertical distance between every two sensors in the vertical column is approximately 0.95 m (i.e., $1.35 \times \cos 45^\circ$) except between sensors 3 and 4, which is approximately 1.2 m. The compaction sensor is located 0.95 m below Sensor V3.



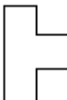
Description		Qty	Description		Qty
	HDPE Cap	12		Router	3
	HDPE Tee	12		Router antenna with 6' cable	3
	1 m pipe	5		Mote	4
	0.25 m pipe	1		Sensor with 6 m cable	12
	4.65 m pipe	4		Gateway	1
	8m	1		WIFI antenna	2
				Outdoor UPS supplied by customer?	1
				Coaxial cable 30 m, 15 m & 5 m	45 m
				Flag	2
				Generator	1

Figure 1: Leach pad sensor installation diagram

Regarding the horizontal row of sensors, the side denoted with a flag in Figure 1 is approximately 2 to 3 m outside the heap's perimeter, while the sensor at location H7 is approximately 7.5 m from that end. The horizontal distance between each two neighbouring horizontal sensors is 5 m.

The mote and moisture sensor were integrated into the sensor array at Scanimetrix's facility and were shipped to the mine site. Additional supplies were provided at the mine site, reducing the installation time required.

For this pilot test, urethane pipes were used solely as conduits for the motes, sensor cables, and coax cables. Four motes and the sensor cables were protected within the urethane pipes, except the sensor at location H12. Coax cables were used for coupling the wireless signals to the edge of the leach pad. Routers were placed at the end of these coax cables so the wireless signal could be re-transmitted to the Internet Gateway. In the future, we expect the motes and sensor cables will not require urethane pipes for protection. Thus, the wireless system installation could be significantly simplified.

As seen in Figure 1, the Internet Gateway is mounted on a light post with its Wi-Fi antenna communicating with an antenna located on a distant tower at the top of a hill. The Wi-Fi antenna is mounted on a lamp post fixed to a generator that provides elevation for the antenna to assure a clear line of sight and provides power to the Internet Gateway as well as the Wi-Fi antenna. In the future, the Internet Gateway can be provided with an optional solar-powered power supply to eliminate the need for a generator.

Following the installation of the sensor array and Internet Gateway, the sensor network was commissioned, and the Internet Gateway was configured to operate on the mine site Wi-Fi network.

After the systems were installed and commissioned, Scanimetrix initiated data aggregation on its cloud. The data was aggregated daily and was accessible to the mine collaborator through a secure web portal provided by Scanimetrix. Data aggregation was performed for an initial 13-week period from the installation date, followed by an additional 13-week period.

Upon network commissioning, Scanimetrix monitored the data received from the sensors. Scanimetrix generated weekly reports on equipment conditions and operational performance based on the data. The data and reports were delivered using the web portal.

To facilitate the objectives and reporting, our mine collaborator provided the following information to Scanimetrix:

- the build schedule for the current lift;
- operational hours for the lift build;
- shift data of the leaching process operators;
- the ore body composition and the density profile of the ore being used to build the current lift; and
- weather reports.

The data was accessible by the mine collaborator throughout the project through a secure web portal provided by Scanimetrix, and contained the following information:

- moisture data;
- moisture dynamics; and
- significant anomalies.

Results

Column test data

The test started on June 5th at two irrigation rates: normal (8 L/hr/m², inner column, C1) and fast (17.56 L/hr/m², outer column, C2). From the mine's standard water weight ratio measurement, the ore sample had an initial average water weight ratio of 2.54%.

Converting the water weight ratio to the Volumetric Water Content:

$$\text{VWC or } \theta = V_w/V_T = (m_{\text{wet}} - m_{\text{dry}})/m_{\text{wet}} \times (d_{\text{wet}}/d_{\text{water}}) = (PH - PS)/PH \times (d_{\text{wet}}/d_{\text{water}}).$$

For the water weight ratio 2.54%, the calculated VWC was approximately 4.22%. Here the specific gravity of the ore sample (d_{wet}) was calculated from the weight ~57.5 kg, and the volume of the ore samples: 6-inch column at the height of ~190 cm and was approximately 1.66.

In each column, there were two moisture sensors. One sensor (#13 in C2 outer column and #15 in C1 inner column) was 15 cm below the top surface; the other sensor (#14 in C2 outer and #16 in C1 inner) was 115 cm below the top surface.

We observed a 3% difference in the baseline sensor moisture values (Figures 4 and 5) using uncalibrated sensors within the sensor's specification.

The column inspection was completed on July 9th. Ore samples were removed from the columns (Figure 2). Under visual inspection, the water content in the ore samples was relatively even, and no significant water pooling was observed. The mine's standard water weight ratio measurement (shown in Figure 3) was used to calculate the ore sample's final water weight ratio, 8.9248% for C1 and 8.9549% for C2. The final water weight ratio to the volumetric water content (VWC), was approximately $16.7\% \pm 0.1\%$ for both columns. Note: the specific gravity of the wet ore in the column was increased to 1.87 due to the compaction.

From the moisture sensor data, the VWC for each sensor location is plotted below.



Figure 2: Ore sample at the end of the test

C 1	C 2
PESO HUMEDO	PESO HUMEDO
61.850 Kg	62.200 Kg
PESO SECO	PESO SECO
56.330 Kg	56.630 Kg
HUMEDAD	HUMEDAD
% 8.9248	% 8.9549

Figure 3: Ore final water weight ratio

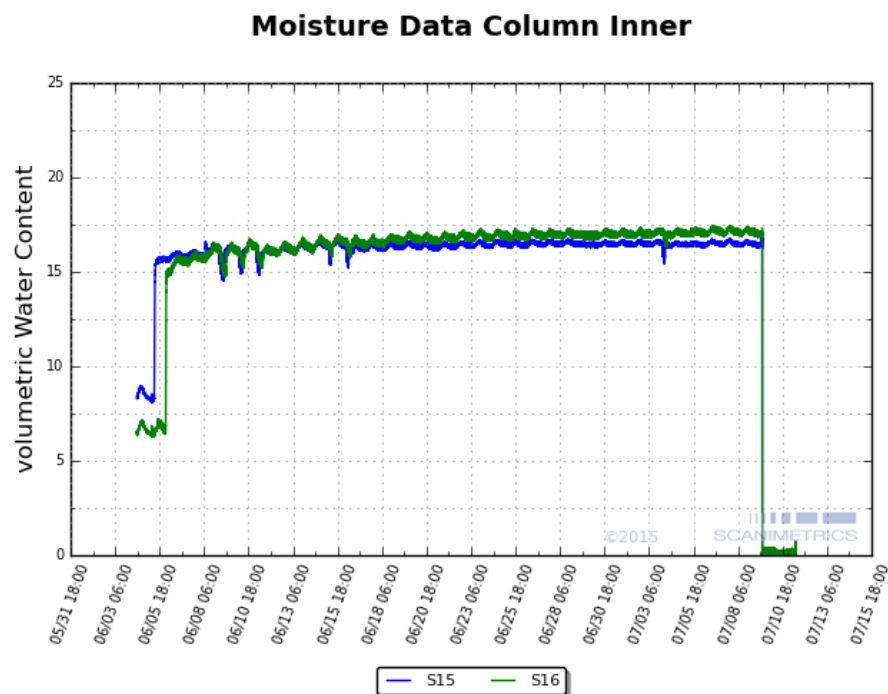


Figure 4: Moisture sensor data in the inner column C1 (8 L/hr/m²).
S15 is 15 cm deep from the top, and S16 is 115 cm from the top

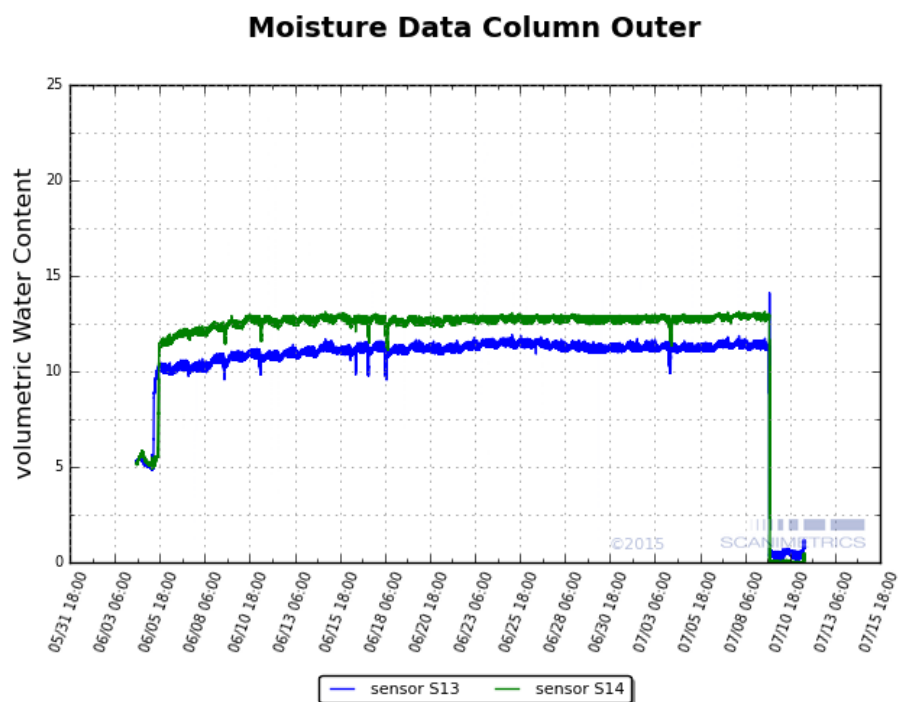


Figure 5: Moisture sensor data in the outer column C2 (17.56 L/hr/m²).
S13 is 15 cm deep from the top, and S14 is 115 cm from the top

A subsequent column test was performed using flow rates of 8 L/hr/m² and 4 L/hr/m² to avoid formation of preferential fluid flow paths. The gold recovery at these applied flow rates is illustrated in Figure 6.

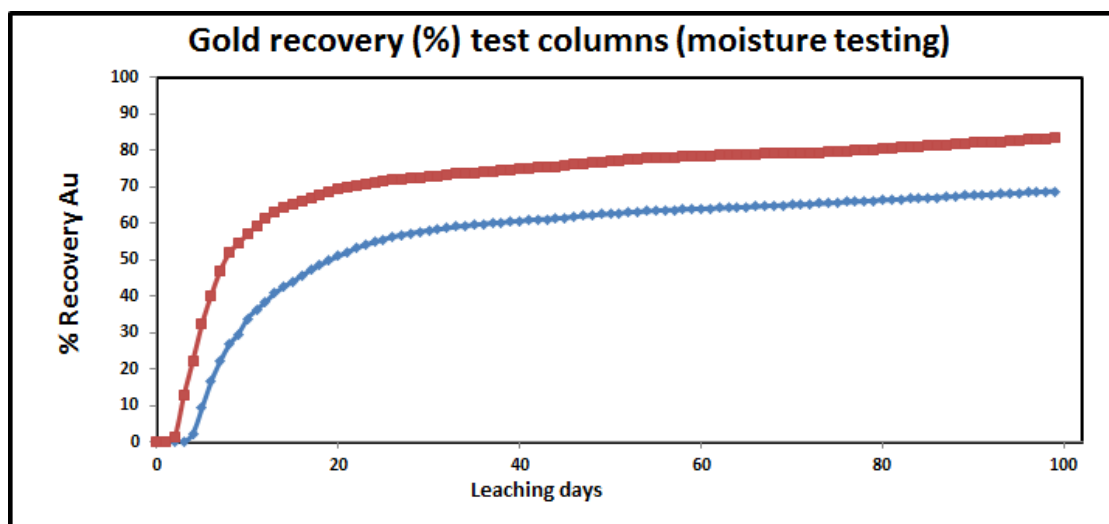


Figure 6: Gold recovery of the test columns (red 8 L/hr/m², blue 4 L/hr/m²)

Another column test was performed using flow rates of 4 L/hr/m², 8 L/hr/m² and 16 L/hr/m² to evaluate elapsed time to gold recovery. These results are illustrated in Figure 7.

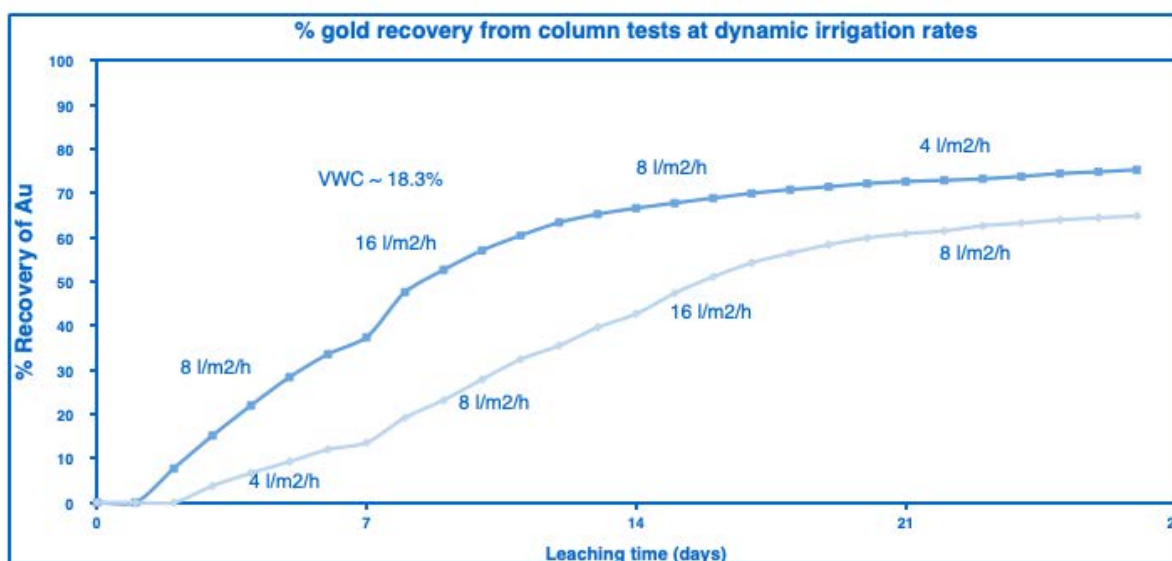


Figure 7: Gold recovery versus time of test columns at 4, 8, and 16 L/hr/m²

Leach pad moisture sensor data

The baseline data from the sensors are illustrated in Figures 8 and 9. The highest VWC is at location H11,

~ 8.5%, which was visually observed during the installation. And the lowest VWC is at location H6 ~2.5%. The average VWC for all of them was approximately 6%. The variation in the observed sensor reading baselines is mainly due to the VWC difference in the ore sample used to backfill around each sensor during installation.

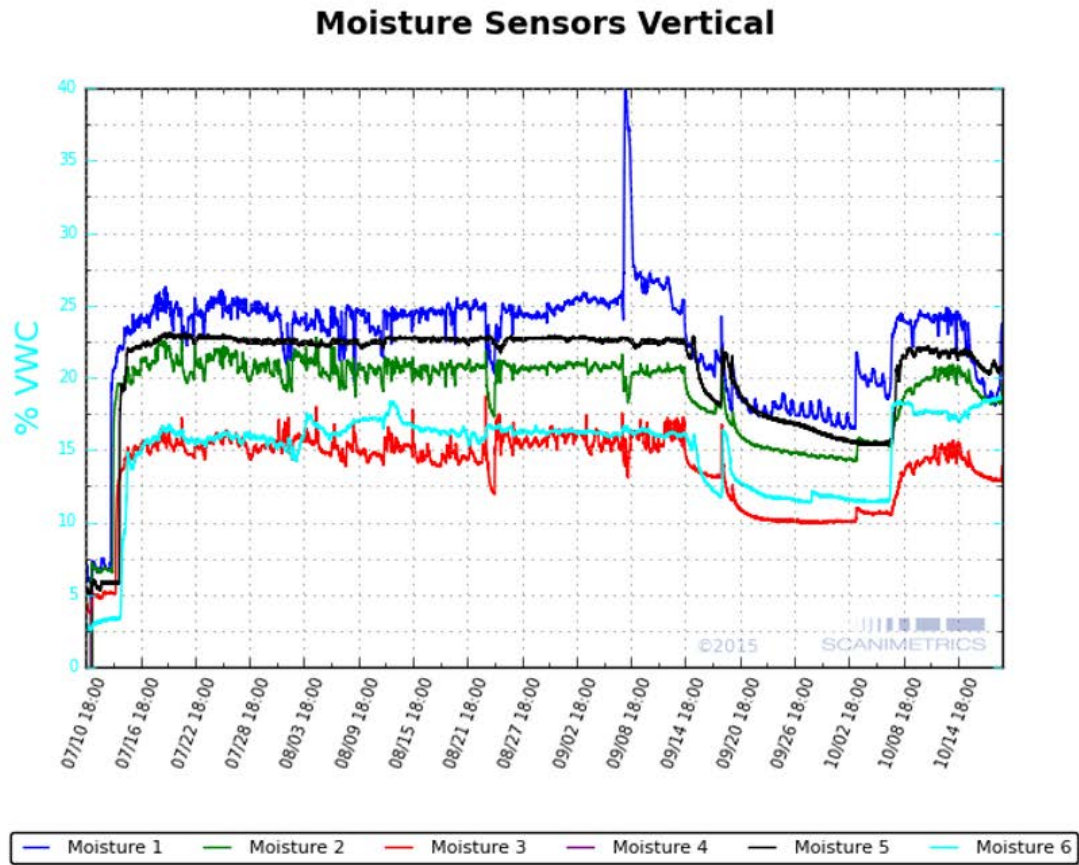


Figure 8: Moisture sensor data from vertical sensor locations

We confirmed with the mine operations that the irrigation drip started on July 13th. All the sensors (except #4) detected the water front within 2 days after July 13th in the order from V1, V2, and V6 to H7-12 (Figure 9).

Rainfall precipitation in millimeters was recorded at the leach pad by the mine operator, and the precipitation during the period of the trial is illustrated in Figure 10. Figure 11 illustrates the response of the compaction sensors during the trial period.

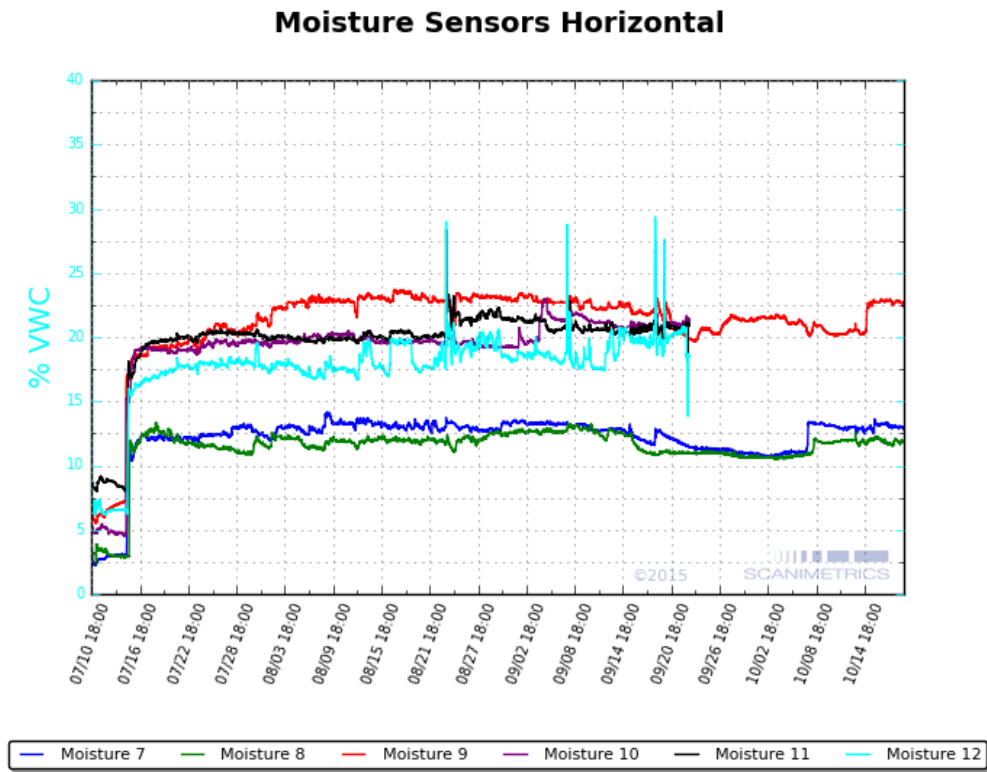


Figure 9: Moisture sensor data from horizontal sensor locations

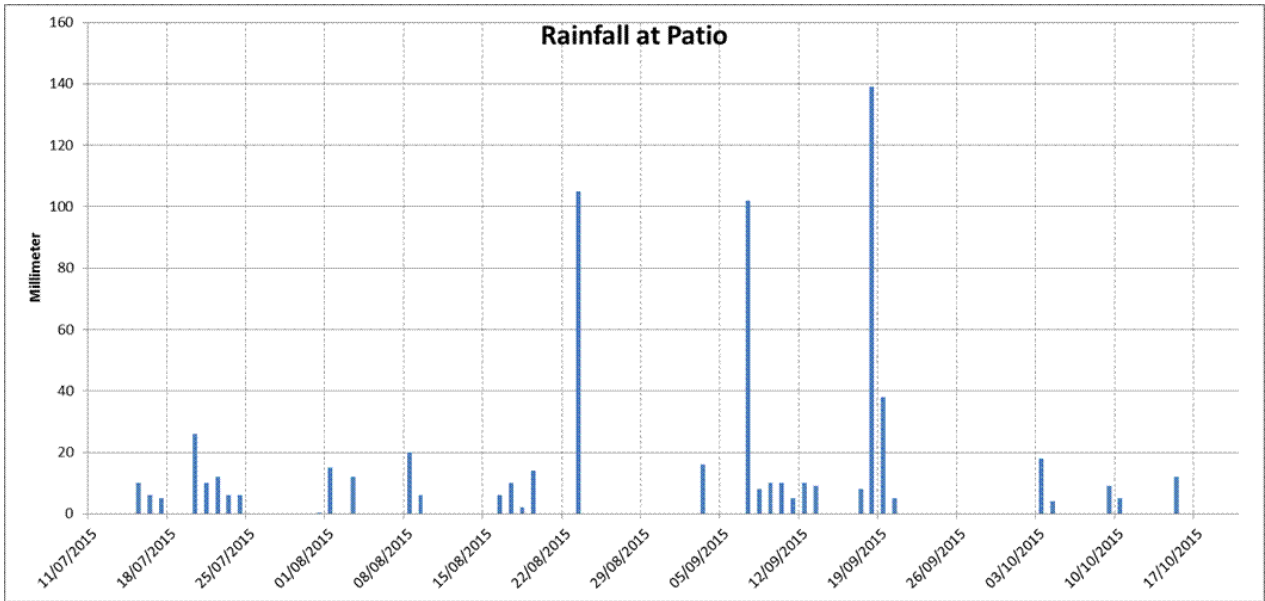


Figure 10: Rainfall at leach pad

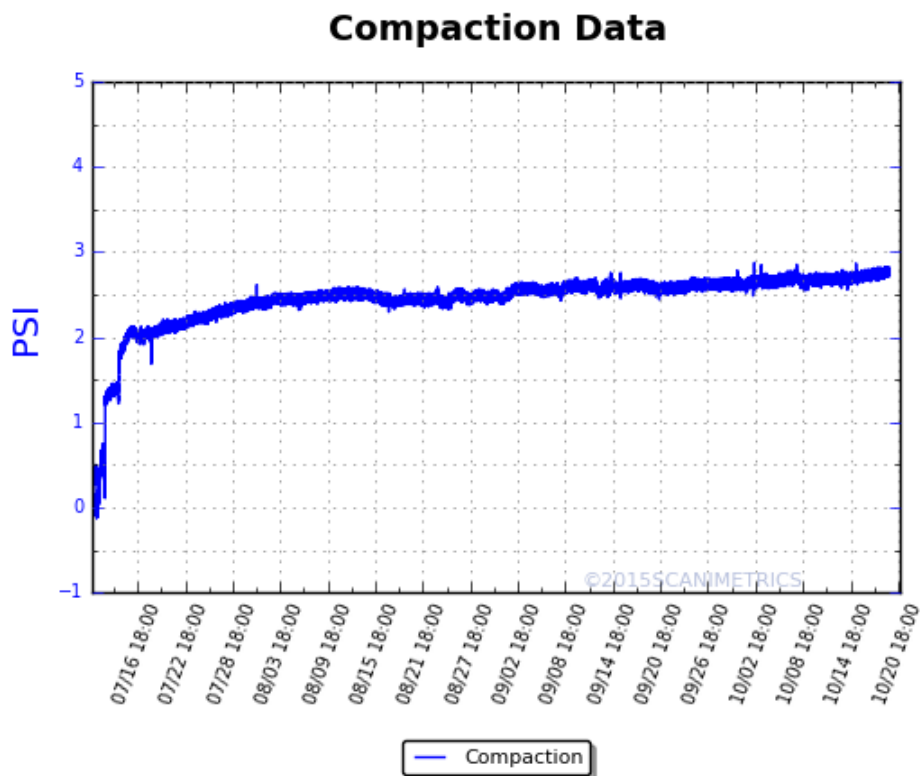


Figure 11: Compaction sensor data

Discussion

Column test data

The evaluation of gold recovery versus moisture content shows that a variation of moisture by 25% affects gold recovery by 69%. Dr. Delwyn G. Fredlund¹ concludes that 50% of saturation is the best moisture for leaching. Assuming that the porosity of the ore is approximately 36.5% at the leach pad, and using the following equation used in soil mechanics and petroleum engineering, θ (VWC) = S_w (water saturation) $\times$ ϕ (porosity). A 50% saturation of a 36.5% porosity ore, $\theta = 50\% \times 36.5\% = 18.3\%$. Thus, the optimum VWC $\sim 18.3\%$. According to this equation, it is obvious that as the ore compaction increases, the porosity ϕ will reduce, and the optimum moisture content for the leaching will decrease accordingly.

Leach pad data

We observed that the moisture content at the horizontal locations shown within the inner area of the leach

¹ Dr. Delwyn G. Fredlund was the Professor Emeritus of Civil Engineering (1968–2000) at the University of Saskatchewan. He has written hundreds of publications in the field of unsaturated soils mechanics.

pad achieved an optimum VWC range for most of the irrigation period. We also noted that the edge of the leach pad was under-irrigated. We recommended the adjustment of the sprinkler system to improve the irrigation coverage at the edge, which resulted in higher VWC within these areas. The additional gold available for recovery from these leach pad areas is 0.7%.

Large rainfall events occurred at the leach pad during August and September, and the impact of these events on VWC is evident in the data. Water saturation occurs especially at locations near the surface. The moisture content recovered as the water drained from the leach pad when the rain event subsided.

Irregular moisture observations suggested over- and under-irrigation at some locations. Over-irrigated observations indicate the existence of direct flow paths and insufficient oxygen for leaching, whereas under-irrigated areas require longer irrigation time. Lower moisture content at some locations suggests that the leaching process is less efficient. More energy must be expended to increase the leaching, or lower gold recoveries are achieved.

The leach pad's compaction (ore density) increased rapidly during ore placement, and as the leach solution penetrated the pad. However, the rate of the increase of compaction (slope of the compaction line) asymptotically approached a steady state of 0 after the first week. There were no observations that indicated a loss of stability of the heap or undesirable settlement events within the heap during the trial.

We also have observed that moisture within a heap is not uniform and varies by as much as 50%. Evaluation of this leach pad suggests that 9.6% more gold is available to be recovered from this leach pad by improving the uniformity during the application of the leach solution.

Conclusion

We have shown that moisture and particle compaction within a heap leach pad can be measured continuously in real-time at multiple locations. Column testing confirms the literature that VWC affects gold recovery. We observed sensors placed inside of the heap could identify excess water and chemical usage. We also observed that sensors along the edge could identify the gold that would otherwise be lost. The mine began irrigating the edges and repaired several problems with the irrigation system that affected the moisture at certain points within the heap. The moisture levels recovered and the column tests and literature described here provide evidence that the mine increased its gold recovery accordingly. Finally, this work suggests that if the leaching process gets off to a bad start, the mine operators can never recover all the gold they would otherwise have recovered if they had properly initiated the leaching process. The most important thing to achieve maximum gold recovery is to ensure the first week is done properly, which means getting the moisture levels to their ideal levels as quickly as possible.

Process Operations Considerations at Victoria Gold Corp.'s Eagle Gold Mine

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Abstract

Victoria Gold Corp.'s Eagle Gold Mine is in the center of Canada's Yukon Territory, approximately 375 kilometers north of the capital city of Whitehorse. The mine is an open pit operation with ore processed via three-stage closed-circuit crushing, and gold recovered through valley-fill heap leaching. The mine poured its first gold in 2019 and has since ramped up production, becoming the largest gold producer in Yukon history.

The operation experienced typical greenfield operation start-up issues; however, many of these were compounded by the extreme climate conditions and often required innovative solutions. This paper summarizes several of the key challenges experienced and lessons learned during the ramp-up phase. The specific and unique challenges in the crushing circuit, leach pad, and ADR circuits afforded by the cold weather operation are discussed with suggestions for consideration in future cold weather operations.

Introduction

Victoria Gold Corp.'s Eagle Gold Mine (Eagle) is the largest gold producer in Canada's Yukon Territory. The mine is located approximately 375 kilometers north of the capital city of Whitehorse and is accessed by a 43-kilometer road intersecting the Silver Trail (Yukon Highway 11) between the towns of Mayo and Keno City. Operations at the mine include conventional open pit mining with ore processed via three-stage closed-circuit crushing, and gold recovered through valley-fill heap leaching with a conventional carbon adsorption-desorption-regeneration (ADR) facility. The first gold was poured in Q3 of 2019 following initial ore placement in the same period, and commercial production was achieved on July 1st, 2020.

The Heap Leach Facility (HLF) is designed to accommodate ninety million tonnes of crushed ore, with design capacity expected to be reached in late 2027. Slightly over twenty-three million tonnes have been placed on the HLF as of July 2022, as stacking has ramped up from commissioning to target a nominal crushed ore placement of over one million tonnes per month with annual gold production targets over 200k troy ounces. Neither ramp-up of operations after start-up, nor processing circuit modifications, are unique

within the industry. However, the location of the site has posed unique challenges and requires special considerations that other operations seldom face.

Located at latitude 64°2' N and longitude 135°50' W, the climate at the Eagle site is classified as northern continental. The mean annual temperature for the area is approximately -3°C , with a range of upwards of 25°C in the summer months to low temperatures in winter occasionally dropping below -50°C . The annual precipitation ranges from 375 to 600 mm, with about half of this falling as snow that accumulates over the period from approximately late September to mid-May. The adverse winter conditions factor heavily in process operations and, while there are multiple leaching operations located in cold weather climates (Smith, 1997; Sinha and Smith, 2015), there is a dearth of documented practical solutions available for operations to leverage in overcoming the prevailing challenges. The discussion below is intended to address specific examples related to cold weather heap leaching process operations and to highlight some of the more commonly neglected considerations, along with practical solutions that subsequent projects may draw upon.

Heap leaching operations



Figure 1: Heap leach facility layout at Victoria Gold Corp.'s Eagle gold mine

The design and layout of the Eagle HLF is largely typical of other valley fill operations in the industry. Figure 1 shows the layout and main infrastructure of the Eagle HLF. The leach pad is a valley-fill-type design with ore stacked behind an embankment spanning a natural basin. The embankment provides a buttress for the stacked ore and creates an in-heap pond for temporary storage of pregnant solution. An overflow spillway connects the HLF in-heap pond to the events pond, which is sized to contain excess solution resulting from an extreme precipitation event or other emergency scenario and provides a means to temporarily store process make-up water. Pregnant solution is pumped to the ADR facility from the in-heap pond by four submersible pumps, with a fifth pump serving as an installed spare. Design flow to the ADR is 2,000 m³ per hour.

After passing through the carbon-in-column (CIC) trains, the barren solution is returned to the leach pad through two main 24" headers running up the east and west sides of the HLF. Distribution piping connects to the headers and progressively reduces to drip emitters used to deliver the leach solution. Relatively little differentiates the Eagle HLF from other heap leach operations in terms of basic process flow or equipment. The unique aspects of the operation are in the details of the execution.

As may be supposed, one of the foremost considerations in operation of the HLF is management of leach solution during the winter. Incorporation of sufficient backup power generation capacity is a manifest requirement, but a general assumption often encountered regarding cold weather leaching is that the circuit must include a means to heat the solution to keep the leach pad from freezing during the winter (Marsden et al., 1995). This is not strictly the case. The ADR circuit at Eagle has an installed heating capacity of 12.5 MBtu provided by two diesel-fired boilers, but, other than for preventative maintenance and readiness testing, the boilers have only been operated intermittently for a short time between December 2019 and February 2020.

During development and construction, they were included as insurance against the pad freezing causing a major failure. In practice they were soon deemed unnecessary and turned off. In operation the HLF acts like a heat reservoir. An insulating layer of snow helps preserve heat absorbed during the summer and, as a result, the leach solution stays above freezing, reaching essentially an equilibrium temperature only a few meters below the surface. Figure 2 shows the daily pregnant solution temperature measured since commissioning in 2019.

Overall the data show a general yearly trend, and periods early in the project with active heating of barren solution are easily identified. The temperature reaches a high of up to 10°C in the late summer, and then trends slowly downwards over the winter months, before dropping to a low of approximately 3°C in late April or early May. The drop results from the relatively rapid influx of melting snow within the leach pad footprint, and does not present a significant risk to the operation in terms of freezing.

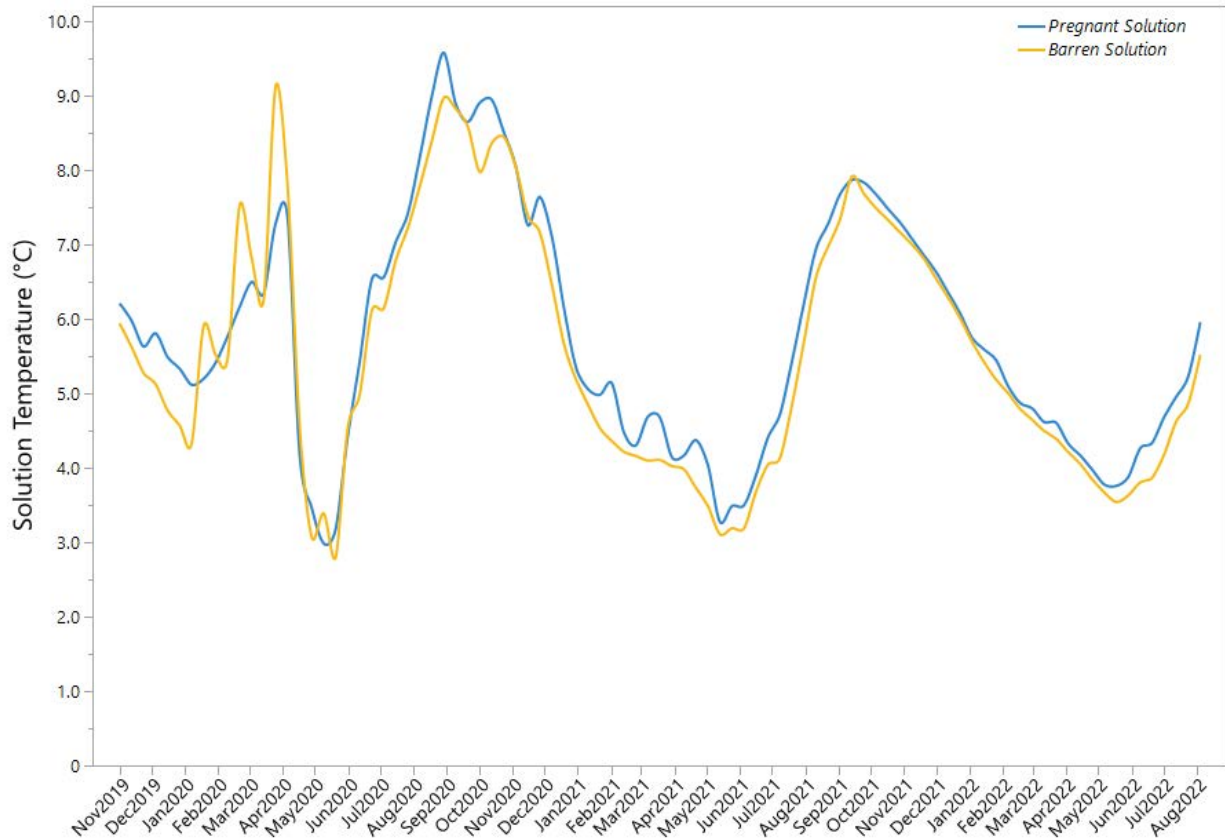


Figure 2: Victoria Gold Eagle mine heap leach solution temperature trends

The primary consideration from a freezing standpoint is preventing potential upsets and minimizing any uncontrolled flow within the solution distribution network. To this end, arranging leach distribution piping on a regular grid that can be plumbed with standard lengths of piping has multiple benefits. Figure 3 shows the convention adopted at Eagle using 90-meter lengths of 12" HDPE as the primary distribution piping, with 45-meter runs of 4" HDPE branching perpendicular feeding the driplines. In this arrangement each 2,100 m² panel can be isolated using valves installed at the 12" to 4" junction, minimizing the overall impact to flow if an area needs to be taken offline to address potential issues or make needed repairs.

This standard arrangement also allows piping lengths to be welded before they are needed, and allows for assembly of “ready kits” that contain all the fittings and associated hardware required to affect any emergency repair to any part of the network. The kits are boxed, clearly labelled, and stored in an area where they can be readily accessed and transported directly to the necessary location. Surveying all piping infrastructure and marking the location of key valves with indicators that are visible above potential snow loads and in dark conditions further expedites an operators’ ability to isolate areas as needed.

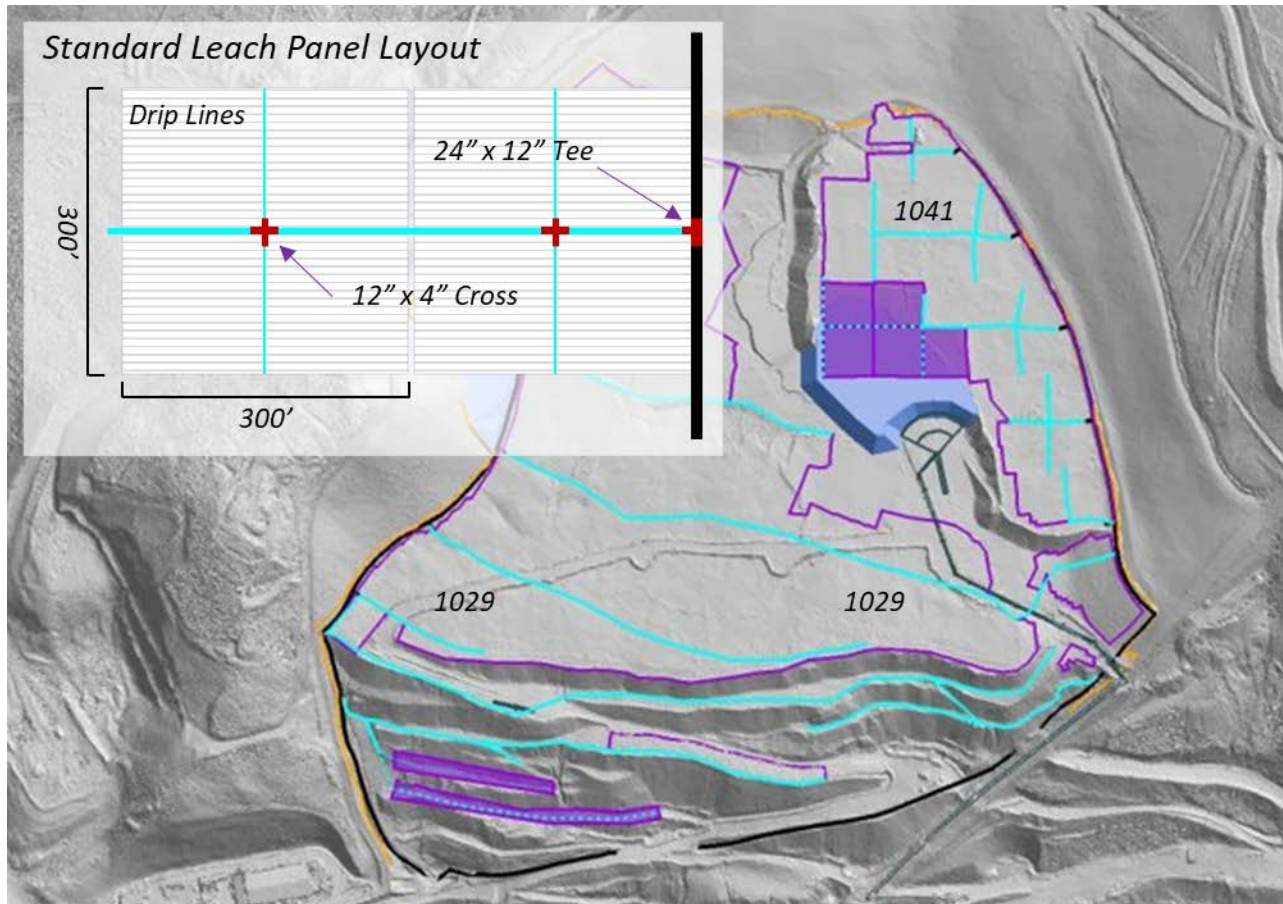


Figure 3: Typical leaching panel true cross layout

Even otherwise small leaks or blowouts can have serious consequences at -40°C . Hence, properly installing drip lines on the HLF is paramount. As in other cold weather applications, driplines at Eagle are ripped approximately one meter into the surface using a modified ripper table attached to a D-6/8-size bulldozer. Modifying the installation sequence to cut and tie off lines before completing the ripping pass allows the ends of the lines to be completely covered. This minimizes the chance for “freeze back”, wherein the exposed end of the dripline freezes and becomes a heat sink for solution further down the line allowing the frozen length to grow. It also minimizes potential ponding and ice formation at the exposed ends.

The drawback to this approach is that there is no direct view to gauge the flow across the leaching panel, and care must be taken to ensure that solution application rate is tightly controlled to prevent ponding at the surface. Similarly, additional care must be taken when attaching driplines to the distribution piping. A single blowout can result in the formation of sizable ice masses in a short time. Once formed, the structures are difficult to clear away. The lines must be attached securely, but there must also be appropriate slack between the distribution piping and buried portion; sufficient to accommodate modest settling and swelling without straining the exposed length, but not so much that it protrudes too far above the surface,

exposing it to the cold. In addition, the overall flow and pressures must be closely maintained at target levels to minimize the risk of driplines blowing off. Any visible ponding or uncontrolled flow must be addressed as soon as it is discovered to prevent areas from freezing or solution from flowing to unexpected areas. This includes any solution from melting of snow that accumulates on the HLF.

A well-laid and maintained leach panel has no visible features protruding from the covering snow layer but will also feel relatively soft underfoot, indicating uniform flow of leaching solution to the area. Figure 4 shows photographs of cross ripping operations and drip line installation as well as an active leach cell during winter operations.



Figure 4: Leaching operations at Victoria Gold Corp.
A) Cross ripping ahead of advancing stacking
B) Installation of buried drip lines
C) Leaching area

Freshet management

The term freshet is generally used to describe the flood of a river resulting from snow and ice melt, but it is broadly applied to the period when there is pronounced and substantial delivery of water to the landscape

by snowmelt, heavy rains, or a combination of the two. This typically occurs at Eagle over a two-week period beginning in mid-April to mid-May. During this time the site manages hundreds of thousands of cubic meters of runoff.

Upwards of 180 km^3 is classified as contact water that must be captured to ensure the parameters are within the allowable discharge criteria. Another 25 to 50 km^3 infiltrates directly into the HLF. Figure 5 shows the combined stored solution volume during the 2022 season in the HLF in-heap pond, events pond, and the control pond used to temporarily store mine contact water runoff from the site. Managing these volumes over such a short time requires a substantial amount of planning, but the topic receives very little consideration in the available literature.

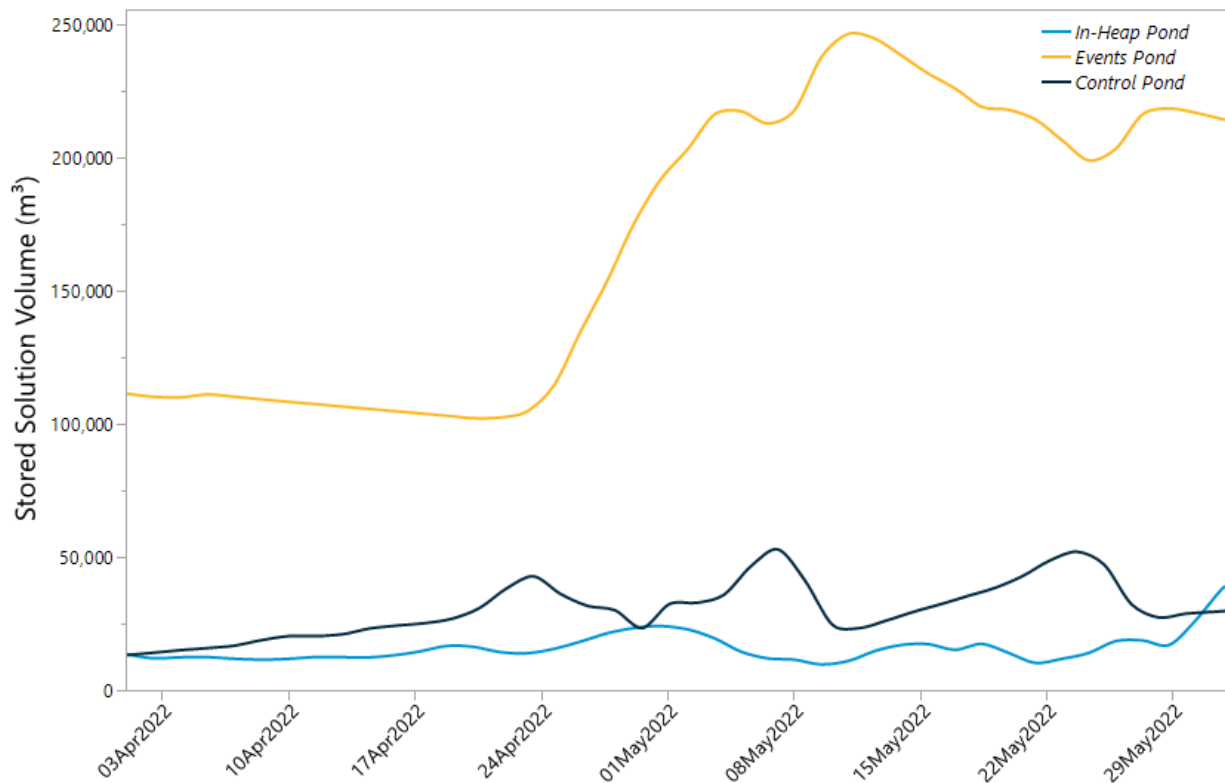


Figure 5: Freshet 2022 stored solution volume trends

Typical stormwater management measures and practices (e.g., sumps, ditches, etc.), are used to direct and control run-off from the site. While the overall volumes are considerable, with the proper measures and pump capacity in place, the impact of freshet to the overall operation is relatively small. The same is true for process operations, but maintaining the overall HLF water balance within safe levels requires additional considerations and tactics. One of the most useful is using sprinklers or hoses deployed during the warmest times of the days before the melt begins in earnest to controllably advance and protract the melt period.

This allows the overall rate that solution must be accommodated to be slowed significantly, providing much more flexibility in managing pond levels.

The most important consideration, however, is ensuring that there is fresh ore available to soak up the influx of melt water and buffer the impact on the operating water balance. At Eagle each tonne of crushed ore requires approximately sixty litres of solution to make up the difference between the placed moisture content and the field capacity at drain down. By strategically delaying placing new areas under leach and minimizing water addition ahead of freshet, the unleached ore becomes a reservoir that can be readily tapped to absorb solution and further dampen potential swings in the pond levels.

As the HLF continues to grow, additional consideration will be given to furthering this approach to deliberately limit leaching of ore during the period from mid-December to mid-March. This would minimize leaching operations during the most difficult time of the year and provide ample ability to absorb the increasingly greater melt with no adverse impact to overall production.

One additional aspect of freshet that receives little attention but has potentially enormous implications is the impact of the essentially daily freeze-thaw cycles during freshet. Aside from the potential safety hazards with changing and slippery conditions, the cycle can lead to the build-up of ice from runoff in areas, particularly those partially shaded during the day, and lead to unexpected flow paths potentially washing out material or resulting in an untoward release of process solution. Incorporating this consideration in the design of infrastructure expansions, stacking plans, and daily operations is standard at Eagle. Careful attention is given to areas prone to freeze-thaw with drains and sumps to direct potential runoff to the interior of the HLF wherever potential problems could arise.

ADR plant operations

The Eagle ADR plant has a conventional design that features two five-tank CIC trains with a combined design flowrate of 2,000 m³ per hour. Each tank is sized to contain four tonnes of activated carbon, and the circuit includes complimenting acid wash and strip circuits. The plant operates according to design. The only noticeable impact of cold weather conditions in operation is a slight reduction in the acid wash efficiency due to the process fresh water used reaching as low as 2°C in the winter. However, the overall impact is negligible, and plant recoveries of over 97% with barren solution grades of less than 0.01 ppm Au are easily achievable.

The primary challenge associated with cold weather operation is the freezing of ancillary systems in the ADR plant. The process water, fire water, carbon transfer, and caustic mixing and distribution circuits have all been upgraded with full heat trace and insulation following multiple freezing episodes in 2020 and 2021. The need for heat trace is particularly acute in any piping that parallels outside walls and should not be neglected in any comparable installation. Similarly, another key recommendation is to locate any valving

for process or freshwater circuits, as well as fuel and reagent delivery systems, inside the main building. This reduces the potential for freezing but also minimizes the risk to operators and maintenance staff in accessing the valves in icy and snowy conditions.

Freezing of reagents is also a consideration as temperatures can drop below the eutectic point for many commonly used reagent mixtures. Providing an enclosed reagent storage area is more than a convenience and should be factored into the overall design of the facility.

Crushing circuit operations

As noted above, the Eagle crushing plant features a three-stage closed-circuit flow sheet. A Superior 5065 MKII gyratory crusher operates as the primary crusher with a target 5.5" closed size setting discharged to either a live coarse ore pile (COP) or a long(er)-term stockpile area. Two Joest Pan feeders reclaim material from the COP. The reclaimed ore is conveyed into the secondary and tertiary (SECTER) crushing building and screened with the oversize discharging to a Nordberg MP 1250 secondary crusher. The crushed product is recombined with the undersize material and then conveyed to the tertiary crushing feed bin. Three additional Nordberg MP 1250s operate in parallel, each with dedicated feeders and double-deck banana screens immediately ahead of the feed chutes. The final crushed product is then transported to the HLF via a 1.5 km overland conveyor, and stacked using a sequence of fixed and mobile conveyors.

In contrast to ADR Plant operation, the impact of cold weather on the operation of the crushing circuit is much more significant. The nominal design throughput for the secondary and tertiary crushing circuits is 2,040 dry tonnes per hour with an overall utilization of 80%. However, achieving these metrics, particularly in the winter months, has proved formidable. The foremost challenges are all related to bulk material flow.

Feeders and chutes at the front end of the circuit are often plagued by “frosties.” These conglomerates of frozen material are often large enough and hard enough to completely choke off the flow of material, requiring operators to shut down and lockout the circuit to manually break and chip them apart. The incurred downtime only increases the likelihood of additional frosties forming in feed piles, chutes, and hoppers.

The most effective preventative measure is to always keep the crushed material moving but this is not possible, and experience has shown that it is crucial to incorporate some amount of localized heating to prevent larger vaults and feed hoppers from freezing up to the point that operations cannot be restarted without manually clearing the frozen obstructions. Likewise, incorporation of convenient access ways sized for appropriate equipment, and designing structures and equipment specifically to hasten and simplify material clean-up, are vital and should be paramount in the design and layout of the circuit.

Freezing material also factors heavily into normal operations and maintenance activities. Clearing build-up of frozen fines during the winter protracts essentially every task. Figure 6 shows the extent to which frozen fines can build up in the primary crusher dump pocket and how they can block screen panels

with openings upwards of four inches. Factoring these impacts into plans and schedules is critical as they can affect crusher operations and increase the time necessary to complete routine maintenance tasks by upwards of 200% above the actual tool time required.

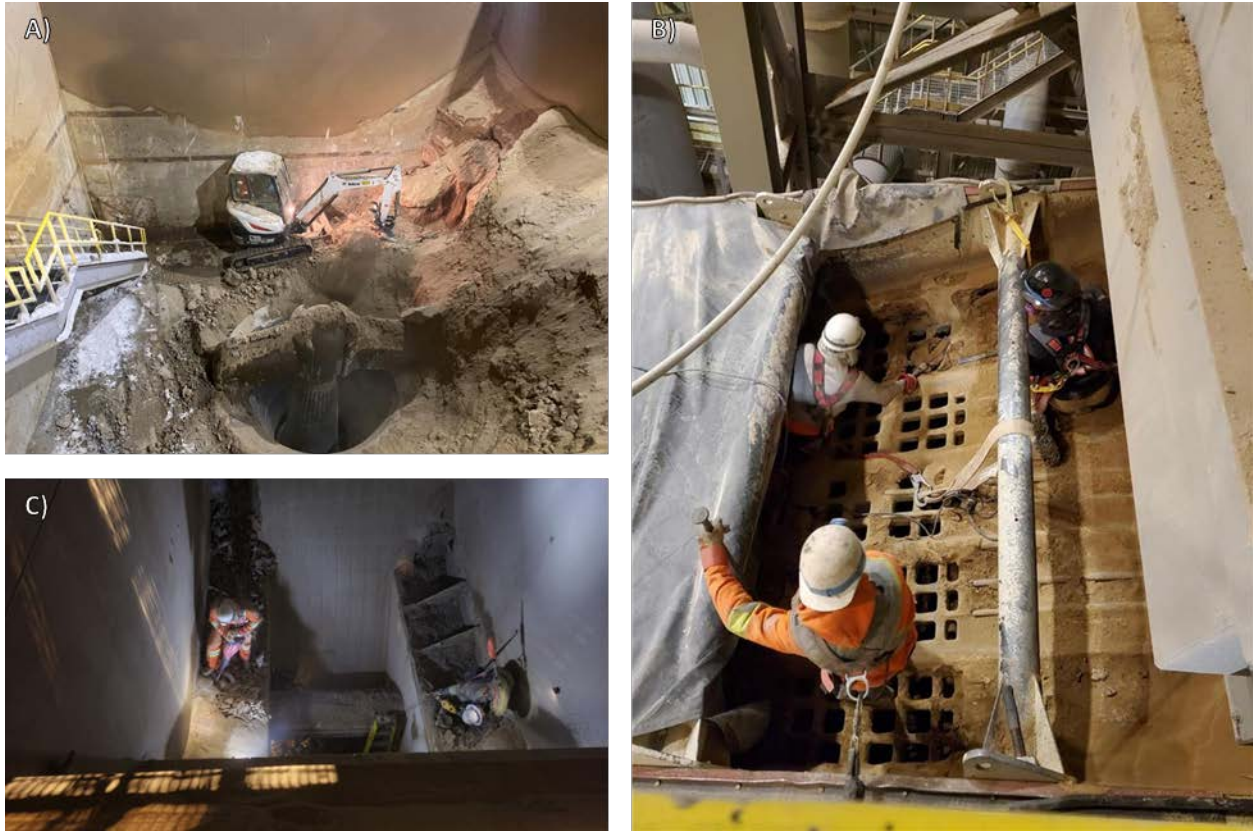


Figure 6: Crushing circuit maintenance operations at Victoria Gold Corp.
A) Clearing the primary crusher dump pocket
B) Chipping fines build up on the secondary screen
C) Scaling the tertiary crusher feed chute

The cold weather also impacts aspects of the operation in ways that are otherwise taken for granted in most cases. Incorporation of cooling systems on equipment lubrication circuits is standard and is necessary during normal operation, but it is also critical to incorporate heating capabilities to ensure that equipment can be safely restarted after prolonged downtime at -40°C . Tooling limitations at these temperatures can also be restrictive. Pneumatic tools quickly freeze up, rendering them useless, and battery-powered units have diminished output capacity and life.

Any electrical designs should account for these limitations and afford multiple receptacles in appropriate voltages near any areas requiring routine maintenance or prone to upsets. Lighting plans should also not be overlooked. During much of the winter the site receives only a few hours of twilight each day

and lighting upgrades have been required in multiple areas, particularly along conveyors and in high traffic areas.

Incorporating lessons learned in the cold weather operation of the crushing circuit to achieve greater consistency in operation is one of the foremost objectives at Eagle as operations continue to ramp up.

Summary

Victoria Gold Corp.'s Eagle Mine is the largest gold producer in Yukon history. The severe winter climate presents unique challenges, but these can be overcome with appropriate planning and tactics. Heap leaching and ADR operations are, for the most part, typical of any valley-fill type operation, but require consideration of the extreme cold and yearly influx of snow melt. These problems are mitigated by diligent operation and planning. Crushing operations during the winter are more heavily impacted by the cold climate. Experience has demonstrated equipment and overall circuit design are key to consistent crusher operation. These issues are foremost in the planning of future operations and increasing overall capacity at Eagle.

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Utilizing Online Cyanide Analyzers in Heap Leach Operations

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Abstract

One of the major operational costs in heap leaching for the purpose of gold extraction is cyanide use. To maximize leach efficiency and save on costs, it is important to effectively control cyanide addition to the heap. A technique that has been widely and successfully used to control cyanide addition and improve cyanide efficiency – especially in milling operations – is the use of online cyanide analyzers. Analyzers such as Cyanco's CCS[®] Cyanide Control Systems have garnered cyanide savings of over 10% in some mills. Mines using online cyanide analyzers can also expect a reduction of approximately 10% in detox costs.

Cyanco's CCS[®] units accurately control leaching processes by analyzing free cyanide, minimizing interference from other cyanide complexes. The units enable continuous adjusting and monitoring of cyanide levels during leaching. Automated monitoring and control of cyanide levels means more predictable results, more accurate forecasts, and better use of resources.

To date, the application of online cyanide analyzers has not been as widely adopted in heap operations as in mills. However, at Cyanco we believe there is a strong cost/benefit argument for porting this technology over to the heap environment – making the costs involved and the dedication of time well worth the investment.

Introduction: the importance of cyanide optimization

For most process facilities, optimizing cyanide usage is one of the most effective ways to lower operating costs while maximizing gold recoveries. Controlling cyanide usage starts with measuring cyanide either manually or digitally, and adding only the necessary amount to reach the target cyanide dosage.

Manual cyanide control requires an operator to collect a sample, conduct a silver nitrate titration, and then manually adjust the cyanide dosage – either by modifying the pump speed or addition valve. This

process can be slow, and often results in either over-usage, which impacts operating costs, or under-usage, which could impact gold recoveries.

Continuous cyanide control requires an online cyanide analyzer that automatically takes samples and measures the cyanide concentration without the involvement of an operator. The cyanide concentration from the online analyzer can then be used to adjust the pumps or valves as needed.

Online cyanide analyzers are classified according to their analytical method. The most widely used methods in gold processing are potentiometric and amperometric. In the potentiometric method, **electric potential** across the solution is measured as silver nitrate is added to the process sample.

Conversely, for the amperometric method, the **electric current** is measured as the silver nitrate is added. The advantage of both methods is that they use silver nitrate and can easily be compared to manual titrations.

Some of the benefits of using online analyzers, as compared to manual measurements, are:

- Higher frequency of cyanide measurement.
- More accurate and consistent measurement of the cyanide concentration.
- Greater precision to control cyanide addition to the desired setpoint.
- Improved control, which may allow for lower setpoints to be established due to a reduction in fluctuations, and more confidence in the results.
- Better cyanide control may also lead to a reduction in detoxification costs. This is especially true for milling operations.

Success of online analyzers in mills

Cyanco has a good track record of installing online cyanide analyzers in milling operations. The examples below demonstrate some of the successes achieved to date:

Case study #1

The analyzer at this customer's site was taken down for a four-day maintenance period. During that time cyanide dosing was done manually. The resultant unstable cyanide dosing cost the client a total of \$186,000 in the four days that the online cyanide analyzer was down.

See Figure 1, which compares the fluctuations in cyanide use between manual control and an online cyanide analyzer.

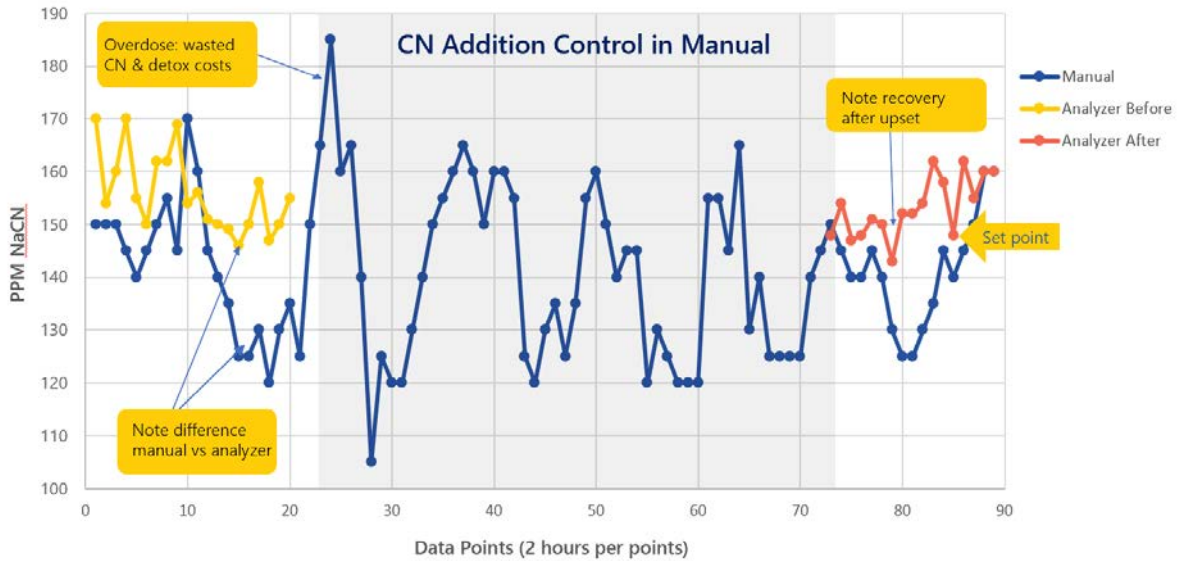
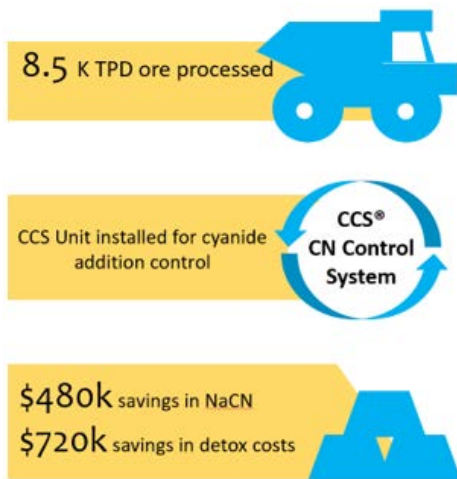


Figure 1: Cyanide control performed manually, vs. using an online cyanide analyzer

Case study #2

Figure 2 below shows another example of major benefits achieved in a milling operation after installing Cyanco's CCS Cyanide Control System. In this example the customer achieved \$480,000 of savings in sodium cyanide addition, and a bonus of \$720,000 savings in cyanide detoxification.



Property: Open pit mine with agitation leach processes TPD of 8,500 tonnes per day

Customer need: After mine was commissioned and in production phase they approached Cyanco to see if they could help them with their cyanide addition control challenges:

- Difficulty in seeing the titration endpoint.
- Time lapses between analysis and addition rate adjustments.

Results: Cyanco designed a CCS to fit their specific needs with a high analysis frequency and multiple sample points to control the various addition points.

- Cyanco converted the results from the analyzer and automated them to feed the mill's DCS system using a 4 – 20 mA signal.
- The DCS was able to vary the NaCN addition pumps to meet a specific setpoint with much greater accuracy.
- ~30% reduction in cyanide and another ~30% reduction in treatment costs.
- Savings of over 2.5 man hours per day.

Figure 2: Example of the benefits of installing a cyanide-control system

Cyanide control in heap leach operations

Challenges/issues/limitations

In general, heap leach operations have more challenges, as compared to milling operations, when it comes to cyanide control, including:

- remote locations;
- light staffing;
- spread-out operations;
- long retention times in the heap, making for difficulties in cyanide control;
- equipment infrequently visited; and
- difficult access to sampling locations and addition points.

Taken as a whole, we believe these issues argue strongly for automation. However, the application of online cyanide analyzers has not yet been as widely adopted in heap leach operations as in mills. For one thing, the return on investment (ROI) is challenging, as heap leach sites do not directly benefit from detoxification savings.

However, cyanide control is extremely important in heap leach operations. The result of today's addition rate may affect outcomes many months in the future.

Continuous cyanide control in heap leach operations

The expansive layout of heap leach operations compared to mills can also be a challenge in designing a control strategy to dose cyanide. Figure 3 shows one example of a cyanide control circuit created for use in heap leach operations.

Best practices

Dedicated personnel and defined maintenance schedules at the mines are prerequisites for the success of online cyanide analyzers. Other recommendations for continuous improvement include:

- better remote access to the analyzer
- simplified and robust equipment
- strategic location of sampling and addition points.

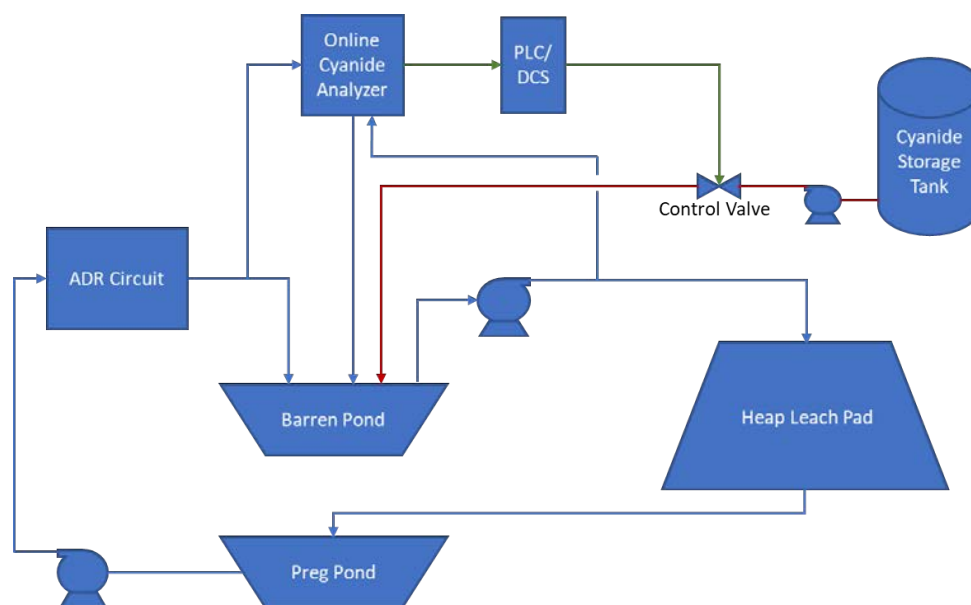


Figure 3: Schematic of cyanide control in heap leach operation

Conclusion

We believe that the successful implementation of online cyanide analyzers in mills, such as Cyanco’s CCS® Cyanide Control System, can be replicated in heaps. Porting over this cyanide control technology can enable significant new cost savings and improved efficiencies across the broader industry, representing a high-potential target for continuous improvement – as well as an opportunity for ESG (environmental, social, and governance) gains.

While Cyanco remains laser-focused on the quality and safety of our products, we also recognize that miners need more support than ever from their key suppliers. This is especially true as mining becomes increasingly advanced and complex, and as we all strive to meet the expectations of the new “circular economy.”

The presentation of this paper is intended to educate the 2022 Heap Leach Conference audience on this cost-saving opportunity, anchored by real-world examples. Cyanco will also share control strategies most suitable for heap leach applications, and new features of Cyanco’s CCS® Cyanide Control System that can increase the operational availability of these systems.

Changes in Water Management during Heap Leach Lifetime in Heavy Rainfall Sites

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Abstract

During the last five years, heap leach pads operating in rainy environments in Peru have successfully faced heavy rainfalls, by applying technical solutions for temporary water management in the construction stage. Successes in these conditions have been due to creativity, and the immediate response and joint work of the engineering and construction teams.

Runoff management in heap leach pads may require changes during construction and operation for different reasons, including economic, topographic, operational, and geotechnical issues that were not anticipated during design, or due to drainage structures being urgently built close to the rainy season. These situations make it necessary to modify the projected water management, which can generate additional expenses and important changes to the whole project.

This paper aims to show a current experience of design adaptation during construction of a heap leach pad located in the Peruvian Andes, where each seasonal transition caused setbacks in both construction and operation of the heap leach pad, with heavy rainfall and runoff the major critical factors. To control these issues, a sequential water management system was adopted. This system works temporally for each rainy season, coupled to the raising of the heap leach pad and connected to permanent hydraulic structures to catch and divert runoff of the impacted valley. This temporary water management used check dams for sediments control, inlets, channels, culverts, energy dissipator ponds, outlets, all lined with either geomembrane or stone masonry, which are considered temporary liners that offer low cost and quick installation.

Based on our experience in these types of projects, we can conclude that the support of a consultant technical team during both construction and operational stages increases the probability of success. This is because this support enables immediate action and response to the unforeseen in stormwater management, avoiding delays in the construction schedule, reducing costs in drainage infrastructure and earthworks,

avoiding costs of emergency works and operational expenses, and avoiding other inconveniences that could impact heap leach pad operation and generate economic losses for the mining operation.

Introduction

Commercial production of the case study mine began in 2016 with a production of 12,000 tpd, increasing to 36,000 tpd in 2019. Permits and studies for construction were obtained in 2014. The mining project area is located in an eastern zone of the Andes. The leach heap pad is located between gorges 1 and 2, as shown in Figure 1.

Water management for the final footprint of the heap is made up of perimeter channels near the top of the hills, which descend through rapids and end in sedimentation ponds located at the toe of the heap leach pad, from where it is discharged into the natural streams.

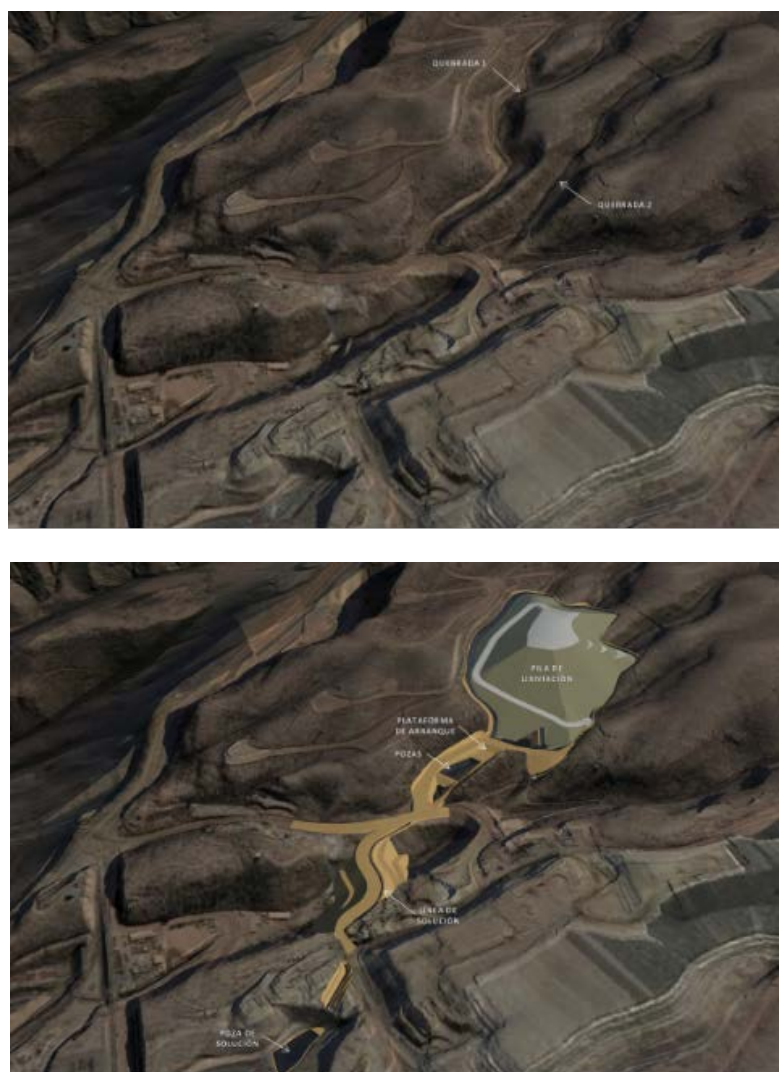


Figure 1: Heap Leach pad location of the case study

Problem to be solved

In this case study the problem to be solved is how to prevent surface runoff from causing delays in the leach pad construction schedule and economic losses during operation. It must be taken into account that in some cases the detailed engineering studies of a leach pad do not consider the water management design of the scheduled raise year-by-year. Instead, they are based either on the ultimate configuration, or on phases intended to facilitate the approval of construction and environmental permits for the whole facility. In these cases, the water management solution is designed for the final or the next phase footprint, when the heap leach pad reaches its corresponding predicted capacity. However, it takes years before the heap reaches its predicted raise; meanwhile, the operator defines the raise annually, according to the operation schedule, or to ensure an area free of runoff for the construction of the next phase. Moreover, the planned designs for water management of the phases further away in the future are taken as a relative reference due to the changes that may occur, so intermediate channels at higher levels can be planned by the operator regardless of that future design, based on other criteria such as constructability advantages, or to avoid very steep alignments. It is not necessary to follow a planned edge that may be adjusted or modified.

In Peru, there is a dry season (from April to October) followed by a wet season (from November to March). According to data from the government meteorological agency (Senamhi), rainfall in the high Andean areas of Peru can be in the form of torrential rains in any year.

In the intermediate phases of the heap leach pad, if the water management design is not done correctly, flooding, erosion, and potential instabilities in components of the heap leach pad may be generated during the wet season. These may generate material losses and delays in the operation schedule of the facility, and consequently significant economic losses.

This case study concerns areas of future heap leach pad expansions upstream of the start platform, which are remote from the perimeter access of the heap that contains the permanent perimeter channel, and to which the temporary channels are to be connected.

Changes in water management of a heap leach pad

The leach pad in its final stage will contain about 140 Mt and will be divided into intermediate phases with an upstream raise. Consequently, it is necessary to review and adjust the performance of surface runoff management from year zero (initial works), through intermediate years and final year. The most common situations that present problems in water management during the development and operation of the heap leach pad, which must be addressed by the supervisors and construction team, are the following:

- During the preliminary works: clearing and grubbing, foundation excavation, underdrain, grading and installation of the liner system, as shown in Figures 2 and 3.

- During the intermediate phases: the same activities, plus removal of the previous phase perimeter berm.
- Complementary constructions such as accesses, platforms, and dikes.

The solution of these problems demands “fast-track” engineering studies to design and construct hydraulic works for water management. To ensure an adequate flow of information to achieve efficient engineering, the following elements must be in place: ongoing updates of hydrological information; knowledge of restrictions due to geotechnical conditions and limitations of permits (e.g. authorized water discharge sites); and timely, scheduled planning, including integration between specialists and mine reviewers.



Figure 2: Heap leach pad – preliminary works



Figure 3: Heap leach pad – start of operation

Update of hydrological criteria

An important aspect is the review of hydrological criteria, to improve accuracy and representativeness. The maximum rainfall frequency curve was revised, since in areas such as that of the case study, there is a relevant limitation: a scarce and widely separated network of meteorological stations with a short recording

period. With the information from the station on the site (local station), for comparison, it was inferred that the rainfall on the site is higher than that reported in nearby stations, although without conclusive statistics on its magnitude. So, combined with the update of information from nearby stations, instead of the frequency curve, it was decided to use the upper limit of the confidence interval, the respective 95% quantile.

The design risk level of the temporary drainage works (channels and culverts) was determined from the risk level of the permanent perimeter channels (in the sense of the duration of the entire operation). According to the practices of the mining industry, in operations of between 10 to 20 years, the return period is estimated at 100 years, which is based on the consequences of the risks. For the temporary channels, a scale was devised according to their operating time, maintaining the level of hydrological risk of the permanent canals (approximately 18%). For the case study, a single scale of 10 years of return period was decided upon for the design of temporary channels, whose function is expected to be 2 to 3 years before they are covered by the raise and construction of the intermediate stages of the heap leach pad. Likewise, it was necessary to review and modify the perimeter channels for intermediate situations, in this case maintaining the 100-year standard, so that any interior overflow can be intercepted by the temporary channels.

Planning and integration between specialists and mine reviewers

For rapid design development there was a need to allocate a design engineer in the mine offices for an extended period, supported remotely by senior staff from the main office. The design engineer reported continuously and simultaneously to the director of the mine operator's planning and development office and worked in conjunction with the field data collection personnel; but had autonomous power to take design decisions. In this way, a rapid flow of information was achieved, including feedback based on direct observation of the performance of the work. This enabled design corrections and adjustments based on the most available construction alternatives, and in general a better alignment of the design with the restrictions of reality.

Recommendations

As a result of this and other experiences, for water management in a heap leach pad in general, it is recommended to consider the following:

- Request updated site information from the operator: topography, soil classification, geodynamic risks and as-built drawings, reflecting nearby structures and components, to identify restrictions for design and construction. This updated information should be coordinated with the design office to

avoid overlaps or interference of plans, such as the construction of new accesses not being communicated.

- Develop an annual water management plan for the work zone, which consists mainly of temporary channels on the edge of the area, in order to divert flows to authorized watercourses without affecting other mining components.
- Evaluate the runoff contribution area that was not considered in the original designs for the temporary channels. Consider the area that is altered, and how it will be in the near future, to evaluate the production of runoff and infiltration.
- As always there will be a gap between what is designed and what occurs later. It is recommended to use a factor on the runoff area, which can be statistically evaluated by comparing the design assumptions with subsequent as-built drawings. However, while that statistical information is developed, based on experience, it is recommended to use a 20% contingency.
- The design should focus on the greater capture and conduction of flows generated by surface runoff through temporary channels with a wide section. The optimum hydraulic section to capture and conduct the flows is the trapezoidal one. Also, consider maintenance needs that require modifying the design and construction of wider sections than those required only by hydraulic conduction, so that cleaning can be carried out with light machinery, since maintenance done without machines does not provide the speed necessary for timely cleaning.
- The design slopes for the temporary channel must achieve the behaviour of flows with a subcritical regime ($Fr < 1$) to avoid high velocities that require the construction of dissipation ponds, which is complicated by the limitations of the steep terrain.
- For the management and control of sediments, consider the construction of ponds located in different direction. The dead volume for sediments in the ponds must be at least 10% of their total capacity.
- It is recommended to use a temporary liner for channels and other drainage works, as these are easy to install and dismantle; this guarantees the protection and liberation of the work area. Smooth HDPE geomembrane can be used to form drainage works; but its use should be avoided in areas susceptible to continuous rockfalls from the cut slopes that impact the liner.
- Temporary water management must undergo constant repair and cleaning to prevent leaks and clogging of structures. Periodic inspection or after each rain event is recommended.
- Pay more attention to the areas that, due to soil conditions, can generate, under the activating effect of infiltrations, mass movements, cracks and displacements. It is recommended, among other measures, to identify them, increase the size of the runoff collection channels, and densify the drainage network in the area.

Conclusions

The construction of a heap leach pad in areas of intense rainfall entails updating the water management plan annually during the stages of execution and operation of the leach pad.

Surface runoff on leach pads causes flooding and instability of existing accesses due to the action of water flows, and infiltrations cause instability in the cut slopes, especially during the initial years of operation.

The impact of surface runoff can be reduced by the coordinated participation of the design engineer on-site with the planning and design of the mine operator to design and execute a water management plan in the shortest time possible.

This case study demonstrated that the cost of a temporary liner using a smooth 1.5 mm HDPE geomembrane for the water management system corresponds to 50% of the costs of a permanent liner, such as stone masonry.

The water management of a heap leach pad should be updated annually before the wet season and designed for each construction phase to avoid the issues presented in this summary. Most of the recommended actions described above can be applied for efficient water management control in a heap leach pad.

Crushed Leached Ore Characterization, Response, and Liquefaction Potential

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Abstract

Heap leach pads are an attractive cost-benefit extraction technique for low-grade gold, silver, and copper ores around the world. In South America the complex terrain features, high altitude, extreme weather conditions, high seismic activity, and limited suitable locations impose a series of engineering constraints for designing a heap leaching operation. Therefore, one of the critical steps is an adequate field and laboratory investigation to perform an optimal geotechnical characterization to ensure a correct geotechnical and especially seismic performance.

This paper presents a practice-oriented methodology for the evaluation of the seismic performance of under-operation leach pads in a case study. Input data for this methodology consists of geotechnical investigations of the stacked crushed ore, such as test pits, geophysical surveying, and CPTu; complemented with a robust geotechnical laboratory testing program. Similar to tailings dam engineering, the crushed ore response is classified as dilatant or contractive depending on the in-situ soil behaviour, drainage (drained or undrained) conditions present within the ore; and appropriate strength parameters corresponding to each type of ore are assigned.

This approach allows one to characterize a heap leach pad, identifying areas of material with contractive or dilatant behavior. Further, the assessment of the liquefaction potential is performed on the locations where CPTus are available, and a combination of CPT triggering procedures and specific site response analyses is suggested to this end. With this information, the geotechnical characterization is complemented by assessing the liquefaction potential of the crushed leached ore within the heap leach pad.

Introduction

A heap leach pad (HLP) is a heterogeneous structure with the stacked leached ore having variable geotechnical properties due to factors that modify their nature such as geology, mineralogy, and alteration of the deposit, ore degradation during leaching, operational issues, engineering design changes, and

environmental factors. From an engineering point of view, understanding the heterogeneity in the ore is key to ensure the correct performance of the facility; this is even more relevant in regions with high seismic activity.

This paper describes the geotechnical characterization and evaluation of the main section of a typical HLP of crushed ore based on cone penetration tests (CPTu), complemented with test pits, boreholes, and laboratory tests. The investigations and analyses are focused on the assessment of the liquefaction potential for a design earthquake scenario. Supporting laboratory tests were performed, and the results were used to determine the ore drained and undrained strengths. It was found that the combination of comprehensive field and laboratory testing campaigns produced a high-quality characterization of the stacked crushed leached ore for liquefaction potential assessment.

In the following sections, representative results and trends are presented and discussed.

Geotechnical characterization of crushed leached ore

The geotechnical characterization of stacked leached ore is challenging. In this study three types of crushed ore were identified depending on the mine production: sand-like, transitional, and clay-like. The crushed stacked ore forming the heap is highly influenced by the production schedule; hence, coarse and fine ores are not uniformly distributed within the HLP. The geotechnical characterization procedure described in this paper helps to identify the location and extent of each type of crushed ore within the HLP and provides an estimate of the type of mechanical response.

The type of response is classified as contractive and dilative; thus, these classifications can be outlined: sand-like contractive/dilative, transitional contractive/dilative, and clay-like contractive/dilative. Thus, beyond the very useful CPTu field data, one-dimensional consolidation tests, consolidated undrained (CU) triaxial tests, consolidated drained (CD) triaxial tests, and hydraulic conductivity tests were performed as part of the investigation program. A series of 09 CPTu were performed along the main cross section of the HLP, located at different elevations, and reaching depths ranging from 10 m to 60 mm as shown in Figure 1. Pore pressure dissipation tests (PPDT) were performed to determine levels of saturation within the HLP. Shear wave velocities (V_s) were estimated based on the Robertson and Cabal (2015) correlation.

To classify the crushed ore into coarse or fine, the soil behaviour type (I_c) of Robertson and Wride (1998) was employed in addition to index laboratory tests. The crushed ore was considered to be sand-like if $I_c \leq 2.5$, transitional if $2.5 < I_c < 2.7$ or clay-like if $I_c \geq 2.7$. The hydraulic conductivity (k_v) tests indicate k_v values of $1.0\text{E-}04$ to $1.0\text{E-}03$ m/s for the coarser ores, and of $1.0\text{E-}08$ to $1.0\text{E-}05$ m/s for the finer ore. These k_v values agree well with the CPT-based characterization. Grain-size distributions and 1D consolidation tests also support the CPT-based classification. The determination of the response of the stacked crushed leached ore was based on tailings dams engineering design, which uses the approaches of

Robertson (2016) and Jefferies and Been (2016) to distinguish between contractive and dilative soils. The Robertson (2016) approach employs the clean sand equivalent normalized cone resistance ($Q_{tn,cs}$) where the limit between contractive and dilative responses is $Q_{tn,cs} = 70$. The approach of Jefferies and Been (2016) employs the state parameter (ψ) with the contractive/dilative threshold located at $\psi = -0.05$ following the findings of Shuttle and Cuning (2007).

The undrained strength envelope was initially estimated from the p' - q plots derived from CU triaxial tests as shown in Figure 2. Then, the trends in the data were verified with a simplified Cam-clay based expression for the normalized undrained strength (S_u/σ_v'), which is found to vary within 0.26 to 0.28.

In addition, CD triaxial tests were considered to evaluate the peak drained strength of the crushed ore (see Figure 3), which was first estimated from the envelope resulting from the test results. Then, the plane stress invariants t and s' of Wood (1994) were used to determine the drained friction angle, which varies within 35 to 38 (see Figure 3). Table 1 summarizes the parameters estimated for the crushed leached ore according to its behaviour.

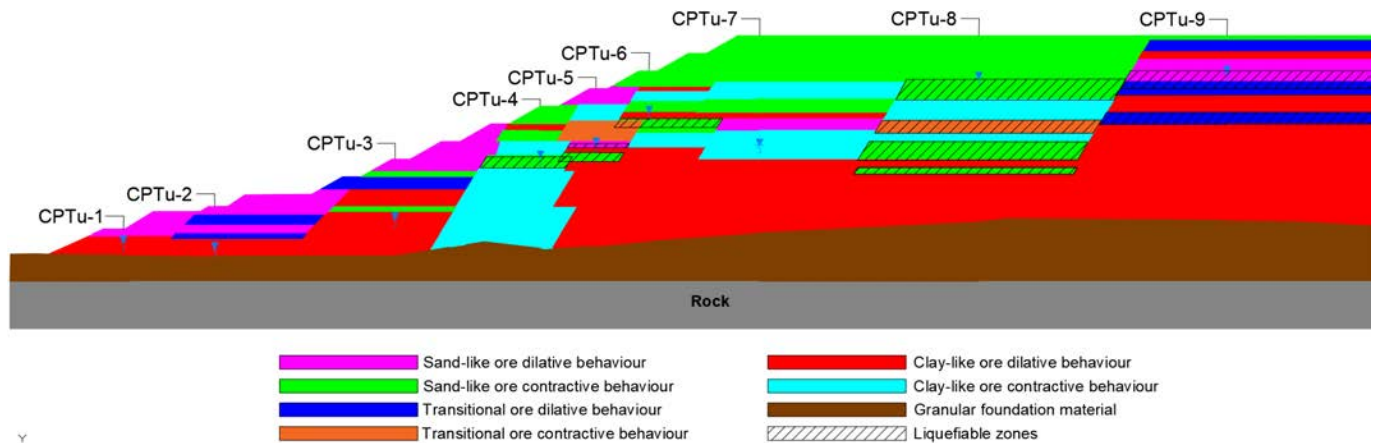


Figure 1: Representative cross section showing material characterization

Table 1: Summary of crushed ore strength parameters

Material	Type of response	Strength parameters		
		c'	ϕ_i'	S_u/σ_v'
Coarse ore	Contractive	–	–	0.28
	Dilative	0	38	–
Fine ore	Contractive	–	–	0.26
	Dilative	0	35	–

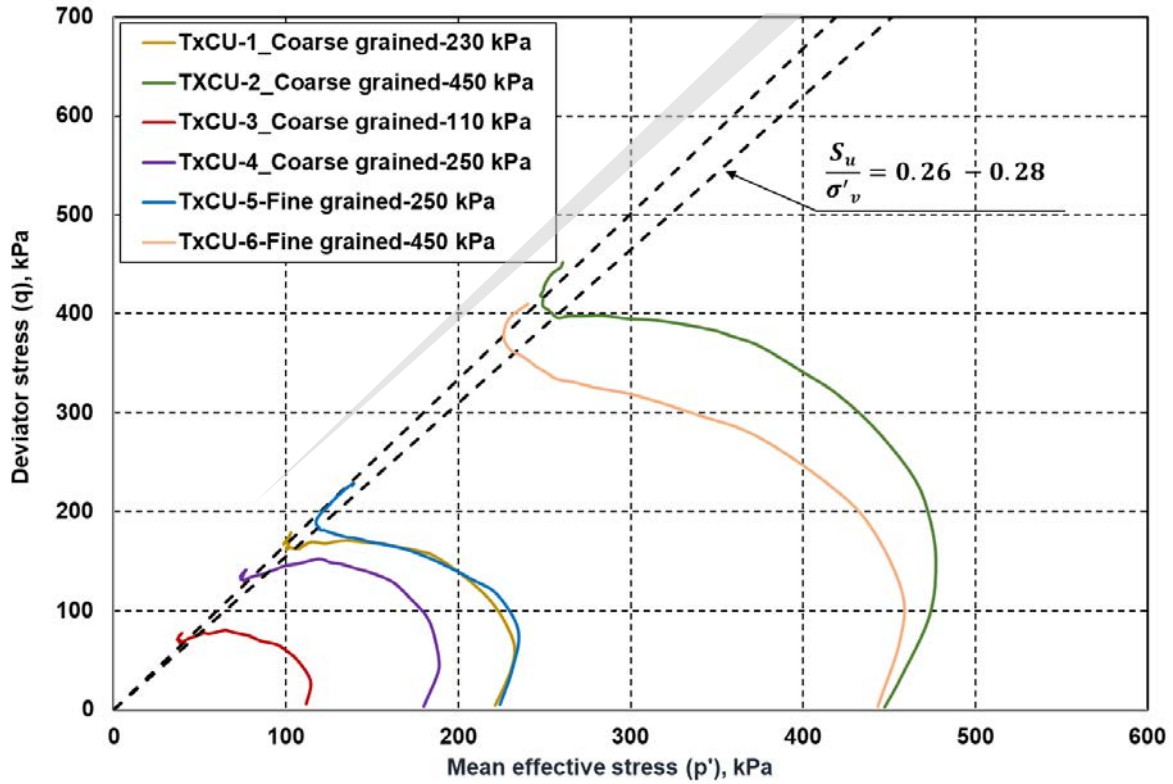


Figure 2: Estimation of crushed ore undrained strength

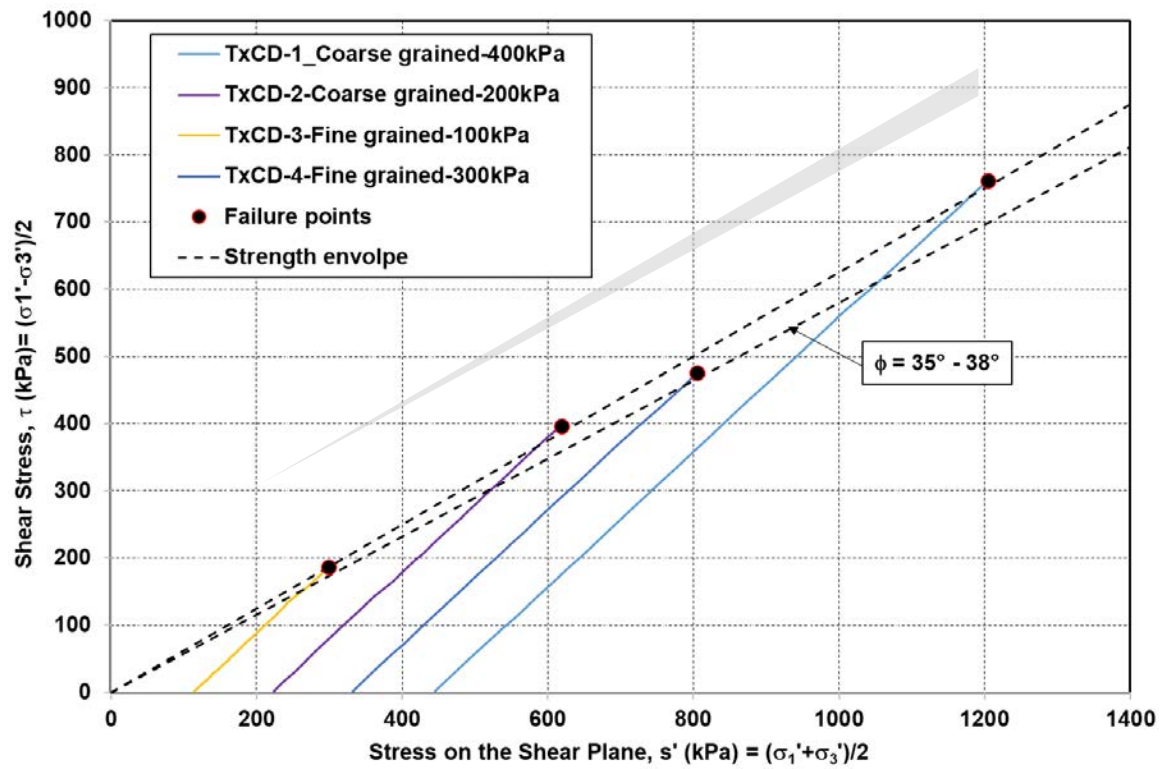


Figure 3: Estimation of crushed leached ore drained strength (Wood, 1994)

Seismic response assessment

One-dimensional nonlinear time-domain site-response analyses (SRA) were carried out at each CPT location using the DEEPSOIL V.6.1 program developed at the University of Illinois at Urbana Champaign, with the objective of developing profiles of induced acceleration and of induced peak shear stresses. Each soil column considered from the bottom up: an elastic half space rock basement, a granular foundation material, and a layered system of fine (clay-like material) and coarse (sand-like and transition material) ore.

To model the stress-strain response, the DEEPSOIL general quadratic model (GQ/H) that considers a nonlinear backbone formulation was employed. An impermeable boundary condition was set at the bottom of the profile with the water table location obtained from the pore pressure dissipation tests. $G/G_{\max}$ and damping curves for the fine and coarse ores were modeled with the Darendeli (2001) formulation, while the granular foundation was modeled with the curves of Menq (2003). The authors analyzed the effect of modeling the low permeability interface between the foundation and the ore using the procedure of Yegian et al. (1988), but smaller amplifications were obtained compared to the models without interface; therefore, conservatively in this study the interface was not modeled in the SRA. A series of seven ground motions were used for the modeling: AT01, CH70, LIM66, LIM74, MI85, PI07, and TAR05.

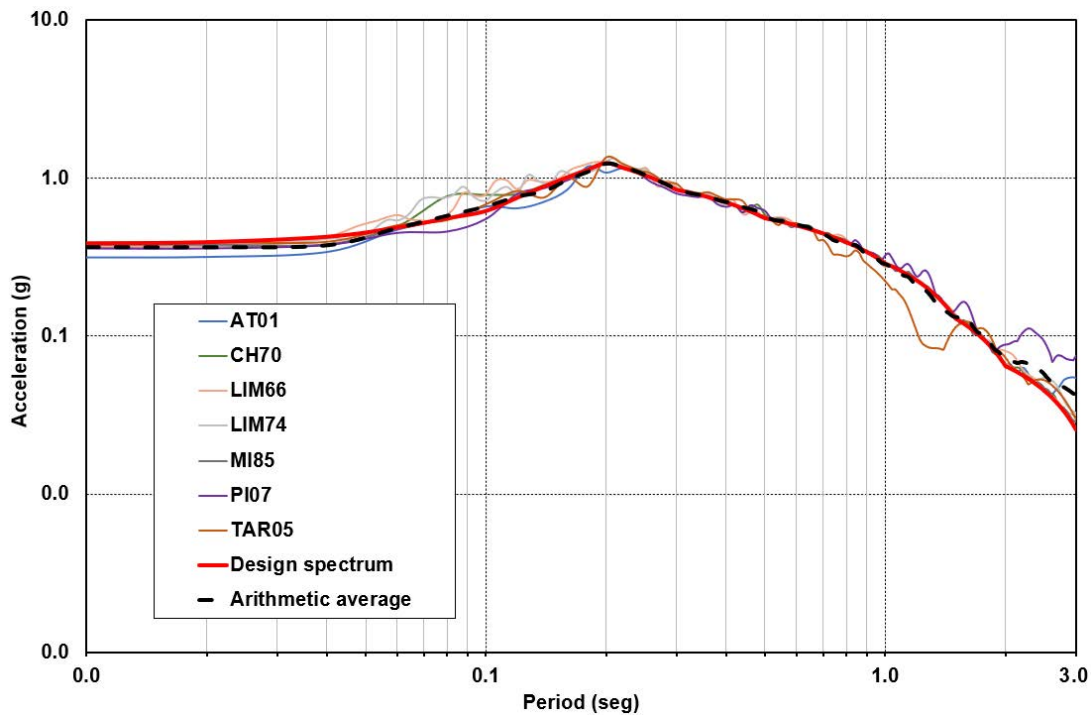


Figure 4: Spectrum adjustment of analysis earthquakes

The seismic demand at the HLP site was estimated from a site-specific seismic hazard analysis. The deaggregation indicates that a M_w 8.8 earthquake and the 475-year Uniform Hazard Spectrum (UHS) are representative for design. To select the suite of ground motions, the arithmetic mean of the spectra of the

seven ground motions was fitted to the target UHS. In the fitting procedure, different scale factors were applied to each ground motion until an acceptable fit was achieved. The fitted response spectra of the selected motions are displayed in Figure 4.

Figure 5 shows one-dimensional SRA results for a representative HLP profile at the CPT-8 location where sand-like, transitional, and clay-like ore are shown with different colours. The distribution of horizontal accelerations within the heap of crushed leached ore is presented in Figure 5(a) for the seven ground motions used in this study. The variability due to the ground motion characteristics and crushed leached ore geotechnical properties in the acceleration distribution is highlighted by the lower and upper bounds, which range in the order of 0.1 g. A significant acceleration amplification is observed at a depth of 50 m, which corresponds to a layer of granular ore sandwiched by two relatively thin layers of softer materials. A similar effect is observed at a depth of about 40 m. At the surface of the HLP the overall amplification factor (AF) is 1.25.

The earthquake-induced cyclic shear stress ratio (CSR) at the same CPT location is shown in Figure 5(b). Similarly, the variability in the CSR is highlighted by the lower and upper bounds. It is observed that the variability in the CSR is smaller than the variability in the acceleration in the whole profile. Also, the CSR is significantly higher in the top 20 m of the HLP, and at the surface the overall amplification factor for the CSR (AF_{CSR}) is 2.7.

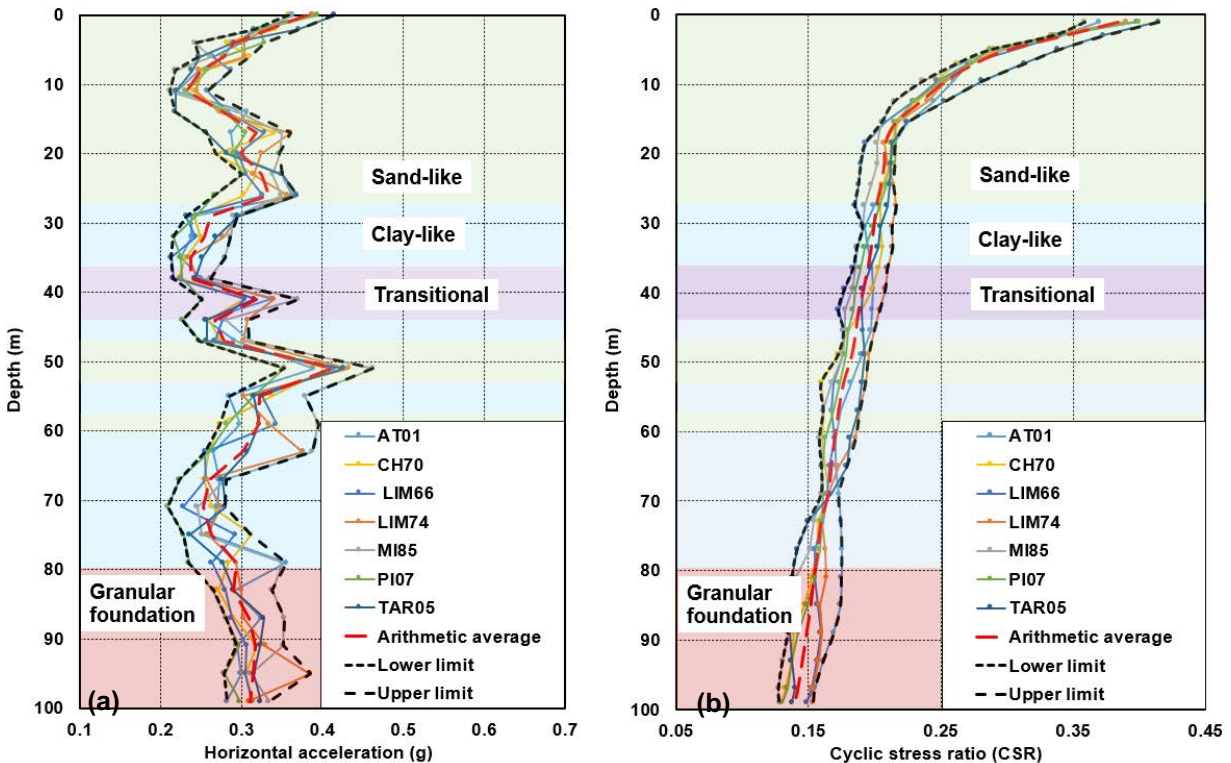


**Figure 5: (a) Horizontal acceleration distribution with depth;
(b) Cyclic stress ratio variation with depth**

Liquefaction potential

The liquefaction potential of the crushed leached ore was assessed using the stress-based method proposed by Seed and Idriss (1971) as presented by Youd et al. (2001). The simplified approach compares the cyclic stress ratio (CSR) to the cyclic resistance ratio (CRR) to obtain the factor of safety against liquefaction (FS_L) as shown in Equation 1.

$$FS_L = \frac{CRR_{7.5}}{CSR} \times k_\sigma \times MSF \quad (\text{Equation 1})$$

Where:

$CSR_{7.5}$ is the cyclic stress ratio for a 7.5 earthquake magnitude;

k_σ is the correction factor for significant effective confinement stresses; and

MSF is the magnitude scaling factor.

The results of the SRA were used to determine the CSR profiles at 9 representative columns in the analyzed HLP. The SRA was performed using DEEPSOIL. CRR profiles were determined based on empirical relations based on the CPT results. Generally, the CRR was calculated for a base scenario with magnitude 7.5 ($CRR_{7.5}$), then k_σ and MSF factors were applied to correct for depth and magnitude effects. The FS_L for coarse ore materials (sand-like and transition soils) were calculated as per Robertson and Wride (1998). Due to the fact that the first 15 m to 25 m are not saturated and thus are non-liquefiable, the static bias factor (k_a) was not incorporated in the evaluated of FS_L .

Also, ore characterized as clay-like with $I_c \geq 2.7$ was initially assumed to not liquefy, and it is recommended to assess the liquefaction susceptibility for these materials using the procedures of Bray and Sancio (2006) and the strain softening using Idriss and Boulanger (2008). Zones of saturated crushed ore that have a FS_L less than 1.1 may undergo cyclic liquefaction. If unsaturated, the material was considered not susceptible to liquefaction. Figure 6 shows the $(Q_m)_{cs}$, state parameter and the liquefaction potential assessment based on FS_L at depth for the coarse ore at the analyzed heap profile.

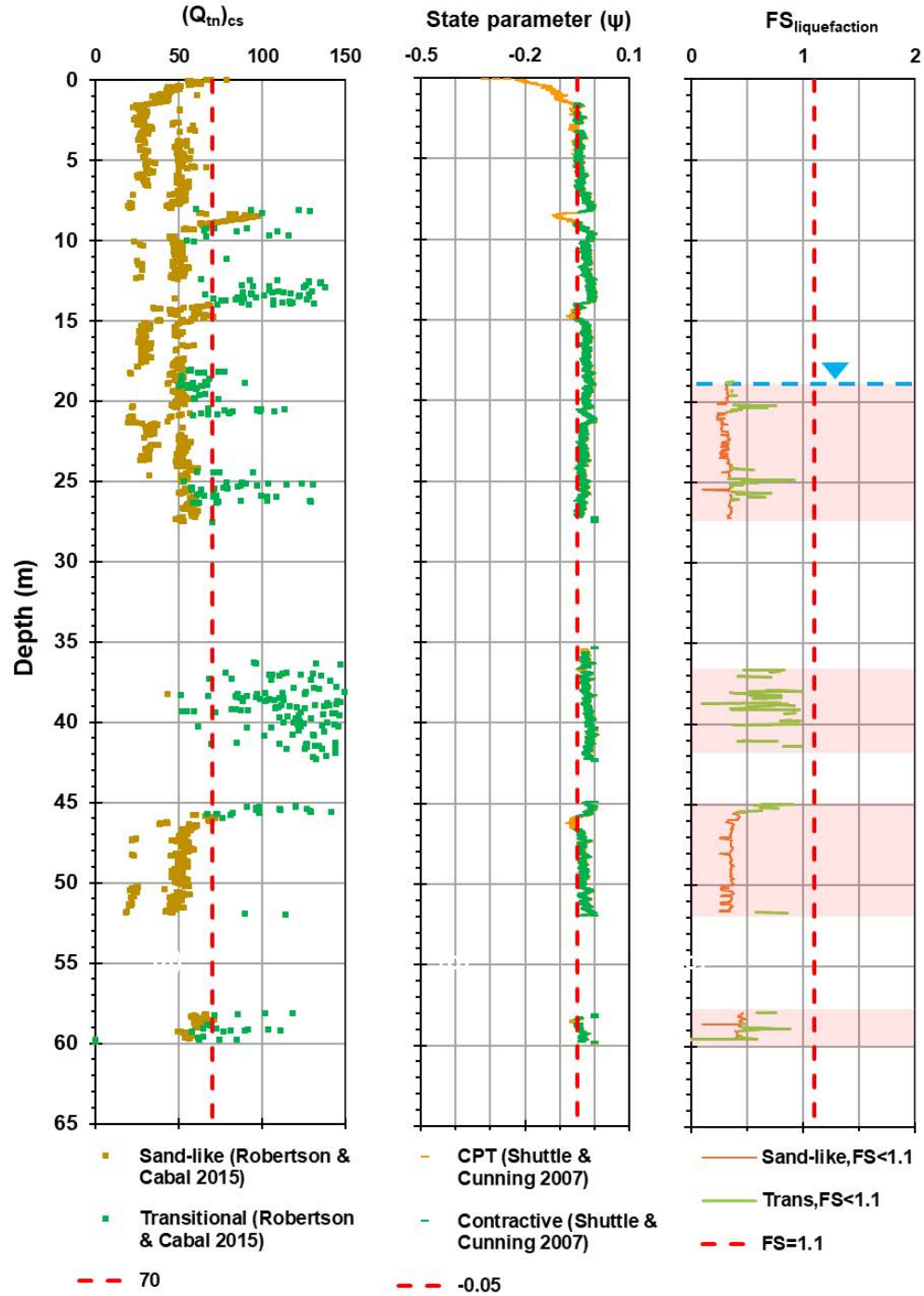


Figure 6: Liquefaction potential assessment results

Based on the results shown in Figure 6, it can be concluded that extensive liquefaction in the HLP profile is expected to occur; therefore final post-earthquake stability analyzed is recommended based on the above and results at other locations in the same main section of the HLP.

Conclusion

This paper presents a practical methodology for the geotechnical characterization of a crushed ore profile of a HLP, analysis of the seismic response and assessment of the liquefaction potential. The methodology is based on the Robertson and Wride (1998) approach, using the I_c parameter to define the ore as sand-like, transitional, or clay-like. The approaches of Robertson (2016) and Jefferies and Been (2016) were used to determine the dilatant or contractive response of the crushed ore. According to Figure 4, both approaches are valid and produce similar results; therefore, it is suggested to use both in a complementary fashion to obtain consistent classifications. SRA suggests that significant acceleration amplification can occur within the HLP and at the surface, depending on the HLP layering. Likewise, an increase of the CSR can be observed in the first 20 m. The analysis of several ground motions compatible with the design spectrum allowed the estimation of the variability in the acceleration and CSR profiles.

To determine the liquefaction potential, a practice-oriented stress-based methodology was used to calculate the CRR and CSR of the crushed ore of the case study. For this paper the CSR was estimated from a series of SRA using the DEEPSOIL program, and the estimation of the CRR for the coarse ore was based on the method of Robertson and Wride (1998). Preliminary, fine ore with $I_c > 2.7$ was assumed to be non-liquefiable. However, it is recommended to assess the liquefaction susceptibility for the fine materials using the procedures of Bray and Sancio (2006) and the strain softening behaviour based on Idriss and Boulanger (2008). Even though the saturation conditions within the HLP allowed a 1D evaluation, the authors are currently performing a series of 2D dynamic analyses for further assessment of 2D effects.

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Chapter 3:

Hydrology and Sustainability



Liquefaction in Heap Leach Piles – A Primer

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Abstract

Even though liquefaction of heap leach piles has not been a very common phenomenon, it has occurred. With increasing demands for and prices of certain metals, their extraction from lower grade ores using heap leaching becomes economically feasible and the number of operations utilizing the process is ever increasing. The pilot heap leaching program of Rio Tinto's Bingham Pit operation in Utah, USA, in spite of the modernized concentrating and flotation operations running successfully since Rio Tinto's acquisition of the mine in 1989, is a good example of this trend. A large number of heap leaching operations in South America and around the world, including the Bingham Pit operation, are in active seismic zones. Moreover, leach pile heights have been steadily increasing, with piles of 150 m or greater becoming common. These facts point to the possibility of liquefaction-induced failures of leach piles becoming less rare, and lately, several heap leach and ripios pile liquefaction issues have occurred.

A number of treatises on heap leaching with some reference to liquefaction and a few technical papers describing the liquefaction analysis of specific heap leach piles have been published, but a complete dossier of various issues related to liquefaction of heap leach piles does not exist. This paper is expected to be a liquefaction primer for heap leach projects, exploring liquefaction potential and functioning, ore types and treatment producing vulnerable heaps or ripios dumps, material characterization and various design and analysis practices currently in use. Some issues of gross mischaracterization of the situation stemming from either generally bad overall practice, failure to adequately consider long-term aging in an acid leach environment and the possibility of liquefaction not necessarily related to seismic loading, will be referenced with the ultimate aim of developing a more complete and safer basis for designing and developing the heap leach piles.

Introduction – what is liquefaction?

Liquefaction of soils is a phenomenon in which the strength and the stiffness of soil are suddenly reduced due to a critical change in loading conditions. Liquefaction is typically a concern only in saturated or near-saturated soils, and the critical change implies an alteration of the external loading resulting in an increase

in the pore water pressure. This increase occurs when the soil particles are loosely packed and the bulk material attempts to contract under the loading; the contraction reduces the pore space, creating excess pore pressure. With sufficient excess pore pressure, the soil can behave like a viscous liquid or a frictional fluid, with little or no shear resistance. In terms of effective stresses, when the excess pore pressure in the soil skeleton equals the total stress, the effective stress contributing to shear strength or shear resistance becomes zero and the soil liquefies.

Various definitions, describing this condition differently, exist in the literature; for example, Seed (1979) described liquefaction as a condition where a soil undergoes continued deformation at a constant low residual stress or with low residual resistance due to the build-up and maintenance of high pore water pressure. On the other hand, to avoid confusion between many related or similar phenomena triggered by different mechanisms, some have labeled them differently and put them in different categories like flow-liquefaction, cyclic mobility, fluidization, etc. The subtle differences between these are important, but will not be focused on or elaborated in this paper

The most common changes in external loading associated with liquefaction in a leach pile or tailings facility are excessive rate of rise and earthquakes, the former leading to what is referred to as static liquefaction, which can be a contributing factor in failures of upstream constructed tailings dams.

From classical to critical state soil mechanics

The term liquefaction was first used in reference to the failure of the Calaveras hydraulic fill embankment dam in California, when the upstream shell slid off as the dam was reaching a height of 64 m (Hazen, 1918). Historically until much later in the century it was considered to be a phenomenon associated with loose coarse-grained soil or sands, and most studies conducted were on sands only. Over time, liquefaction has also been observed in gravel and silt, and much emphasis has been placed on establishing firm criteria for determining the soil susceptibility to liquefaction. For example, an analysis of the major earthquakes in China for a period of ten years established the common characteristics of the liquified soils in terms of limited ranges of some index properties (Wang, 1979). Dubbed as “Chinese Criteria”, this became the norm for judging the susceptibility of soils to liquefaction for some time, with the limits being modified later as “clay content $\leq 15\%$, liquid limit ≤ 35 and water content $\geq 90\%$ of liquid limit,” and the standard set as, “liquefaction of fine-grained soil will occur only if all three criteria are met with” (Seed and Idriss, 1982).

Later, Seed et al. (2003) suggested a criterion based on Liquid Limit and Plasticity Index, separating the soils into three groups of varying liquefaction susceptibility to cyclic pore pressure increase, with the applicability limited to certain ranges of fines content (FC) and plasticity index (PI).

Subsequently Bray and Sancio (2006) came up with still other criteria similar to Seed et al. by way of defining three different levels of liquefaction susceptibility (susceptible, moderately susceptible, and not susceptible) for soils with $FC \geq 50\%$. Figure 1 shows a graphical representation of the three criteria mentioned above. There have been some other similar developments, but the three mentioned above seem to have been or are being the most commonly used.

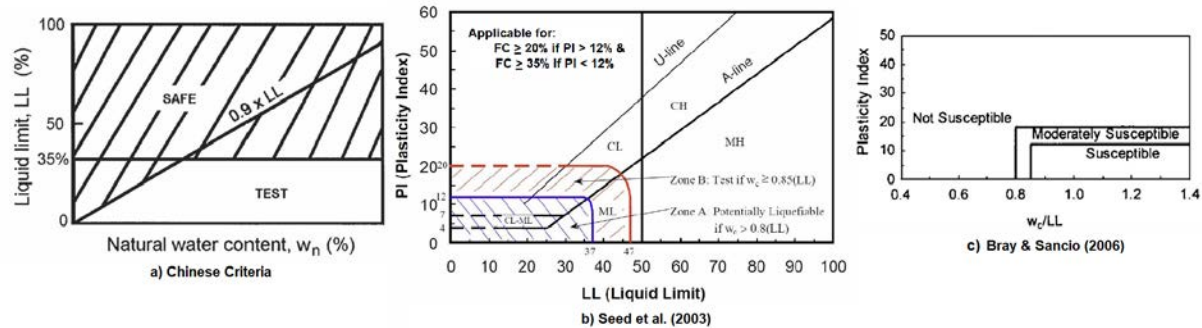


Figure 1: Graphical representation of some liquefaction susceptibility criteria (reproduced and adapted from Green and Ziotopoulou, 2015)

With more earthquakes and cases of liquefaction around the world since then, any criterion focusing mainly on moisture content and plasticity of soil seemed to be inadequate to explain the phenomenon in every case and, thus, focus has shifted. Research, analyses, laboratory testing, and field investigation early in the last century had led to the finding that coarse-grained soils or sand in an initially loose state have a tendency to contract upon shearing under a constant normal stress or pressure, while the same sand in initially dense state tends to dilate, both approaching the same steady state void ratio, as the shearing continues. This steady state void ratio, labeled as the “critical void ratio”, was found to be a function of the normal pressure applied during the shearing process, giving rise to a unique curve on the void ratio (e) versus logarithm of the normal pressure ($\log p$) plot, identified as the Critical State Line or CSL (Casagrande, 1938). This was in a sense the onset of the field of Critical State Soil Mechanics and a new approach to the treatment of liquefaction.

Figure 2 shows a modified version of the diagram mimicking some direct shear test results that Casagrande used to explain these characteristics of loose sands leading to liquefaction. A soil consolidated under an effective pressure p_1 to a void ratio e_1 represented by the point C_1 above the CSL in Figure 2, if sheared, will tend to contract, while the same soil with the initial conditions represented by the point C_2 with consolidation pressure p_2 and void ratio e_2 , below the CSL, if sheared will tend to dilate. However, if the volume changes in the two cases are prevented, or the shearing is too fast to allow the re-adjustment of the granular structure, the soil responds by

transferring part or all of the inter-granular stresses to the pore water – the stipulated contractive and dilatative tendencies until the critical state is reached, giving rise to positive and negative pore pressures, respectively. As shown by the line C_1D_1 representing the undrained shearing (no volume change allowed) of the soil with contractive behaviour, this results in a significant drop in effective stress and thereby a loss of strength, causing liquefaction. On the contrary, the path C_2D_2 representing the undrained shearing of the soil with dilatative tendency and the corresponding negative pore, will not be conducive to liquefaction.

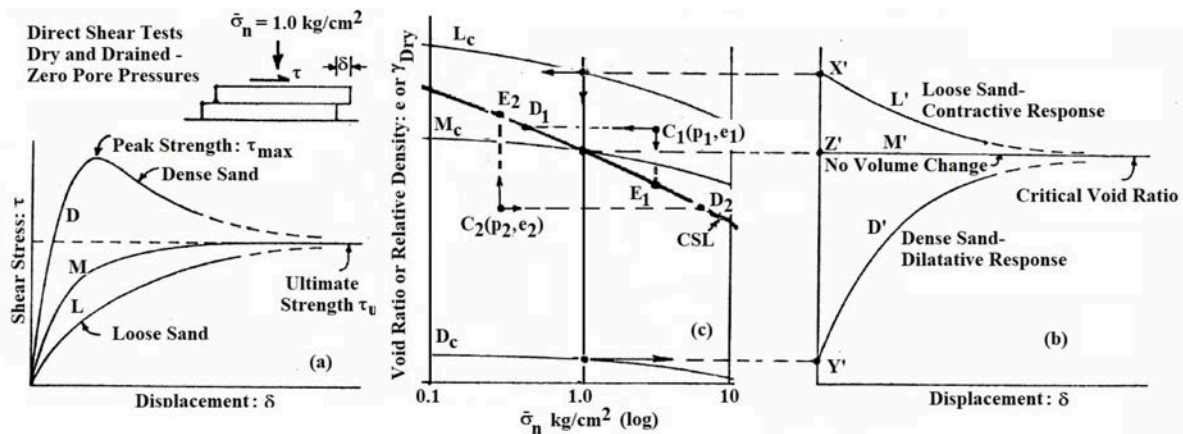


Figure 2: Graphical demonstration of Critical State Line from shear test result (after Casagrande, 1938)

Thus, it is clear that liquefaction is not only a function of the saturation and the plasticity of soil, but also depends on the effective normal stress applied. Casagrande concluded that all combinations of void ratio and effective normal stress located below, or to the left of the CSL in Figure 1(c), represent states in which the sand would develop dilatative response and thus be safe against liquefaction, while the points to the right or above the CSL would correspond to a contractive response resulting in a slump of limited dimensions, or a flow slide, depending on the resulting drop in effective stress.

History of liquefaction in heap leaching

Although liquefaction flow slides under conventional static loading conditions are well documented for mining wastes (e.g., Davies et al., 2002), these accounts have been invariably in reference to tailings impoundments and coal (fly ash) waste dumps. Most heap leach ores are well drained even after leaching, which makes heaps generally of low susceptibility to liquefaction. The only well documented case of heap liquefaction is for the Cuajone copper mine in Southern Peru, which was triggered by the M_w 8.4 Moquegua, Peru earthquake on June 23, 2001 (Rodriguez-Marek et al., 2004). Although located just 60 kilometres (km)

from the epicentre, the failure and the resulting lateral spreading were restricted to the uppermost lift of the heap (Figure 3). Another nearby heap suffered a slope failure but the cause is disputed. Both heaps consisted of low permeability ore with high degrees of saturation.

The liquified lift shown in Figure 3 was under active leaching at the time and the likely cause of liquefaction was determined as the high solution application rate, in conjunction with the relatively fine gradation of the ore and the presence of an underlying inter-lift liner inhibiting pore pressure dissipation. It is worth mentioning, however, that the presence of an inter-lift liner is likely the reason the liquefaction was restricted to only the upper lift.



Figure 3: Liquefaction of the Cuajone Copper Heap, Peru

On the other hand, liquefaction of spent ore (commonly called ripios in South America) dumps is becoming common for Chilean copper mines using dynamic heaps (on/off leach pads). Sulfuric acid leaching of copper ores often results in significant chemical degradation of the ore, producing ripios of very high fines content, low permeability, and low shear strength. Leach cycles for dynamic heaps with poor quality ores can be very long, in part due to the low permeability. For example, at one mine in Northern Chile some of the ore can be irrigated at a maximum of 4 litres per hour per square meter (L/h/m^2), for just 12 hours per day, for an effective irrigation rate of 2 L/h/m^2 . The ripios from such heaps is often saturated and very weak. Figure 4 shows ripios at a Chilean mine that has liquefied under its own weight (static liquefaction) during deposition in the ripios dump. In some cases, static liquefaction of the ripios dump has created serious safety risks for operators and traffic at near-by roads. Another copper mine in Chile is currently upgrading ore blending and geotechnical monitoring at its conventional (permanent) heaps to mitigate the risk of liquefaction.

Heap leach projects are getting larger, with heap heights greater than 150 meters (m) becoming common. Mining rates are increasing ten-fold every three decades (Robertson, 2011) and there is now at

least one copper mine moving over 1 million tonnes daily. Ore quality is getting worse as miners search for new reserves, and longer leach cycles result in more chemical degradation. Dynamic heaps are increasingly used for poor quality copper ores, and are the prime method considered for nickel ores. Most heap leach projects across the world, including Chile, Peru, Turkey, Nevada, and California, are in seismically active areas. Thiel and Smith (2004) have described in some detail the various reasons for and implications of increased liquefaction susceptibility in heap leaching, including static liquefaction, and earthquake-induced flow slides.



Figure 4: Static liquefaction of ripios, Chile

Triggering mechanisms

Liquefaction triggering is a function of the properties of the soil or leach ore itself, and the mechanism inducing the appropriate loading conditions. The triggering mechanism for liquefaction could be static or dynamic loading. Monotonically increasing shear load imposed by increasing embankment heights, oversteepening the slope, toe erosion, construction equipment loading, reservoir filling, and progressive failure leading to steeper slopes are some of the examples of static loading leading to liquefaction flow (Jefferies and Been, 2016; Sadrekarimi, 2014). In upstream tailings dams, static liquefaction can be triggered if the rate of rise of the crest of the dam is excessive (Davies et al., 2002). As demonstrated by the ripios dump liquified under its own weight in Figure 3, static liquefaction can be a problem with nickel leach heaps or ripios dumps (Smith and Christie, 2015). While seismic shaking, blasting, and pile driving are the most obvious and common examples of dynamic loading that could trigger liquefaction, machine vibrations, effects of water waves, rapid sediment accumulation and groundwater movements are also considered to be potential triggering mechanisms (Owen and Moretti, 2011). In this paper the only dynamic trigger discussed is seismic. Once triggered, the strength of a soil susceptible to flow liquefaction is no longer

sufficient to withstand the static stresses that were acting on the soil before the disturbance, and failure ensues.

Based on the flow slides observed in tailings piles and coal waste dumps, Thiel and Smith (2004) gave some indicators of liquefaction and flow slide susceptibility applicable to heap and dump leach piles as well (Table 1). In the years since then, some of the indicators are found to be less relevant and the respective thresholds either too stringent or too liberal. The changed perspective and the considerations leading to changes are listed alongside in the column added to the original table. However, more recent experience with Chilean ripios dumps suggests the height threshold can be as low as a few meters and the foundation slope may not be relevant within the range of heap leach pad geometries.

Table 1: Indicators of flow slide susceptibility (updated from Thiel and Smith, 2004)

Parameter or characteristic	Threshold (approximate)	Changed perspective
Maximum height	≥ 100 m	CPT work shows the liquefaction triggering below 30 m being hard due to high initial stress and relatively low induced dynamic stress.
Foundation slope	≥ 15 degrees	May not be relevant within the common range of heap leach geometries, but still relevant because steeper slope increases the runout distance
Location, terrain	Incised valley	Existing heap leach operations with liquefaction issue are not in incised valley and this restriction is also from the runout consideration
Inter-bench slopes	Near angle of repose	–
Heaped moisture content of ore	$\geq 5\%$	Too low; can be increased to $\geq 10\sim 12\%$ but is very ore dependent
Saturated permeability of ore	$\leq 1 \times 10^{-2}$ cm/sec	Perhaps too high; < 2 to 3 times the irrigation rate may be a better threshold
Saturation (at any point in the heap)	$\geq 85\%$	–
Other factors:	• No toe support • Finer material near the base • Water, impermeable layer at base	Water on a low permeability layer anywhere in the heap; most of the critical sections have shown high-risk zones well above the base

Dynamic liquefaction hazard assessment

The cyclic stress approach (Seed and Idriss, 1967; 1971), often dubbed as the “simplified approach” or “simplified method,” evaluates the earthquake induced liquefaction susceptibility from estimates of induced shear stresses and the shear stresses required to cause liquefaction expressed by two non-dimensional ratios-

cyclic stress ratio (CSR) and the cyclic resistance ratio (CRR), respectively. If the CSR is close to or greater than the CRR, liquefaction can occur.

In the simplified method, the CSR is based on the observation that the effect of the non-uniform stress cycles during an earthquake could be equated to that produced by an equivalent number of uniform stress cycles with 65% of the actual peak shear. Thus, the expression for CSR is given as:

$$\text{CSR} = 0.65 (a_{\max}/g) \cdot (\sigma_{v0}/\sigma'_{v0}) \cdot r_d \quad (1)$$

Where: $a_{\max}$ = Peak horizontal ground surface acceleration during the earthquake (generally determined from deterministic or probabilistic seismic hazard assessment)
 g = Gravitational acceleration
 σ_{v0} = Total overburden stress
 σ'_{v0} = Effective overburden stress, and
 r_d = Stress reduction coefficient guided by the flexibility of the soil profile and depth

Peak horizontal acceleration is based on the earthquake under consideration. The stress reduction coefficient, r_d at any depth is known to vary with the flexibility of the soil profile and the range of variation increases with depth. The National Earthquake Engineering Research Council has considered the following empirical bilinear relationship between r_d and depth (Liao and Whitman, 1986) to be suitable for use in routine engineering practice (Youd and Idriss, 2001):

$$r_d = \begin{cases} 1.0 - 0.000765 \cdot z & \text{for } z \leq 9.15, \text{ and} \\ 1.174 - 0.0267 \cdot z & \text{for } 9.15 < z \leq 23. \end{cases}$$

Where: z = Depth below surface in meters

The soil's cyclic strength, or CRR, can be estimated from either laboratory or field testing, and often both are used. While the classical approach to determining liquefaction susceptibility from laboratory testing is based simply on the moisture content and standard index properties including particle size gradation (viz., Seed et al., 2003; Bray and Sancio, 2006), the critical state approach requires direct measurement of dynamic or cyclic strength. Cyclic triaxial or cyclic direct shear tests are performed over a range of initial in-situ stress conditions to determine the number of loading cycles required to initiate failure under a range of earthquake loads.

The two most common field techniques used to determine CRR are the standard penetration test (SPT) and the cone penetration test (CPT). It is customary to use an earthquake of magnitude $M_w=7.5$ as the base or reference in such determinations, though a more site-specific event can be substituted. A number of empirical equations to estimate CRR from SPT blow count (N) and CPT tip resistance (q_c) have been proposed in the last forty some years. One relationship widely used to compute CRR from SPT data is (Youd and Idriss, 2001):

$$CRR_{7.5} = \frac{1}{\{34 - (N_1)_{60,CS}\}} + \frac{(N_1)_{60,CS}}{135} + \frac{50}{\{10 \cdot (N_1)_{60,CS} + 45\}^2} - \frac{1}{200}$$

Where:

$(N_1)_{60,CS}$ is the SPT blow count with appropriate corrections for energy, borehole diameter, sample barrel, rod length, and grain characteristics (fines) applied to the raw N-value recorded in the field (Youd and Idriss, 2001).

The equation is considered valid for $(N_1)_{60,cs} < 30$ under the assertion that for $(N_1)_{60,cs} \geq 30$ represents soils *too dense to liquify*.

Commonly used expressions for deriving CRR from CPT data, for an earthquake of magnitude $M_w=7.5$, are (Robertson and Wride, 1998),

$$CRR_{7.5} = 0.833 \left\{ \frac{(q_{c1N})_{cs}}{1000} \right\} + 0.05; \quad \text{for, } (q_{c1N})_{cs} < 50$$

and,

$$CRR_{7.5} = 93 \left\{ \frac{(q_{c1N})_{cs}}{1000} \right\}^3 + 0.08; \quad \text{for, } 50 < (q_{c1N})_{cs} < 160$$

Where:

$(q_{c1N})_{cs}$ is the cone penetration resistance obtained after applying corrections for overburden stress and grain characteristics to raw value q_c as obtained in field.

Once estimates of the earthquake loading, CSR, and the soil's strength, CRR, are obtained, the factor of safety (FOS) against liquefaction is determined in the classical way of dividing the strength by the stress:

$$FOS = CRR / CSR$$

Since the CSR is estimated from the seismic hazard assessment, either deterministic or probabilistic, the uncertainty in that parameter is tied to the hazard assessment and can be quantified accordingly. There is also uncertainty in the determination of CRR using the in-situ test data and, to this end, the Bureau of Reclamation (USBR, 2015) recommends that the CRR obtained using in-situ data is not to be considered a hard lower bound on the cyclic loading that could cause liquefaction; instead, it should be treated as an estimate with a probability of liquefaction of 15 to 30%. For structures such as tailings and water dams, there is a wealth of literature to guide designers in adopting suitable minimum factors of safety. For heap leach facilities (heaps and ripios dumps), this is not the case. Thus, the designer should establish acceptance criteria based on the specific facility's consequences of failure.

Static liquefaction hazard assessment

Static liquefaction can occur when loads are applied too quickly. The most common incidents in the mining sector are upstream raises to tailings dams, where the rate of applied load due to the increasing height of the dam can exceed the strength gain in the underlying tailings. However, this can also occur in ripios

dumps, or in waste rock dumps placed over weak, wet soil foundations. Plant residue “dry stack” dumps for nickel processing can also be subject to this mode of failure, as the residue is wet and requires time to consolidate and gain sufficient strength for stable slopes. For leach heaps, static liquefaction should generally not be a significant risk even with very poor-quality ores.

Static liquefaction potential is generally assessed using the critical state approach, where the state parameter (Ψ) is defined as the difference between the current void ratio and the void ratio at critical state ($e_{\text{current}} - e_{\text{critical state}}$), at the same effective stress (Jefferies and Been, 2006). The state parameter can be calculated directly from the CPT data, and for this reason, CPT investigations often include producing profiles of Ψ versus depth for each hole. Davies (1999) and Davies et al. (2002), suggest the limiting value of Ψ to guard against static liquefaction as -0.1 , implying that values of Ψ greater than approximately (-0.1) could be subject to liquefaction, even though $\Psi < 0$ suggests a dilatant state.

However, the CPT approach only works for existing facilities. For facility planning, the risk of static liquefaction is assessed by investigating pore pressure build up due to rapid stacking of a ripios dump (or, in rare cases, the leach heap) through a combination of laboratory testing and numerical modeling very similar to that used for designing upstream tailings dam raises. Just as with tailings dams, as a ripios dump is raised with sequential lifts of waste, the underlying waste needs to consolidate to dissipate the pore pressures, resulting in an increase in effective stress contributing to strength. The rate of consolidation in turn is controlled by the permeability of the material and the distance to a drain. Because some ripios, especially those from sulfuric acid leaching of copper and nickel ores, can have very low permeabilities, the rate of consolidation and the resulting pore pressure dissipation and strength gain can be very slow. If the rate of rise is too quick, the increasing stresses from the freshly placed material can exceed the strength and a static liquefaction ensues.

Modeling and software for liquefaction analysis

A variety of constitutive models like UBCSAND, Hypoplastic, and CASM (Zardari, 2011) have been implemented in commercially available finite element or finite difference computer software to analyze static as well as dynamic liquefaction. Castillo et al. (2005) have demonstrated the use of UBCSAND coupled with the finite difference software FLAC to predict dynamic liquefaction potentials during an earthquake for heap leach pads. Computer analysis using these models and software allow the designer to both estimate the risk of static liquefaction and design effective counter measures. Such measures may include the use of vertical or horizontal drains to speed consolidation and thereby strength gain, buttressing the high-risk areas with better quality material such as waste rock, slowing the rate of rise by constructing a larger area for disposal, or mixing the ripios with better quality material (this is currently being used at

one Chilean copper heap leach project, and was previously used at a gold valley fill heap leach project in Peru).

Heap leach analogues

Since there have been no major liquefaction failures in heaps, and only a few in ripios dumps, analogues become important. Dawson et al. (2011) mention a number of flow slide failures over the past quarter century in Rocky Mountain coal mine waste dumps with heights of 100 to 400 m. These dumps were constructed by end-dumping and stabilized at their natural angle of repose of approximately 38°. Initial collapse of structure in some nearly saturated sandy gravel layers within many of these dumps are believed to have caused high pore pressure build up and liquefaction in spite of safe designs produced by conventional static slope stability analyses.

Among the breaches in coal ash ponds, the infamous dyke failure at Tennessee Valley Authority's (TVA) Kingston plant in 2008 is considered as one of the most expensive environmental and industrial disasters. The Kingston failure was the first tailings-like failure to exceed \$1 billion in remediation costs. Kingston is an important reference for designing dumps of wet, lower permeability material such as heap leach ripios. Irrespective of whether or not the TVA disaster and other coal ash failures could be decisively attributed to liquefaction, for the dykes being constructed of coal ash, often in the near-saturated state, the liquefaction susceptibility is certainly a reality and is drawing attention. The analogy to heap leaching or building a ripios dump of saturated, weak material such as acid leached copper or nickel ore is self-evident.

Finally, tailings facilities are good analogues, if not at face value, certainly from consideration of the fundamental soil mechanics. The failure of tailings deposits due to liquefaction has been observed for at least sixty years (WISE, 2020). One of the most talked about tailings dam failures due to liquefaction is the El Cobre copper mine dam slide in Chile, triggered by a strong earthquake in 1965 and resulting in more than 200 deaths with 2.25 million cubic meters (m³) tailings released and travelling 12 km downstream. In 1966 at the Merthyr Vale Colliery-owned coal mine in Wales, UK, a liquefaction-induced dam failure following a heavy rain caused the release of 162,000 m³ of tailings, killing 144 people. In 1994 the Merriespruit gold tailings dam in South Africa failed, causing 17 deaths and considerable damage. Forensics established that a good percentage of the material in the failed area was of a contracting nature, in a metastable state and the overtopping in conjunction with the erosion of the impoundment wall resulted in static liquefaction of the tailings and a consequent flow failure. The 2015 static liquefaction failure of the Fundão tailings dam at Germano mine in Brazil killed at least 19 people, flooded the town of Bento Rodrigues with polluted slurry which travelled 663 km to the Atlantic Ocean, and caused some \$5 billion in damages. This failure was also attributed to inadequate drainage leading to liquefaction. Still fresh in our memories is the January 2019 devastating failure of the tailings dam B-1 at Vale S.A.'s Córrego do Feijão

Iron Ore Mine in the state of Minas Gerais, Brazil that released 12 million m³ of tailings flowing down 7 km and killing at least 259 people. The expert panel investigating the case released its final report in December 2019, which declared the failure and the resulting flow slide to be caused by static liquefaction within the tailings body in the dam itself and linked it to a history of marginal designs, inadequate water management, poorly understood tailings characteristics, and excessive rain in the region.

Summary and lessons to be learned

Increasing heights of the heaps and ripios dumps, in conjunction with many drastic failures in the sibling arenas attributed to liquefaction, demand a full understanding of and greater attention to this phenomenon by heap leaching sector professionals. The general principles and the susceptibility criteria for static as well as dynamic liquefaction remain the same as adopted in the sibling arenas of tailings dam, described above, and the failures have a lesson for the heap leach industry. They strongly point to the need for proper consideration of liquefaction susceptibility for ores and ripios of relatively low quality, especially for acid leach operations such as copper, nickel, and uranium. For most operations that have liquefaction susceptible materials serious problems can be avoided if recognized in the design. However, when this is neglected, the problem is much harder to mitigate once in operations and risk to property, the environment and human health can be significant. Operational changes required to circumvent the liquefaction potential, where possible, may even have drastic financial consequences. Some tools helpful to identify and mitigate liquefaction risk are given below:

Ore and ripios characterization: Laboratory testing for grain size analysis with focus on percentage fines, plasticity characteristics, unit weight, permeability and shear strengths of the ore forming the heap, are components of standard testing currently practiced, and must be continued. However, while degradation of both the physical and mechanical properties of the ore upon leaching is a well-known phenomenon, by and large, leach ore tests are generally performed on fresh or slightly aged ore. Such tests can be misleading, and this is one of the reasons that several Chilean heap leach operations are now spending hundreds of millions of dollars to retrofit their facilities. It is recommended that these tests be performed on aged (leached) as well as fresh ore samples.

Additional laboratory testing: Cyclic triaxial or direct shear testing are not part of routine testing for heap leach piles. Understandably, these are more difficult as well as expensive tests. The sample size restrictions and the elaborate setup to get meaningful results from these tests impose additional constraints and not every testing laboratory is equipped to perform these. Nevertheless, these tests should be considered essential for any project with liquefaction-susceptible ore and ripios and the industry must work to eliminate the constraints in conducting these tests and getting meaningful results. As with the other testing, it is

important that these tests be performed on properly aged samples to simulate the long-term performance in the heap or ripios dump.

Heap design: Designing heaps and ripios dumps for free draining slopes is a practice that at least one major mining company have adopted, and is best practice for any heap that may have limited long-term permeability. Drained slopes can be achieved by selectively stacking the best quality ore in these areas, by constructing drains similar to a chimney drain in a dam, or by installing vertical drains such as wick drains as the heap is raised (these approaches are in use in two and several mines, respectively).

Operational checks: Once in operation, the heaps and ripios dumps should be evaluated at some routine frequency, ideally using cone penetrometer tests in conjunction with geophysical surveys to detect high saturation zones. Installing an array of piezometers in the heap and regular monitoring to keep track of the phreatic surface is also a good practice and is now being followed in some heap leach facilities. Also, any increase in the irrigation rate during the operation, perhaps for improved kinetics or other metallurgical needs, must be evaluated for the geotechnical impacts and potential contribution to liquefaction risk.

Additional analysis: Considering the large distances to which the liquefied materials have flown in various flow slides, for risk assessment and potential risk management, a run-out analysis should be considered an integral part of operational control as the leach pile height rises. Since the first of its kind analysis method was developed by Lucia (1981), a number of empirical-statistical and analytical models for two and three-dimensional run-out analysis have been developed, viz. Dan-W (Hungr Geotechnical Research, Inc.) and Flo-2 D (Flo-2 D Software, Inc.).

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Pulp Agglomeration Process and Application for Gold and Silver Heap Leach Operations

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Abstract

Pulp agglomeration is a hybrid heap leach and milling process in which a milled, high-grade ore slurry is blended with crushed low-grade ore and agglomerated with cement to allow for further processing and metal recovery by heap leaching. The main advantage of pulp agglomeration compared to a conventional cyanidation mill is that the pulp agglomeration circuit does not require a dedicated tailings storage facility because mill tailings are stored in the heap. Pulp agglomeration is a viable and proven alternative to conventional cyanide milling that has recently been implemented by Pan American Silver at their Dolores Mine, and has also been successfully used in the past at the Castle Mountain and Ruby Hill mines.

In a typical pulp agglomeration circuit, low and high-grade ore is crushed in a two- or three-stage conventional crushing circuit, and stockpiled. High-grade ore is reclaimed and ground using conventional grinding technology. The resulting slurry may be pumped to an agitated filter feed tank, which functions as a single-stage leach tank, followed by filtering, or the slurry may be sent directly to a thickener. Pregnant filtrate or thickener overflow solution is processed in a Merrill-Crowe or ADR plant that is shared with the heap leach circuit. The resulting filter cake or thickener underflow is mixed with crushed low-grade ore from the low-grade stockpile, agglomerated with cement, and stacked onto a leach pad for cyanide leaching. Pulp agglomeration provides several operating advantages, including faster and increased recovery of gold and silver values from the high-grade ore compared to heap leaching only, reduced capital and land requirements compared to a conventional mill with agitated leach and tailings impoundment, and operational flexibility for processing low- and high-grade ore.

This paper provides a general overview of the pulp agglomeration process, important design parameters and considerations for pulp agglomeration, and an overview of projects where pulp agglomeration has been used.

Introduction

Pulp agglomeration is a hybrid heap leach and milling process that has only been sparsely utilized in the

mining industry. The process as discussed in this paper was first known to be utilized in the early 1990s at Viceroy Gold's Castle Mountain mine (Zaebst, 1994). However, a similar process, which considered combining a milled product with crushed material in an agglomeration drum for heap leaching, was previously implemented successfully by Comco, with assistance from KCA, in the late 1980s at Cerro Rico de Potosi in Bolivia (Rose et al., 1990). The novel approach allowed Castle Mountain to achieve higher recoveries from high-grade ore by grinding without having to construct a tailings pond, which would have taken considerable time and effort in permitting. Since the implementation at Castle Mountain, pulp agglomeration has been applied successfully at the Ruby Hill mine and Dolores mine, and has potential to be applied elsewhere.

The pulp agglomeration process consists of blending a ground product slurry (coarse or fine grind) with a crushed product and binding them with cement to form stable agglomerates for heap leaching. This can be accomplished using many different processing methods depending on the specific ore and project requirements. A typical flowsheet considers two product streams for low and high-grade ore. The low-grade ore is crushed as required for regular heap leaching, which may be accomplished using a conventional jaw and cone crushing plant, mineral sizers, gyratory crushers, high pressure grinding rolls (HPGR), etc. depending on the ore type, final crushed product size required, and throughput.

High-grade ore is similarly crushed, which can be accomplished using a shared crushing circuit and campaigning low and high-grade ore onto separate product stockpiles, or by separate dedicated crushing circuits, followed by grinding in a mill. Milling equipment and stages are dependent on the target grind size of the product. The milled product is dewatered, typically by filtering or thickening, before being recombined with the low-grade ore product and agglomerated with cement. If cyanide is present in the milling circuit, it is common to leach a significant portion of the recoverable metal values from the ore before final processing by heap leaching. Pregnant leach solution from the mill is collected as filtrate from the filtration system or thickener overflow solution and sent to the recovery plant. Filtrate solution is typically dirty, and may require a clarification step prior to processing in the recovery plant.

Agglomeration is typically accomplished using an agglomeration drum. The blend of low and high-grade ore, as well as cement addition requirements, are determined through test work on the material with typical blend ratios ranging from 2.5:1 to 4:1 low-grade to high-grade, but may be even higher. Depending on the characteristics of the filtered material, additional equipment, such as a mixer to break up the filter cake and blend cement or a re-pulping system to control moisture, may also be required.

Capital and operating costs for pulp agglomeration is largely dependent on the processing requirements and throughput of the project. In general, capital costs are lower compared to conventional cyanidation mill because the agitated leaching circuit and tailings storage facility are not needed. Depending on the tailings disposal method and site topography, the overall capital savings can be significant. Cement

is the most significant added operating cost. This cost is typically in the range of US\$0.50 and US\$1.00 per total tonne processed, but can exceed US\$2.00/tonne. This is partially off-set by cyanide destruction savings, which is not required for pulp agglomeration and lime requirements as the cement acts as the pH buffer for the heap.

Pulp agglomeration design considerations and parameters

Important design considerations for pulp agglomeration include recovery improvements for gold and silver by grinding versus conventional heap leaching, ratio of low-grade to high-grade ore (both available, and required), dewatered high-grade ore moisture content, cement addition requirements and material properties. These design considerations must be supported by the mine plan (with respect to low-grade and high-grade material available for blending) and project economics. There will also be some logistical challenges, such as the availability and delivery of the large quantities of cement that may be required. In order to evaluate and develop a potential pulp agglomeration project, a detailed and thorough laboratory test program is necessary.

Column and bottle roll leach tests, including grind size versus recovery tests, are necessary to estimate final recoveries and select target crush and grind sizes. These tests will also enable designers to determine if there is an economic benefit for milling with cyanidation of the high-grade ore versus conventional heap leaching or other recovery method (such as gravity or flotation). Characterization work including abrasion and work indices, specific gravity, and multi-element analyses of the material can be performed as part of the initial leaching program. Once a target grind size is selected, rheology, filterability and thickening test work should be performed to determine the dewatering potential for the high-grade slurry.

Heap permeability is the most significant risk for pulp agglomeration due to the introduction of a large quantity of fines to the heap. In order to mitigate this risk, a detailed compacted permeability test program is critical. Compacted permeability testing simulates heap conditions under simulated loads and evaluates the performance of the heap based on permeability, material slump, and solution clarity. In a typical compacted permeability test program, material is tested under the simulated load of the maximum heap height at various cement addition rates; stage loading at different heap heights may also be considered. A failure in solution flow (permeability) or slump indicate that the heap will most likely not perform well under the expected conditions. A failure in solution clarity suggests a migration of fines, or degradation of agglomerates; however, if the sample passes based on solution flow and slump, a failure for solution clarity does not necessarily mean the heap will not perform well. A typical compacted permeability test apparatus is presented in Figure 1.

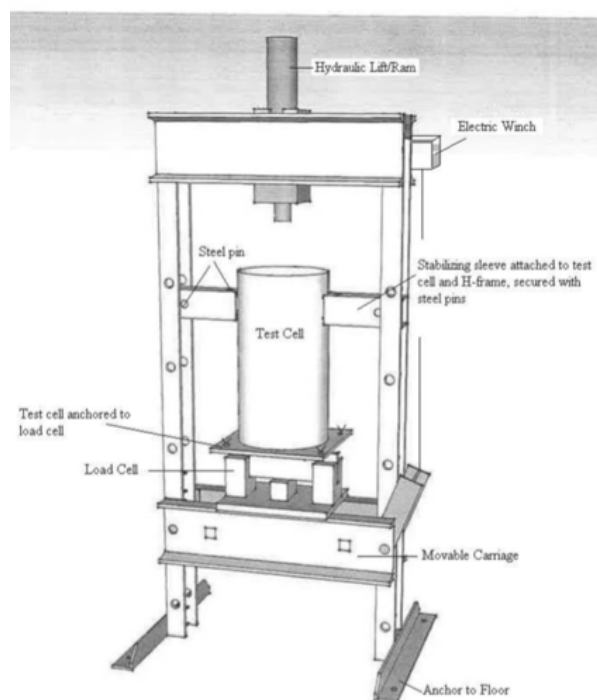


Figure 1: Compacted permeability test apparatus

Compacted permeability testing for pulp agglomeration should include multiple candidate blending ratios of high-grade ore at the target grind size and crushed low-grade ore at different cement addition rates. Depending on the results, adjustments to the ratio of low-grade to high-grade ore, cement addition rate or moisture could be considered to improve the simulated conditions. Once a target blend ratio and cement addition rate is selected, confirmatory tests should be performed to ensure the results are repeatable. It is important that all test work be performed on representative samples.

Pulp agglomeration projects

Castle Mountain

Operating parameters and process description for the historical Castle Mountain pulp agglomeration process are summarized directly from Zaebst (1994). A simplified flowsheet is presented in Figure 2.

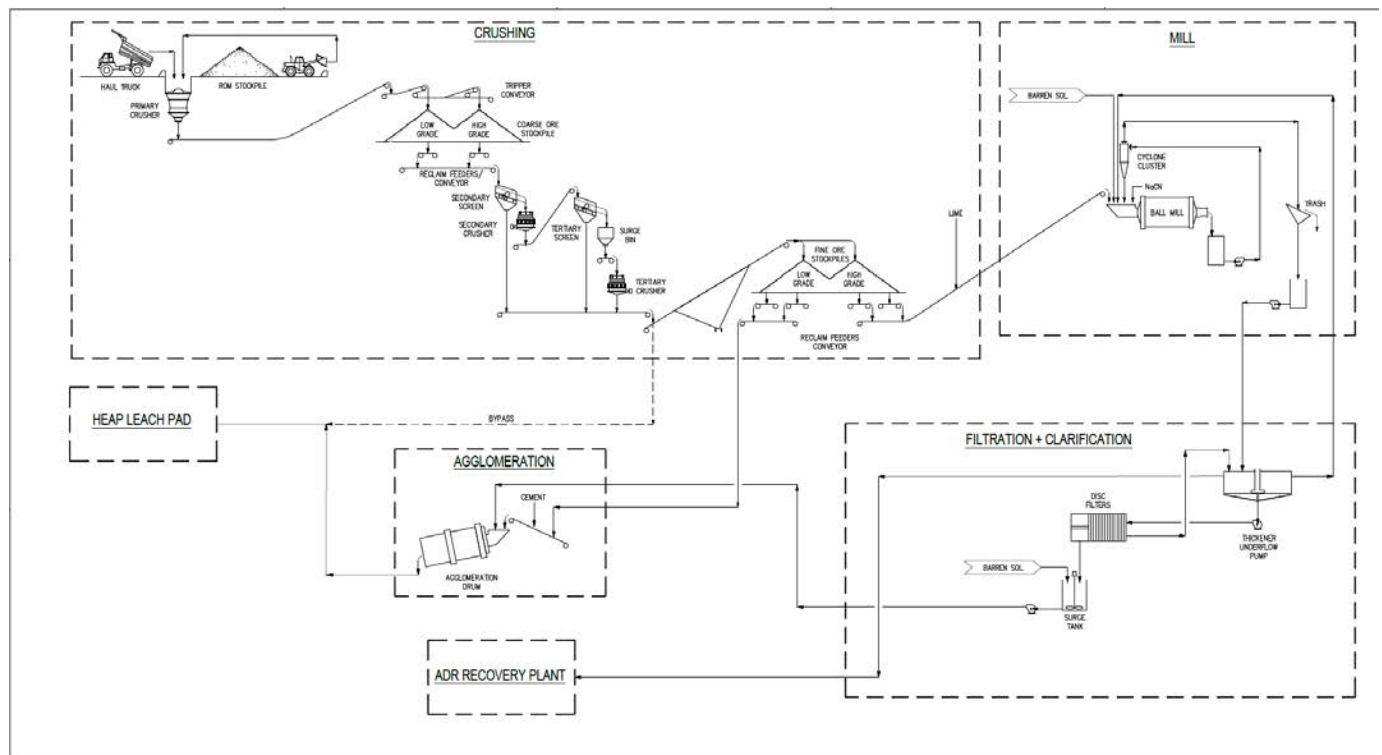


Figure 2: Castle Mountain pulp agglomeration simplified flowsheet

Key design parameters

Key design parameters for the historical Castle Mountain pulp agglomeration process are summarized in Table 1.

Table 1: Key design parameters for the historical Castle Mountain pulp agglomeration process

Item	Design parameter
Low-grade production rate	8,000 tons per day
Low-grade crushed product size	P80 3/8"
High-grade production rate	1,200 tons per day
High-grade crushed product size	P80 100 mesh
Low-grade : high-grade ratio at agglomeration drum	3.5:1
Overall low-grade : high-grade ratio	6.7:1
Cement addition	8 lb per ton of ore

Process description

Low- and high-grade ores at the Castle Mountain project were crushed to 80% passing 3/8" in a shared three-stage, open crushing circuit. Primary crushing was accomplished using a gyratory crusher with secondary and tertiary cone crushers. The final crushed product was conveyed directly to the leach pad by an overland conveyor, or was stockpiled onto an 8,500 ton fine ore stockpile. A radial stacker was used to direct ore to the high-grade stockpile for further processing by milling, or to the low-grade stockpile to feed the agglomeration drum.

Grinding for high-grade ore was accomplished in a single stage using a 1250 HP 12' × 16' ball mill. Ore was ground in a barren process solution with lime added for pH control. The ball mill product was pumped to a cyclone cluster for classification to produce a final product at P₈₀ 100 mesh. The cyclone overflow discharged across a vibrating trash screen before being pumped to a thickener, and cyclone underflow was recycled back to the ball mill. The mill product was thickened to approximately 65% solids before being pumped to two disk filters. Filtrate from the disk filters reported back to the thickener and the resulting filter cake was then re-pulped to 69% solids using process solution for agglomerate moisture control, before being pumped to the agglomeration drum. Overflow from the thickener solution was directed to the carbon adsorption circuit for recovery of gold values from solution.

Agglomeration of crushed low-grade ore and high-grade slurry was accomplished in a 9' × 34' agglomeration drum, with cement added at 8 lb per ton to produce agglomerates with a moisture content of 9 to 10%. Discharge from the agglomeration drum was combined with other low-grade ore and conveyor stacked onto a heap leach pad in 30-ft lifts, before being leached with a low strength sodium cyanide solution. Gold and silver values from the pregnant heap leach solution were recovered in the carbon adsorption circuit. Loaded carbon was then acid washed, stripped using a high-pressure, high-temperature desorption circuit, and regenerated. Gold and silver from the pregnant strip solution were recovered by electrowinning and smelted to produce doré bars.

Ruby Hill

Operating parameters and process description for the historical Ruby Hill pulp agglomeration process are summarized directly from Jones (2000). A simplified flowsheet is presented in Figure 3.

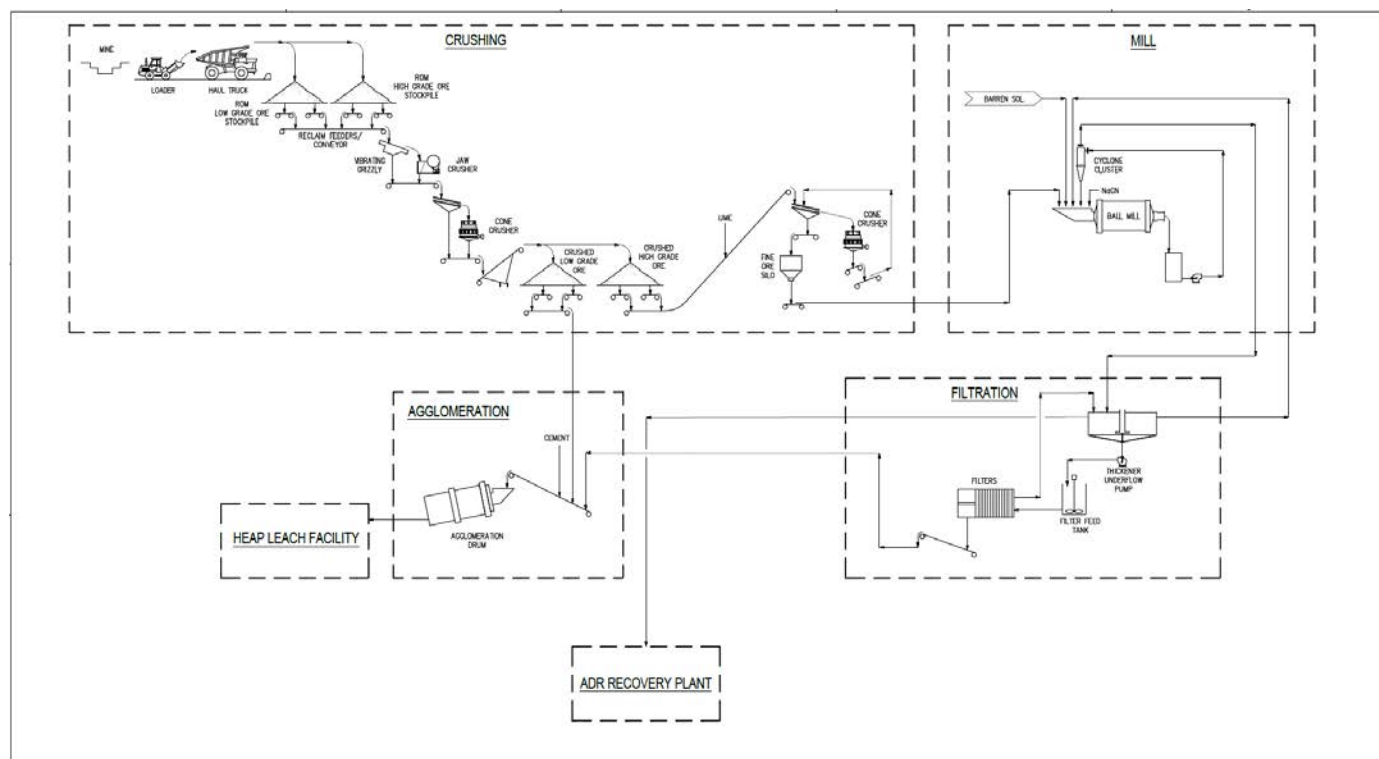


Figure 3: Ruby Hill pulp agglomeration simplified flowsheet (Jones, 2000)

Key design parameters

Key design parameters for the Ruby Hill pulp agglomeration process are summarized in Table 2.

Table 2: Key design parameters for the Ruby Hill pulp agglomeration process

Item	Design parameter
Low-grade production rate	1,600 to 3,150 tons per day
Low-grade crushed product size	P100 1.5 inch
High-grade production rate	900 tons per day
High-grade crushed product size	P85-90 65 mesh
Low-grade : high-grade ratio	1.8 to 3.5 : 1 (depending on low-grade availability)
Cement addition	8 lb per ton of ore

Process description

Low and high-grade ROM ores at Ruby Hill were crushed in a shared two-stage crushing circuit consisting of a primary jaw and a secondary cone crusher. Ore was crushed to minus 1.5" and then stockpiled by grade using a radial conveyor. High-grade ore was reclaimed and further crushed to minus 3/8" using a tertiary cone crusher before grinding with cyanide solution to 85 to 90% passing 65 mesh in a 10.5 × 16' ball mill. The ball mill product was pumped to a cyclone for size classification with the cyclone overflow being thickened to 50% solids. Overflow from the thickener was pumped to the carbon column (CIC) circuit for gold recovery, and thickener underflow was pumped to an agitated, air sparged leach tank before filtering in two vacuum-assisted belt filters. Approximate gold extraction averaged 90%, with 85% of high-grade ore extraction occurring in the mill and thickener.

Filtered cake from the belt filters (approximately 21% moisture) was combined with low-grade crushed ore at an average ratio of 1:2.5 and agglomerated with cement in an agglomeration drum. The pulp agglomerates were then conveyor stacked in 30 feet lifts and leached using barren cyanide solution. Moisture content of agglomerates averaged 12%.

Pregnant solution from the heap was pumped to the CIC circuit from a pregnant solution tank along with overflow from the thickener circuit. Loaded carbon was acid washed with nitric acid before being stripped in a high temperature high pressure desorption circuit. Gold from pregnant strip solution was recovered by electrowinning with the resulting gold sludge being processed in a retort for mercury recovery and smelting to produce doré.

Dolores

Operating parameters and process description for the Dolores pulp agglomeration project are shared with Pan American Silver's permission, based on work completed by Kappes, Cassiday & Associates (KCA) and the Pan American Silver team. KCA was responsible for the detailed Engineering, Procurement and Construction Management of the Dolores Pulp Agglomeration Mill, which was completed in 2016. A simplified flowsheet is presented in Figure 4.

PULP AGGLOMERATION PROCESS AND APPLICATION FOR GOLD AND SILVER HEAP LEACH OPERATIONS

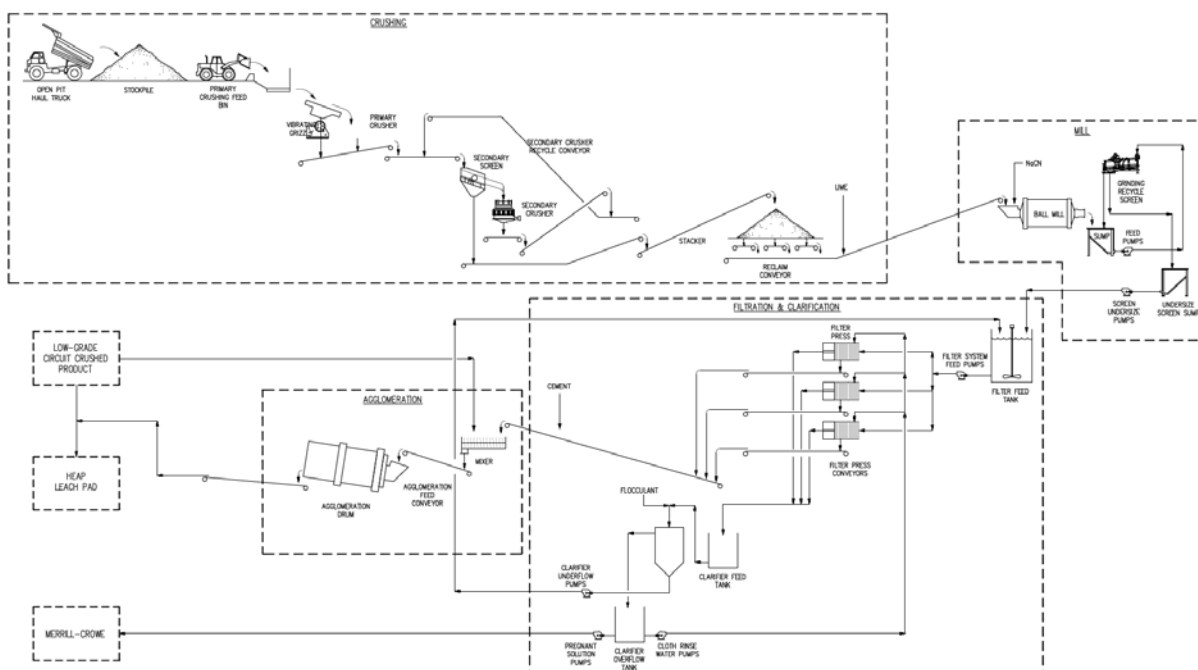


Figure 4: Dolores pulp agglomeration simplified flowsheet

Key design parameters

Table 3

Item	Design parameter
Low-grade production rate	15,860 tonnes per day
Low-grade crushed product size	P ₈₀ 8 mm
High-grade production rate	5,440 tonnes per day
High-grade crushed product size	P ₈₀ 425 micron
Low-grade : high-grade ratio at Drum	1:1
Overall low-grade : high-grade ratio	2.9:1
Cement addition	20 to 30 kg per ton of high-grade

Process description

Pulp agglomeration at the Dolores Mine was designed as an expansion of their existing heap leaching circuit to process high-grade underground and open pit ore. High-grade ore is crushed at an average rate of 5,440 tonnes per day to 100% passing 23 mm in a dedicated two-stage, closed crushing circuit consisting of a primary jaw crusher and secondary cone crusher. Crushed ore is stockpiled before being reclaimed and milled in two stages to a final product of 80% passing 425 microns. The grinding circuit includes an open

circuit primary rod mill and closed circuit secondary Vertimill (tower mill) with a vibrating wet screen for material classification; cyanide is added to the rod mill to start the leaching process.

The final milled product is pumped to an agitated, aerated filter feed tank, which acts as a single-stage leach tank, and is then pumped to three vertical plate filter presses (2.5 m × 2.5 m) operated in parallel. The ground ore is filtered to 86% solids by weight before being conveyed to a mixer where cement is added at a rate of 20 to 30 kg per tonne of high-grade ore along with low-grade ore in a one-to-one ratio.

Initially, the process also included a “delumper” for breaking up the filter cake ahead of the mixer, but this equipment was later determined to not be necessary. The high-grade ore, low-grade ore and cement are mixed and large filter cake lumps are broken up in the mixer before being fed to a 3.6 m × 10 m agglomeration drum. The ore is agglomerated to 11% moisture by weight and combined with crushed low-grade material on an overland conveyor for an overall low-grade to high-grade ratio of 2.9:1, which feeds the existing conveyor stacking system.

Pregnant filtrate from the filter presses is transferred to a pin bed clarifier to remove suspended solids, which are pumped back to the filter feed tank. The clarified pregnant leach solution is pumped to an existing Merrill-Crowe recovery plant and refinery, where it is processed along with pregnant leach solution from the heap leach to produce the final doré product.

A picture of the Dolores pulp agglomeration mill is presented in Figure 5.



Figure 5: Pan American silver pulp agglomeration mill expansion (designed and built by KCA)

Conclusion

Pulp agglomeration, although not applicable to every milling and heap leach operation, is a proven and viable processing method, and should not be discounted when evaluating process options. Pulp agglomeration provides several capital and operational advantages compared to conventional heap leach and milling operations, including faster and improved recovery of gold and silver values from the high-grade ore by grinding compared to heap leaching, reduced capital and land requirements compared to a conventional mill with agitated leach and tailings impoundment, and operational flexibility for processing low- and high-grade ore. Despite the benefits, pulp agglomeration has several distinct risks, primarily the increased risk of permeability issues in the heap. Test work is critical for a successful operation and should include blending optimization for high- and low-grade ore and compacted permeability tests to ensure stability and permeability of the heap.

Overall, the pulp agglomeration process has been well demonstrated, and although the process is site/deposit specific, it is a valid application that may be underutilized in the industry.

Acknowledgements

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A Laboratory and Numerical Approach to Assess Ore Consolidation and Solution Distribution during the Ramp-up Period

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Abstract

Slow ramp up of initial irrigation rates is a commonly used scheme at the start of heap leaching to reduce ore consolidation and promote solution distribution. The phenomenon of ore consolidation during early irrigation is not well understood, and operators typically apply ramp-up irrigation schedules from experience. A combined approach using laboratory tests and numerical simulations has been developed to investigate ore consolidation and to optimize irrigation ramp-up schedules. Laboratory consolidation tests define the ore critical solution content, which is defined as the solution content at which the ore experiences major collapse/consolidation. Ore hydraulic properties are measured and applied in numerical solution flow models simulating various ore irrigation ramp-up schemes. The model-predicted solution content distributions are then evaluated to assess the leach solution content distribution laterally and vertically at different time steps under the irrigation ramp-up schemes. The relative observed differences help to identify ramp-up schemes that do not exceed the laboratory measured critical solution content and minimize the time needed to achieve the full target irrigation rate.

Results indicate ramp-up irrigation schemes can improve the solution distribution and reduce the potential to exceed the critical solution content. Solution redistribution from on/off irrigation schemes is more effective at minimizing ore solution contents at shallow ore depths and creating more uniform solution distribution than using constant low irrigation rates. We hypothesize that ramp-up irrigation minimizes shallow ore consolidation by allowing better solution redistribution over time, which creates a more uniform solution flow field that reduces the potential for reduced soil strength and ore collapse.

Introduction

Heap leaching is a cost-effective process to extract gold, copper, and other metals from low-grade ores. In addition to metallurgical properties and metal recovery rates, the leach ore particle size distribution, consolidation characteristics under loading, and the resultant hydraulic properties, are primary factors to consider for heap leach design and operations. Crushed ores (i.e., 0.5-inch diameter or smaller) typically yield faster and higher recovery rates due to a higher surface area to volume ratio; however, excessive fines generation can reduce permeability. Ore consolidation at depth within a heap can also significantly reduce leach ore permeability, increase solution bypass, and result in reduced recovery; the extent of consolidation and loss of permeability is a function of the ore physical properties, particle sorting, and overburden pressure (Milczarek et al., 2013; Guzman et al., 2013).

Agglomeration can be used to improve the permeability and strength of ores, by adhering the fine particles to the surface of larger particles and generating a low bulk density porous media consisting of large macro-pores between the agglomerated particles. The strength of agglomeration and its effect on the in-situ characteristics of the heap is not well understood, and frequently laboratory column results do not scale well to the field (Bouffard, 2005). Whereas gold and silver leach ores can use strong binders such as cement or lime to maintain agglomerate structure, copper leach agglomerates have a weak agglomerate structure (Lewandowski and Kawatra, 2009), and may experience loss of agglomeration and permeability due to gangue mineral dissolution and resultant ore decrepitation (Ghorbani et al., 2015).

Many crushed and agglomerated heap leach operations have observed shallow ore collapse directly under drip emitters at the heap surface and resulting metal recovery losses. To minimize this effect, these operations typically employ an irrigation ramp-up period to slowly wet the ore prior to initiating full irrigation rates. The phenomena of near-surface leach ore collapse and its effect on heap leach operations has not been well studied and is not observed with current consolidation-permeability testing methods. Column experiments by Briseño (2018) indicate that wetting of agglomerated copper leach ore at full irrigation rates resulted in shallow surface ore collapse, and that slow ramp-up wetting reduced ore collapse and improved solution distribution within the heap leach material. Because ramp-up wetting increases the leach-cycle time, better understanding of shallow leach ore collapse, and optimization of ramp-up wetting, could improve heap leach economics.

Within the soil sciences, collapsible soils have been extensively studied; Li et al. (2016) provided a summary review of loess (silty) soils in arid climates, which have a loose structure that is susceptible to collapse upon wetting. Soil collapse can be attributed to wetting reducing the strength of contact between particles and causing the soil structure to fail with particle rearrangement and loss of larger pore space. In general, significant collapse starts to occur at a critical water content, which varies depending on the initial water content and vertical stress under which the soil is wetted. As the soil water content increases beyond

the critical value, significant volume change is experienced prior to reaching saturation. Li et al. (2016) cite several studies that observed full soil collapse prior to 70% of saturation. Whereby, mechanical elastoplastic breakage models have been developed to predict soil collapse, the parameters required by these models are hard to determine and simple constitutive relationships that can be used to predict soil collapse have yet to be identified (Li et al., 2016).

Whether loess soils provide an analogue to leach ore agglomerates deserves further study. Conceptually, an agglomerated ore that is stacked to achieve low bulk density conditions has a leach ore structure held in place via contact forces between the agglomerated particles. As the leach ore is initially irrigated, the leach ore solution content directly below the dripper increases rapidly. A rapid increase in solution content could decrease the shear resistance and the capillary tension that holds particles in place, in addition to potentially loosening the particles bound to the agglomerate surface, and result in a collapse (increased bulk density, reduced porosity) of the leach ore structure. Briseño (2018) observed that shallow leach ore collapse can propagate through the leach ore profile and result in differential settlement between the wetter and dryer zones, in addition to uneven solution distribution.

Consequently, the objective of an irrigation ramp-up schedule is to apply leach solution in intermittent (short time) intervals, which allows the solution sufficient time to redistribute and slowly wet the leach ore. Slow wetting reduces the potential for agglomerate structure collapse and should maintain more uniform hydraulic conductivity of the leach ore.

Once the leach ore is uniformly and fully wetted, its structure should be more stable to allow leach solution to be applied with full irrigation rates without further ore solution water content increase and ore collapse. Because the potential for collapse is dependent on the material properties of the leach ore and the irrigation rate, experimental methods to determine wetting and consolidation behaviour for agglomerated or non-agglomerated ores are needed to guide heap leach designs and operations to minimize ore collapse at the surface, maximize solution distribution throughout the leach ore profile, and minimize leach ore wet up time.

Methods

Four copper ore samples crushed to 0.5-inch diameter nominal particle size and with similar particle size distributions (Figure 1) were selected from a larger hydrodynamic testing program to evaluate hydraulic properties and potential ore collapse at shallow near-surface conditions. Sample S4 had the lowest (16%) and S3 had the highest (19%) passing #100 mesh.

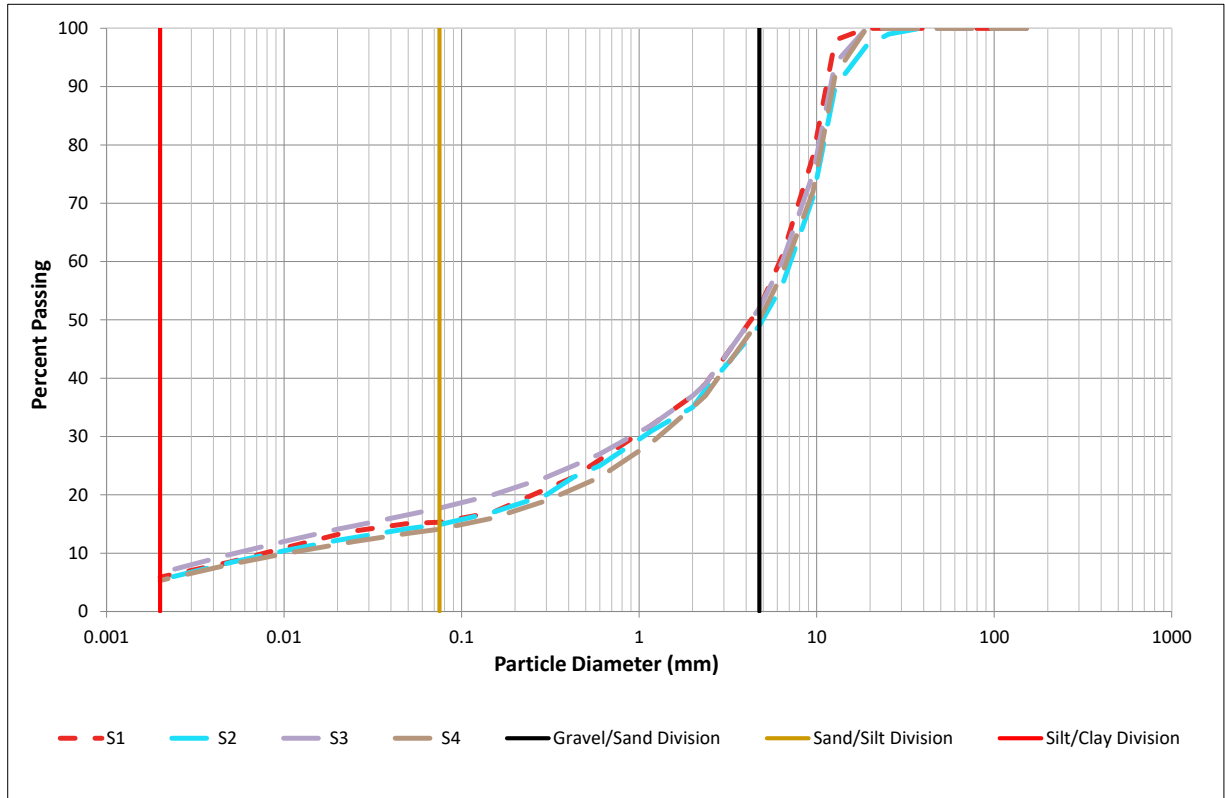


Figure 1: Ore sample particle size distribution

Laboratory test methods

Laboratory tests performed on the four ore samples included:

- Hydraulic properties: saturated hydraulic conductivity (K_{sat}), unsaturated hydraulic conductivity (K_{unsat}), and moisture retention characteristics (MRC).
- Leach ore stability: wet pack ore collapse tests.

Tests were designed to identify the saturated and unsaturated hydraulic properties and the approximate solution content at which collapse/consolidation may occur. K_{sat} and K_{unsat} were performed in dual wall permeameters using the methods described in Milczarek et al. (2013). MRC tests followed ASTM method 6836 – 02 (ASTM, 2008). Leach ore samples were agglomerated with raffinate and 10 kg/ton concentrated sulfuric acid at the optimal gravimetric solution content (GSC) between 5.8 to 7.2%. Ore samples were packed into 15-cm diameter, 5-cm high, rigid-wall Tempe cells. The packing bulk densities were targeted to represent a 1 m heap height from consolidation permeability tests.

Wet pack ore collapse/consolidation tests

Wet pack ore collapse tests consist of preparing agglomerated leach ore samples at different GSC using raffinate solution, packing the leach ore samples into 15-cm diameter by 30-cm tall dual wall permeameters

and then compressing at increasing pressure steps from 0.5 to 6 m over burden pressure to determine the critical volumetric solution content (VSC) at which the maximum consolidation is observed. Leach ore samples were loaded into dual wall permeameters at an initial approximate bulk density of 1.5 g/cm^3 at GSCs of 5.6%, 7.8%, 9.5%, 12.2%, and 14.3%, which correspond to initial volumetric solution contents (VSC) of 8.4%, 11.7%, 14.2%, 18.2%, and 22.1%. The samples were then compressed at increasing pressures approximating 0.5, 1, 1.5, 2, 4, and 6 m overburden pressure. Bulk density changes were monitored with an automated system using pressure transducers to monitor the consolidation of the ore every five minutes.

Numerical modeling

Two-dimensional, axisymmetric variably saturated fluid flow in the leach ore was simulated using the finite element numerical code HYDRUS 2D/3D (Simunek et al., 2018). The focus of the numerical modeling was to identify ramp-up irrigation rates/schemes that would:

1. Minimize ore solution contents between 0 to 3 m depths during the wetting up phase.
2. Maintain VSC values below the critical solution content, defined as the solution content at which the maximum compression or maximum bulk density is achieved.
3. Maximize solution distribution between 0 to 3 m.
4. Obtain the most even wetting profile within the upper 50 cm of the domain.

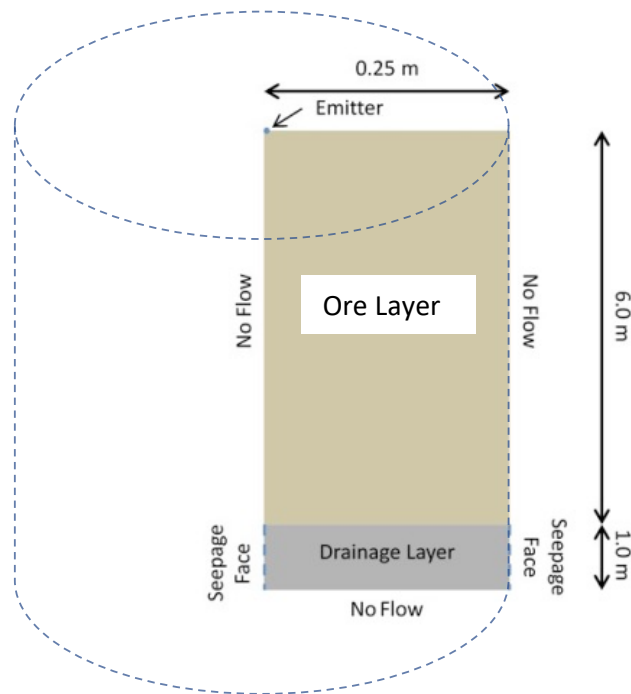


Figure 2: Schematic of model domain (not to scale)

A schematic representation of the model domain is provided in Figure 2. The model consisted of a 7 m high by 0.25 m radius domain with variable finite element grid spacing ranging from 0.01 m to 0.05 m, with finer grid cells near the location of the solution input (solution emitter). Ore hydraulic properties were assigned from the surface to a depth of 6 m below ground surface (bgs). A one-meter-thick drainage layer material type was assigned from 6 m to 7 m bgs (Figure 2).

Measured saturated and unsaturated hydraulic parameters were used to estimate the van Genuchten parameters needed for the numerical modeling via the RETC code (van Genuchten et al., 1991). Solution flow into the model domain was simulated as a variable flux point source representing a single emitter applied to the domain through a 5 cm radius with 1.5 L/hr solution flux. The bottom boundary of the model and the side boundaries of the leach ore were assigned as no flow boundaries. The side boundaries of the drainage layer were assigned a seepage face boundary, allowing solution to exit from the model when the drainage layer becomes saturated (Figure 2). Initial conditions were set to the agglomerated ore solution content at 0.06 GSC or 0.09 VSC for both ore and drainage layer.

To guide the ramp-up schemes design for modeling, the following schemes were considered. A no ramp-up scheme and a slow ramp-up scheme (15 days) were first selected for the test. Scheme 1 was designed to be similar to the slow ramp-up scheme but with an accelerated schedule (15 days versus 8 days ramp-up period).

To evaluate whether we could improve the solution distribution by lowering the irrigation rate, Scheme 2 (7 days) was designed to have an identical daily average irrigation rate as Scheme 1 but with half the dripper irrigation rate and half the on/off period time.

To evaluate the irrigation distribution by reducing the irrigation amount for each irrigation period but faster ramp up, Scheme 3 (4.25 days) was designed to have an accelerated ramp-up schedule with shorter (a) and regular (b) on/off times.

Scheme 4 (12 days) was designed to have a similar scheme at the early ramp up but accelerated toward the end.

The HYDRUS model was then used to simulate six ramp-up irrigation schemes, as shown in Table 1.

Table 1: HYDRUS simulation ramp-up schemes

Scheme	Ratio	Duration (h)	Cumulative time (h)	Cumulative irrigation (cm)	On period (h)	Off periods (h)	Irrigation rate during on periods (L/m ² /h)
No ramp-up	N/A	216	216	130	216	0	6.00
Slow scheme	1/48	24	24	0.30	0.5	23.5	6.00
	1/24	72	96	2.10	0.5	11.5	6.00
	1/8	96	192	9.32	0.5	3.5	6.00
	1/4	48	240	16.5	0.5	1.5	6.00
	1/3	48	288	26.2	0.5	1	6.00
	1/2	72	360	47.8	0.5	0.5	6.00
	Full rate	48	408	76.7	48	0	6.00
Scheme 1	1/32	48	48	1.20	0.5	15.5	6.00
	1/16	48	96	2.71	0.5	7.5	6.00
	1/8	24	120	4.51	0.5	3.5	6.00
	1/4	24	144	8.12	0.5	1.5	6.00
	1/2	24	168	15.3	0.5	0.5	6.00
	Full rate	48	216	44.2	48	0	6.00
Scheme 2	1/32	48	48	0.90	0.25	3.75	3.00
	1/16	48	96	2.71	0.25	1.75	3.00
	1/8	24	120	4.51	0.25	0.75	3.00
	1/4	24	144	8.12	0.25	0.25	3.00
	1/2	24	168	15.3	24	0	3.00
	Full rate	48	216	44.2	48	0	6.00
Scheme 3 a, b (15^a and 30^b min on period)	1/8	54	54	4.06	0.25 ^a 0.5 ^b	1.75 ^a 3.5 ^b	6.00
	1/4	24	78	7.67	0.25 ^a 0.5 ^b	1.75 ^a 3.5 ^b	6.00
	1/2	24	102	14.9	0.25 ^a 0.5 ^b	1.75 ^a 3.5 ^b	6.00
	Full rate	48	216	43.7	48	0	6.00
Scheme 4	1/48	24	24	0.30	0.5	23.5	6.00
	1/24	72	96	2.10	0.5	11.5	6.00
	1/16	48	144	3.91	0.5	7.5	6.00
	1/8	48	192	7.52	0.5	3.5	6.00
	1/4	48	240	14.7	0.5	1.5	6.00
	1/2	48	288	29.2	0.5	0.5	6.00
	Full rate	48	336	58.0	48	0	6.00

Results

Laboratory testing results

Hydraulic property testing

The estimated van Genuchten parameters are provided in Table 2, based on the measured MRC data and saturated and unsaturated hydraulic conductivity measurements at 3 m equivalent heap height. Drainage layer hydraulic parameters were estimated from the GSA database of hydraulic properties.

Table 2: Leach ore and drainage layer van Genuchten hydraulic parameters

Sample	Saturated hydraulic conductivity	Unsaturated hydraulic parameters				
		Saturated solution content	Residual solution content	Alpha	N	L
	(cm/s)	(cm ³ /cm ³)	(cm ³ /cm ³)	(1/cm)	(–)	(–)
S1	2.00E-03	0.250	0.07	0.080	1.500	–2.0
S2	1.50E-02	0.270	0.09	0.700	1.400	–1.0
S3	8.00E-02	0.265	0.075	0.500	1.200	–2.0
S4	6.50E-02	0.233	0.06	0.536	1.224	–1.0
Drainage layer	3.10E-01	0.330	0.08	1.150	1.470	–2.9

Wet-pack leach ore consolidation tests

Wet pack leach ore consolidation tests determine the critical VSC at which the ore structure collapses. This is identified by the solution content at which a sharp increase/decrease in the bulk density is observed. To reduce the potential for ore collapse, the leach ore VSC should be kept lower than the critical VSC during the ramp-up period. Figure 3 show the results of the wet pack consolidation tests at five solution contents and six estimated heap heights (0, 1, 1.5, 2, 4, and 6 m) for each of the samples.

With the exception of Sample S3, all samples showed increasing consolidation with solution content and increasing depth in the heap profile (Figure 3). Of note, the lowest and greatest consolidation were observed at the shallowest and deepest ore depths respectively, but for each specific ore, the critical VSCs were similar, regardless of ore depth. Samples S1 and S3 had critical VSCs of at least 0.22 cm³/cm³, except for S3 the critical VSC above 4 m depth was approximately 0.19 cm³/cm³. Samples S2 and S4 had critical VSCs of 0.20 and 0.19 cm³/cm³, respectively. At solution contents higher than the critical VSC, sample bulk densities decreased with the increasing solution content.

A LABORATORY AND NUMERICAL APPROACH TO ASSESS ORE
CONSOLIDATION AND SOLUTION DISTRIBUTION DURING THE RAMP-UP PERIOD

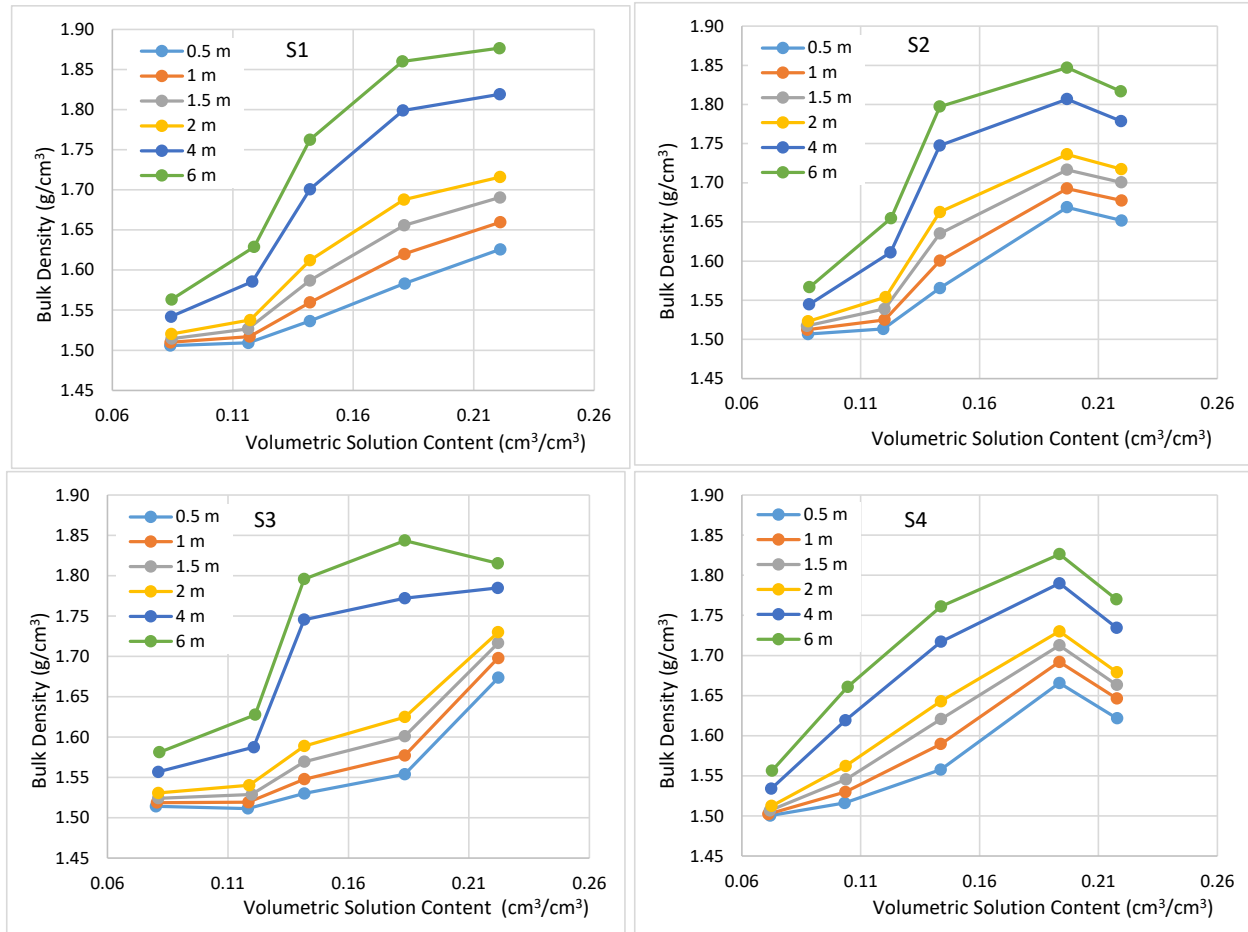


Figure 3: Solution content vs. dry bulk density at estimated depths within the heap

Numerical modeling results

The wet-pack consolidation results identified critical VSC values of 0.22, 0.20, 0.22, and 0.19 cm³/cm³ for samples S1, S2, S3, and S4, respectively. Unsaturated flow modeling results were interpreted by analyzing the predicted VSC distribution within 1 m of the surface to determine whether the irrigation ramp-up scheme exceeded critical VSC values for the leach ore being simulated. The time periods and corresponding cumulative irrigation at which the critical VSC is exceeded were identified for each simulated scheme and compared to determine the optimum irrigation ramp-up scheme(s).

Simulated VSC profiles from 0 to 3 m below the emitter after 18 cm of cumulative irrigation are shown in Figure 4. For the no ramp-up scheme, the critical VSC is exceeded to depths greater than 80 cm for all four ores. Schemes 1 to 3 generally show more similar wetting depth profiles to the no-ramp scheme. The Slow scheme and Scheme 4 were predicted to have lower VSCs throughout the profile than the other schemes and generally with lower VSC than the critical VSC below the depth of 20 to 50 cm; in addition, the depth of solution penetration is greater, indicating that these slower ramp-up schemes provide more

uniform solution distribution throughout the profile. These solution distribution characteristics were maintained as cumulative irrigation increased (data not shown).

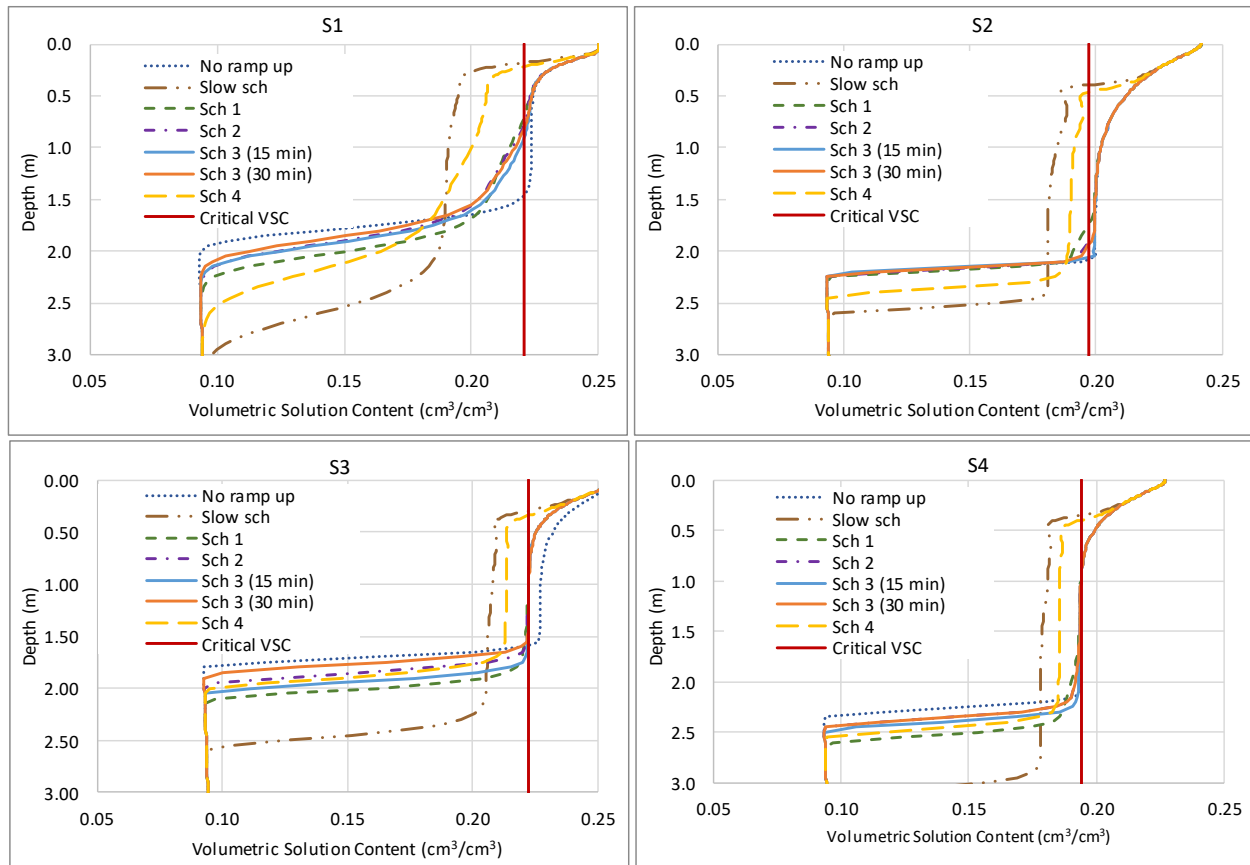


Figure 4: Model predicted solution content below the emitter at cumulative irrigation of 18 cm

The predicted horizontal distribution of VSC at depths of 15 cm and 30 cm bgs after cumulative irrigation of 10.2 cm are shown in Figure 5 for all four ores. The predicted VSCs are higher and less evenly distributed for the no ramp-up scheme compared to the ramp-up schemes. With the exception of S1, Scheme 2 showed lower and more evenly distributed horizontal VSC values at the 15 cm depth than all other model scenarios.

The no ramp-up scheme and Scheme 3b (30 min) showed the highest VSC horizontal values, whereas the Slow scheme and Scheme 4 showed intermediate predicted VSCs below the emitter and lowest horizontal VSC values. At 30 cm bgs, all schemes show more even lateral distribution than at 15 cm depth. The no ramp scheme and Scheme 3b (30 min) had the highest VSC and with the majority of the profile over the critical VSC. Schemes 1, 2, and 3a had the intermediate VSC with a much lower percentage profile over the critical VSC. Scheme 4 and the slow ramp-up scheme had the lowest VSC and most of the profile was lower than critical VSC.

A LABORATORY AND NUMERICAL APPROACH TO ASSESS ORE
CONSOLIDATION AND SOLUTION DISTRIBUTION DURING THE RAMP-UP PERIOD

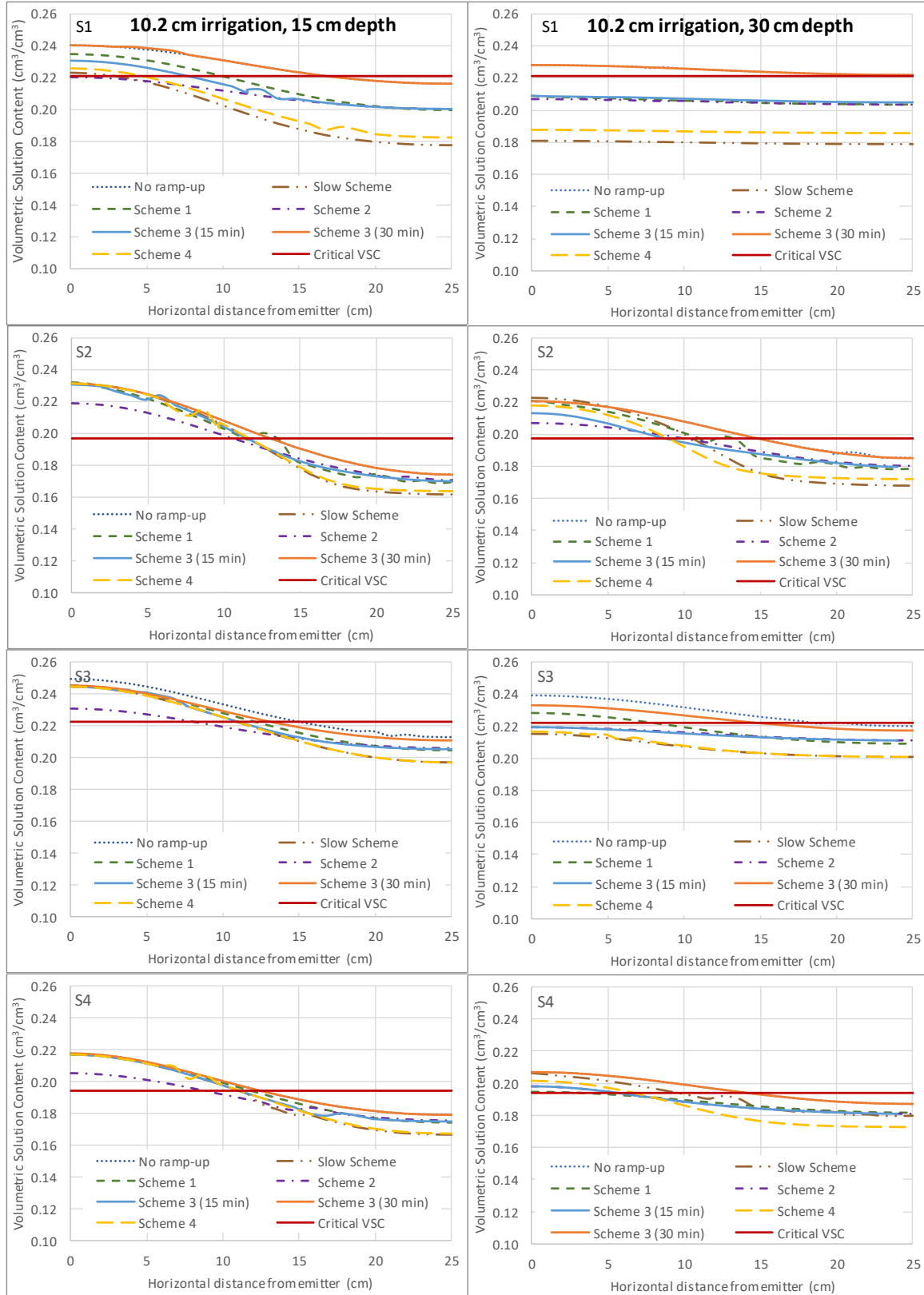


Figure 5: Predicted VSC horizontal profile for four ores at 10.2 cumulative irrigation at 15 cm (left) and 30 cm (right) depth

Overall, shorter on period cycles (Scheme 3b, 15 min) and/or lower irrigation rates (Scheme 2) could result in better redistribution of solution and less potential for ore collapse. Although Scheme 2 and Scheme 3b (15 min) could achieve an approximately 50% reduced ramp-up time than the Slow scheme, 15-minute on periods or lower irrigation rates may not be practical. Scheme 4 showed lower VSC values at the 15 and 30 cm depths and similar horizontal VSC distributions to the Slow scheme at later irrigation steps, indicating that similar leach ore wetting results could be achieved with Scheme 4 and a reduced ramp-up time (288 hours vs. 360 hours).

Conclusions

Hydrodynamic laboratory tests and wet pack collapse tests were performed on four copper leach ore samples, and unsaturated flow modeling was used to determine optimum irrigation ramp-up schedules that will minimize ore collapse at the surface, maximize solution distribution throughout the leach ore profile, and minimize leach ore wet up time.

The wet pack testing results indicate that VSC values exceeding 0.22, 0.20, 0.22, and 0.19 cm³/cm³ for ore samples S1, S2, S3, and S4, respectively, resulted in ore collapse and loss of agglomeration structure. These observed VSC critical thresholds were used to evaluate the unsaturated flow irrigation ramp-up models.

Unsaturated flow modeling results showed the predicted critical VSC directly below the emitter was exceeded from 0 to 1 m bgs for all four simulated ores for all but Scheme 4 and the slow ramp-up scheme. Model predicted VSC values from the slow ramp-up scheme and Scheme 4 were less than the critical VSC at depths of 20 to 50 cm bgs or greater for all ores.

Simulated VSC profiles from 0 to 1 m below the emitter at 10.2 cm of cumulative irrigation indicate that Schemes 1, 2, and 3 have similar wetting depth profiles. The slow ramp-up scheme and Scheme 4 were predicted to have lower VSCs below the emitter than the other schemes for all 4 ores at depths greater than 40 cm bgs. After 10.2 cm of irrigation, the no ramp-up scheme and Scheme 3b showed the highest VSC horizontal values, whereas the slow ramp-up scheme and Scheme 4 showed intermediate predicted VSCs below the emitter and lowest horizontal VSC values.

Simulated results for the slow ramp-up scheme and Scheme 4 were very similar. This indicates that similar leach ore wetting results could be achieved with Scheme 4 at a reduced ramp-up time (288 hours vs. 360 hours). The shorter but more frequent 15-minute on periods for Scheme 3a also resulted in lower VSC values below the emitter and a more uniform horizontal distribution, compared to Scheme 3b (30 minutes). Although Scheme 2 and Scheme 3a (15 min) could achieve an approximately 50% reduced ramp-up time, 15-minute on periods or lower irrigation rates may not be operationally feasible.

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Mining Practice and Capacity Building Based on ESG Principles

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Abstract

Multiple governmental agencies around the world are involved in regulating the mining industry. This is particularly true in the United States because of the vast amounts of government owned/managed lands in the west, and the extent of mining on tribal lands. Mining on tribal lands is almost always subject to approval by both tribal governments and the United States government. In the United States, 30% of the nation's coal reserves west of the Mississippi River and 50% of all uranium reserves are on tribal lands. Tribal governments in the United States are often poorly understood by the mining industry, but they play an important role in mineral development. In almost all cases tribal governments can exercise sovereign authority over mining on their lands and can significantly impact mining on public lands because of laws and policies requiring consultation. Increasingly, tribal communities are demanding that mining be done in a way that respects their view of the environment and culturally significant resources, both on their own lands and on federally managed public lands. Indigenous communities have also gained considerable power internationally, and in some cases regulating agencies will not issue mining permits if the affected indigenous communities object. The long history of mining-induced damages and the enduring societal impacts have, in many cases, created an atmosphere of mistrust and opposition with some environmental organizations and indigenous communities – from northern Canada to the southern Andes. The ongoing – and in some areas growing – objections to mining from the affected communities and environmental organizations speaks to the truth that the mining industry must do better.

For these reasons, capacity building (CB) premised on the notion of environmental, social, and governance (ESG) requirements, including a social license to operate, has become a significant part of the mining industry's practice. A World Bank summary of participatory processes refers to capacity building as the improved ability to make decisions about a project and transfer information between groups. The focus generally is on building a community's or group's capacity to participate in decision-making about a

certain subject, as opposed to identifying capacities in a community and strengthening these elements (Gibson, 2001).

However, capacity building needs to continue to evolve beyond its current framework and implementation. The focus must shift from just encompassing the idea of overall social benefit, to specifically including the building of technical capacity within affected communities, enabling them to fully understand the implications of the mining operation. The enhanced technical capacity must be integrated with mining operations from the conception through operations and to the post-closure and monitoring stage. Capacity building must also be expanded to enable affected communities to share existing information about culturally important resources that could be impacted by mining, and where necessary support efforts by these communities to engage in new research and data collection regarding culturally significant resources.

Introduction

The current issue of *Global Mining Outlook* (KPMG, 2022), an annual report put forward by the international consulting company KPMG with strong ties to the mining industry, ranks “environmental risks, including new regulations” as the number one risk for the mining industry. “Community relations and social license to operate” ranked third on this list. For the first time in the 12-year history of this report environmental issues ranked at the top of the list, jumping from no. 5 in the previous year; community relations and social license also jumped a step up. The mining industry itself is aware of the changing social dynamics. In a recent interview, the Society for Mining, Metallurgy and Exploration president Ronald Parratt said, “On one hand, it would seem like the new green economy could be a threat to the mining industry, but in reality, there is a lot of opportunity” (Mining Engineering, 2022a).

In the context of mining, ESG refers to the industry’s approach to managing air, water, and soil pollution, treatment of and attitude towards employees and the broader community, and management practices like diversity, transparency, and good relationships with regulatory and non-governmental organizations (MarshMcLennan, 2022). In recent years, the mining industry has embraced these concepts as fundamental components of mining (Natural Resources Canada, 2015). In a recent survey of nearly 4,000 business executives and analysts, the international mining company Newmont Corporation, a mining industry leader, scored very high on social responsibility and management quality fronts (Mining Engineering, 2022b).

The term CB (capacity building) was originally conceived in the educational arena and subsequently adopted by the United Nations Development Program (UNDP) in reference to personal or professional development. Since then, UNDP applies capacity building or development principles in practically all of its programs and even has a special team, called the Community Development Group (CDG, 2009). In

reference to the mining industry, CB points to improving communities' understanding of a project's long- and short-term goals and implications and a concerted effort to eradicate the hurdles caused by lack of this understanding. This is necessary in part because mining operations are rarely locally owned or managed businesses, and rapid globalization has further intensified the gap between the understanding and expectations of the local community surrounding a mine and the planners and developers of the operation.

This paper discusses capacity building in the context of ESG and a social license to operate, and how it is currently being applied, and then discusses what more needs to be done, including some recommendations for how capacity building can be expanded.

How is it actually working?

Capacity building has acquired a broad definition with the focus ranging from developing individual skills and ability for accomplishing goals to improving the capacity of an organization or team to “produce, perform or deploy” (Merriam-Webster, 2022). Different institutions have applied it in the context of decision-making, information transfer among various stakeholders of a project, and community support. In the mining industry, capacity building has generally occurred through funding community development and educational opportunities and holding town-hall-style workshops. In the context of education, mining companies routinely fund schools, scholarships, and mentorship programs.

Community development efforts have included infrastructure planning and development, job training, workshops for local entrepreneurs, and even funding micro-enterprises through loans and equity funding. While employment opportunities for the local community surrounding a mining operation is an assumed beneficial impact of most new mines, capacity building efforts have expanded on this concept to include employment outside of the mining sector as well. As part of capacity building efforts, the mining industry is supporting locally generated endeavours like bakeries, butcher shops, cooperative farms, tire repair shops, as well as helping to establish business ventures supporting the mining projects (Government of Canada, 2015). CB efforts have also included meetings with community councils and inviting community input into developmental, health standards, and educational issues.

These examples are important and significant steps forward, and should be supported, but they all focus on capacity building in the context of overall social benefit such as better opportunities for employment, and more small businesses. Extremely limited efforts have been focused on building capacity within affected communities to advocate for mining practices that will better protect their communities. Few of the current capacity building measures will result in the affected communities being better able to bargain effectively with the mining industry on a technical level about the nature of its operations, such as the type of wastes generated, impoundment designs, monitoring programs, or reclamation plans. These are areas that are ripe for continued community participation. For example, in a recent Canadian decision, the

owners of two operating mines were ordered by the Yukon government to more than double the amount of their combined closure securities, though the prior securities had been vetted by both the government and the affected First Nations (Crawford, 2022).

Further, these are the types of issues that frequently result in the most significant concerns and impacts to the community, and where disputes are most likely to occur. There is a limited number of experts willing and able to provide technical support to affected communities, and often the affected communities and environmental organizations lack the funds necessary to hire technical experts. As a result, community members and staff working for the government of the affected communities are often in the position of having to educate themselves, as much as possible, about the impacts of mining operations based on information provided by the mining industry, whose interests frequently do not align with the affected communities’.

In addition, mining operations are rarely local. Usually, regulators are physically located far away from the actual mining operations, in some cases thousands of miles away. Local communities are generally invited to participate in the decision-making by offering comments on planning documents, engaging in consultation, or in some cases serving as cooperating agencies. However, they often lack resources to offer technical feedback, and even when they do have resources and provide technical comments their comments are often ignored. Sometimes they lack the resources to even attend the project meetings. Ultimately, the decisions are principally made by the mining companies and distant regulators. We note also that often even the regulators may lack specific experience with the type of hard rock mining operation at issue. This is especially true in regions without a robust mining industry.

Empowering local regulatory authorities

A recent advance in capacity building is engaging independent consultants or experts to better equip the regulatory authorities and educate the affected communities. One of the authors of this paper has previously been hired by a territorial government in Canada to assist in the development of mining practice guidance documents, and hold workshops on these issues, including participation with the impacted First Nations. This is a prime example of how capacity building can and should be expanded. While in this case it was the regulatory authority that bore the cost of the specific CB efforts, there are valid arguments that the mining industry should bear some or most of the responsibility and costs of such efforts.

Taking this example one step further, this same author has also directly provided *pro bono* technical expertise and support to a tribe (Smith, 2011). In this case, with the backing of a well-recognized expert, the tribe was able to better engage with both the operator and the state regulatory authority. The work provided by the author has continued to be leveraged and relied upon by the tribe and affiliated NGOs to advocate for changes in the mineral processing operation itself. The ability to advocate on a technical level

has tangible long-term benefits for affected communities. It can allow them to hold operations accountable for promises of environmental sustainability and provide more specific and useful input about all stages of the operations, including closure. However, cases like this example are rare.

To fully protect their interests, and to ensure a meaningful social license to operate, impacted communities must be able to offer informed feedback to better align mining operations with community values. Affected communities frequently have regulatory and natural resource professionals, as well as political leaders, who are fully aware of what the community wants in terms of land management, environmental protection, and public health. As an example, the Navajo Nation has its own environmental protection agency, and the Ute Mountain Ute Tribe and the Confederated Tribes of the Colville Reservation have their own environmental departments. All three communities have been impacted by mining operations. These leaders and professionals also generally understand mining operations; however, they may not have technical expertise about the specific types of hard rock mining at issue. However, if they had access to technical experts, they would be able to more effectively offer technical feedback that incorporates their understanding of the community's needs and values. This is not to suggest revamping of regulatory structures; rather, it is to advocate for a collaborative role for affected communities within the existing regulatory structures.

Community-based monitoring

Another aspect of technical capacity building with significant potential is community-based monitoring (CBM). While community-based monitoring is occurring to a limited extent within the mining sector (Pareja et al., 2019), it has not generally been incorporated within the concept of capacity building. CBM is described as a collaborative process where stakeholders from different sectors partner to monitor, track, and respond to environmental and social issues. The overarching purpose of CBM is to share monitoring data with influential stakeholders in order to contribute to decision-making processes. Globally, the CBM approach has been recognized and is being accepted in different arenas in developing countries, as well as in North America, Russia, and Australia (Conrad and Hilchey, 2011; Conrad and Daoust, 2008; Sader, 2016).

Previous research has cited funding as a limitation and also found that in some cases volunteer training is inadequate. These are the very types of limitations that capacity building can directly address. CBM has the potential to significantly benefit the mining industry by increasing trust between affected communities and mining operations, as well as by focusing energy and resources on the most important issues. On a practical level, at least in some cases, CBM could be achieved through partnering with regulatory entities established by indigenous communities. These entities generally lack the funding to hire the additional personnel and purchase the equipment necessary for effective CBM, but have high levels of regulatory and

oversight expertise, and significant experience with impacted resources such as groundwater, soil, and air. Overall, CBM fits well within the overall goals of capacity building and is an area where capacity building can and should be expanded.

Concerns about mining's impact on food resources

Finally, one of the biggest concerns of indigenous communities, particularly in communities with high rates of subsistence hunting, fishing, and gathering, is how mining may impact critical food resources. South American indigenous groups commonly cite the loss of native species supporting their food habits (fishing, hunting, nuts, seed gathering) during mining as a significant concern, and push to have these returned at closure. Impacts to ceremonial areas are another major concern of these communities. The following examples demonstrate the type of positive outcomes that can be achieved by listening to such concerns and collaboratively working with indigenous communities.

One of the co-authors of this paper worked on a project where an affected indigenous community was concerned about the potential impact of the mining operations on caribou migration. Caribou had not been seen in this particular area for decades, but caribou were still important to the community. Specifically, the community was concerned that if the reclamation plan attracted moose, caribou would not use the area. To address this concern, the mining operation worked closely with the First Nation to develop topographic features and utilize native plants that were caribou friendly and either disliked by, or neutral, to competing animals such as moose. This work was prescient, because two years into the project study caribou returned to the area.

Another project, currently under development in Brazil, has adopted as part of its reclamation policy specific actions that will be implemented jointly with the affected communities to better incorporate traditional knowledge about land use and food supplies. This type of collaborative work is only possible when the affected communities have the resources necessary to effectively engage with the mining industry and regulators.

What more needs to be done?

While the increasing acceptance of the concepts of ESG and CB by the mining industry have been important and have resulted in mutually beneficial developments, there is more that can and should be done:

- Capacity building should be expanded to include opportunities and funding for affected communities to develop and share information to help mitigate damage to important cultural resources and foster resource management in a way that is meaningful to the affected communities.

- Specifically, the mining industry can fund additional staffing dedicated to advocating for protection of cultural resources and engaging with the mining industry, and can also fund community-based research focused on cultural resources and on mitigating harm from mining operations.
- In countries like the USA where consultation and public participation are conducted by the federal or state governments, the industry should consider voluntarily seeking direct input from affected communities about cultural resources and integrating this input into its operational plans.
- The mining industry must be willing to invest more in the capacity of communities, NGOs, and regulatory agencies. In order to facilitate this educational process, the industry should help fund independent technical experts who are accessible to surrounding communities. This technical capacity is essential both to obtain a meaningful social license to operate and also to ensure meaningful community engagement on an ongoing basis.
- Capacity building should be expanded to include community-based monitoring. In some situations affected communities, particularly indigenous communities, have sophisticated and well-developed regulatory agencies and well-trained staff within their own governments. In these cases, the existing community expertise can be utilized to develop monitoring and compliance data.

Conclusion

Capacity building has been a valuable tool for the mining industry and affected communities. It has helped the companies achieve ESG goals, and it has improved engagement by the communities, NGOs and, to a lesser degree, the regulatory agencies. Expanding on CB's success will increase trust between the various stakeholders which, in turn, will help build better mines and better communities. But it must be expanded upon to encompass technical capacity, allowing affected communities to not only benefit from the end results of the mining operations but shape the very nature of the operations themselves to align with community values and needs. Moreover, it must more actively engage affected communities in oversight such as monitoring. To this effect, industry must commit to expanding upon CB to build technical capacity; to share resources and information, particularly about cultural resources; and to gain community trust.

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Accounting for Complexity in Draindown Modeling for Heap Leach Closure

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Abstract

Active management of residual leach solutions, which no longer have commercial value, represents a significant financial liability for the closure of heap leach operations, and the duration of active management should be carefully planned both technically and financially. The state-of-practice in developing leachate management plans for closed facilities includes deterministic models such as the Heap Leach Draindown Estimator (HLDE) developed in the state of Nevada, USA. Such models rely on an approximation of the unit draindown rate using theoretical formulations of unsaturated flow, which is then extrapolated to a regular orthogonal geometry and a single defined date for cessation of irrigation. When adequate calibration data is available this method can yield satisfactory results; however, such calibration data is typically not available.

An improved method is proposed in this paper that takes into account the important variables that describe a heap leach pile such as geometry, material domains, and time. The method consists of a combination of numerical modeling of unsaturated flow and analytical modeling to predict the time required to achieve passive water management. To account for the non-linearity of draindown flow a unit draindown rate is determined for each domain through a rigorous finite element numerical modeling based on the physical and hydraulic properties of the leach material. This step can also account for different material types that may exist in the leach pile and create families of individual curves for each material. The geometry of the heap leach pile is taken into account by discretizing the pile into geometric domains, grouping areas of similar height.

The resulting draindown curves are then multiplied by the representative surface area to compute the expected volume of draindown from each of these domains. In the third step of the assessment, the individual draindown volumes are assembled in a time series honouring the actual schedule of when irrigation has been completed in the various regions of the facility leading up to the final closure date.

Various complexities can be added to the model to enable a complete water balance of the heap as well as integration with the site-wide closure water management plan. Some examples are recirculation to

some areas and not others, deployment of enhanced evaporation methods, treatment and discharge of draindown solutions, or reduction in infiltration by covering the heap or portions thereof.

An example of the method applied to a mine in Argentina is described in this paper, detailing the assumptions that were made with respect to the closure water balance and the functional parameters used for modeling the final draindown of the heap leach facility.

Introduction

Heap leaching is one of the lowest cost processing methods and is increasingly used in projects in all climates with large low-grade reserves. Heap leaching facilities vary widely in size and shape, and large solution inventories, often in the order of millions of cubic meters, must be managed in order to sustain economically viable operations.

Leaching of the ore is typically achieved in pre-determined leach cycles, where the leach solution is recirculated onto the heap continuously for a number of weeks or months. Once the metal concentration in the leach solution falls below an economic cut-off value, irrigation is moved onto a new leaching area and the process is repeated. The portion of the heap that is no longer to be irrigated transitions into the passive draindown phase.

Draindown is defined here as the period of time during which the flow rate is gradually reduced from the operational level to the residual steady-state value. Draindown is highly non-linear, with an initial quick reduction in flow rates, followed by a gradual slowing down. The example in Figure 1 illustrates the flux decay for unit surface columns (one square meter cross-section area).

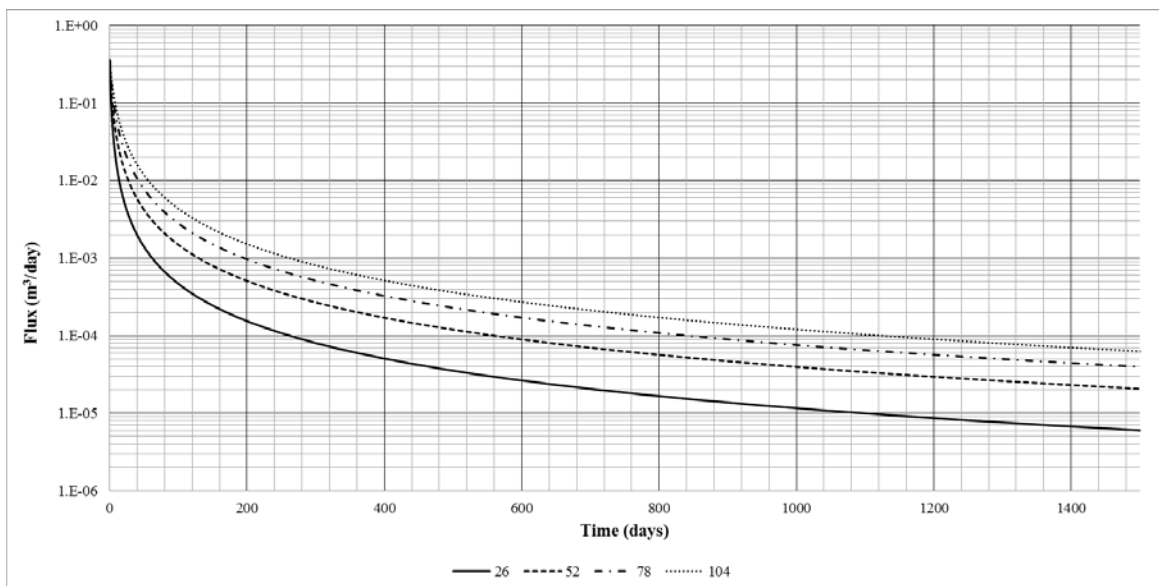


Figure 1: Example of unit draindown curves for various column heights (26 m, 52 m, 78 m, and 104 m)

The draindown of a heap as a whole can be described as a complex system of actively leached portions and portions of the heap in various degrees of draindown, more often than not including consecutive lifts irrigated several times before closure. It is therefore difficult to predict the fluxes of leach solution that will need to be managed in closure and post-closure periods, as well as the duration of such sustained fluxes.

Management of the leach solution inventory of a closed heap could be divided into distinct active and passive stages. During the active stage recirculating the solution to the heap may be necessary to prevent the existing ponds from overflowing while inventory reduction methods such as treatment-and-discharge or enhanced evaporation are implemented.

In the passive stage, recirculation is no longer necessary to prevent the ponds from overflowing. In some situations passive water management can only be achieved once the outflows from the leach pile, consisting of draindown and net percolation from precipitation, become less than the achievable evaporation flux from the surface of the water management ponds, i.e., the ponds will not overflow. Where excess storage capacity is available, temporary storage and subsequent evaporation may shorten the active management period.

Challenges of predicting draindown

When closure liabilities must be estimated significant costs (beyond already existing equipment and facilities) are associated with the duration of active water management (site presence, i.e. increased indirect costs) and the complexity of the water management system (necessity for water treatment, seasonal discharge, etc.). The draindown model is a tool that can provide the appropriate inputs in estimating these closure costs.

Accurately predicting final draindown of an entire heap is a process that must overcome challenges related to hydraulic properties of the heap material, geometry, heterogeneity of the heap leach ore, operational history of the heap, and the typical lack of adequate calibration data. The following subsections provide details on these parameters.

Throughout the entire life cycle of the heap leach pile, the flow regime within the heap is ideally unsaturated. Even during the active leaching period the irrigation rates are generally kept below values that would cause saturation of the heap material.

There are various practical reasons for this; for example, the need for air circulation for the lixiviation process to occur at reasonable efficiency, or slope stability limitations. It is therefore natural to approach the draindown as an unsaturated flow problem, where the moisture content of the pile changes continuously in time while it also varies spatially in the different regions of the heap.

The flow rates required to efficiently leach the metal from the ore are determined in the laboratory based on column experiments where flow rates and extraction coefficients are observed. These column

studies are the basis for successful operations, but it was noted in various studies that column studies tend to drain faster than the full-scale heaps (Kappes, 2002; Muller and Newton, 2008).

To quantify the relationship between saturation levels and drainage quantities, the soil water characteristic curve (SWCC) is used. The SWCC, also referred to as the water retention curve, is a graphic expression of the relationship between the moisture content and the matric suction in a soil, and provides an indication of the manner in which a soil gains and releases the stored water.

The SWCC is normally determined experimentally, but theoretical methods were also developed to estimate it when experimental results are not available (van Genuchten, 1980; Brooks and Corey, 1964; Fredlund and Xing, 1994).

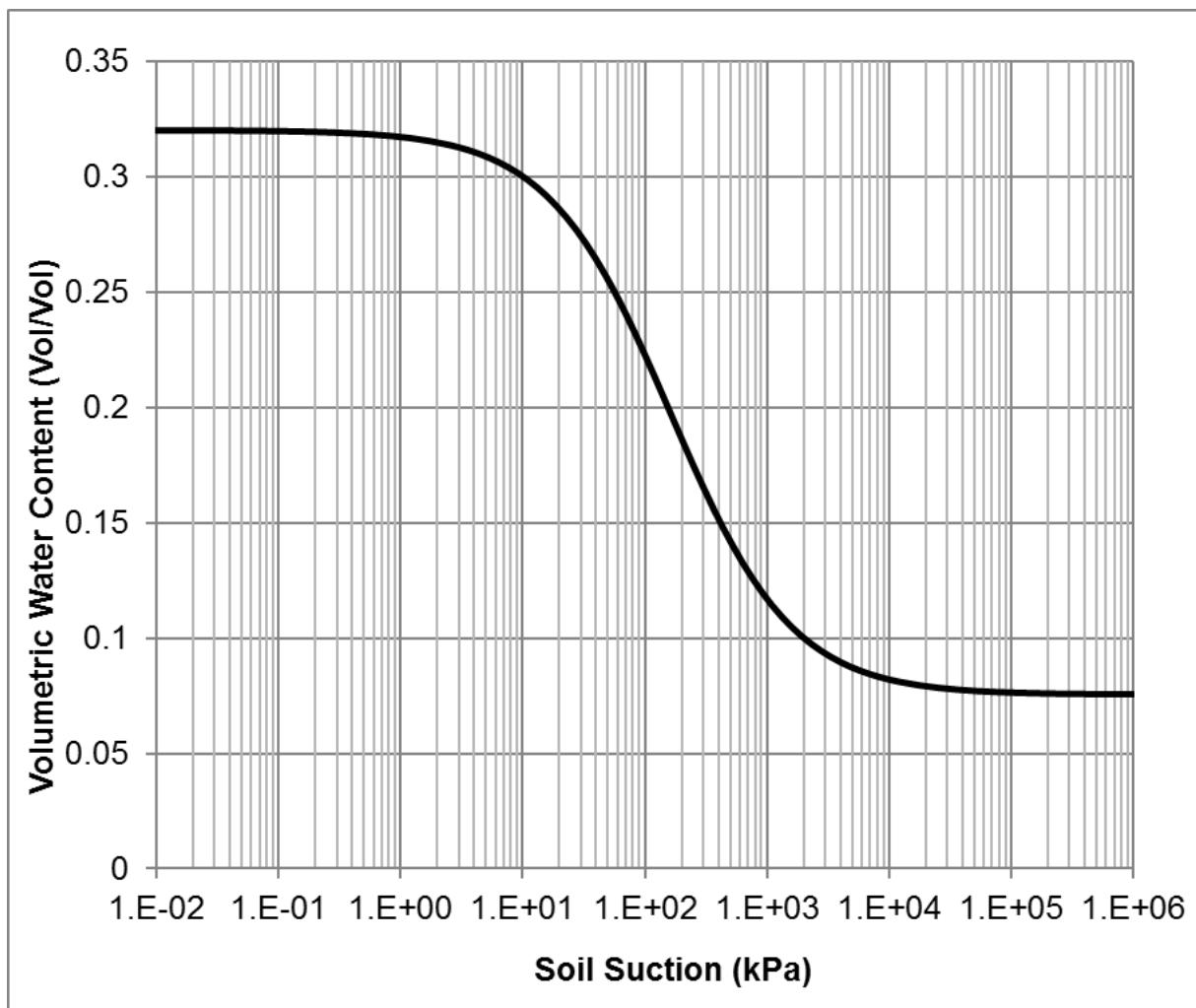


Figure 2: Example SWCC for heap leach materials

As suction increases, smaller and smaller pores become empty. Conversely, larger and larger suction is required to remove less and less water until a residual water content is reached where essentially no more

water can be removed through normal natural processes. When irrigation of the heap leach pile occurs, the infiltrating water first fills the smaller pores (having a higher suction applied to them) and then gradually larger pores fill up as suction is reduced. Then, once irrigation is stopped and draindown begins, the largest pores start emptying water first, followed by successively smaller pores. The level of suction at which the largest pores begin to drain of water is called the air entry value (AEV).

Ideally a set of distinct SWCC curves would be determined to characterize the variations in the ore type (where required) and the various physical changes the ore goes through over the life of the heap (self-weight consolidation, decrepitation, pore clogging by mineral precipitation, etc.) (Greaser et al., 2019).

The pre-mining topography influences the shape and configuration of the heap. Some operations are located in areas where flat land is available for development, and therefore the shapes are regular. Other operations are located in difficult topography, where the constraints of the terrain will produce complicated shapes as the heap will be developed using the available footprint, as shown in Figure 3. This will yield highly variable overall 3-D shapes with sharply changing height, footprint, and slope angles (on the base liner).

Overlaying the 3-D shape, the internal geometry of the pile is an additional complicating factor, with the heaps typically built in successive lifts and commonly in successive phases adjacent to each other. The larger the heap, the more opportunities for the shape to become more complicated, as frequently the heap size is expanded beyond the original design to accommodate increased ore quantities.

Applying a unified draindown model to the entire heap would require a complex 3-D model domain of large dimensions and relatively small mesh size due to the non-linearity of the equations to be solved, making the model computationally excessively expensive.

Simplifications must therefore be made, in which the heap geometry is discretized as described in the following section, while unit draindown curves are applied to each sub-domain. The unit draindown curves are determined a priori based on the set of SWCC to be representative of the subdomains.



**Figure 3: Example of heap leach pile in plan view.
Each colour represents a different operational phase**

Another challenge in predicting the heap draindown is heterogeneity in the physical and hydraulic properties of the heap ore material. While the heap in general is thought of as a unified extraction unit, the geology of the orebody may yield leach ores with distinctly different properties. The primary source of heterogeneity in properties is often the mine plan, i.e., ore with distinct properties being stacked on the heap in various geometric configurations. Also, stacking plans with distinct construction phases being built over time may further increase heterogeneity.

Density of the heap material will normally increase with depth due to self-weight compaction, influencing hydraulic properties in a relatively consistent continuum. Layering during deposition will result in density gradients repeating at each layer, with a higher density trafficked layer near the surface, and with gradually increasing overall density increasing with depth.

Weathering characteristics of the various ore portions may also be different, resulting in changes in the properties after the material is placed on the heap. As these changes occur after placement, relatively similar initial properties may change over time, and become quite different by the time the heap is spent and final draindown can begin.

The history of heap operations will have a direct influence on the saturation state of the various regions at the end of operations. Application rates of the leach solution will also influence the saturation state of the heap at the end of operations. During normal operations, various areas of the heap are irrigated in a pre-determined sequence, resulting overall in zones that were irrigated for a long time in the past (sometimes years) compared to more recently irrigated sections.

Another challenge in heap leach draindown modeling is scarcity of suitable calibration data. Typically, reliable calibration datasets are limited or more often not available (the typical metallurgical estimates do not honor unsaturated flow principles – mass balance approach). Heap drainage values typically available are normally limited to a small subset of the site (but this is better than nothing).

The followed approach

The reality is that there really is no “best practice” for predicting heap leach draindown times, and such planning often relies solely on the experience of long-time operators and allowing for a wide margin of error.

A multitude of models are used to predict draindown time, ranging from simplified empirical methods to computational fluid dynamics integrating leach solution flow and metallurgical recovery. All these models are suitable to provide information for specific issues related to heap leach operations and closure.

A brief list would include the following methods:

- Simplified empirical methods, i.e. Heap Leach Draindown Estimator (HLDE) based on the implementation of the Brooks and Corey equation.
- Numerical models of saturated flow, i.e., conventional groundwater flow models.
- Numerical models of unsaturated flow, some of the most common codes used being HYDRUS, Seep/W and SVFlux.
- Computational fluid dynamics (CFD) modeling, typically used for recovery modeling, with a flow component; and
- Complex modeling environments including geochemical, metallurgical, and flow components such as Metsim and Goldsim.

Arguably multidimensional (geometry and heterogeneity) unsaturated flow numerical modeling is the superior approach. However, complicated models requiring days or weeks to run often far exceed the need for information. It is therefore important to keep in mind the “fit for purpose” approach, where simplified and cruder models could provide the information needed to make the required decisions.

Overall concept

Considering all the challenges presented in the sections above, SRK has developed an analytical model to predict the final draindown time of a heap leach pile. To overcome these challenges, the draindown analysis is conceptualized by breaking the heap into multiple smaller building blocks. Draindown analysis is conducted on each of these columns independently as if they have unit dimensions, and then the separate draindown curves are combined to yield a single composite curve, as shown schematically in Figure 4.

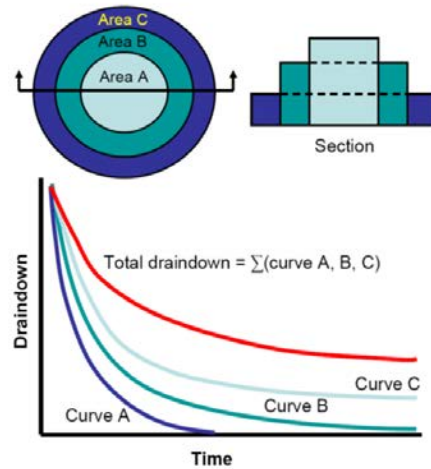


Figure 4: Conceptualized geometry used in draindown model

Each of these blocks represents a unique portion of the heap in the form of a column that extends from the base to the top of the heap, as illustrated in Figures 5 and 6. In this approach the dimensions (surface area and height), material properties, and draindown state (time passed since last production irrigation) of each column can be uniquely varied and the sequence in which they are summed up can be set based on an understanding of the construction and operating (i.e. irrigation) sequence. In essence, this approach presents a simplified three-dimensional representation of the problem.

Conceptually, the model manages leach solution inventory as if the heap consists of two sections: (1) passively draining phases; and (2) an active recirculation phase. This is a crucial feature of the model, as it allows recirculation to be included in the calculations while keeping track of real-time inventory on a daily time step. If recirculation is selected, the drained inventory from the passive phase is recirculated as part of the active phase, which means the passive phase reaches a drained state first.

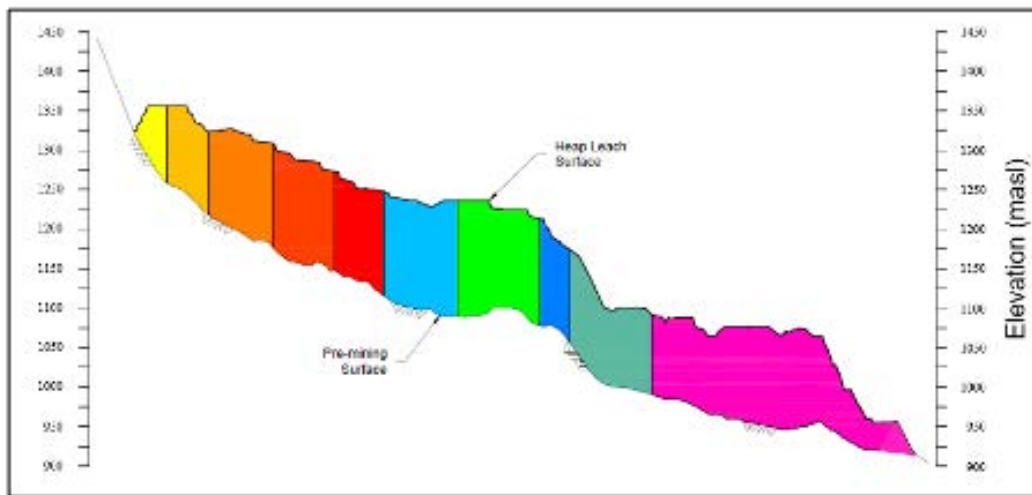


Figure 5: Elevation profile through an example valley-fill type heap leach facility

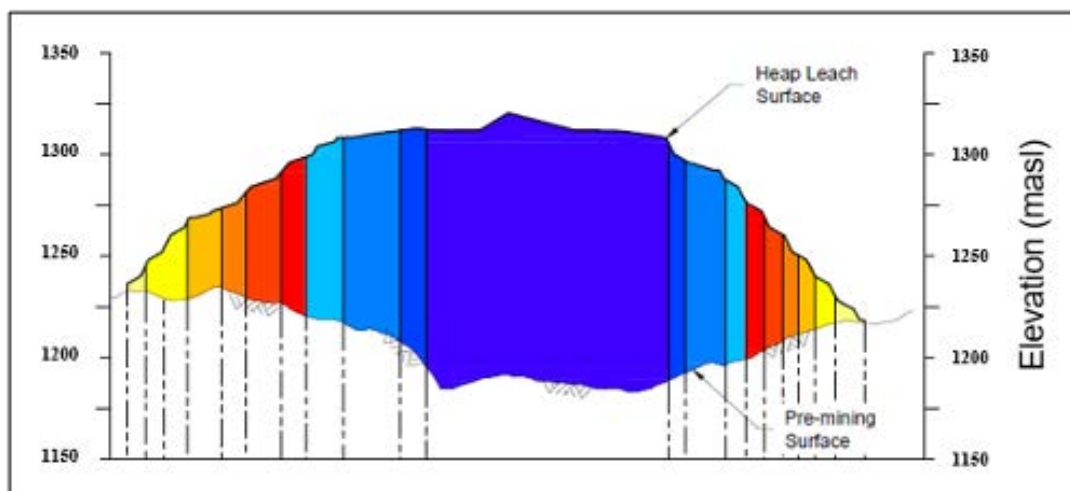


Figure 6: Cross-section of an example heap leach facility showing column discretization by height

The drainable inventory is calculated as the difference between the volume of water at the initial time in the model and the residual water expected to never drain. This takes into account the time passed since the last day of irrigation for each cell and computes the water content based on the draindown curves previously modeled. The inventory of leach solution is calculated on the initial model date based on the moisture content within each phase, which is a function of the time passed since the last irrigation of each particular phase was completed.

The chosen computation platform is Microsoft Excel, building on the wide distribution of, and familiarity with, the software. The choice was guided by the desire to create a tool that could be widely used by the site personnel of SRK's client corporations. The model includes four interconnected functional modules: draindown, inventory, recirculation, and evaporation. The computed water balance includes a series of intermediate terms dealing with assumed pond storage capacity.

Typical model inputs

Draindown curves

At its core, the model is based on the fact that assuming soil properties are homogeneous, the height of the column will cause large differences in draindown flux from different areas of the pile. The draindown curves were therefore created for the unit surface area, which is then multiplied by the footprint of each column, phase by phase. This was done to avoid the need to create a large matrix of curves for each height for each surface area, and to facilitate updating the model in the future when more refined heap leach configurations become available.

The draindown unit curves can be created using various methods. Unsaturated flow modeling using Hydrus or Seep/W would yield the most accurate unit draindown curves, although spreadsheet models solving the Brooks and Corey equation could result in satisfactory curves.

Climate parameters

Precipitation and potential evaporation values can be included in the model, to allow quantification of the soil-atmosphere interaction and the long-term moisture regime in the heap. Site precipitation data is highly desirable, but synthetic precipitation data obtained through reanalysis methods are also routinely used by SRK. Potential evaporation (PE) values are typically computed based on a range of models, with the most common being the FAO model based on the Penman-Monteith equations (FAO, 1998) and Priestly-Taylor model (Priestly and Taylor, 1972). A simple snow sublimation routine can also be included if appropriate.

Operational history

Operational history of a heap is captured in the model by the final irrigation schedule input, meaning the last complete cycle of irrigation for each of the zones in the heap. The idea is that the final irrigation schedule allows quantification of the time elapsed since the last active irrigation on any particular location on the heap. This will in turn allow the model to find the corresponding moisture content in that location and thus compute the inventory for the initial and each subsequent time step. The fundamental assumption is that the model time starts once the final irrigation is complete and continues until the outflow from the heap diminishes to a pre-determined value, allowing long-term passive water management.

Ore volume isopach

Based on drawings (as built or design schematics) and ore deposition schedules, an isopach is produced for the heap in an effort to obtain the characteristic footprint of the number of lifts placed within each phase. The isopach is developed into a matrix of reference values for each height and for each zone within the heap, which is then multiplied by the unit draindown flux to compute the overall flux for any particular time step.

Model results example

The heart of the model is the water balance sheet. On this sheet the inflow and outflow terms are brought together and the water balance for each time step is computed. The water management methods, such as treat and discharge or enhanced evaporation, are accounted for in the water balance.

Pinpointing the date of transition from active to passive water management is a trial-and-error process. Although a goal seek function is available to direct the search, due to the many assumptions one must consider in such a complex system a multi-variable chart was found to be most useful. Although some

parameters cannot be plotted against a native scale, i.e., volumes and fluxes cannot be plotted on the same scale, the changes of the parameter values against time and against each other are the meaningful outcome.

An example of a modelled draindown scenario is shown in Figure 7. This is a fictitious scenario in which active water management is required for the first seven years, with the first four years of continuous recirculation followed by three years of seasonal recirculation. The available pond capacity increases as new ponds become available, although the pond volume (red curve) is shown to be well below the existing capacity.

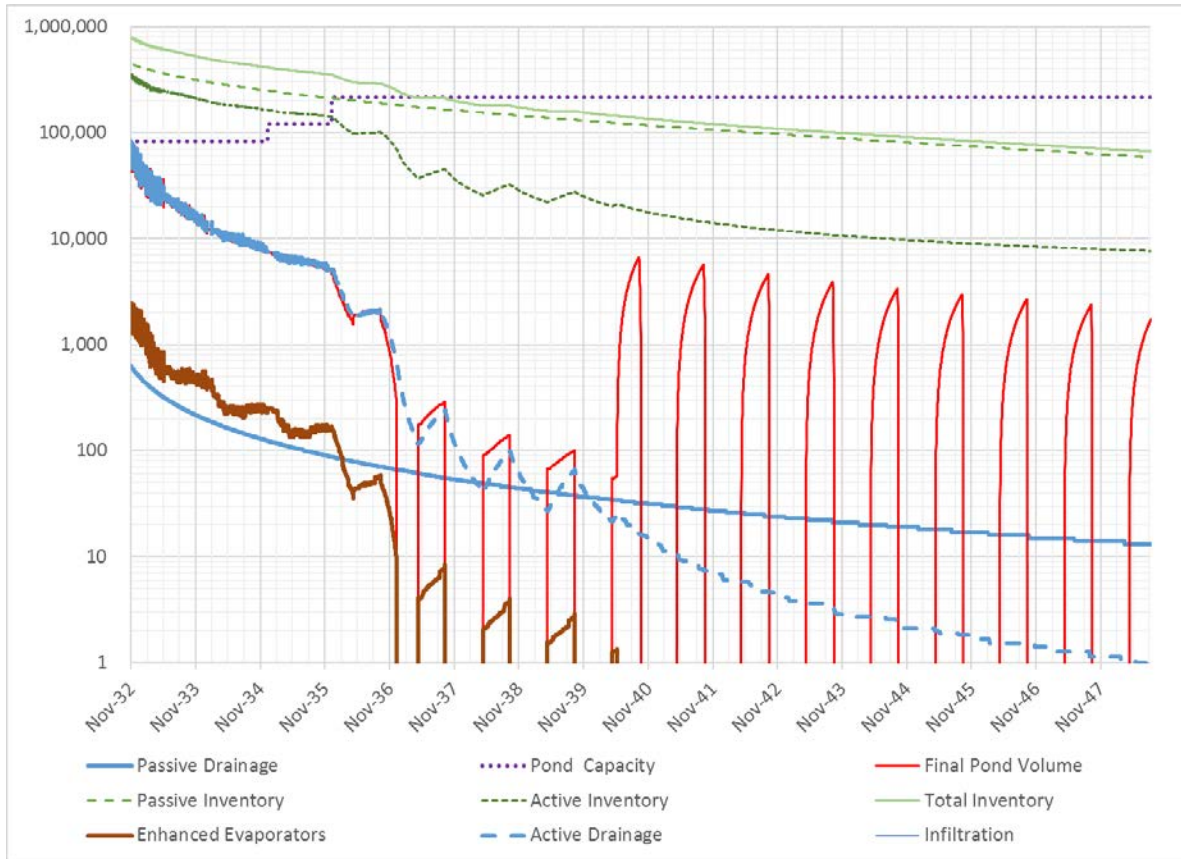


Figure 7: Example of draindown model results

Discussion and conclusion

The model can be implemented for any climate, from cold arid to wet moist, with the appropriate surface water management assumptions being included in the inputs. This flexibility is allowed by the fact that run-off and net percolation are dealt with as separate independent terms in the water balance. This fact is also one of the limitations of the model, where the direct effect of precipitation on the moisture content of the heap cannot be assessed.

During implementation of the model for a mine in South America it was noted that discretization of the entire heap into phases as defined by the mine plan, as opposed to distinct irrigation grid rotations, can result in overestimation of drainable inventory. Optimizing discretization within the limitations of the Excel file size could improve the accuracy of the predicted draindown flux and in turn the necessity of excess pond capacity.

When considering a facility covered with a geomembrane, the draindown problem becomes simplified since net percolation is nil. The passive draindown concept still applies, with all other challenges discussed above still present. However, as shown in the previous sections, the model is suitable to predict draindown for such facilities.

The analytical model presented here can also be used to predict draindown from other mine waste facilities such as tailings deposits and waste rock dumps. Material heterogeneity is present in these other waste materials, although differences in particle size distribution and SWCC of tailings may range on a narrower scale. Geometry, the same as for heap leach facilities, is dictated by the original topography. The notable difference is the relatively uniform moisture content the tailings typically present at the end of active deposition. The time scale for tailings draindown is also expected to exceed the time scale of heap leach draindown by a wide margin, possibly orders of magnitude, while waste rock draindown is likely to be similar.

The appeal of this modeling approach resides primarily in its simplicity, as it leverages user familiarity with the Excel platform and computing a series of building blocks. No computational fudge factors such as constants or internal adjustment factors are used. The internal structure of the model is fully transparent to allow easy review and query of assumptions, although the large number of computations and referencing loops requires a thorough understanding of the working principles.

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Chapter 4:

Operations and Design



Gold Leaching Problems and Solutions at the Altan Tsagaan Ovoo Mine

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Abstract

The Altan Tsagaan Ovoo gold mine is mining a gold-polymetallic deposit that is comprised of oxide, sulfide, and transitional ore that originated from epithermal breccia pipes. The mine is located in the continental rift zone of the Mesozoic era in eastern Mongolia.

Mining and stacking of the heap leach pad started in 2019, and leaching operations were initiated in 2020. During warm weather the irrigation system consists of a sprinkler, and during cold weather a dripper system is used. The gold and silver is recovered using a carbon-in-column plant. The process is continuously enhanced through research and laboratory experiments.

Due to restrictions related to the COVID-19 pandemic, the processing plant was faced with a long-term shortage of the main key chemicals, especially cyanide, during 2020. Notwithstanding the shortages of key chemicals during that time, the solution continued to be circulated through the heap and plant. It took approximately six months for the cyanide content to be fully used up, allowing for the recovery of some silver and gold each month.

Due to the lower cyanide content, copper adsorption dramatically increased, leading to a significant deterioration in gold adsorption. The carbon and the cycle solution became highly contaminated, to the point where the carbon contained 10% copper, and the solution contained approximately 270 ppm of copper. In order to stop copper adsorption from obstructing gold adsorption, research was conducted into the following method: using carbon as a filtering agent and removing metal copper by desorption after adsorption of gold, silver, and copper in activated carbon. The gold sludge containing silver and copper mix was dissolved in nitric acid two to three times in order to recover copper; a caustic soda was used; and sugar to recover silver sugar was used in the nitric acid. Once digestion was complete, a fine gold powder remained.

Following extensive research, it was found that when cyanide is insufficient, copper mono-cyanide and di-cyanide are formed, which is actively adsorbed in the carbon. This active copper compound was

observed to create stronger bonds with activated carbon than gold, which has the potential to even displace gold adsorbed in the carbon.

Based on this research, an experiment was conducted that changed the cyanide content of the solution and measured the copper adsorption amount in solutions after different time durations. This resulted in increased cyanide content and decreased copper adsorption, and required up to three days to complete the reaction of cyanide and copper.

This paper will focus on the process of stopping copper adsorption and copper removal from cycle solutions, and gold refining from high copper-content ores.

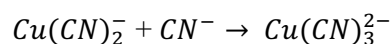
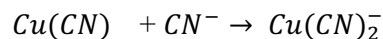
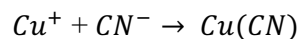
Introduction

This table shows an analysis of the oxide gold ore.

Table 1: An analysis of the oxide gold ore

Hole ID	Sample ID	Au, ppm	Ag, ppm	Cu, ppm	Pb, ppm	Zn, ppm
ATO-01	Z10002	2.32	7.70	215.00	2,320.00	261.00
ATO-01	Z10003	1.04	7.70	342.00	2,110.00	280.00
ATO-01	Z10004	1.18	5.60	168.00	6,080.00	196.00
ATO-01	Z10005	1.14	7.40	232.00	3,890.00	236.00
ATO-01	Z10006	1.19	7.00	407.00	5,410.00	495.00
ATO-01	Z10007	1.95	7.80	637.00	4,450.00	549.00
ATO-01	Z10008	1.84	8.80	702.00	5,100.00	844.00
ATO-01	Z10009	2.61	11.50	1260.00	11,000.00	1,045.00
ATO-01	Z10010	1.81	10.00	1220.00	2,820.00	1,096.00
ATO-01	Z10011	0.95	4.50	423.00	2,660.00	400.00
ATO-01	Z10012	1.26	5.20	227.00	2,600.00	183.00
ATO-01	Z10013	1.40	7.90	179.00	1,870.00	178.00
ATO-01	Z10014	1.65	10.90	462.00	2,570.00	280.00
ATO-01	Z10015	1.25	8.20	382.00	2,260.00	227.00
ATO-01	Z10016	1.51	7.80	495.00	2,780.00	295.00
ATO-01	Z10017	1.07	5.20	480.00	1,100.00	283.00
ATO-01	Z10018	1.77	10.10	779.00	5,670.00	522.00
ATO-01	Z10019	1.58	9.20	490.00	5,870.00	394.00

As can be seen, even though the amount of copper in the low-grade gold ore is high, it is economically inadequate. Moreover, during the process of leaching the ore with cyanide, copper interacts with cyanide, and mono-, di-, and tri-copper cyanide compounds are formed.



Pregnant solutions for cyanide leaching are often contaminated by impure metals such as copper (Marsden and House, 2006). Cyanide also forms complexes with these impure metals solubilized from host/gangue minerals, increasing the consumption of cyanide in leaching.

For example, under typical cyanide leaching conditions, copper dominantly presents in cuprous form (Cu^+), which readily forms cyano-complexes of $\text{Cu}(\text{CN})_2^-$, $\text{Cu}(\text{CN})_3^{2-}$ and $\text{Cu}(\text{CN})_4^{3-}$ depending on the concentration of cyanide and copper, pH, and temperature (Xie et al., 2013). Metal-cyanide complexes are classified, based on their stability constants, as strong ($\log K > 30$) and weak ($\log K \leq 30$) complexes (Table 2) (Van der Merwe and Breuer, 2013).

Table 2: Weak and strong metal-cyanide complexes

Reaction equation	Stability constant
Strong acid dissociable cyanides (SAD)	
$\text{Co}^{3+} + 6\text{CN}^- \rightarrow \text{Co}(\text{CN})_6^{3-}$	$\log \beta_6 = 64.0$
$\text{Fe}^{3+} + 6\text{CN}^- \rightarrow \text{Fe}(\text{CN})_6^{3-}$	$\log \beta_6 = 43.6$
$\text{Fe}^{2+} + 6\text{CN}^- \rightarrow \text{Fe}(\text{CN})_6^{4-}$	$\log \beta_6 = 35.4$
$\text{Au}^+ + 2\text{CN}^- \rightarrow \text{Au}(\text{CN})_2^-$	$\log \beta_2 = 38.8$
Weak acid dissociable cyanides (WAD)	
$\text{Ni}^{2+} + 4\text{CN}^- \rightarrow \text{Ni}(\text{CN})_4^{2-}$	$\log \beta_4 = 30.22$
$\text{Cu}^+ + 4\text{CN}^- \rightarrow \text{Cu}(\text{CN})_4^{3-}$	$\log \beta_4 = 23.10$
$\text{Cu}^+ + 3\text{CN}^- \rightarrow \text{Cu}(\text{CN})_3^{2-}$	$\log \beta_3 = 21.66$
$\text{Ag}^+ + 2\text{CN}^- \rightarrow \text{Ag}(\text{CN})_2^-$	$\log \beta_2 = 20.48$
$\text{Cu}^+ + 2\text{CN}^- \rightarrow \text{Cu}(\text{CN})_2^-$	$\log \beta_2 = 16.26$
$\text{Zn}^{2+} + 3\text{CN}^- \rightarrow \text{Zn}(\text{CN})_3^-$	$\log \beta_3 = 16.05$
$\text{Zn}^{2+} + 4\text{CN}^- \rightarrow \text{Zn}(\text{CN})_4^{2-}$	$\log \beta_4 = 19.62$
$\text{Cd}^{2+} + 4\text{CN}^- \rightarrow \text{Cd}(\text{CN})_4^{2-}$	$\log \beta_4 = 17.92$
Free cyanide (CN^-, HCN)	
$\text{H}^+ + \text{CN}^- \rightarrow \text{H}(\text{CN})$	(pKa 9.3 at 20°C)

The ratio of weak acid dissociable (WAD)/weak acid cyanide/cyanide versus Cu glycinate in the system depends on the redox cycling between cuprous (Cu(I)) copper and cupric (Cu (II)) copper.

Oraby and Eksteen (2016) demonstrated that the slow leaching rate of glycine can be counteracted by adding small amounts of copper and cyanide (as cuprous-cyanide complex) to the glycine solution, resulting in leach rates as much as five times higher than those of cyanide-only leaching. In all cases where cyanide is present, it is at starvation levels. That is, the cyanide is utterly insufficient to dissolve cyanide-soluble copper minerals, and free cyanide is not measurable at the end of the leaching process, although low levels of WAD cyanide may remain. The use of small amounts of cyanide is usually not problematic as environmental effects are minimized, safety for workers is maximized, and operating costs are reduced significantly.

In addition to carbon, its elemental composition, which to a large extent depends on the source raw material, comprises hydrogen, nitrogen, sulfur, and oxygen. Active carbons have a tridisperse microcrystalline structure comprising pores of varying sizes classified as: micropores (<2 nm), mesopores (2–50 nm), and macropores (>50 nm). Macropores found on the outer surface of the active carbon particles act as a gateway for adsorbate molecules to pores in the innermost part of the active carbon. Mesopores, also known as transitional pores, branch directly off macropores and their function is to channel the adsorbate into the micropores.

Micropores, which make up about 95% of the total surface area of the active carbon, largely affect the adsorption capacity of a given active carbon. Most activated carbon adsorbents that are used commercially have a specific surface area in the order of 800 to 1,500 m²/g. The adsorption efficiency of active carbons is also determined by the chemical structure of the carbon's surface.

Maintaining a low free cyanide concentration increased the carbon's selectivity of gold over copper. Increasing the CN:Cu molar ratio from 1:1 (i.e. 123 ppm cyanide) to 2.5:1 (i.e., 476.3 ppm cyanide) reduced the gold loading capacity by half due to the increased adsorption competition of CN⁻ ions and copper. This is related to the presence of the WAD cyanide $Cu(CN)_2^-$. It is better to minimize the WAD cyanide in glycine-cyanide-copper in order to have higher gold adsorption. This may be done by sulfide precipitation, solvent extraction, or ion exchange.

Methodology and discussion

Copper compound deactivation

In order to stop the copper from being adsorbed into activated carbon and reducing gold loading, the cyanide content of the solution was increased from 18 ppm to 400 ppm, and adsorption tests were carried out for varying amounts of time on each of the different mixing times. As shown in Table 3, additional cyanide significantly decreased the copper adsorption in activated carbon and enhanced the gold content.

Table 3: Copper absorption in pregnant cyanide solution

	Pregnant solution cyanide content, ppm	Absorption time, min	Feed solution total Cu, ppm	After absorption solution total Cu, ppm	Cu absorption, %
Bottle roll absorption test	18	5	310	250	19.35
	50	10	280	233	16.78
	100	15	250	229	8.4
	250	25	280	277	1.07
	300	30	170	170	0.0
	400	40	280	280	0.0

Real life practice

Beginning in March of 2022, the processing plant was fed with cyanide. When the cyanide content was low, copper adsorption was very high and gold adsorption was very low. After adding cyanide into the cycle solution in March, copper adsorption decreased, and gold adsorption increased significantly, as shown in Table 4.

Table 4: Metal adsorption in activated carbon in the processing plant

Strip no.	Pregnant solution cyanide	Load carbon, Au	Load carbon, Cu
	ppm	ppm	ppm
130	7.81	908.2	133,231
135	5.2	711	148,860
139	5.2	679.8	107,328
175	250	4,670	31,316
179	270	2013	15,900
180	210	3,103	17,150
181	290	3,010	16,129
182	225	2,330	16,122

Date	Carbon grade			Cyanide content of pregnant solution, ppm
	Au	Ag	Cu	
1 Feb 2022	711.0	3,811.7	148,860.8	0
4 Feb 2022	652.5	4,601.9	66,892.1	0
7 Feb 22	594.2	2,985.3	91,923.9	0
11 Feb 22	679.8	3,178.9	107,328.6	0
13 Feb 22	649.5	3,610.3	120,996.7	0
16 Feb 22	655.1	3,271.8	124,315.4	0
21 Feb 22	549.8	3,695.8	75,558.4	0
10 Mar 22	5,270.1	9,173.9	9,153.6	250
13 Mar 22	3,476.5	7,045.6	16,210.0	250
16 Mar 22	2,295.9	6,116.1	19,989.8	250
19 Mar 22	1,638.7	4,204.7	20,555.8	250
1 Apr 22	2,396.2	5,359.2	23,911.5	250
3 Apr 22	2,134.1	5,795.7	26,864.2	250
05 Apr 22	1,865.0	6,338.3	23,958.0	250

Removing copper from the waste acid solution

Copper and silver in the gold concentrate was dissolved in nitric acid during the digestion stage to separate the gold. After silver was recovered by precipitation with the help of NaCl to form silver chloride, copper remained in the nitric acid solution. The metal content is presented in Table 5.

Table 5: Metal content of waste acid from digestion

SAMPLE ID	Au	Cu
	ppm	ppm
	Au-AA44	Cu-OG43n
Sample 1	4.47	121,763.1
Sample 2	2.44	108,517.7

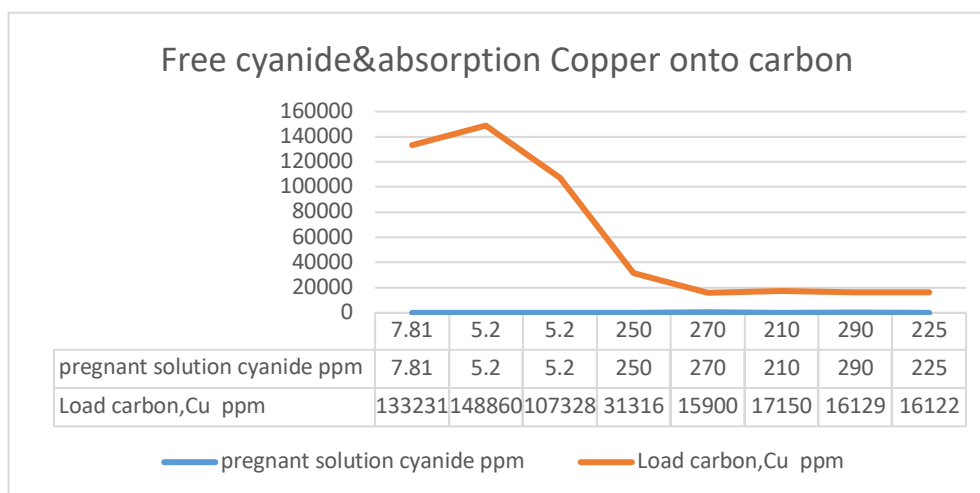


Figure 1: Free cyanide and adsorption of copper onto carbon

On average, the sample solutions produced approximately 1,500 kg of 115,140 ppm copper solution from every digestion stage, which was then poured into the mines barren pond. While circulating the high copper content, the solution was absorbed into activated carbon, which obstructed the gold absorption.

Therefore, there was a need to separate copper from the waste acid solution at a low cost and without installing new equipment.

Five millilitres of copper-containing solution was taken for comparison, and 1 gram of caustic soda was added to the first glass of solution. After mixing, the solution to which caustic soda was added turned deep blue with pure water on the top, meaning that it was fully separated from copper.



Figure 2: Caustic soda added to the solution



Figure 3: Precipitated copper sludge being filtered



Figure 4: Smelted copper out of waste solution

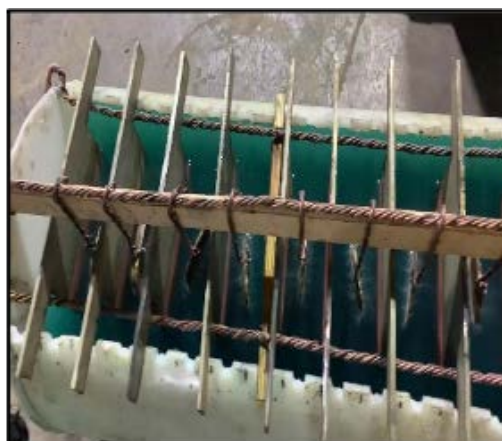


Figure 5: Ongoing electrolysis



Figure 6: Refined gold powder



Figure 7: Before and after comparison of refined bar

Copper removal was tested using a method in our actual process, and it purified the solution until its copper concentration reached 3 ppm. The precipitated copper was dried and smelted into pure copper.

Refining of high-copper-content gold ore

Two mini-scale electrowinning cells were built with a volume of 40 to 50 litres each. We prepared 12 pieces of 20×20 cm stainless steel board as cathode to carry out the experiment while the plant's stripping process was undertaken, in order to monitor and adjust the test process precisely. For instance, we adjusted the cathode size corresponding to the alloy bar size. The bars were supposed to be used as an anode. However, it was too thick for electrolysis; therefore, we re-smelted and cast them as 2 kg bars.

Welding apparatus was connected to the EW cell's anode and a cathode with doubled copper wire in order to supply electric current.

Copper electrolysis

We hung three alloy bars as an anode, and 12×12 cm stainless-steel boards as a cathode in the prepared

electrowinning cells.

Each EW cell was filled with 40 to 50 litres of water, 4 litres of nitric acid, 2.5 litres of ~15% copper nitrate solution, and was supplied with a 42 A electric current. While the process was being carried out simultaneously in the 2 EW cells, the chemicals were added to keep electrolysis proceeding effectively. Also, the operational solution was replaced with a new solution every other day, so that it would not become contaminated and slow down the process. The old solution had to be filtered to catch any gold powder.

During electrolysis, the alloy bars eroded by acid and electric current due to copper transfer into cathode, and gold became a compact powder stuck on the bar surface. Then, the gold powder covering the alloy was scraped in order to speed up the process. The alloy bars were scraped twice a day. The bars were hung like hooks, to make it easier. The electrolysis process lasted for 13 days in order to completely extract the gold inside the bars.

- *Gold cleaning:* the extracted gold powder was washed in nitric acid twice to remove trace copper. This was followed by washing in plenty of water to neutralize its acidic condition.
- *Smelting:* the fine gold powder was dried in a calcine oven for five hours and smelted.

As a result, a total of 14.38 kg of 4 pcs fine gold bars were produced from the high copper content alloy bars. It is calculated that the initial fineness of the bars was 314°.

Conclusions

For this project, research on removing the copper cyanide compounds that are formed during the heap leaching of low grade oxidized ores was carried out. The following conclusions were drawn.

The pregnant cyanide leach solutions of complex ores contain impure metals that form cyanide complexes. A certain level of free cyanide is required to be present in leach solutions for the efficient leaching of gold. Therefore, the content of free cyanide in the solution was adjusted to between 50 to 400 ppm. The most suitable cyanide content is >250 ppm, when the copper cyanide in the solution is contained in the pond in the form of a solution that does not adsorb on the activated carbon.

If this problem continues for a long time, adverse impacts such as cyanide consumption increase. Moreover, due to the increase of pollution in the solution ponds, the leaching process may be slowed down.

If the leaching process slows down, it is necessary to either change the solution in the solution ponds, or install a filter. When the copper amount is <300 to 500 ppm and the pH is high (pH>12.5), precipitating the copper and cyanide is the most efficient solution. But, it is clear that is not the case for the Altan Tsagaan-Ovoo mine project.

Also, the most economical solution might be to accumulate the copper-rich solution in which the desorption was done, to prevent the copper amount from increasing in the solution pond to the tailing storage pond.

Selective leaching of gold over copper from a copper–gold concentrate containing significant amounts of cyanide soluble copper has been achieved using cyanide solution and caustic soda as a pH modifier. The results showed that gold dissolution can be enhanced, and cyanide consumption can be reduced, if the leaching parameters are optimized. The important leaching conditions include cyanide concentration, CN/reactive copper ratio, and the pH of the leach solution.

Cyanidation is the dominant method of recovering gold from ores. In recovering gold from copper-containing gold ores by cyanidation, some copper minerals readily dissolve in cyanide. These can have a detrimental effect on the process efficiency and economics due to high cyanide consumption, reduced gold leach rates, and poor gold recovery.

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Column Experiments to Assess Consolidation Behaviour at Different Initial Irrigation Rates in Copper Heap Leaching

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Abstract

It is common practice to initiate irrigation in heap leach facilities at low rates, followed by a gradual ramp-up to the final target irrigation rate. This presumably improves the leach ore permeability and the metal recovery. However, no detailed study has been performed to understand the processes that may lead to better recovery using this technique. Large diameter column tests were performed with crushed and agglomerated copper heap leach ore to study solution flow and ore consolidation behaviour under three different initial irrigation schemes: baseline, slow ramp-up and fast ramp-up. The columns were monitored with moisture sensors, a neutron probe, and tensiometers. In addition, tracer tests were performed to evaluate flow uniformity. The baseline test used the full target irrigation rate from the beginning. In the baseline test, the leach ore experienced significant agglomeration structure collapse, localized ponding, consolidation, and preferential flow to depths of 30 cm. The slow ramp-up test exhibited minimal ore consolidation, no ponding, and reduced preferential flow. The faster ramp-up test showed some ore consolidation, but no ponding and intermediate preferential flow. Water content appears to be a determining factor in the agglomeration structure collapse; however, other factors such as pore size variation, bulk density (depth of heap), and pore solution velocity may also contribute. Consolidation tests can be performed to evaluate the ideal ramp-up schedule for specific heap leach ores.

Introduction

Heap leaching is a process to extract metals via irrigation with chemical solutions of rocks that are piled in heaps. The method has been used for centuries to recover precious and base metals from low grade ores. In recent years, its use has been expanded, due to favourable costs compared to the high cost and

environmental impact of ore milling and smelting. Numerous studies have investigated heap leach ore processing and irrigation methods, focusing on the unsaturated hydrology nature of this method (Decker, 1996; Orr, 2002; Galla, 2007; Guzman et al., 2013; Milczarek et al., 2013; Silver, 2013; Robertson, 2017). However, there is much improvement that can be made in the recovery process.

Leach ore hydraulic properties can play a significant role for metal recovery besides the ore grade. In particular, the presence of low permeability ores or the presence of preferential flow in the heap results in poor leachate distribution, leaving unleached or incompletely leached rock under normal irrigation cycles (Rucker et al., 2009). Many heap leach operations initiate solution irrigation at slow rates, ramping-up until a final target irrigation rate is achieved. This has been shown in some cases to improve the metal recovery; however, a limited number of detailed studies have addressed the flow mechanisms that cause increased leaching efficiency (Vries, 2013; Fernando et al., 2018). The purpose of this study is to investigate the flow mechanisms of this initial phase in the leaching process through a series of instrumented large-scale laboratory column experiments.

Methods

Heap leach ores from two copper mines were mixed, obtaining a particle size distribution that was expected to have a lower range of acceptable hydraulic properties according to the criteria described by Milczarek et al. (2013). The resulting ore blend was agglomerated with raffinate (leaching solution) at a gravimetric water content (GWC) of 0.06 g/g and concentrated sulfuric acid at 6 kg H₂SO₄/ton of ore. After agglomeration, the samples had a curing period of between 1 and 2 weeks. The agglomerated and cured ore was used for all tests in this study, except for the particle size distribution (PSD, ASTM D6913), which was performed on subsamples of air-dried ore.

Saturated hydraulic conductivity (K_{sat}) tests were carried out on 15-cm-diameter dual-wall cores using the long column method as described by Milczarek et al. (2013). This method simulates the confining pressures that the leach ore experiences at different depths in the pad and assists in determining the maximum heights of heaps in minimize permeability constraints. In this study, wall pressures were increased to simulate heap depths between 0 and approximately 20 m.

Large columns (150 cm tall × 53.8 cm in diameter, 341 L) were packed with agglomerated and cured leach ore at target dry bulk densities of 1.5 g/cm³. The column tests consisted of irrigating the columns with raffinate at a constant application rate of 9.1 L/h/m² (38 mL/min), through a single drip emitter at the center of the column, with different initial irrigation schedules, as shown in Table 1. Test 1 was the baseline, with constant irrigation at the target application rate. Slow ramp-up was applied in Test 2, whereas fast ramp-up was used for Test 3. Similar to heap leach operational practices, ramp-up irrigation rates were achieved by

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turning the pumps on and off. For example, for a rate of $1/16^{\text{th}}$ of the target irrigation rate, the columns were irrigated at the target rate for 0.5 hour and stopped for 7.5 hours, in 8 hour cycles (Table 1).

Table 1: On and off irrigation periods for each of the simulated irrigation rates, and schedule for ramp-up periods on all tests

Fraction of the target irrigation rate	Application (irrigation) rate (L/m ² /hr)	Time on (hr)	Time off (hr)	Duration of each irrigation phase (hr)		
				Test 1	Test 2	Test 3
$1/16^{\text{th}}$	0.57	0.5	7.5	–	48	–
$1/8^{\text{th}}$	1.14	0.5	3.5	–	48	–
$1/8^{\text{th}}$	1.14	1	7	–	–	25
$1/16^{\text{th}}$	0.57	1	15	–	16	–
$1/16^{\text{th}}$	0.57	0.5	7.5	–	–	23
$1/8^{\text{th}}$	1.14	1	7	–	–	24
$1/4$	2.28	1	3	–	24	24
$1/2$	4.56	1	1	–	24	–
$1/2$	4.56	4	4	–	24	24
1	9.12	Continuous	–	382	432	408
Total test duration (hr)				382	616	528

Heap leach solution was applied using calibrated high precision flow pumps; inflow rates were checked by measuring changes in the inflow reservoirs over time. Due to limited solution supply, heap leach solution was recycled throughout the tests; inflow reservoirs were refilled once or twice every day using the outflow solution. The column outflow rates were measured by obtaining the volume accumulated in the outflow reservoirs twice daily.

Moisture content was measured with nine capacitance soil moisture sensors (ECH₂O-10HS, Decagon, Pullman, WA) installed every 15 cm, through the center of the column. Sensor readings were cross-checked with GWC (oven dry weights) measurements before and after the tests. Soil moisture sensors were calibrated using the same ore material and the leaching solution as the column tests. Calibration was performed in a box by adding leaching solution in 0.03 volumetric water content (VWC) increments to the ore uniformly until reaching saturation. Raw readings were recorded and correlated to the known moisture content. Soil moisture calibration equations were adjusted for the solution density (1.17 g/cm³), and solution high salts content (20% by wt).

Moisture content sensors were paired with micro-tensiometers (GeoSystems Analysis, Tucson, AZ). Data from the sensors was collected with a Campbell Scientific CR10X data logger, at 5-minute intervals (CSI, Logan, UT).

Surficial ore collapse was assessed by measuring the depth difference from the original packed surface inside the column. In Test 1 the column center and three other outer locations were monitored. Depths from a reference height were measured approximately every hour for the first six hours, and more were taken while the test continued. For Tests 2 and 3, depths were tracked in a more comprehensive manner. Ten locations were monitored across the column area: five along the West-East axis and five along the South-North axis.

To assess preferential flow, Dye Blue FD&C 1 was injected at a concentration of 1 g/L at the end of the test when full target irrigation was achieved. After approximately two pore volumes, the irrigation was terminated.

The columns were drained by gravity and unpacked by removing 10 cm of material in each step and taking photographs to capture the dye distribution at each 10 cm depth. Ore samples were taken for GWC testing every 20 cm vertically at the: center; each cardinal direction at the edge of the column; and another four samples between the center and the edge.

In Test 2, bulk density samples were also taken with 10 cm diameter by 10 cm high PVC cores at each location, in order to verify the bulk density estimates. In Test 3, bulk density samples were taken every 20 cm vertically at the center and at the four positions on the edge of the column.

Results and discussion

Saturated hydraulic conductivity (K_{sat})

Results from the dual wall K_{sat} test showed the agglomerated leach ore increased bulk density from 1.5 g/cm³ with no loading pressure (top of the heap) to 2.0 g/cm³ with 18.7 m of equivalent loading pressure; K_{sat} values ranged from 1.99×10^{-1} cm/s to 2.2×10^{-3} cm/s over the same heap heights (Figure 1). A K_{sat} value $> 100\times$ the target irrigation rate is considered good ore permeability (Milczarek et al, 2013), consequently at heap heights greater than 5 m, the K_{sat} is below the ore permeability threshold (2.53×10^{-2} cm/s) and the leach ore is likely to have permeability constraints at depth within the heap.

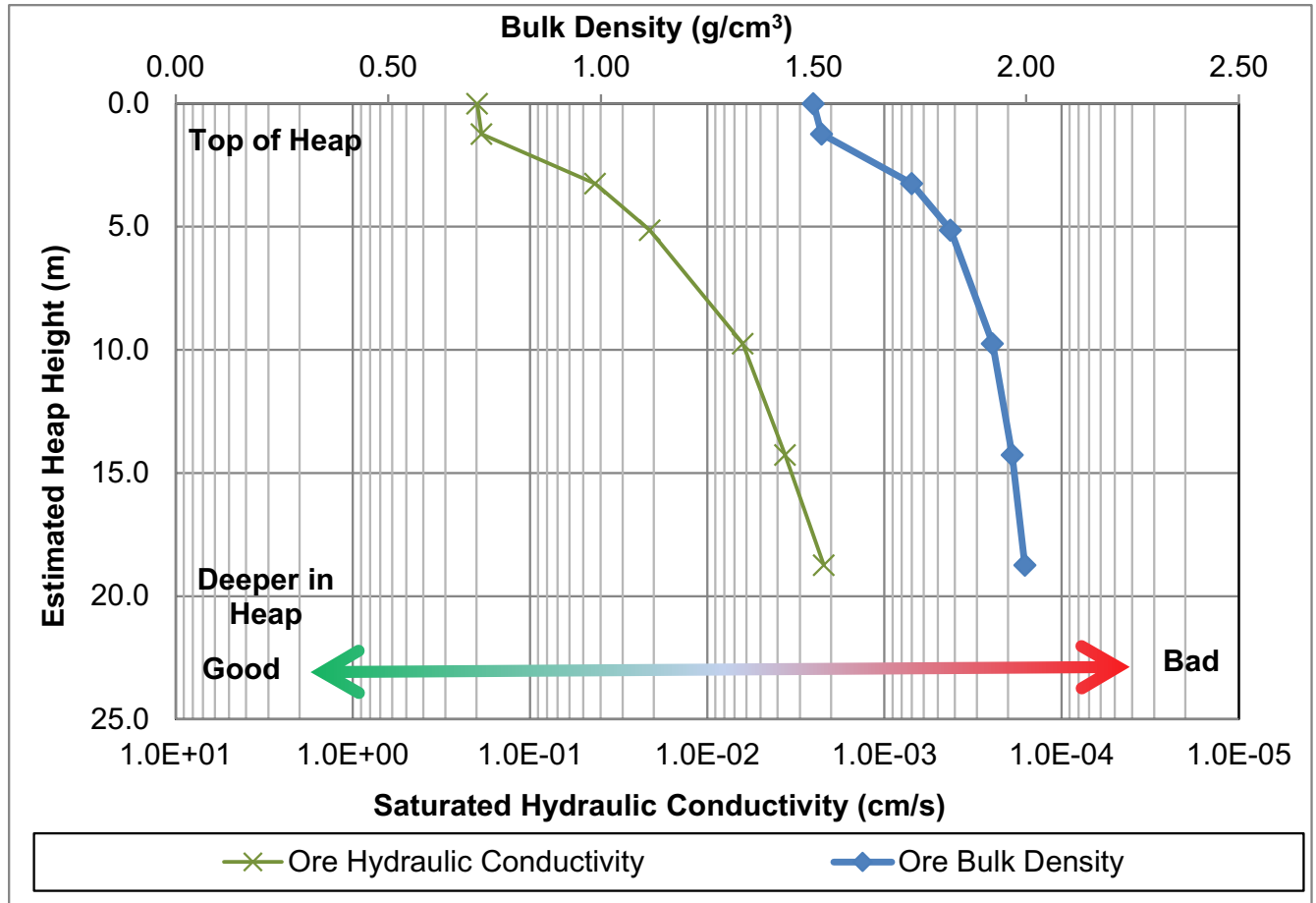


Figure 1: Leach ore saturated hydraulic conductivity at depth

Column tests

Consolidation/ore collapse

Figure 2 shows the change in depth at the ore surface for the three tests with total applied solution in litres (L) on the X-axis. Test 1 showed the most slumping, followed by Test 3 and Test 2. Test 1 also showed ponding of solution and the fastest ore collapse, settling to a final depth at approximately 38 hours, or 69 L of applied solution. Test 2 settled to an approximate final depth at 326 hours (430 L), and Test 3 at 260 hours (375 L) without ponding. Tests 1 and 3 showed greater ore settlement in the center versus the outer part of the column, whereas Test 2 showed even settlement.

For this reason, there is only one settlement curve for Test 2. These results indicate significantly greater ore collapse occurred with the faster ramp-up schedules compared to the slow ramp-up Test 2 which appeared to preserve the agglomeration structure.

Figure 3 shows the density profiles in all tests as linear regression from the bulk density samples collected at 20 cm intervals from the top to the bottom of the columns. Test 1 showed the highest bulk densities in the center and outer portions of the column: 1.6g/cm³ at the top (10 cm deep) and 2.20 g/cm³ at

the bottom. Test 2 showed the lowest bulk densities, from 1.50 to 1.68 g/cm³ and Test 3 showed bulk densities from 1.50 g/cm³ to 1.88 g/cm³ at the bottom.

The observed consolidation in Test 1 at depths greater than 40 cm ($> 1.8 \text{ g/cm}^3$) is equivalent to depths greater than 5 meters measured in the consolidation K_{sat} test, which also indicates a 10× or greater loss of ore permeability below 40 cm.

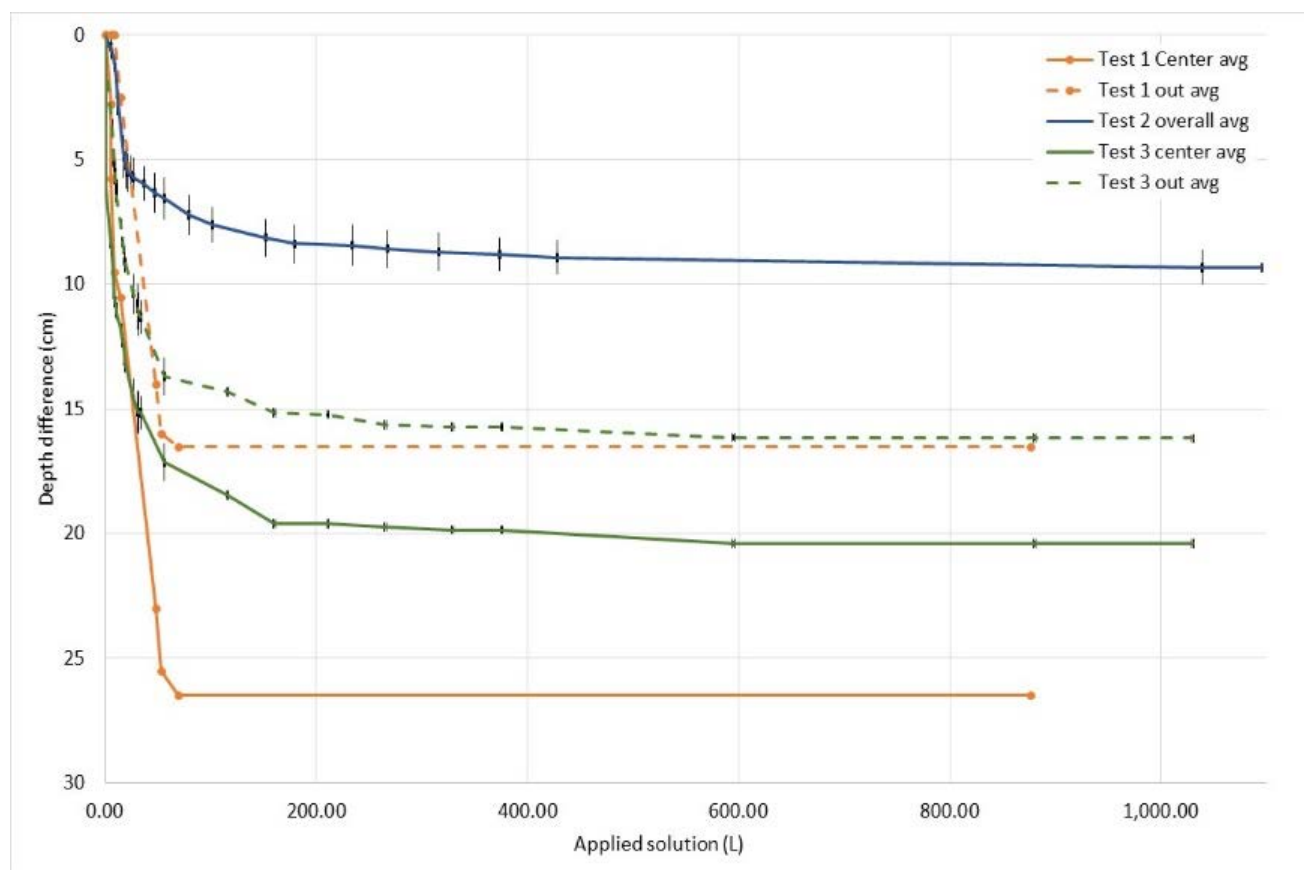


Figure 2: Slumping from original surface in Tests 1, 2, and 3 vs. time as applied solution

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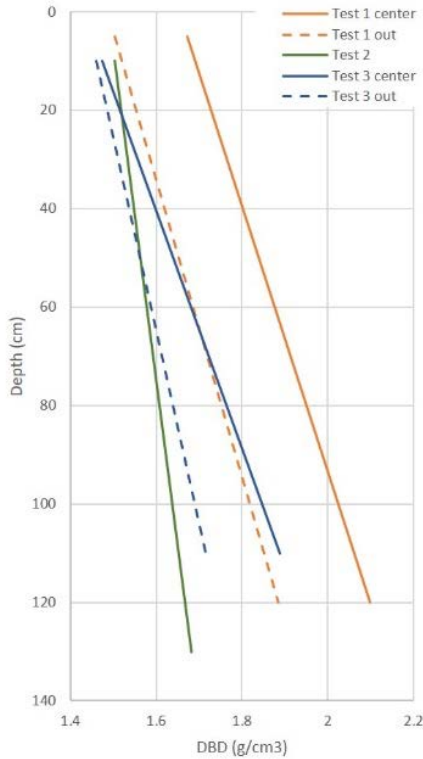


Figure 3: Estimated final bulk densities in Tests 1, 2, and 3

Moisture content

Initial VWC values from the nine moisture content sensors indicated a range of 0.085 to 0.111 with an average of 0.100 g/cm³, in Test 1. Figure 4 shows the wetting front progression in the Test 1 (baseline) column. The wetting front progressed faster in the first 45 cm than in the rest of the column, where the solution velocity was relatively constant, most likely due to spreading of solution away from the dripper. Observed step-shaped curves, such as sensors at depths 15, 30, 75, and 135 cm are interpreted to be the result of the ore collapse that occurred in the column. Significant variations can be seen in the moisture content data at 45, 75, and 90 cm in the first 150 hours.

The wetting and drainage behaviour from the Test 2 slow ramp-up period is shown in Figure 5. Exfiltration from the column did not occur until after the flow rate was changed to 1/8th of the target rate, more than 48 hours after the irrigation started (Test 1 was irrigated for 382 hours). Smooth water content curves were observed in Test 2 compared to Test 1. Moisture content values slowly increased with every irrigation pulse, whereas drainage from sensors deeper than 30 cm deep were greater than the 15 cm sensor most likely due to less consolidation and ore collapse at these depths. Overall, the VWC (average sensor data) was approximately 0.02 g/cm³ lower than in Test 1.

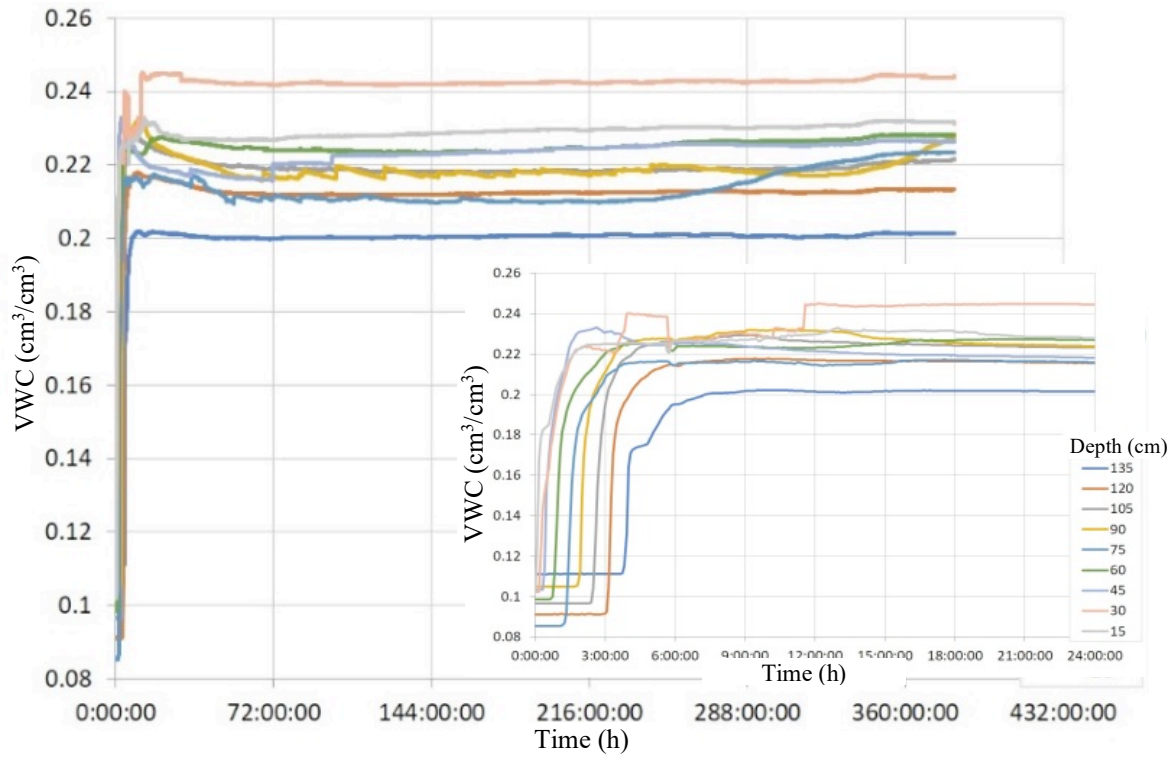


Figure 4: Volumetric water content at nine depths vs. time in Test 1

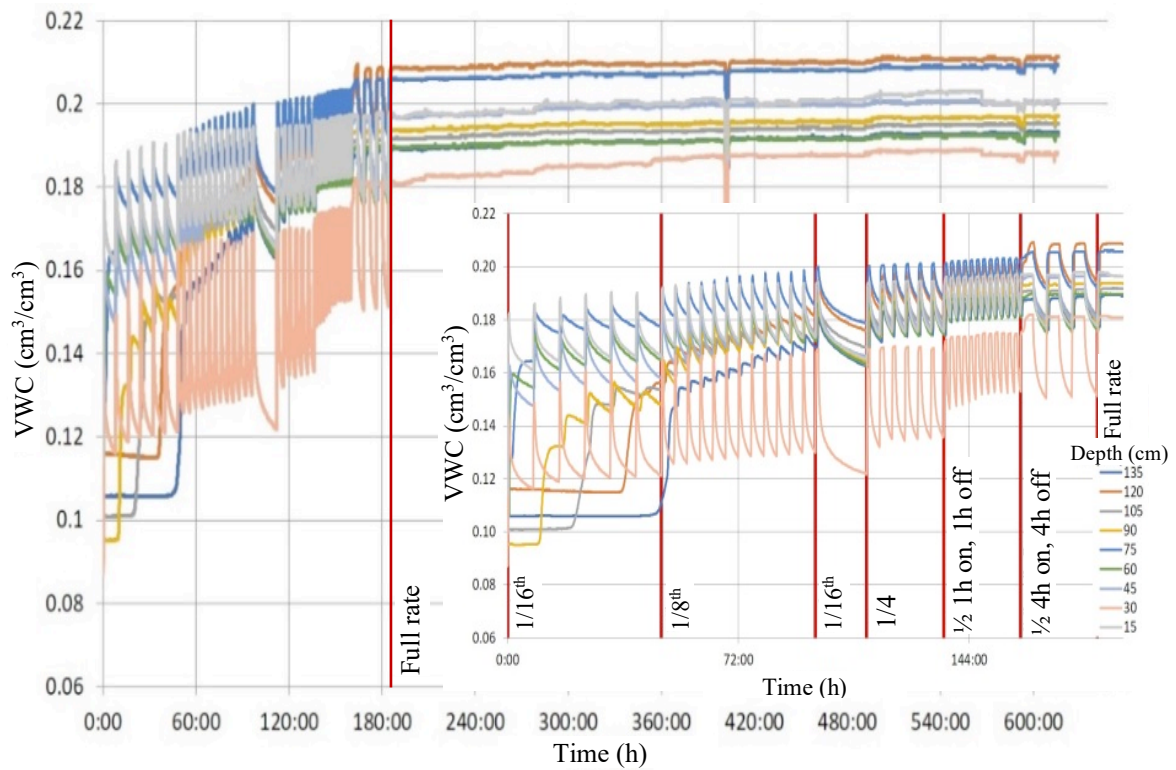


Figure 5: Volumetric water content at nine depths vs. time in Test 2

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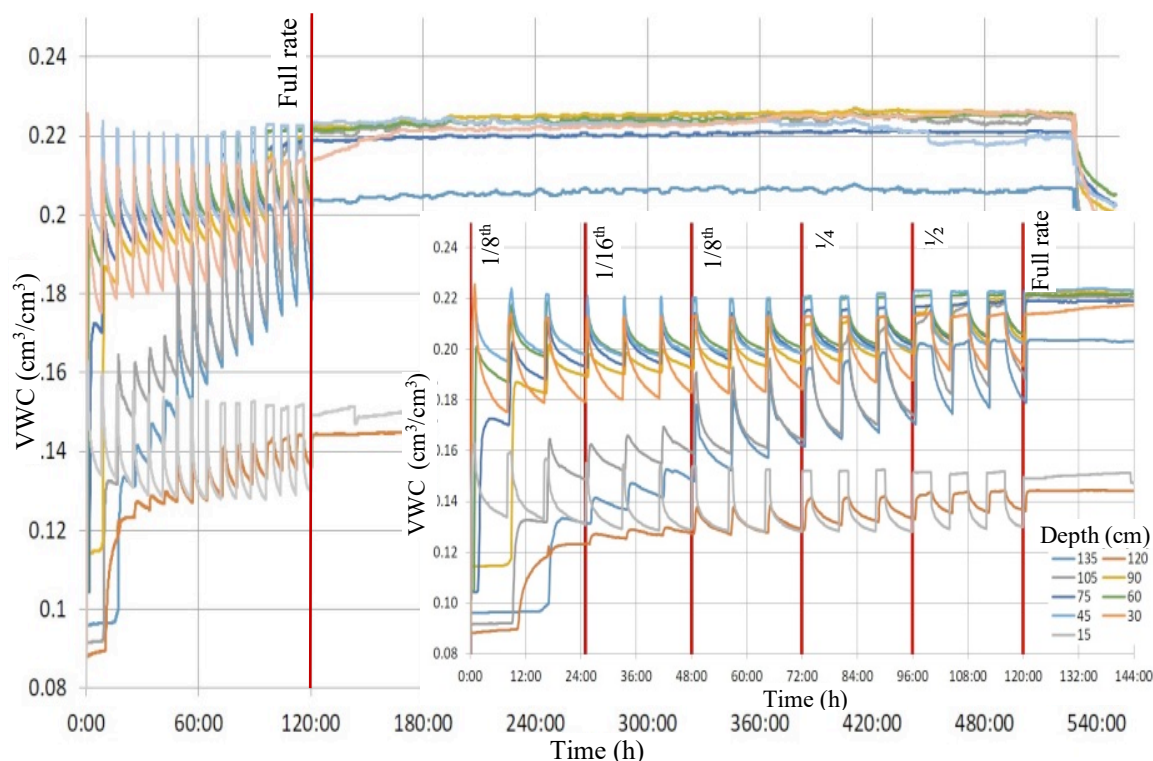


Figure 6: Volumetric water content at 9 depths vs. time in Test 3, from moisture sensors

Test 3, which started with a higher irrigation rate ($1/8^{\text{th}}$ target irrigation) than Test 2, resulted in higher moisture contents over time (Figure 6). The exceptions are sensors 105 and 135 cm, which had lower moisture from the start. After significant slumping was detected in Test 3 after the first day of irrigation, a $1/16^{\text{th}}$ rate was used to assess if reducing the rate would assist in stopping the ore collapse. This was not the case, so at 48 hours, the $1/8^{\text{th}}$ irrigation rate was used again. Sensors at depths 15 and 120 cm showed lower VWCs throughout the test, possibly due to solution bypass (see Dye tracer test results). Although intermediate ore collapse was observed in this Test 3, compared to Tests 1 and 2, the VWC was stable.

Dye tracer test

Figure 7 shows the ore and dye staining from solution contact at different depths. Due to the short period of dye tracer application (two unsaturated pore volumes) the amount of dye was insufficient to fully penetrate the entire column and thus dye tracer staining is indicative of the predominant flow paths within each column. Due to limited ore availability, no sorption coefficients or retardation factors were measured for the dye. However, prior research has reported sorption of Blue FD&C 1 by various porous media (Flury and Flühler, 1995; Perillo et al., 1998; Germán-Heins and Flury, 2000; Koestel et al., 2008). The estimated volume of influence (dyed volume/solution volume) are 60.03, 50.93, and 67.06 L in Tests 1, 2, and 3, respectively.

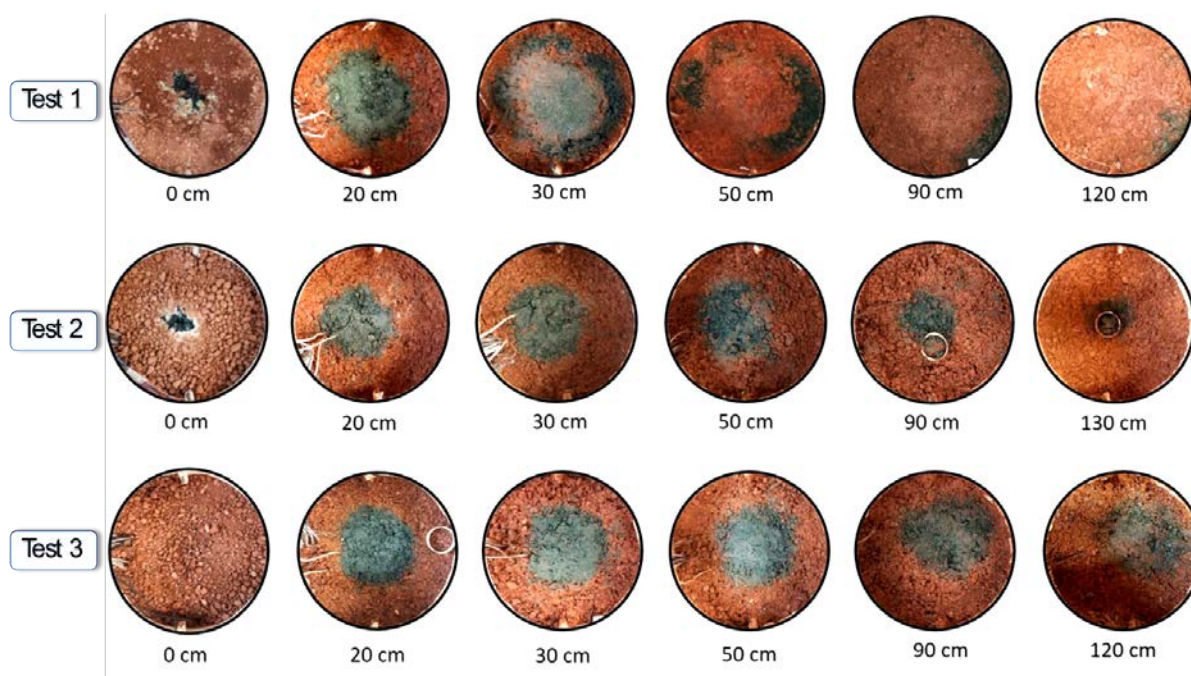


Figure 7: Ore images at different depths after the dye test

Test 1 showed two dye patterns. The surface of the column shows a small centroid directly under the emitter. By 20 cm, the dye spread through the center of the column and spreading increased by 30 cm with notable staining towards the outside of the column. By 50 cm, staining is only observed around the outer portions of the column until only a small portion of ore was dyed at 120 cm depth. This indicates that below 30 cm, solution flowed around the collapsed center of the column into the outer pore space with higher hydraulic conductivities. Nevertheless, limited, and slower flow may still have occurred in the center of the column, as seen from moisture content data slowly increasing at the final stage of Test 1 (Figure 4).

Test 2 showed more uniform staining patterns. With the exception of the deepest moisture sensor in Test 2, there is correlation with the measured VWCs in all tests with the dyed areas, i.e., the smaller the dyed area, the lower the moisture content. Although ore settlement in Test 3 was intermediate to Tests 1 and 2, the dyed areas show that the ore collapse in the center of the Test 3 column was not enough to reduce the ore permeability to cause diversion of flow to preferential pathways around the outside of the column. Dye staining remained in the center, but showed more lateral spreading, as seen by the slightly larger dyed areas compared to Test 2. This indicates that the partial collapse of the agglomerate structure may have promoted more spreading of solution in Test 3 versus Tests 1 and 2.

Mass balance

The mass balance results indicate that in Test 1, storage increased quickly to greater than 30 L during the initial wetting period, and then it decreased between 100 to 200 of solution applied (Figure 8). Although

additional slumping was not observed during this time, the redistribution of solution draining from the collapsed ore space to higher conductivity pore spaces may have been the cause of this loss in storage. After 200 L, storage stabilized and slightly increased from 25 to 27 L. Storage in Test 2 increased slowly during ramp-up and had a sudden increase when the full irrigation rate was applied. It experienced a second increase after tracer injection started and then decreased slowly until the end of the test. Storage in Test 3 increased up to 21 L during the initial ramp-up and increased slowly up to 27 L. Overall, Test 2 showed the greatest solution retention between the three columns.

Conceptual model of leach ore initial irrigation

Rapid wetting of leach ore and subsequent ore collapse is most likely due to a combination of physical forces. Solution flux rates near the drip emitter are very high compared to the target rate of 9.1 L/m²/hr; a 5 cm² area below the dripper is subject to 2,000× the target irrigation rate. As solution then flows under gravity into agglomerated leach ore, it rapidly increases saturation and weakens the capillary and direct contact forces that keep particles together.

Rapid saturation may also loosen fine particles from the agglomerated surface and break down the structure, in addition to chemical decrepitation which breaks down larger particles. The resultant ore collapse should be localized under the emitter and result solution flow bypass (Figure 9). High localized flux rates should decrease with depth as the wetting front advances and capillary forces pull the solution into smaller pores both laterally and downward. These processes are further discussed in Yao et al. (2022, in these proceedings).

The results of these column studies indicate that initial irrigation at the target flux rates resulted in ore collapse and solution bypass below 30 cm depth. The ramp-up irrigation column results indicate that during the off-irrigation period, drainage can be expected to occur initially from larger pores and then from smaller pores spreading laterally and slowing down and redistributing the wetting front. When irrigation is reinitiated, the underlying leach ore is more uniformly wetted and better able to conduct solution due to higher hydraulic conductivity than in the initial irrigation phase. This process was repeated at 8 hour intervals for the initial ramp-up step and then subsequently with lower off period intervals. Since the on periods were maintained at 0.5 hours of irrigation, the shorter drainage periods result in less time for redistribution. However, if the column or leach ore profile is initially wetted during the initial ramp-up phase, additional solution distribution is likely to occur more uniformly.

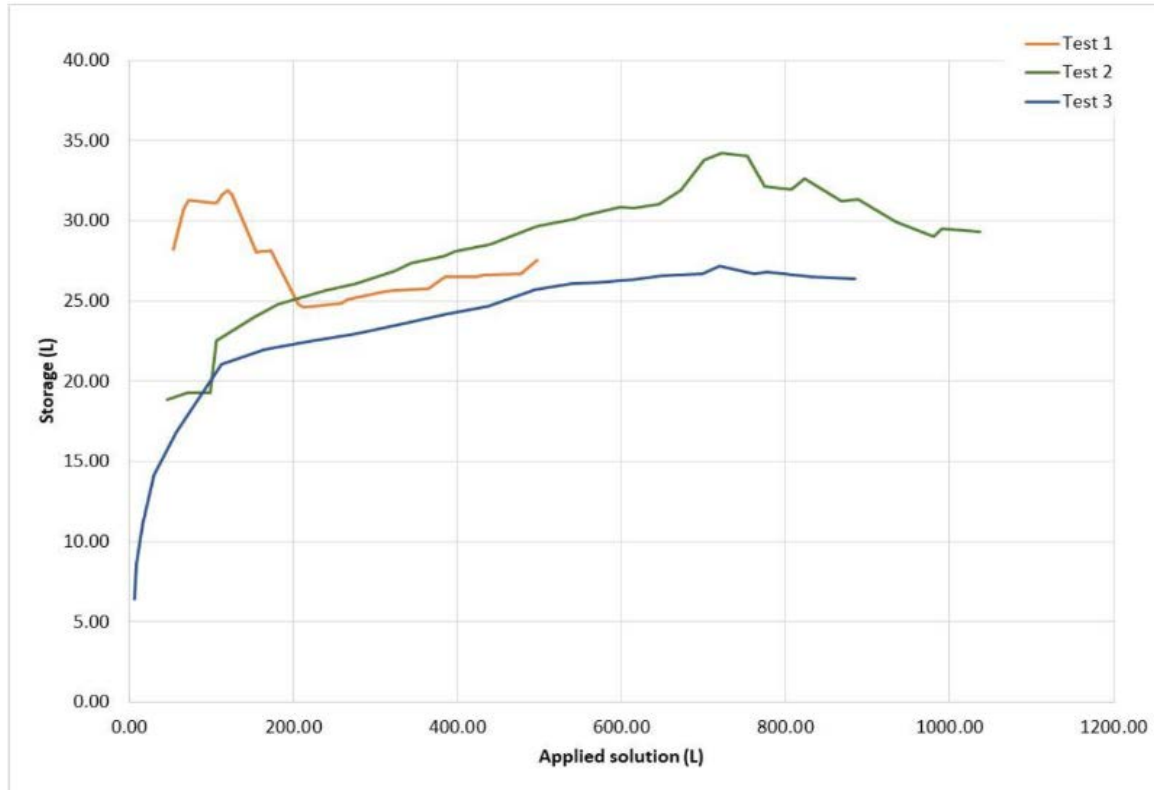


Figure 8: Solution storage over time in Tests 1, 2, and 3

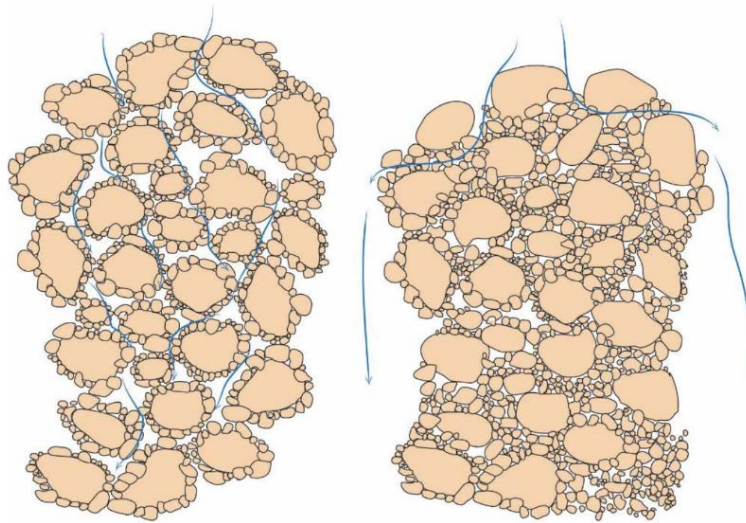


**Figure 9: Conceptual process of redistribution of particles after agglomeration break-up.
Left: Before irrigation. Right: During and after initial irrigation and wetting**

Conclusions

A large copper leach ore sample was agglomerated and tested for hydraulic and physical property parameters. Three large column tests were performed to assess the effect of different initial irrigation

schemes on ore consolidation and solution distribution. The baseline test (Test 1) irrigated the column at a target irrigation rate of 9.1 L/m²/hr, which showed significant ore collapse, ponding at the surface, and preferential flow. Test 1 showed a rapid increase in solution storage and then decline after ore collapse; overall solution storage retention was smaller in this test than in the ramp-up tests, which indicates lower solution contact with the ore. Although Test 1 showed larger dyed areas in the upper 30 cm of the column, significant dye channeling was observed in the lower half of the column. The dye staining indicated solution bypass occurred around a collapse zone in the center of the column.

The slowest ramp-up test (Test 2) showed minor ore settlement, and the highest solution retention. Dye tracer results indicated flow was present through the central portion and not diverted as in Test 1. The higher storage retention is most likely due to improved moisture redistribution during the off-irrigation periods. The fast ramp-up test (Test 3) showed intermediate slumping between the baseline Test 1 and slow ramp-up Test 2; however, the degree of collapse was not as extreme, and solution flow remained in the central portion of the column. Given that solution to ore contact is important for leaching, the Test 1 and Test 2 column results indicate good conditions for copper recovery.

The results of this study indicate that initial irrigation conditions can significantly affect the solution distribution and consolidation of agglomerated copper leach ore at the near surface. To maintain desirable conditions for agglomerated ore heap leaching, irrigation ramp-up schedules should be evaluated and incorporated into copper heap leach system design.

Acknowledgements

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Modeling the Iodine-Assisted Chalcopyrite Leaching in Ferric Sulfate Media in Response to Operating Parameters in Heap Leaching

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Abstract

The iodine-assisted chalcopyrite leaching process involves the addition of iodine in ferric sulfate media to accelerate the dissolution of chalcopyrite at ambient temperature. This process is developed to mainly treat low-grade copper sulfide ores by heap leaching. To scale up the process from laboratory to industrial-scale heap leaching, it is essential to understand the fundamental processes that control the performance of the iodine-assisted chalcopyrite leaching in a heap leach environment. These fundamental processes include hydrodynamics, reaction kinetics, and heat transport, which can be controlled by manipulating various key design variables.

In this study, the effect of key design variables, including irrigation rate, ferric concentration, total iodide concentration, temperature, and heap height, on the copper extraction and the solution potential were investigated using the HeapSim 2D software. The HeapSim 2D model was calibrated using a kinetic equation developed for iodine-assisted chalcopyrite leaching and experimental data collected from the 1-m column leach tests. The sensitivity study shows that optimum copper extraction could be achieved by adjusting these key design parameters. The iodine-assisted leaching slowed down at potentials lower than 650 mV vs SHE, at which the predominant iodine species was iodide ions (I^-) and the leaching rate was similar to that of the conventional ferric sulfate leaching. Interestingly, passivation was not observed in the course of leaching and leaching was resumed with fast kinetics when the solution potential was sufficiently high (above 650 mV vs SHE).

Introduction

Heap leaching is considered to be an effective and economically viable technology for treating low-grade copper sulfide ores. Among all oxidants studied, ferric sulfate is the most commonly used lixiviant because

of process simplicity, proven downstream processes, and low capital and operating costs. However, the leaching process exhibits poor leaching performance when the dominant copper sulfide is chalcopyrite. The slow dissolution kinetics is related to a passivating layer, which has been reported to be metal-deficient sulfide ($M_{1-x}S_2$) (Buckley and Woods, 1984; Lu et al., 2000; Mikhlin et al., 2004), copper-rich, iron-deficient polysulfides (MS_n , $n > 2$) (Hackl et al., 1995), elemental sulfur (Dutrizac, 1989; Munoz et al., 1979), and iron precipitates (Antonijević et al., 2004; Córdoba et al., 2008).

JX Nippon Mining and Metals Corporation has developed a process called the iodine-assisted heap leaching of chalcopyrite ore to overcome the slow dissolution. This process involves the addition of 50 to 200 mg/L iodide to significantly enhance the dissolution of chalcopyrite in ferric sulfate media at ambient temperature (Manabe, 2012). The iodide ion is thought to be oxidized by ferric to diiodine (I_2), which can act as a strong oxidant (Eq. 1). Diiodine oxidizes chalcopyrite to release copper and iron and meanwhile is reduced to iodide (Eq. 2). Diiodine is continuously regenerated by ferric during leaching; hence, theoretically, the process does not require a continuous addition of iodine. Ferric ions are regenerated separately via ferrous oxidation assisted by iron-oxidizing bacteria (Kuwano et al., 2011).



The key factors that control the reaction rate and the mechanisms by which iodine accelerates the leaching kinetics have been investigated in previous studies by the authors (Winarko et al., 2022; 2020). Those studies reported that the solution potential was the primary factor affecting the process performance through influencing the iodine speciation. The dissolution rate was not hindered by the leaching product and iron precipitate. The dissolution rate of chalcopyrite in the process was described by Eq. 3.

$$X = 1 - \left(1 - 49.712 \exp \left(\frac{43,096}{R} \left(\frac{1}{298.15} - \frac{1}{T} \right) \right) \left(\frac{[Fe(III)]}{[Fe(II)]} \right)^{0.322} \left(\alpha [I_{total}] + \frac{2}{3} \beta (1 - \alpha) [I_{total}] \right)^{1.167} \cdot t(\text{day}) \right)^{3.236} \quad (\text{Eq. 3})$$

Where:

X is the fraction of the total copper dissolved,

t is the leaching time (in days),

R is the gas constant ($8.314 \text{ J mol}^{-1} \text{ K}^{-1}$),

T is the leaching temperature in Kelvin,

$[Fe(III)]$, $[Fe(II)]$, and $[I_{total}]$ are the total concentration in mol/L of ferric, ferrous, and iodide, and

α and β are conversion factors of iodine ranging from 0 to 1.

This kinetic equation is essential to optimize the performance of the iodine-assisted leaching process in an industrial-scale heap leach pad.

To assist the simulation and prediction of heap leaching performance, the HeapSim 2D model has been developed, which incorporates solution transport, heat transport, and kinetics of reactions occurring inside a heap (Bouffard and Dixon, 2009). In the current study, the HeapSim 2D model was used to simulate the iodine-assisted chalcopryrite leaching. Two sets of 1-m column leaching tests were performed: one for calibration and the other for validation of the HeapSim 2D model. The calibrated model was then used to understand the effect of various heap leach design parameters, i.e., irrigation rate, solute concentration, temperature, and heap height on the leaching performance. The findings of this study will assist the optimization of the iodine-assisted heap leaching of chalcopryrite ore by manipulating the key design factors that determine the process performance.

Methodology

Column leaching tests for model calibration

Sample preparation

The copper ore used for the test had the following particle size distribution: 46% > 11.2 mm, 25% – 11.2 + 4.75 mm, and 29% < 4.75 mm. The copper content was analyzed using the sequential leaching method to obtain the acid soluble copper, the secondary copper sulfides or cyanide-soluble copper, and the primary copper sulfides or insoluble copper. The chemical analysis was performed using inductively coupled plasma-optical emission spectrometry (ICP-OES). The percentage distribution of copper is shown in Table 1.

Table 1: The distribution of copper minerals in the ore used for the column tests for model calibration

	Total Cu	Acid soluble Cu	Secondary Cu sulfides	Primary Cu sulfides
Grade (%)	0.616	0.026	0.026	0.564
Cu (%)	100	4.22	4.22	91.56

Column leaching test procedure

The column leaching tests were carried out in polypropylene columns of 10 cm in diameter and 1 m in height. The inner surface was coated with a PTFE film to prevent iodine loss by adsorption onto the plastic material. Each column contained 12 kg of primary copper sulfide ore.

The column leach tests were performed at two total iodide concentrations, i.e., 100 and 200 mg/L. For comparison, a control test similar to conventional ferric sulfate leaching in the absence of iodine was also performed. All column leach tests were conducted at the atmospheric pressure and the room temperature (~25 °C) and at an initial ferric concentration of 5 g/L. The conditions under which the column tests were carried out for the model calibration are shown in Table 2. For the tests with iodide addition, iodide and

ferric solutions were introduced separately to prevent loss of diiodine by evaporation before iodine came into contact with the ore. The total solution flowrate was set at 1 L/day.

Table 2: Experimental design of the column leaching tests used for the model calibration

Column leaching tests	[Fe(III)]	[I _{total}]	Solution flow rate, L/day	
	g/L	mg/L	KI	Fe ₂ (SO ₄) ₃
Control test	5	0	0	1
Total iodide concentration	5	100	0.5	0.5
	5	200	0.5	0.5

The column leaching tests were run in three sequential stages: acid leaching, ferric leaching, and iodine leaching.

1. Acid leaching stage: the columns were initially irrigated with sulfuric acid solution at pH 1.2 for 8 days. The ratio of the total volume of leaching solution to the mass of the ore was 0.7 m³/tonne ore for this stage.
2. Ferric leaching stage: the columns were then irrigated with a ferric sulfate solution at 5 g/L Fe(III) for 14 days. The ratio of the total volume of leaching solution to the mass of the ore was 1.9 m³/tonne ore for this stage.
3. Iodine leaching stage: all columns continued to be irrigated with a ferric sulfate solution, but two of the three columns (except the control test) were also irrigated with an iodide solution. Ferric sulfate and iodide were supplied to the columns via two separate feed solutions to prevent iodine loss via volatilization during solution preparation. The flow rate of the two feed solutions was fixed at 0.5 L/day, which gave a total solution flow rate of 1 L/day. The tests were completed when the ratio of the total volume of the leaching solution to the mass of the ore reached 10 m³/tonne ore (125 days).

Solution samples were collected weekly and analyzed by ICP-OES for concentrations of the total Cu and the total Fe. The solution pH and ORP and the total iodide concentration were measured weekly. At the end of the column leach tests, the residues were digested and analyzed for the copper and iron contents by ICP-OES.

Column leaching tests for model calibration

Sample preparation

A different copper ore was used in the column leaching tests for the model validation. The ore has a particle size (P₈₀) of 20 mm. The total copper content in this ore was 0.57% – 0.60%, of which 0.02% Cu was acid soluble, 0.01% – 0.02% Cu was secondary, and 0.53 – 0.57% was primary copper sulfides.

Column leaching test procedure

Column leaching tests for the model validation were carried out in a column of a different dimension and at a flowrate and ferric and iodide concentrations different from the tests used for the model calibration. The column leaching tests for the model validation were carried out in columns of 1 m in height and 15 cm in diameter at the atmospheric pressure and the room temperature (~25 °C). The column contained around 25 kg of the ore sample and was irrigated at a solution flowrate of 1.9 L/day.

Two tests were performed: one without iodide addition similar to conventional ferric sulfate leaching as control and the other at 100 mg/L total iodide. The initial total iron concentration was set at 5 g/L. The column test with iodide addition was first irrigated with only ferric sulfate solution for 21 days to remove secondary copper sulfides. Then, ferric sulfate and iodide solutions were supplied separately to the column. The flowrate was adjusted to 1.7 L/day for the ferric sulfate solution and 0.2 L/day for the iodide solution.

Model calibration and validation

The HeapSim 2D model was used in this study to simulate the 1-m column leaching tests. The kinetic model developed in the stirred reactor tests (Eq. 3) was rearranged to (Eq. 4), which was then incorporated into the HeapSim 2D model.

$$r = \frac{dC_{cpy}}{dt} = \rho_b \frac{k_{cpy}}{C_{cpy,0}^{(\varphi-1)}} \exp \left[\frac{43,096}{R} \left(\frac{1}{298.15} - \frac{1}{T} \right) \right] \left(\frac{[Fe(III)]}{[Fe(II)]} \right)^{0.322} \left[\alpha[I_{total}] + \frac{2}{3}\beta(1-\alpha)[I_{total}] \right]^{1.167} C_{cpy}^{\varphi} \quad (\text{Eq. 4})$$

Where:

r is the rate of chalcopyrite dissolution,

k_{cpy} is the rate constant at the reference temperature (25 °C),

ρ_b is the bed packed density (kg/m³_{heap}),

$C_{cpy,0}$ is the concentration of chalcopyrite in ore at $t = 0$ (kmol/kg),

C_{cpy} is the concentration of chalcopyrite in ore at time t (kmol/kg),

φ_{cpy} is the topological exponent corresponding to the degree of liberation.

In the model calibration, the intrinsic parameters controlling the leaching reaction, including the activation energy and the reaction orders, were maintained the same as derived from the reactor tests. Because the size fraction and the characteristics of the ore used in the column tests were different from those in the reactor tests, the topological exponent (φ_{cpy}) and the rate constant (k_{cpy}) that could represent these differences were adjusted in the model calibration. The model calibration was carried out using the first set of column leaching data. Then, the calibrated model was validated with the second set of column tests. During model validation, the rate constant (k_{cpy}) was the only parameter adjusted.

Sensitivity test

The calibrated model was used to simulate column leaching performance under different key design parameters, i.e., irrigation rate, solute concentration (including ferric and iodide concentration), temperature, and column height. The data obtained from the simulations included the percentage of copper extraction and the solution potential. The conditions under which the simulations were performed are shown in Table 3.

Table 3: The variations of key design parameters investigated in the sensitivity test

Parameters	Design parameters				
	Irrigation rate (L/m ² /h)	[Fe(III)] (g/L)	[I _{total}] (mg/L)	Temperature (°C)	Column height (m)
irrigation rate	2.65	5.0	200	25	1.0
	5.30	5.0	200	25	1.0
	7.95	5.0	200	25	1.0
	10.60	5.0	200	25	1.0
Ferric concentration	5.30	2.5	200	25	1.0
	5.30	5.0	200	25	1.0
	5.30	10.0	200	25	1.0
	5.30	15.0	200	25	1.0
Iodide concentration	5.30	5.0	50	25	1.0
	5.30	5.0	100	25	1.0
	5.30	5.0	150	25	1.0
	5.30	5.0	200	25	1.0
	5.30	5.0	250	25	1.0
Temperature	5.30	5.0	200	15	1.0
	5.30	5.0	200	25	1.0
	5.30	5.0	200	35	1.0
	5.30	5.0	200	30	1.0
	5.30	5.0	200	35	1.0
Column height	5.30	5.0	200	25	0.5
	5.30	5.0	200	25	1.0
	5.30	5.0	200	25	2.0
	5.30	5.0	200	25	4.0

Results and discussion

Model calibration

Figure 1a shows the copper extraction over time at three different total iodide concentrations for the 1-m high column tests for the model calibration. For all three tests, the conventional ferric sulfate leaching was carried out for the initial 21 days to remove acid soluble copper minerals and secondary copper sulfides. From day 21 onwards, copper was dissolved very slowly in the absence of iodine, with a dissolution rate of $< 0.05\%$ copper per day. By contrast, the addition of iodide significantly accelerated the kinetics of the copper dissolution. The copper extraction reached 54.7% in the presence of 200 mg/L total iodide on day 125, whereas the conventional ferric sulfate leaching achieved only 6.5% copper extraction during the same leaching period.

Figure 1b depicts the solution potential of the pregnant leach solution over time during leaching. The solution potential measured of the conventional ferric sulfate leaching test was the highest among the three tests, because the least amount of ferric was consumed by chalcopyrite oxidation. The potentials were similar in the presence of 100 mg/L and 200 mg/L total iodide. This was attributed to the existence of a potential window, within which the oxidized iodine species (diiodine and triiodide) are predominant and act as the oxidants for chalcopyrite dissolution. Once the solution potential decreased to below the lower limit of the potential window, ferric became the main oxidant and the conventional ferric leaching kicked in.

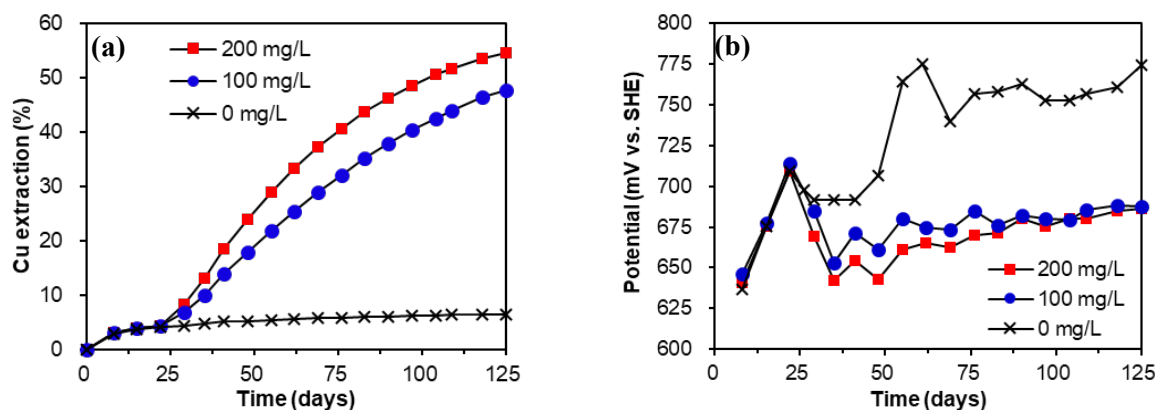


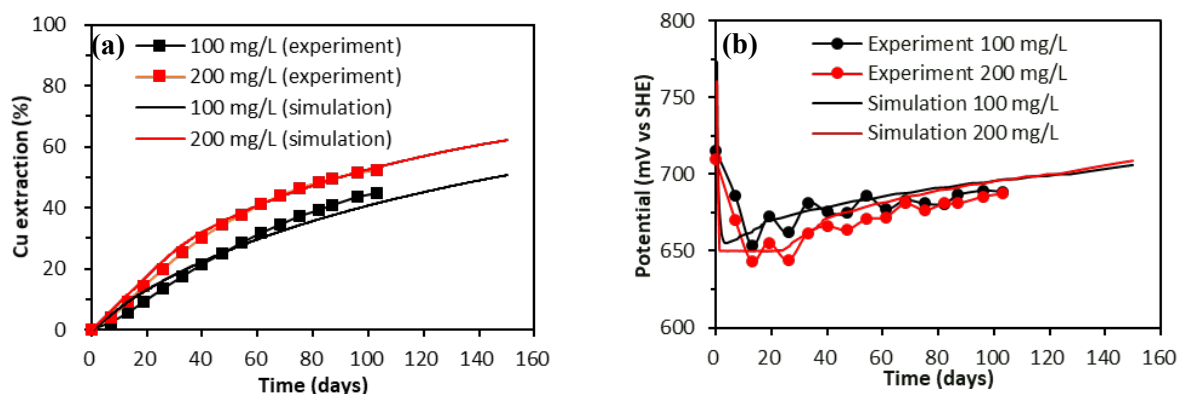
Figure 1: Experimental results of 1-m column leaching tests used for model calibration without and with the addition of 100 and 200 mg/L of iodide at room temperature ($\sim 25^\circ\text{C}$): (a) cumulative copper extraction; (b) solution potential during leaching

The model calibration used the column leaching data in the presence of iodine collected from day 22 onwards. The copper extraction was adjusted to account for the extraction from only chalcopyrite, which contained 0.564% of the total copper, because the acid soluble and secondary copper were removed in the first 21 days of leaching. The calibrated parameter values are presented in Table 4.

Table 4: The calibrated values of all leaching kinetic parameters

Parameters	Description	Value	Unit
φ_{cpy}	Topological exponent, corresponding to degree of liberation	3.6	–
k_{cpy}	The rate constant at reference temperature (25 °C)	34.56	1/day

The model with these calibrated parameters could satisfactorily simulate the copper extraction and the solution potential during leaching (Figure 2). The pregnant leach solution has lower potential values in the presence of iodine than the control test, because a higher amount of chalcopyrite was dissolved for the same period of time. With the addition of 200 mg/L iodide, the potential reached 650 mV vs SHE and remained stable at this value for the first 50 days of leaching. This suggests that there was not enough ferric in the column to maintain the potential in the range where the oxidized iodine species were predominant. When the potential was at 650 mV, conventional ferric leaching was responsible for chalcopyrite dissolution. As more ferric was provided to the column via irrigation and the leaching became slower, the potential eventually started to increase to around 688 mV vs SHE.

**Figure 2: The calibration of the kinetic parameters in the 1-m column leaching tests used in model calibration: (a) copper extraction; (b) solution potential****Table 4: The calibrated values of all leaching kinetic parameters**

Parameters	Description	Value	Unit
φ_{cpy}	Topological exponent, corresponding to degree of liberation	3.6	–
k_{cpy}	The rate constant at reference temperature (25 °C)	34.56	1/day

Model validation

Figure 3 shows the copper extraction and the solution potential during leaching without and with the addition of 100 mg/L total iodide for the model validation. The columns were initially irrigated with only ferric sulfate solution until day 25. The copper extractions achieved in both tests during this period were

similarly low (around 4% Cu). However, the copper extraction was significantly enhanced by the addition of 100 mg/L iodide (Figure 3a). In the presence of iodine, the rate of copper dissolution was faster (37.1% Cu extraction at the end of leaching) and the leaching reaction did not cease after 144 days (Figure 3b), while the control test (0 mg/L iodide) was very slow and only achieved 8.5% copper extraction in 144 days. It indicated that the chalcopyrite particles were not passivated in the course of leaching and iodine can significantly enhance the dissolution rate of chalcopyrite.

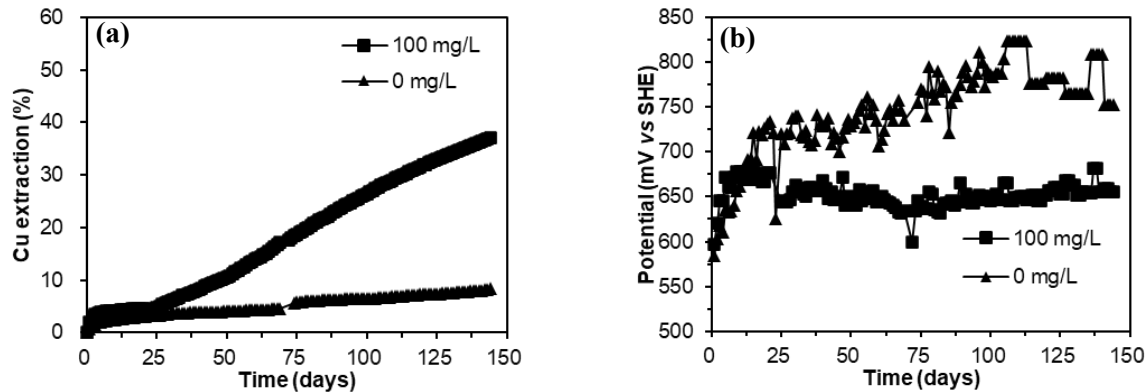


Figure 3: Experimental results of 1-m column leaching tests used in model validation without and with the addition of 100 mg/L of iodide at room temperature (~25 °C): (a) cumulative copper extraction; (b) solution potential during leaching

Note that the copper dissolution rates in this test were lower than those obtained from the 1-m column leaching tests for the model calibration. For example, on day 100, 28% copper extraction was achieved in this test as opposed to 42% in the test used for model calibration. This was attributed to the lower ratio of iodine to the mass of ore used in the test for model validation. Figure 3b shows that the solution potentials with the addition of 100 mg/L iodide were lower than those without iodide addition, because the former had higher levels of ferric consumption and ferrous generation from chalcopyrite oxidation.

By using the same topological exponent, the model was validated using the experimental data of the 1-m column leaching test. To satisfactorily describe the leaching kinetics of this 1-m column test, the rate constant was reduced from 34.56/day (model calibration) to 10.37 /day (model validation) to account for the lower ratio of iodine to the ore mass for this column test. The simulation results of the copper extraction and the solution potential for this column test are shown in Figure 4.

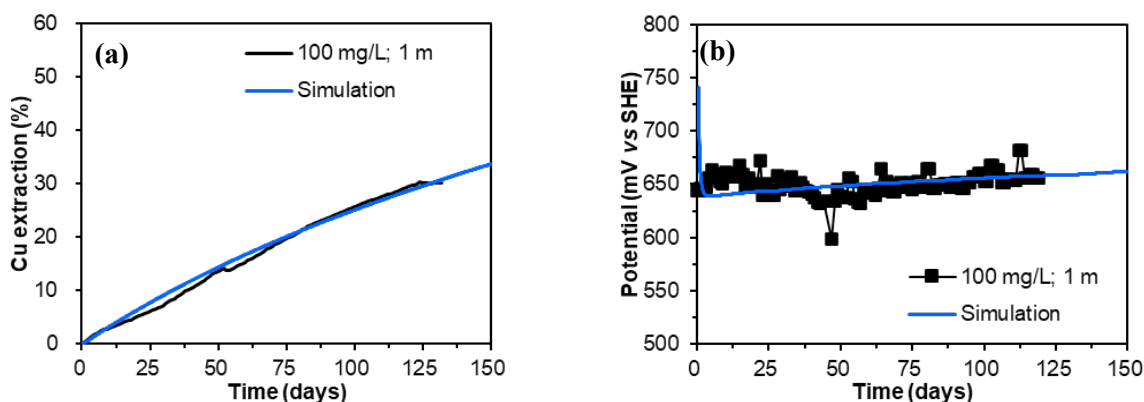


Figure 4: The calibration of the kinetic parameters in the 1-m column leaching test used in model validation: (a) copper extraction; (b) solution potential

Sensitivity tests

Sensitivity tests were performed to investigate the response of the leaching performance to variations in key design and operating parameters. The key parameters investigated were irrigation rate, ferric concentration, total iodide concentration, temperature, and column height.

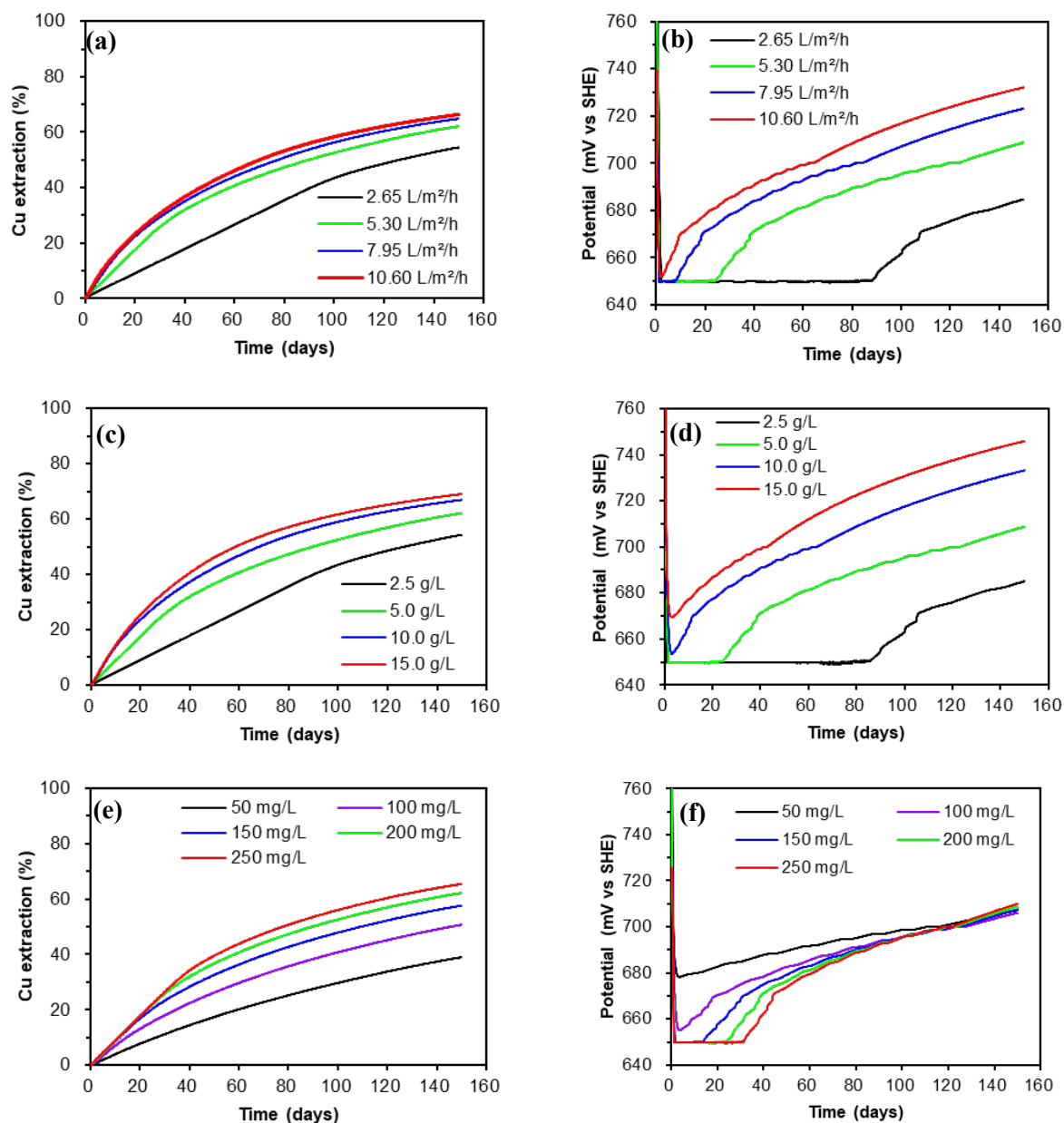
Figures 5 a and b show the leaching performance in response to changes in the irrigation rate. The results indicate the existence of an optimal irrigation rate, beyond which the leaching performance can no longer be improved. A similar conclusion can be drawn on the ferric concentration that the copper extraction increased with the ferric concentration up to a certain value, as shown in Figure 5c and d. This corresponds to the potential window within which the oxidized forms of the iodine species are predominant and act as the oxidants for chalcopyrite dissolution.

To further improve the leaching performance, a higher concentration of iodide must be added, along with sufficient ferric concentration or irrigation rate to maintain the potential value in the suitable range (Figure 5e and f). Raising temperature leads to an increase in the copper extraction, as shown in Figure 5g and h. The increase in the column height lowers the rate of copper extraction because of the decreasing amount of the oxidant available per unit mass of ore (Figure 5i and j). Reduction in the particle size is thought to increase the degree of liberation of chalcopyrite, thereby enhancing chalcopyrite dissolution (Winarko et al., 2022). However, the actual choice of these design and operating parameters must take into consideration the geotechnical stability of the heap and the possible occurrence of iron precipitation at higher ferric concentrations.

Figure 5b, d, f, h, and j show that the solution potential drops to 650 mV vs SHE when the ratio of ferric concentration to the mass of ore is low. Under these conditions, the column is segregated into two regions, the iodine leaching region and the conventional ferric leaching region. The former appears when the solution potential is above 650 mV vs SHE, whereas the latter occurs at potentials lower than 650 mV

MODELING THE IODINE-ASSISTED CHALCOPYRITE LEACHING IN FERRIC SULFATE MEDIA
IN RESPONSE TO OPERATING PARAMETERS IN HEAP LEACHING

vs SHE. The iodine leaching region expands as leaching progresses. When the dissolution becomes slower over time, the potential of the pregnant leach solution (PLS) gradually increases.



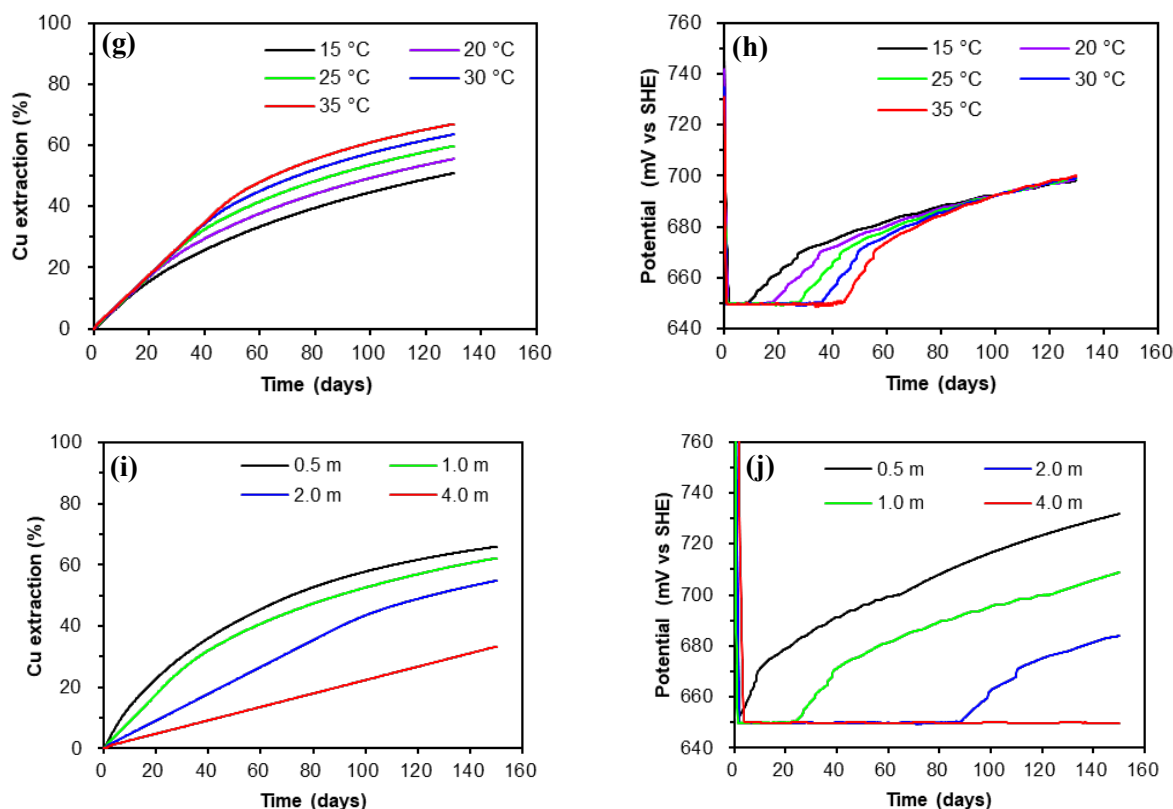


Figure 5: The simulation of copper extraction and solution potential trends for column leaching under the following conditions (5.30 L/m²/h of irrigation rate, 5 g/L of ferric concentration, 200 mg/L of total iodide concentration, 25°C, and 1m in height), except the parameter under investigation: different irrigation rate (a-b), different ferric concentration (c-d), different total iodide concentration (e-f), different temperature (g-h), and different column height (i-j)

Conclusions

Chalcopyrite dissolution was enhanced in the presence of iodine in ferric sulfate media. The kinetic modeling of the column tests shows that the performance of the iodide-assisted chalcopyrite leaching is dependent on the solution potential. When ferric was insufficient, the potential measured in the PLS dropped and maintained stable at 650 mV vs SHE, at which the leaching kinetics was very slow. In this case, the predominant iodine species becomes iodide as the solution percolates towards the end of the column; thus, the leaching process becomes conventional ferric sulfate leaching. Therefore, it is important to supply sufficient ferric to maintain the potential in the entire column within the range where diiodine or triiodide are predominant. All copper extraction curves show that the leaching did not plateau, meaning that the leaching would proceed as long as a fresh ferric solution is continuously supplied.

The simulations indicate that there is an optimal value of irrigation rate and ferric concentration, beyond which further increases do not lead to significant improvements in the copper extraction. To further improve the leaching performance, a higher concentration of iodide must be added, along with sufficient

ferric concentration or irrigation rate to maintain the potential value in the suitable range. In general, the copper extraction increases with increasing temperature. The actual choices of these design and operating parameters must take into consideration the geotechnical stability of the heap and the possible occurrence of iron precipitation at higher ferric concentrations.

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<https://doi.org/10.1016/j.hydromet.2020.105481>

Updated Statistics on Leaks per ha, Based on 22 M m² ELL Surveys Conducted in Six Different Countries over the Last Decade

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Abstract

Technology is constantly evolving at a high pace, decade after decade. This includes methodologies and equipment for geomembrane liner fabrication and installation, as well as for testing, both before and after placement on mining facilities. In fact, geomembranes are relatively new compared to other lining materials such as bitumen, clay, concrete and asphaltic concrete (Giroud and Bonaparte, 1989).

Geomembrane liners have been used significantly in the mining industry since about 1970 for lining solution and evaporation ponds, tailings impoundments and heap leach pads. The most extensive use of geomembrane liners has been, and remains, in the construction of evaporation ponds, starting in the early 1970s, and heap leach pads starting in the late 1970s.

Nowadays, the same questions asked forty years ago are still valid: how critical are the defects in the geomembrane due to punctures, inadequate seams, cracks, wrinkles, etc.? What range of “reasonable” leakage rates should be considered in regulations and specifications? How many holes should be expected when following state-of-the-art quality assurance practices? Is it possible to get a zero-leak facility?

Based on a number of Electric Leak Location (ELL) projects overviewed by the authors over the last ten years, and as an update to a previous research study conducted five years ago by some of the same authors, this paper provides a detailed analysis of the findings of ELL surveys of over 22 million square meters (M m²) of geomembrane liner installed in six different countries, including metal mining impoundments, lithium evaporation ponds, and other types of facilities. It also provides a benchmark to current liner installation practices and shows a number of leaks detected on each site, classified by type of facility, ELL methodology, geomembrane material, country, and liner installation stage.

In order to look for a trend line, current ELL results are compared with those of previous papers written by other authors in the past.

New facilities, where liner is installed following industry-standard construction-quality assurance practices, should expect, at least, not to exceed the average number of defects per hectare presented in this paper. Consequently, the statistics presented in this research work could be used to evaluate the construction quality of any new or existing facility in comparison to the 219 cases included in this study.

Introduction

Despite public perception, in the technical community it is commonly acknowledged that installed geomembranes typically contain holes, which in most cases can be located for repair using electrical leak location (ELL) technologies (Gilson-Beck, 2019; Thiel, 2013).

The main goal of this paper is to offer project record-based statistics to be used as a benchmark for predicting the number of holes to be expected under different scenarios, with an emphasis on the need for Construction Quality Assurance (CQA) and ELL during the installation of geomembrane liners. Reinforcement of these good practices certainly favours a reduction in the number of holes and mitigates the level of risk for future shutdown of facilities.

Data from previous research indicates that the average number of holes reported in sites where CQA was performed during the installation stage is about 0.5 per hectare, whereas the number of holes without CQA can increase up to 107.11 per hectare (Forget et al., 2005; Beck, 2006; Prota et al., 2017). A summary of these values is presented in the following table, where WL indicates Water Lance methods:

Table 1: Hole frequency from different authors

Type of facility/application/ construction phase	Holes per hectare with CQA	Holes per hectare without CQA	Source
Overall leak frequency	2.5	>25.0	1
Exposed geomembrane	4.0	22.0	2
Covered geomembrane	0.5	6.0	2
Pond	4.0	22.0 (small 10 mm ²)	3
Pad with both WL and dipole	4.0 (small 10 mm ²)	22.0 (small 10 mm ²)	3
	0.5 (big 1,500 mm ²)	1.0 (big 1,500 mm ²)	
Pad with WL only	4.0 (small 10 mm ²)	22.0 (small 10 mm ²)	3
Pad with dipole only	3.0 (small 10 mm ²)	15.0 (small 10 mm ²)	3
	0.5 (big 1,500 mm ²)	3.0 (big 1,500 mm ²)	
Leak frequency (holes/ha)	1.95	107.11	4
Leak frequency (holes/ha)	0.128		5

Sources: Giroud and Bonaparte, 1989; Forget et al., 2005; Beck, 2006; Prota et al., 2017; Gilson-Beck, 2019.

Methodology

The authors have shared their own field project experience, resulting from 22.5 million square meters of geomembrane liner surveyed over 219 ELL projects, where key data from each of them has been recorded, including type of facility, surveyed area, number of leaks and/or electrical anomalies (e.g., potential leaks) detected, among other aspects.

The analysis includes projects executed in Argentina, Bolivia, Chile, the Dominican Republic, Peru, and the USA with the use of ELL technologies such as the Dipole method (ASTM D 7007) (ASTM Subcommittee D35.10, 2016), Arc Testing (ASTM D 7953) (ASTM Subcommittee D35.10, 2020), the Water Lance method (ASTM D7703) (ASTM Subcommittee D35.10, 2022), the Water Puddle Method (ASTM 7002) (ASTM Subcommittee D3510, 2022) and hydroGEOPHYSIC'S proprietary ELL Methodology for geomembranes covered with a highly conductive solution.

The facilities in the analysis include tailings dams, leach pads, water ponds, solutions ponds, evaporation ponds, swimming pools, biodigestors, landfills, and other facilities, surveyed from 2010 through 2022, some of them with double or triple liner systems.

The analysis has also taken into consideration whether the liner system has been installed under proper CQA supervision, and if covered or exposed geomembrane liner has been evaluated.

Finally, the results have been compared with previous papers, both from others and from the same authors.

Table 2: Paper database details

Aspect	Details
Number of projects	219
Surveyed area (m ²)	22,561,911
Countries	Argentina, Bolivia, Chile, Dominicana Republic, Peru, USA
ELL technologies	Exposed geomembrane Arc testing, water lance Covered geomembrane Dipole on soil-covered materials, Dipole on water-covered materials, HGI alternative proprietary dipole methodology
Type of facility	Tailings dams, leach pads, water ponds, process ponds, solution ponds, evaporation ponds, landfills, biodigesters, swimming pools

Data and discussion

Overall results

The overall leak frequency is 9.45 holes per ha, based on the total surveyed area and the total number of leaks detected, as summarized in Table 3.

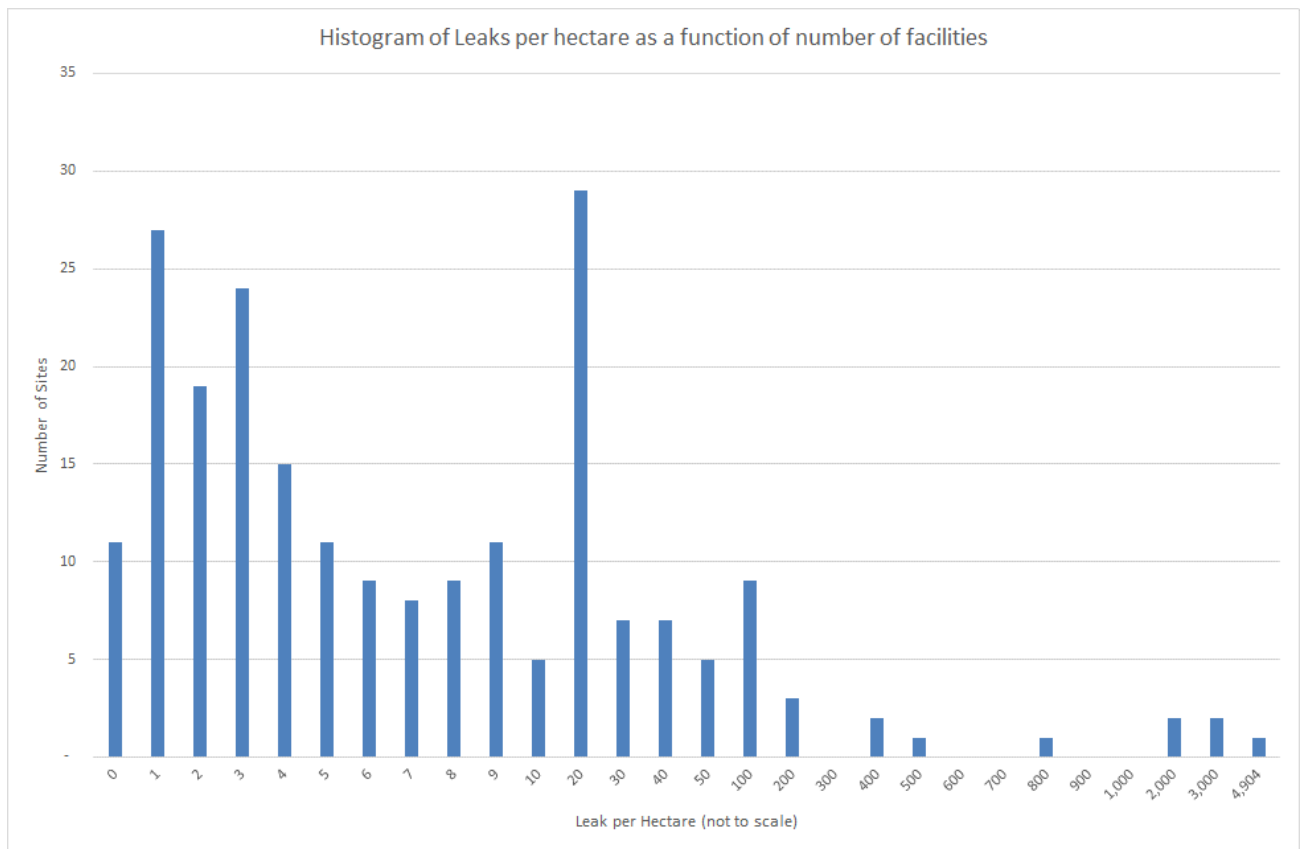
Table 3: Overall leak frequency results

Number of surveys	Total surveyed area (m ²)	Number of leaks detected	Leaks per ha (average)	Max. (leaks/ha)	Min. (leaks/ha)
219	22,561,911	28,221	9.45	4,903.8	0.0

Although the average value of leaks per ha is close to those obtained by previous authors, some specific cases present a significant deviation from those figures: six projects registered over 1,000 leaks per ha, with a maximum of 4,903 leaks per ha. It should be noted that those sites are very small, with installed geomembrane surface areas ranging from 100 m² to 1,100 m², where strict CQA procedures were not followed.

This in turn triggers a relatively high value for the standard deviation (512.4), as a result of the significant dispersion of the leak-per-ha value across the different projects.

The following graphs illustrate the dispersion in the number of leaks per hectare:

**Figure 1: Histogram of leak-per-hectare frequencies**

UPDATED STATISTICS ON LEAKS PER HA, BASED ON 22 M M² ELL SURVEYS
CONDUCTED IN SIX DIFFERENT COUNTRIES OVER THE LAST DECADE

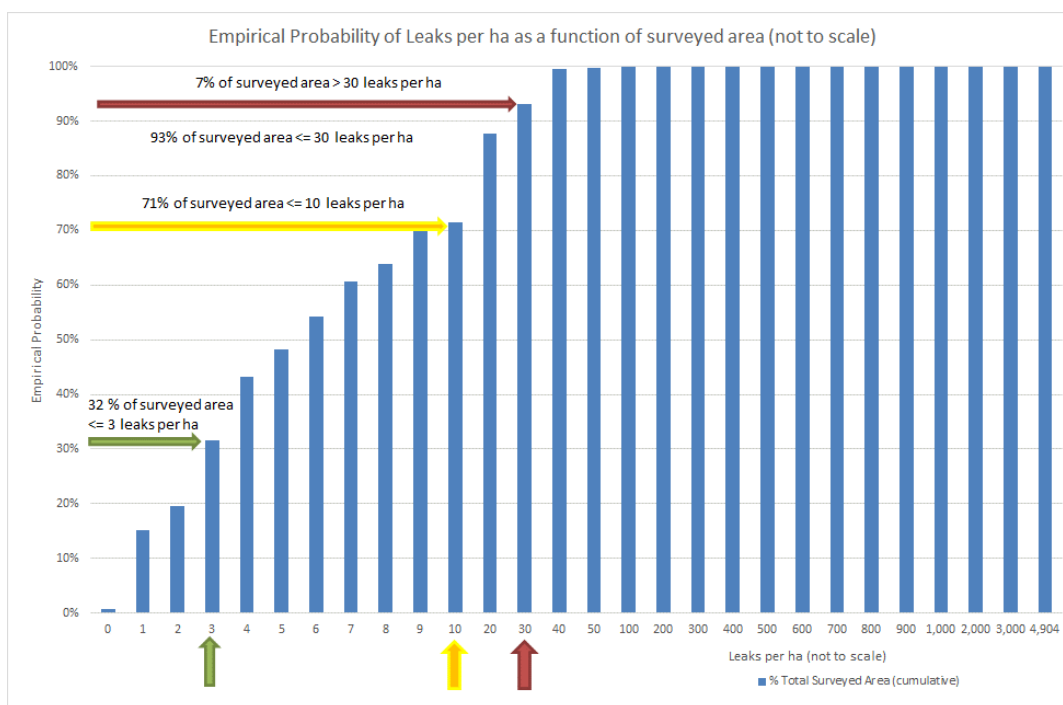


Figure 2: Empirical probability of leak

Based on surveyed data, approximately 32% of the installed geomembrane should have less than 3 holes per ha; around 71%, less than 10; 93%, less than 30, and 7% might exceed 30 holes per ha.

Results grouped by categories

In the following tables and figures, we provide a detailed evaluation of the field data.

CQA considerations

In addition to the big dispersion driven by a few projects with a high hole frequency, as outlined by previous investigations, there is a significant distinction between the projects with CQA and those with no CQA.

Table 4: Leaks classified by quality control status during construction

Leaks per ha category	With CQA	Without CQA
Exposed geomembranes (average)	7.4	28.3
Covered geomembranes (average)	1.4	3.2
Total exposed and covered geomembrane (average)	6.6	26.9
Maximum	1,056.3	4,903.8
Minimum	0.0	0.0

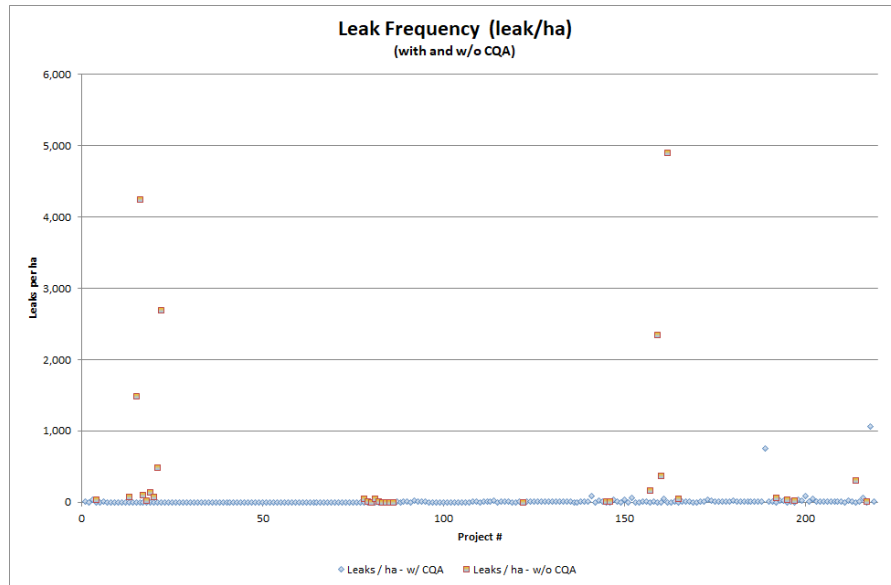


Figure 3: Leak frequency categorized by CQA

In the next graph the vertical axis is switched to a log scale, in order to get better visibility of the sites with less than 100 leaks/ha.

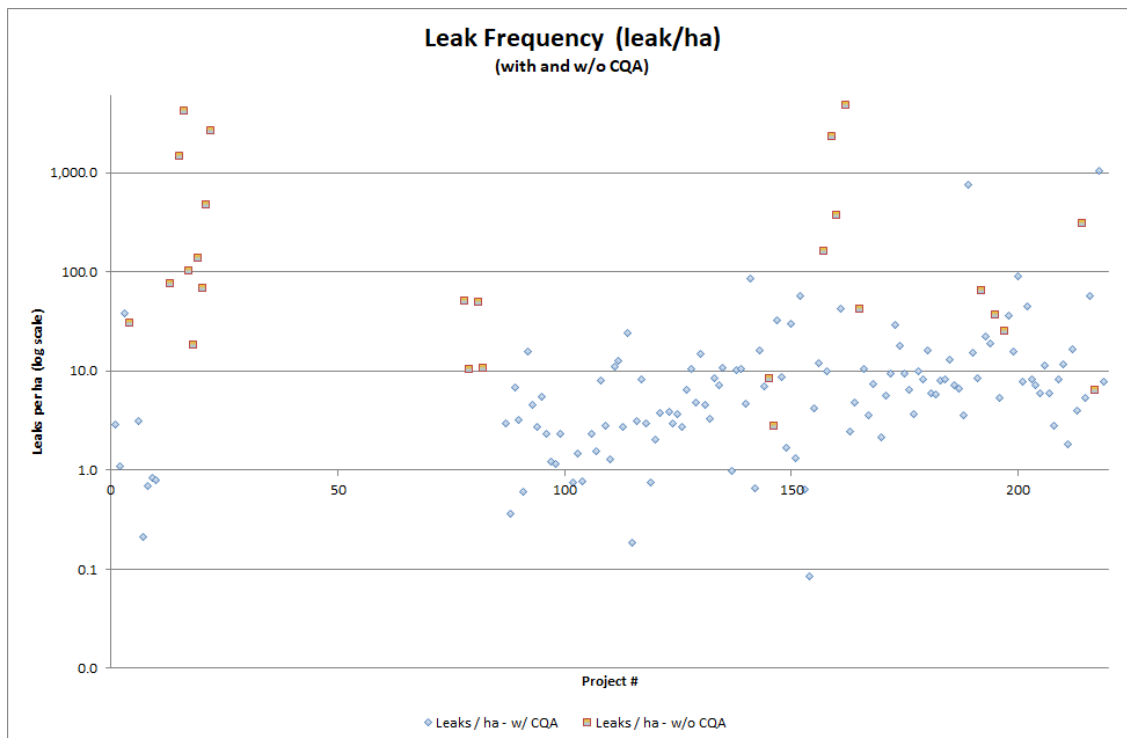


Figure 4: Leak frequency categorized by CQA (log scale)

Note: When using the log scale, it is not possible to include data points with a zero value on the y axis; therefore, projects with zero leaks are not represented in the upper graph.

UPDATED STATISTICS ON LEAKS PER HA, BASED ON 22 M M² ELL SURVEYS
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The majority of the projects with no CQA during liner installation are above the 100 leak/ha frequency, while most of the projects with CQA are below that threshold.

Results by ELL standards

Table 5: Leaks classified by ELL technology

Technology	Surveyed area (m ²)	Number of leaks detected	Leaks per ha (average)	Number of facilities	Standard deviation
Arc testing	18,106,977	20,177	11.1	118	679.2
Arc testing and water lance	357,481	457	12.8	4	18.9
Dipole	2,172,933	337	1.6	41	11.2
Dipole in water	40,000	7	1.8	1	–
Alternative dipole	1,884,519	345	1.8	55	10.8
Total	22,561,911	21,323	9.5	219	

Direct methodologies, such as the arc testing and the water lance, show a much higher leak detection frequency than the indirect ones, such as the dipole. However, and based on the authors' experience, the size of the leaks detected with the dipole might be extremely large (over several cm²), while the ones detected with arc testing or the water lance are usually below 1 cm².

Results by project location

Table 6: Leaks classified by project location

Country	Surveyed area (m ²)	Number of leaks detected	Leaks per ha (average)	Number of facilities	Standard deviation
Argentina	12,083,006	8,714	7.21	71	614.4
Bolivia	1,185,372	642	5.42	1	n/a
Chile	6,071,539	10,947	18.03	78	620.0
Dominican Republic	101,172	7	0.69	1	n/a
Peru	1,466,974	680	4.64	15	21.7
USA	1,653,848	333	2.01	53	10.9

Note: Data for Chile includes both QC and QA survey results, while data from the other countries is only QA. The authors assume this explains most of the increase in the leaks frequency detection, when compared to other countries in this paper.

When analyzing by country, there is significant variation in the leak-per-ha frequency, presumably driven by differences in liner installation procedures, CQA or QC, and/or enforcement practices.

*Results by facility type***Table 7: Leaks classified by type of facility**

Type of facility	Surveyed area (m ²)	Number of leaks detected	Leaks per ha (average)	Number of facilities	Standard deviation
Tailings dam	221,360	510	23.04	5	32.4
Leach pad	4,003,159	1,306	3.26	31	2.0
Pond	4,541,027	3,363	7.41	104	35.2
Evaporation pond	13,403,477	15,728	11.73	62	649.9
Landfill	6,000	5	8.33	1	n/a
Other	386,887	411	10.62	16	1,303.6
Total	22,561,911	28,221	12.5	219	

Over the last ten years, the number of sites and total installed area for lithium evaporation ponds has consistently increased, except for 2020, and has become the predominant market compared to other minerals and type of facilities.

*Results by liner material***Table 8: Leaks classified by type of geomembrane liner material**

Material	Surveyed area (m ²)	Number of leaks detected	Leaks per ha (average)	Number of facilities	Standard deviation
Bituminous	68,028	626	92.02	4	131.3
HDPE	5,029,201	7,825	15.56	65	667.1
PVC	1,561,201	2,608	16.71	9	9.4
Polyurethane	578	61	1,056.28	1	—
LLDPE	12,889,382	8,078	6.27	55	4.8

Note: The material specification was not available for some of the projects; therefore, the total surveyed area in this table is smaller than the total in Table 1.

In the installation of most of the lithium evaporation ponds surveyed by the authors, LLDPE geosynthetic liner is being used, which drives the dominant position of this material in these statistics.

This material is offering the lowest leak-per-ha leak average value.

UPDATED STATISTICS ON LEAKS PER HA, BASED ON 22 M M² ELL SURVEYS
CONDUCTED IN SIX DIFFERENT COUNTRIES OVER THE LAST DECADE

Chronological evolution of ELL

Table 9: Surveyed area classified by year

Year	Surveyed area (m ²)	Number of leaks	Leaks per ha
2010	40,000	7	1.75
2011	–	–	N/A
2012	1,093,904	819	7.49
2013	1,281,980	1,023	7.98
2014	2,165,304	1,329	6.14
2015	604,265	599	9.91
2016	1,541,910	4,927	31.95
2017	1,338,225	3,310	24.73
2018	1,584,247	677	4.27
2019	2,912,260	2,568	8.82
2020	601,772	85	1.41
2021	6,206,046	3,719	5.99
2022	5,068,769	4,374	8.63

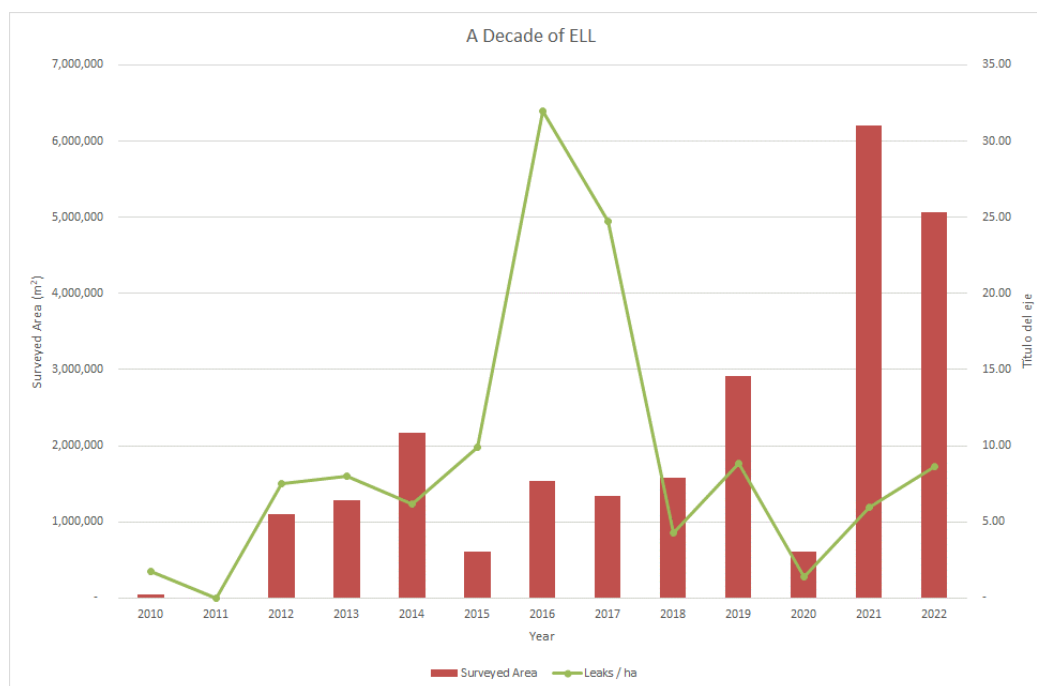


Figure 5: ELL annual data

Note: The 2022 annual survey and leak data was extrapolated based on 2022 S1 actual figures.

Although there is no linear correlation for any of the variables, there is a clear positive trend on the number of square meters surveyed per year, except for 2020, when the COVID-19 pandemic impacted all activities worldwide. In terms of leaks per ha, it is very difficult to establish a defined trend, with consecutive peaks and valleys. This is not what would be expected for an activity where the installation companies are supposed to be going up the learning curve, and might rather be associated with the lack of experience of the new welders joining the labour market every year, in the clearly demanding sector of geomembrane liner installation, mainly driven by lithium exploration and production booming.

Conclusion

Multiple conclusions can be drawn from the above results.

Significant variations in the leak-per-ha occurrence have been detected when evaluating the available data from different perspectives, which indicates that several parameters such as material type, proper QC/QA, project location, facility type, liner material, geomembrane application and ELL methodology can lead to very different results.

Even when proper QC and QA practices are observed, leaks may be present in installed geomembrane liner, with an average of 6.6 leaks/ha. This parameter rises to 26.9 leaks/ha when no proper QC/QA has been conducted.

From a statistical perspective, the leak-per-ha function cannot be defined as normal distribution and in fact shows a significant dispersion.

The results might indicate that installation quality varies significantly from one country to another, and apparently in some countries installation practices are not improving over the years.

The rapid expansion of lithium extraction projects might not be accompanied by an appropriate increase in the number of trained and qualified field engineers and technicians, whose work is key for the zero-leak goal.

Leak-per-ha data recording methodology presents significant variations among consultants, which makes it difficult to group results from the surveys of different companies and obtain wider and representative statistics based on comparable and compatible data. It might be useful to develop a voluntary uniform data recording protocol, in order to facilitate future updates of this kind of statistic.

Acknowledgements

The authors would like to thank each of their clients for their continuous support and their belief in the value of ELL as a tool to increase productivity, reduce costs and protect the environment.

The development of the ELL to the state-of-the-art technology level in some countries would not have materialized without the help of Abigail Gilson-Beck, who generously shares her knowledge and provides valuable support as needed.

We would also like to acknowledge the commitment of our employees towards quality improvement even under extreme and adverse field conditions.

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New Excess Flow Valves for Preventing Side Slope Washouts

Zachary Felkey, Newmont Cripple Creek & Victor, USA

Abstract

Newmont's Cripple Creek & Victor (CC&V) mine operates two valley fill heap leach pads. One of these, VLF1 (Valley Leach Fill), is a 372M ton leach pad that is over 700 feet (213 m) deep in some areas. As the leach pad was built, it has created over 12M square feet (1.11M m²) of side slope area, which comprises nearly 80% of its leachable area. While these side slopes have been leached intermittently since 1995, they remain a potential contributor to CC&V's production.

There is limited access to VLF1's side slope areas, with most benches just wide enough to fit a pickup truck. This makes them difficult to leach. Moreover, there is a high potential for washouts caused by irrigation drip lines coming loose from adaptors. These washouts can affect multiple benches and access roads on the leach pad, creating safety and environmental concerns and generating costly remediation efforts and production loss.

To address this problem, leach pad metallurgist Zachary Felkey has developed two new excess flow valves (EFVs). EFVs isolate flow when the flowrate through a pipe increases beyond a specified set point. The "Check" and "Eddy" excess flow valves utilize two new mechanisms to achieve flow isolation. These EFVs allow for efficient operation by providing quick interconnections to drip line adaptors, while providing a reliable and robust safeguard to reduce the potential for washouts.

Introduction

Plumbing and installing drip line on the side slopes of heap leach pads presents many challenges, depending on the environmental and physical conditions surrounding the slopes. VLF1 is located at a 9,500 ft (2,900 m) to 10,400 feet (3,200 m) elevation in the Rocky Mountains of Colorado. It presents particular environmental challenges including high wind speeds, thunderstorms, snow, and freezing conditions. VLF1 was also stacked to maximum extents for many years, which has resulted in narrow benches at roughly 100 feet (30 m) elevational lifts.

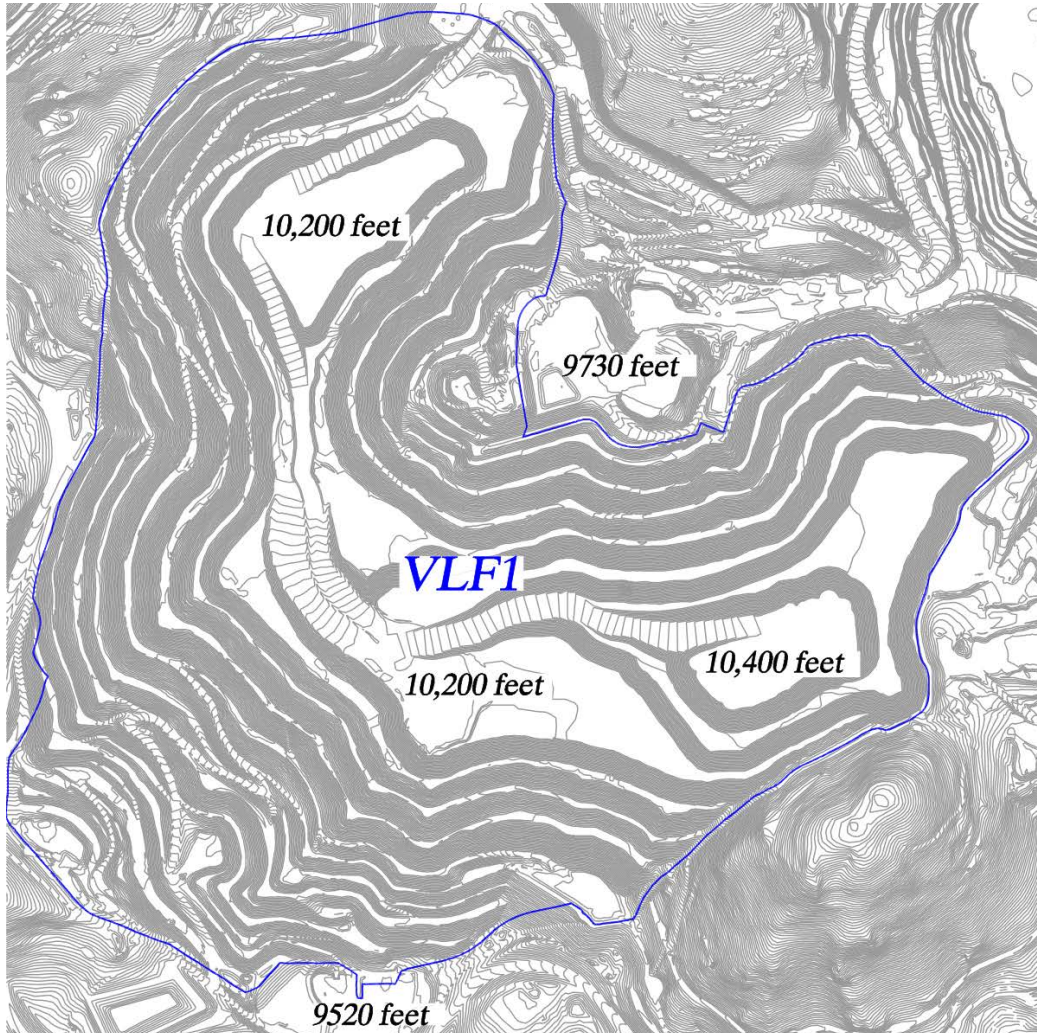


Figure 1: Plan view of VLF1

The process team at CC&V have developed many methods for laying out drip tube irrigation, including winch systems, remote-controlled skid steers, and rebar anchors. Each of these systems has advantages, but the end result is drip tubes on a slope that are exposed to high wind speeds or, if buried, strain from the weight of ore pulling down on the drip tubes. With this tension comes the possibility that drip tubes may be pulled off their adaptors (pop-off).

The resulting high flow rate from a pop-off on a narrow bench side slope panel can either cause a small washout directly below the adaptor, or build a large pond on the surface and result in a large washout. This may cause further pop-offs on either side of the adaptor as the moving ore pulls on additional drip tubes, or in some cases pulls the feed header to the point of breaking.

In either case the resulting damage could cause a geotechnical safety hazard or an environmental release off liner. Remediation including regrading the slope or access road is costly and time consuming, which in turn can cause delayed production.

While training pad crew members to be vigilant in their installation of the drip tube to the adaptors eliminates some of the hazard of pop-offs, inevitably, there will be a poorly installed drip tube that will result in a pop-off. The necessity to leach slide slopes despite these risks facilitated the development of alternative connections of the drip tube, such that pop-offs would not cause washouts. Two alternative connections were invented that act similarly to traditional excess flow valves, which isolate flow when the flowrate through a pipe increases beyond a specified set point.

The Check excess flow valve

The first EFV developed uses a swing check valve and a bushing, which is modified to accommodate a drip tube adaptor. The modified bushing is installed into the feed header in place of a typical adaptor using a close nipple. A coupling and swing check valve are attached to the bushing such that the drip line adaptor fits inside the coupling. For installation, the drip tube is fed into the swing check valve and coupling part while it is separated from the bushing. The drip tube displaces the flap of the swing check valve into an open position, thus holding the flap open when the drip tube is installed onto the drip line adaptor. The coupling and swing check valve part is then threaded onto the bushing, completing installation.

When the drip tube comes off the drip line adaptor due to pressure or stress, the drip tube is pushed past the flap in the swing check valve by the flow of solution, allowing the flap in the swing check valve to close and isolate the flow.

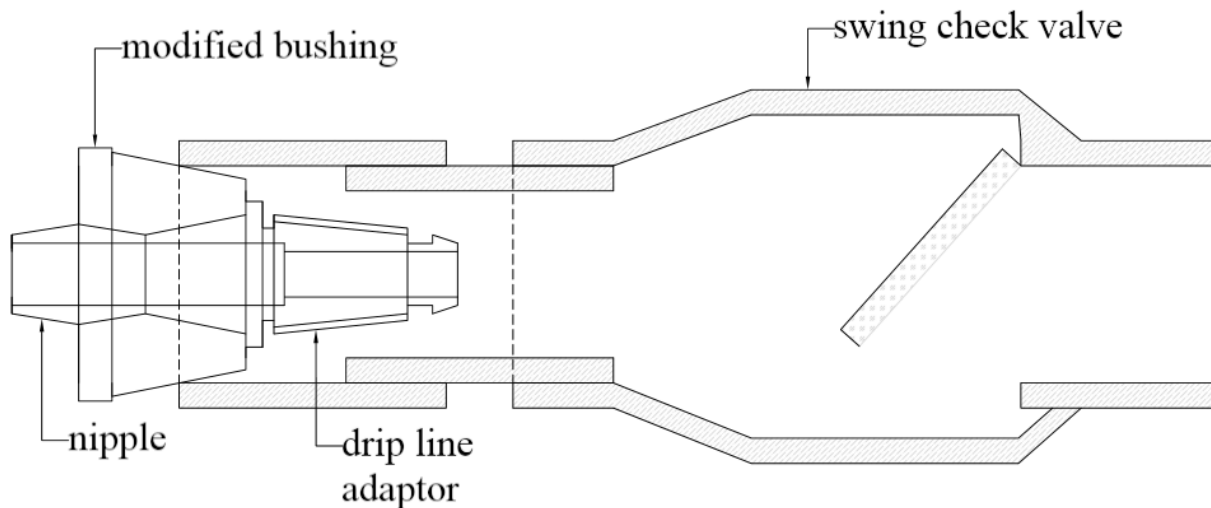


Figure 2: Design of Check excess flow valve



Figure 3: Photograph of the Check excess flow valve

Check EFV findings

The Check EFV is effective in its simplicity of design. In all trial cases, the drip tube would either entirely exit the swing check valve causing it to close, or be lodged in the swing check valve flap causing some solution to flow from the end at a low rate. Full implementation for all side slope panels was commenced in 2017 and continued for about two years before the use of injection wells and sprinklers for VLF1 production overcame the need to leach side slopes.

Before implementation of the Check EFV, side slope leaching required continuous monitoring for washouts from pop-offs that occurred almost daily. This was especially true for panels that were recently plumbed and needed to be subjected to fluctuations in plant feed pressure and settling of ore. After implementation, close monitoring of side slope panels continued, but the occurrence of washouts was reduced significantly. Over the two years in use, the Check EFVs prevented numerous washouts from occurring.

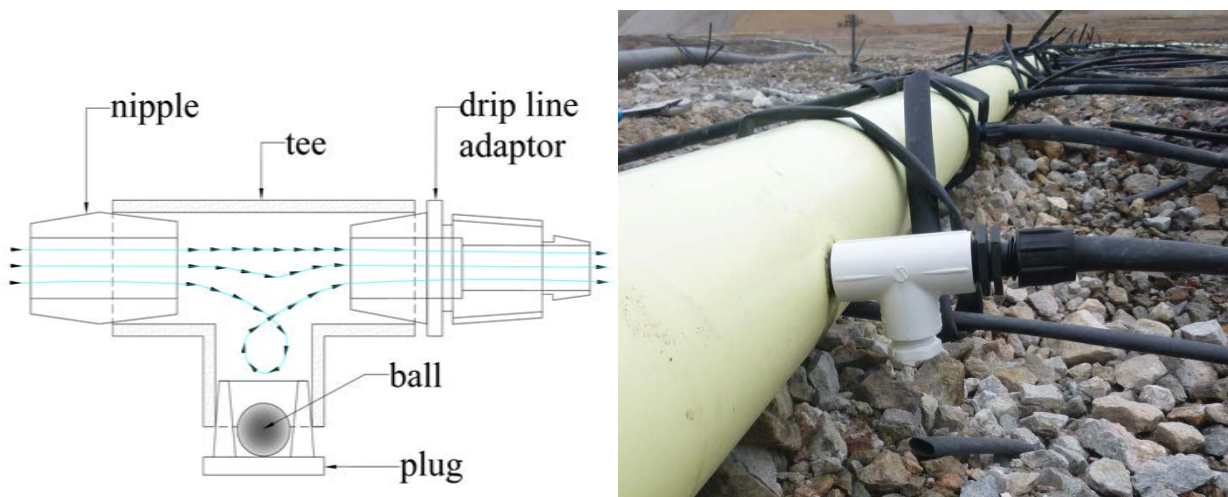
A design flaw was discovered in that the swing check valve would freeze if filled with solution after closing from a pop-off. The frozen swing check valve would crack and spray solution, requiring the swing check valve to be replaced. This was remedied by drilling a small hole in the swing check valve of each assembly. If the swing check valve filled because of a pop-off, the hole would allow solution to flow at a low rate and prevent freezing of the assembly.

Other disadvantages to the design include the necessity for modifying the bushing and assembling the device. A bushing tapped to accommodate an adaptor is not readily available on the market and a method for tapping bushings efficiently was required. A steel template that could hold 12 bushings securely was used to quickly tap them. The cost of each Check EFV is approximately \$14. This EFV is considered optimal for plugging solutions since its mechanism is not affected by debris or scale.

The Eddy excess flow valve

The second EFV developed is a tee or wye fitting with a ball in the side outlet of the tee that is plugged off. The tee is installed into the feed header using a close nipple and an adaptor is threaded into the opposite end. When a pop-off of the drip line occurs, the velocity increases through the tee creating an eddy current, or swirling motion; this is sufficient to lift the ball into the straight section of the tee. The ball then lodges into the inlet side of the adaptor, isolating flow.

The size and material of the ball as well as the distance of the ball from the straight section of the tee can be changed for the desired maximum flow rate and solution being used. Upon activation of the Eddy EFV, the ball can be reset by isolating the feed header. If the EFV is faced slightly upward, the ball will roll back to its original position when the line pressure is removed. The Eddy EFV design is a departure from typical excess flow valves in that the flow path of the fluid is minimally affected. No directional change occurs, and no moving parts operate directly in the flow path. The design and operation of the Eddy EFV may be appropriate to other applications beyond drip tube plumbing.



Figures 4 and 5: Design and photograph of the Eddy excess flow valve

Eddy EFV findings

The Eddy EFV operated consistently in all trial cases. Due to the timing of development and testing, the Eddy EFV was never fully implemented at CC&V. However, the test panel on which they were installed operated for six months. During this time there were several successful activations of Eddy EFVs and upon testing of operability at this point, the EFVs were found to still be successful. The small amount of scale that had built up in the EFVs did not have any negative effect on their operation.

A concern that the EFVs may engage prematurely if the panel was turned on too fast was eliminated in testing as the eddy effect of the solution filling the drip tubes was not sufficient to lift the ball. The cost of each Eddy EFV is approximately \$4. This design is considered an improvement from the Check EFV in

that its size is considerably less, and they are easier to install. However, a concern remains that high scaling and plugging solutions could prevent the ball from lifting.

Conclusion

Washouts caused by drip tube lines separating from adaptors are a significant safety and environmental hazard that can require costly remediation and can delay production. At CC&V, these washouts were a near daily occurrence, requiring continuous monitoring. After implementation of the Check EFV, the occurrence of washouts was reduced drastically.

The Check and Eddy excess flow valves can provide the ability to safely leach side slopes that require feed headers to be near the crest of narrow benches. The Check EFV is advantageous in its simplicity of operation and its ability to operate independent of the plugging and scaling potential of the solution used.

The Eddy EFV is a low profile and cost-effective device for cleaner solution types. The Eddy EFV can also be adjusted in design to accommodate varying flow setpoints for operation and fluid types. It is the hope of the inventor that this design can be used for multiple excess flow valve applications beyond drip tube irrigation that require a simplistic and less flow-path-intrusive design.

Altan Tsagaan-Ovoo Mine's Mine Closure Concept Plan and Results of the Trials and Research

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Abstract

The influence of the mining sector on the Mongolian economy is huge. For example, 24% of gross domestic product, 72% of total industrial factory output, 89% of exports, and 74% of foreign investment is related to this sector. Of these, the gold mining sector is the largest contributor.

The Ministry of Mining and Heavy industry has forecast that the gold mining sector of Mongolia will reach one billion dollars in profit every year for the foreseeable future. According to this forecast and plan, the alluvial deposit dominated gold mining sector will be shifted into a primary deposit dominated gold mining sector within the next five years.

The use of cyanide chemical technology was the primary gold extraction method for the mining industry during the last 100 years. However, for the last ten years, there has been research and use of biotechnological extraction methods for gold. The use of biotechnological gold extraction methods from ore bodies in Mongolia is difficult because of weather conditions and the local climate.

Currently, there are five companies using cyanide on primary gold deposits in Mongolia. In accordance with the national plan, gold primary deposit mining will increase. Because of the increase, environmental issues, in particular mine closure, will become a key consideration in mine planning. Of these five companies, only a few are focusing on mine closure and conceptual plans as part of early-stage development.

Steppe Gold LLC (Steppe Gold) has become the sixth company to use cyanide for gold recovery in Mongolia by starting the Altan Tsagaan-Ovoo gold-polymetallic primary deposit project in 2019. Steppe Gold is planning and implementing several actions directed at mine closure during the operational years of the mine, in order to be responsible and pro-active with respect to the ongoing development and closure planning for the project. One of these considerations is the mine closure concept plan and the development

of a detailed mine closure plan based on international standards, as well as further research and analysis based on site-specific conditions.

The Altan Tsagaan-Ovoo mining project is located in the Mongolian vast steppe ecosystem, a Central Asian part of the Euro-Asian steppe dominated steppe where the nomadic culture has remained intact. This Dornod-Mongolian steppe is one of the main characteristics of the region and has been very important for the sustainability of the local ecosystem that has supported the nomadic tribe culture for the past one hundred years. Therefore, it is necessary to plan and implement Altan Tsagaan-Ovoo mine's closure in cooperation with stakeholders by making plans that contribute to the local ecosystem's diversity, growth, and development.

The environmental team from Steppe Gold developed a mine closure and preliminary rehabilitation concept plan concurrent with start of mine operations and started research and trials along with the plan.

Focuses of Altan Tsagaan-Ovoo mine's closure plan

Steppe Gold's mine closure concept plan contributes to, and is integral with, the mine operations and actions of the current mining phase. Some closure activities will be completed during the operational phase. Moreover, the necessary reclamation funding is being set aside by based on detailed estimates of the mine closure costs. Altan Tsagaan-Ovoo mine's closure plan has the following focuses:

- Management of the mine operation through the end of the mine life, closure and post-closure actions, and closure rehabilitation.
- Selection of the most appropriate mine closure concept based on research and stakeholder input. Depending on the project's infrastructure and construction, this closure concept may include tourism, semi-farming operations, and traditional livestock grazing.
- Management of closure, future infrastructure usage, and stakeholders' participation.
- Actions after closure, including post-closure monitoring.
- Site cleanup, removal, and detoxification of all chemical remnants, chemically contaminated equipment, and other materials.
- Tailing management and closure plans.
- Calculation and estimates of the required funds for mine closure, and establishment of the fund as required by local regulations.

Based on the team's research and results, Steppe Gold are planning to implement an example mine closure in Mongolia by contributing to sustainable development of the local ecosystem in a way that provides balance and diversity, and promotes the social and economic well-being of the local citizens. The

plan is to improve the plan during the operational phase and implement some of the closure actions during the operations phase.

These are some of the trials that will be done as part of the research:

1. Tree nursery trials.
2. Trials of seed germination of local pasture perennial plant seeds and planting.
3. Trial to reduce cyanide concentration in the heap leach pad ore by recirculating water.
4. Vegetation trial on cyanide-containing ore stockpile.

Management is planning to carry out the above-mentioned trials for five years, and this research paper contains some of the preliminary results of those trials.

Two stages of the mining operation spanning eleven years are planned. Oxidized ore will be processed using heap leaching during the first five years. Fresh rock will be processed with flotation processing technology from the sixth year to the end of mine life (Knight Piesold, 2018). The mine's infrastructure and construction work began in 2018 and was completed in March of 2020. The mine is now fully operational.

Geography and weather

The Altan Tsagaan-Ovoo mine is located in the vast steppe of Mongolia dominated by the East-Mongolian steppe ecosystem. The mine site is located 650 kilometers east of the capital city Ulaanbaatar, 152 kilometers west of Choibalsan city, the capital city of Dornod province.

This region has low snowfall during the winter, and air humidity is low, while the wind speed is high during the spring, and the temperature difference between the seasons is great. The lowest air temperature reaches -45°C and the maximum air temperature reaches $+39.8^{\circ}\text{C}$. The annual average temperature is -0.4°C . The coldest time of the year is in January, which has an average temperature of -21.8°C , while the hottest time of the year is in July with an average temperature of 19.1°C . The average annual precipitation is 250 mm. The mean average of wind speed for many years is 2.5 to 3.0 m/sec, with the highest wind speed in most months of the year being more than 15 m/sec reach. The wind reaches speeds of up to 28 m/sec in the spring (Make Green, 2019).

Tree nursery trial results

A total of 362 hectares of land will be disturbed throughout the mining operation. As of 2022, 115 hectares of land has been disturbed.

Steppe Gold plans to rehabilitate the land disturbed from mining operations using local perennial plants and trees, making sure rehabilitation work is suitable for the environment and weather conditions of the region. For this reason, a tree nursery was established in 2018 and the trial work was started.

Methodology

Due to the severe weather conditions in Mongolia, trees and bushes are planted in both spring and autumn seasons. A total of eight species of woody and bush plants were planted in spring and autumn, namely blue honeysuckle (*Lonicera caerulea*), Siberian currant (*Ribes diacanthum*), blackcurrant (*Ribes nigrum*), hackberry (*Prunus asiatica*), Siberian peashrub (*Caragana arborescens*), sea-buckthorn (*Hippophae rhamnoides*), *Malus pallasiana* Jiz, and willow (*Salix L.*). Tree planting followed the required Mongolian National Standards (MNS, 2011).

During tree planting, a distance of 1.5 m was maintained between individual trees and a distance of 2 m was maintained between lines of trees. In addition, the distance between the plots were fixed at 3 m, with 64 trees planted in each plot, as shown in Figure 1.

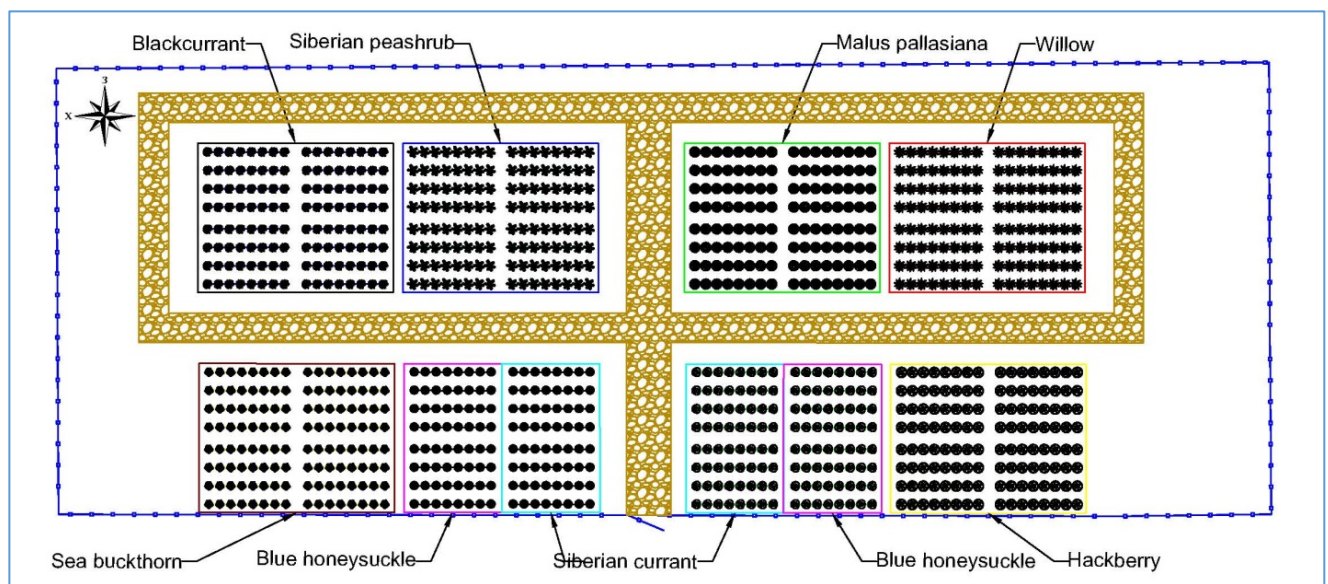


Figure 1: Tree nursery trial field scheme

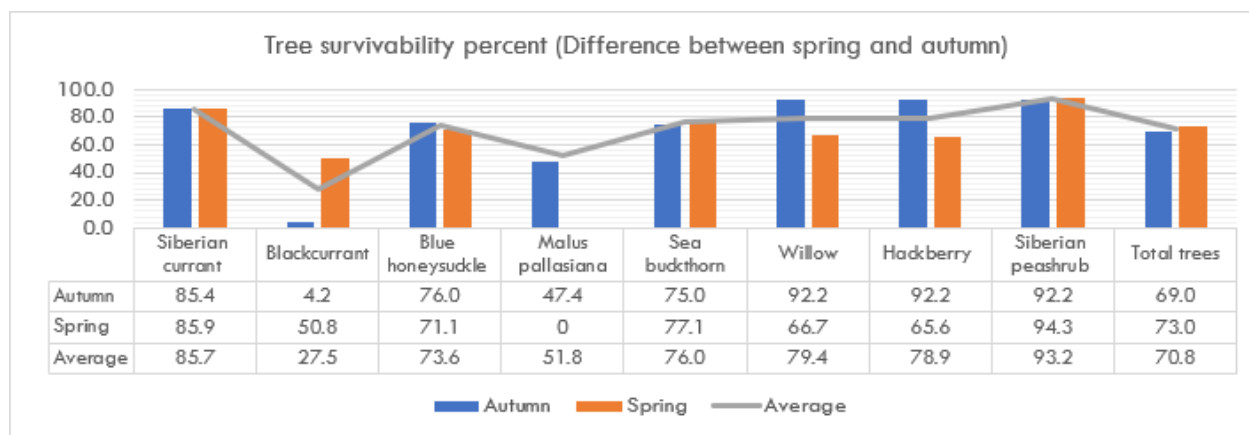
From 2018 to 2020, 1,024 trees were planted in 16 plots in the tree nursery during the spring and autumn seasons.

Results

Tree survivability counts were conducted every spring and autumn. The results of the tree survivability count are shown below in Table 1 and Graph 1.

**Table 1: Tree survivability average percentage
(from 2019, 2020 and 2021 survivability results)**

Planted time	Survivability percentage								Total trees
	Siberian currant	Blackcurrant	Honeysuckle	Hackberry	Sea-buckthorn	Willow	Siberian crabapple	Siberian peashrub	
Autumn	85.4	4.2	76.0	47.4	75.0	92.2	92.2	92.2	69.0
Spring	85.9	50.8	71.1	—	77.1	66.7	65.6	94.3	73.0
Average	85.7	27.5	73.6	51.8	76.0	79.4	78.9	93.2	70.8



Graph 1: Tree survivability average (From 2019, 2020 and 2021 survivability count results)

Based on the chart and figure above, out of eight species of woody and bush plants planted in the tree nursery, the species with the lowest survivability rate is the blackcurrant, and the species with the highest survivability rate is caragana. The average survivability rate of all the trees is 70.8%. By MNS standards (MNS, 2011), trees can be planted in spring and autumn. As indicated by this standard, the eight species of trees were planted in the spring and autumn resulting in an average survivability rate of 69% and 73%, respectively, for trees planted in the spring and autumn. While the difference between planting in spring and autumn is not great, a slightly better survivability rate was noted in the spring.

Tree seedling survivability was notably affected by rabbits and small rodents. The blackcurrant species was the most eaten and had more small rodent holes in the leaves than the other trees. This impacted the survivability and germination of the blackcurrant trees.

Seed germination of local pasture perennial plant seeds and planting results

Steppe Gold aims to rehabilitate the disturbed land by using the perennial plant seeds collected from local plants, thereby generating seed bank and biological rehabilitation of the disturbed land as closely as possible to the prior natural form and appearance.

Methodology

The mine site is situated on the border of the forest-steppe and steppe region. There are a total of 118 species of plants belonging to 84 kinds, with 35 families identified. Out of these species, five plants were chosen, namely *Helictotrichon schellianum*, *Stipa glareosa*, *Bromus inermis*, *Agropyron cristatum*, and *Chloris virgata*, which dominantly grow in the region. Seeds from these species were collected for seed germination tests in the mine site laboratory and were planted on a topsoil stockpile.

The seeds were planted in a topsoil stockpile in plots with dimensions of 2×2 m, and each of four plant species were planted in two repeats, with seeds sown by hand in a total of eight plots.



Figure 2: Collection of plant seeds, collected seeds of *Stipa glareosa* and *Helictotrichon schellianum*



Figure 3: Growth of *Stipa glareosa* and *Helictotrichon schellianum*



Figure 4: Growth of plants on topsoil stockpile research plantation

Results

Germination tests of all collected plant seeds were conducted at the Plant Protection Institute's laboratory at the Mongolian Science Academy with the following success rates: *Stipa glareosa* 95%; *Helictotrichon schelliamum* 93%; *Bromus inermis* 91%; *Agropyron cristatum* 52%; and *Chloris virgata* 13%. While collecting seeds of *Agropyron cristatum* and *Chloris virgata*, some of the seeds were not fully ready for collecting. This might have affected the quality of the seed. Further seed collection is necessary.

In addition, of all the collected seeds, collecting seeds of *Stipa glareosa* and *Helictotrichon schelliamum* was the easiest and time-efficient, and it was noted that these seed varieties can be collected in large quantities to be used in rehabilitation programs.

Table 2: Laboratory germination results and outdoor germination results

Plant species	Plant germination, %	Outdoor germination results, %
<i>Helictotrichon schelliamum</i>	93%	90%
<i>Stipa glareosa</i>	95%	85%
<i>Agropyron cristatum</i>	52%	48%
<i>Bromus inermis</i>	91%	81%
<i>Chloris virgata</i>	13%	—

Results of trials to decrease the cyanide concentration of heap leach ore using water

In accordance with the Altan Tsagaan-Ovoo mine's feasibility study, 5.2 million tons of ore will be stored on a 22.4 hectare heap leach pad. The heap leach pad will consist of 3 lifts of stacked ore with each lift 8 m high. The heap leach pad will be stacked in accordance with the design prepared by Knight Piésold LLC (2018).

Column leach tests were conducted to research closure of the ore heap and to evaluate the reduction of cyanide concentration by fresh water rinsing and recirculation. The heap leach pad will be closed in accordance with the conceptual plan shown in Figure 5.



Figure 5: Conceptual plan for heap leach pad closure

Methodology

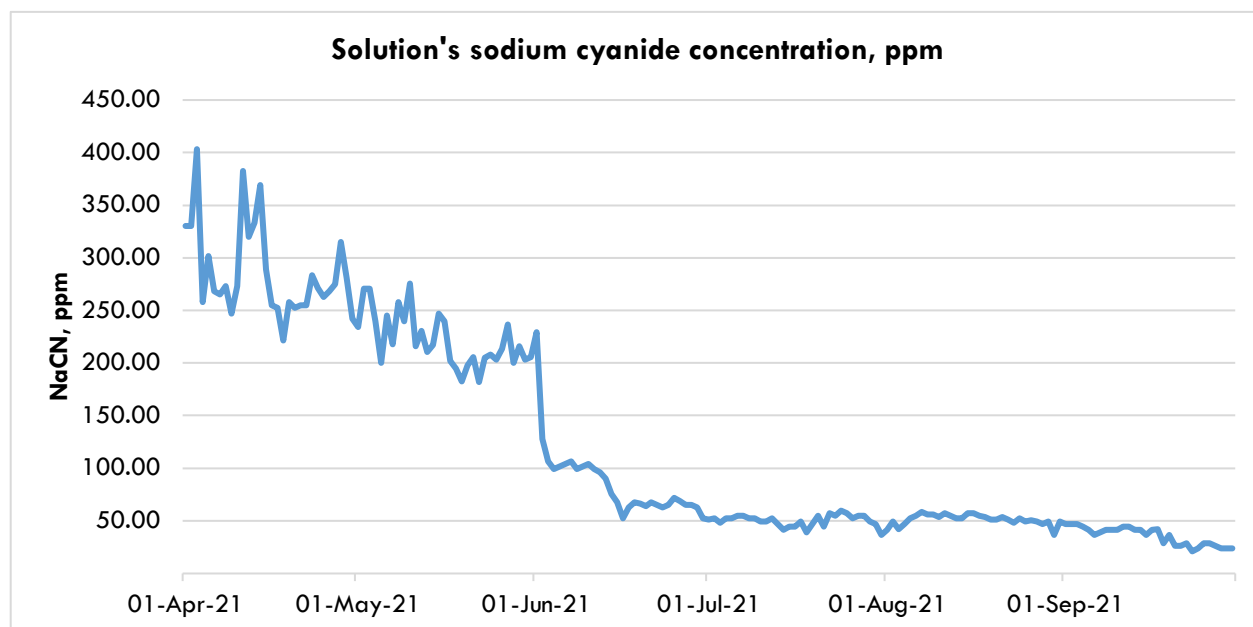
Water circulation tests were done for a total of 20 days on spent ore from column leach tests. These tests were conducted in a metallurgical laboratory in the processing plant to determine the duration required to reduce cyanide concentrations in the ore to acceptable levels. The processing plant's column leach test was done using a barren solution with a cyanide concentration of 243 ppm and a pregnant solution with a cyanide concentration of 235 ppm, with a total solution quantity of approximately 25 L used in the test. The volume of ore used was 147 kg. The water was recirculated at a rate of 25.15 ml per minute over the 20-day test period.

Result

For the 20-day test duration, a total of 12 L of fresh water was added to keep the solution volume constant. During the test period, the cyanide concentration decreased from 235 ppm to 41 ppm, a decrease of 82%. A total of 724 L of solution, including freshwater addition, were recirculated through the test column.

During the year this column leach test was done, due to the COVID-19 pandemic situation, the border and customs were closed and cyanide shipments to the mine were stopped temporarily. Therefore, the processing plant circulated the solution without adding any cyanide to the solution ponds for five months.

During that period, a total of 18,871 m³ of water were used for 476,000 tons of ore. A total of 790,560 m³ of solution were recirculated from June to October on the heap leach pad, with a solution pond pumping rate of 270 m³ per hour. The cyanide concentration was 228.98 ppm in June and after recirculating the solution for 5 months, the cyanide concentration decreased to 23.4 ppm, a decrease of 89%.



Graph 2: Solution's cyanide concentration reduction

Table 3: Test to reduce cyanide concentration of solution and real situation results

	Solution concentration, ppm	Solution recirculation duration	Water pump rate	Total recirculated water volume	Solution concentration, ppm	Additional water volume
Column leaching test	235	20 days	25.15 ml/min	724 L	41 ppm ↓82%	12 L
Heap leaching with no cyanide addition	228.9	150 days	270 m ³ /h	790,560 m ³	23.4 ppm ↓89%	215 mm precipitation*

* Average precipitation amount during the heap leaching.

Based on the results of the column test and field-scale recirculation on the leach pad, it is shown that cyanide concentrations can be reduced significantly from the ore by rinsing and recirculation using fresh water.

Results from vegetation test done on ore from cyanide heap leach

According to the mineralogy research, the dominant minerals in the deposit are chalcopyrite, sphalerite, galena, and pyrite. Based on laboratory analyses, the content of lead, zinc, gold, and silver are high and the copper content is low. In addition, cadmium, helium, bismuth, selenium, molybdenum, and antimony minerals are present in the deposit as associated or by-product minerals. Based on the research, the economically valuable minerals are lead, zinc, gold, and silver.

Gold and silver will be extracted from the oxidized ore. Lead and zinc concentrate will be produced from processing fresh rock by flotation technology (Knight Piésold, 2018).

The Altan Tsagaan-Ovoo mine is a gold and polymetallic deposit and due to its mineralogical characteristics, some heavy metal elements content can possibly be high in groundwater and soil near the mine site.

The purpose of this test was to plant seeds collected from local plants and purchased from seed producers on a reclaimed ore stockpile contaminated by cyanide produced from the test heap mentioned above; calculate the growth rate and the heavy metals amount in grown plants; and thus determine which plant is most suitable for future rehabilitation and closure of the heap leach pad.

Methodology

A total of six plant species were evaluated in the test plot including *Bromus inermis*, *Medicago varia*, *Helictotrichon schelliamum*, *Agropyron crystatum*, *Prunus dillus*, and *Stipa glareosa*. These were grown on a reclaimed ore heap that was first covered by topsoil. The ore contained a solution with a cyanide concentration of 41 ppm. A total of 120 kg of ore containing cyanide was split into six 20-L containers. Ore was then covered with about 20 cm of topsoil.

Results

An assessment was conducted two months after planting. Based on field observations, the seeds collected from natural plants *Helictotrichon schelliamum* and *Stipa glareosa*, and the seeds purchased from seed producer *Medicago varia*, had the highest growth and germination rates. Sample were taken by combining all the plants grown from this test and then analyzed for heavy metals in the laboratory. In addition, a background sample was taken from natural plants in the area that were not part of this test plot.



Figure 6: Ore covered with topsoil and plant seeds planted on the topsoil



Figure 7: Growth of planted seeds (two months after planting)

Table 4: Some heavy metals analyses results of natural plants, plants on cyanide ore, MNS standard requirements, topsoil, mine ore and solution samples

Heavy metals	Ore sample	Feed solution into plant	Topsoil	* MNS 5850:2019	Natural plant	Plant grown on cyanide ore	Difference between natural plant and plant grown on cyanide ore %
ICP	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	percent
Pb	722.00	0.08	13.90	100.00	7.76	8.23	6.06
Cu	88.00	267.00	7.19	100.00	4.26	6.95	63.15
Li	40.00	—	19.52	—	2.39	6.96	191.21
As	155.00	0.00	14.41	20.00	0.00	0.00	0.00
Cr	6.00	0.02	24.45	150.00	2.92	8.27	183.22
Mo	6.00	0.18	5.00	5.00	2.15	0.40	—81.40
Se	—	0.03	5.00	—	0.87	0.01	—98.85
Ni	3.00	0.35	11.67	150.00	0.83	2.23	168.67
V	25.00	0.00	44.13	—	3.10	14.47	366.77
Co	1.00	0.46	9.00	50.00	0.36	1.92	433.33
Cd	1.40	0.00	1.00	3.00	0.15	1.05	600.00
Be	0.50	0.00	<5.0	—	0.15	0.01	—93.33
Zn	33.10	0.15	39.43	300.00	14.30	37.07	159.23

*MNS 5850:2019 Soil quality, maximum allowed amount of pollutants in soil.

Heavy metal analyses were conducted in the laboratory on plant samples taken from natural plant and plants grown on the test heap leach plot with a cyanide concentration of 41 ppm (covered with about 20 cm of topsoil). There results are presented in Table 4. Based on the test results, some of the heavy metal content in plants grown on the test heap plot is high.

Conclusion

The conclusions of Altan Tsagaan-Ovoo mine's research and trials are as follows:

1. For mine rehabilitation, protection tree lines will be established. Woody plants Siberian currant (*Ribes diacanthum* Pall), Honeysuckle (*Lonicera caerulea*), sea-buckthorn (*Hippophae*), Siberian peashrub (*Caragana arborescens* Lam.), Willow (*Salix* L.), and Siberian crabapple, with survivability of more than 70%, are more suitable. It is recommended that more woody plant species should be added to future test plots to evaluate their suitability.
2. *Helictotrichon schelliamum* and *Stipa glareosa* plant seeds can be collected to make seed banks and can also be used for mine rehabilitation. Weeds were growing from the soil's seed bank during

the topsoil stockpile vegetation test. It is recommended that more soil seed bank analysis tests are conducted in future test plots.

3. Recirculation of water on cyanide-containing ore heap leach and solution can be a feasible method to reduce the cyanide concentration. However, an extended period of water recirculation might result in the release and uptake of previously stable constituents into the plant species (Parshley et al., 2012). Therefore, further research and analyses are recommended on water-recirculated ore and solution.
4. Judging from the preliminary results of vegetation trials, the heavy metal content of plants grown is higher than the plants grown on natural soil. Steppe Gold's environmental team plans to continue field trials to assess the results of increasing the thickness of the soil covering the ore, varying the plant species being used in the trial, and analyzing the heavy metal content in each of the plants grown on the reclamation test plot over the ore heap.

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Governance and Management: Using KPIs and Early Indicator Dashboard to Manage and Communicate Heap Leach Pad Facility Performance

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Abstract

Management of a Heap Leach Pad (HLP) facility involves the use of critical control measurements and Key Performance Indicators (KPIs) to detect, monitor, and mitigate significant safety and environmental risks to the performance of a HLP. Freeport-McMoRan Inc.'s operations with HLP use KPIs and an Early Indicator Dashboard to guide operational and technical teams in managing the day-to-day performance of a HLP and to drive clear communications and good practice actions. The KPIs form part of the operational, maintenance, and surveillance guidance in managing and monitoring the physical stability of the HLP. KPIs are used to show that day-to-day operational construction meets the Engineer of Record's (EOR) design criteria and good practices for HLP safety. Guidance for managing KPIs can be found in the *Operational, Maintenance and Surveillance Manual (OMS Manual)* together with Standard Operating Procedures (SOP) and Work Instructions (WI) to help to smoothly onboard newly hired geotechnical engineers. Each Early Indicator KPI has a Work Instruction developed to define the purpose of the Early Indicator KPI, how it is measured / calculated and describe how the scores are obtained.

The site geotechnical engineering group, in collaboration with Leaching Operations, Leaching Technical Service (LTS), Crush and Convey, and the EOR, will monitor, summarize, and interpret different sources of data. These data will generally consist of the following: Pore Pressure, Superficial and Deep Movements, Pond Management, Material Placement, Crushed Ore Material Quality, Solution Balance, and Visual Inspections. The Dashboard also includes progress on implementing recommended actions from EOR and third-party Crushed Leach Stewardship Team inspections.

The Early Indicator Dashboard provides site Senior Management teams with information to proactively drive performance and ensure objectives are consistently met effectively and efficiently. The Early Indicator Dashboard also keeps Corporate Senior Leadership and Technical Management apprised of day-to-day conditions and trends of the HLP.

Introduction

Heap leach pad facilities consist of ore from the pit that is crushed, agglomerated, and stacked on an engineered pad in designed lifts. A weak acid solution is then circulated through the heap and target metal is extracted in the form of a pregnant leach solution (PLS). Heap leaching is a fully coupled geotechnical-hydrometallurgical process (Wooten et al., 2015). It is critical that the geotechnical stability of the facility is managed closely while seeking optimal metal recoveries throughout the life cycle of the facility.

The current industry practice of HLP management involves developing an internal framework that guides risk management, often tailored to site-specific conditions. Examples of these management plans are well documented in the literature (Strata Gold Corporation, 2019; YukonZinc Corp, 2011). These management plans consist of measurable key performance indicators (KPIs) used to assure the daily operation of the HLP meets the design intention at a frequency approved by the Engineer of Record (EOR). KPI data provides operational and design performance information and consistently ensures objectives are met based on a scoring system that classifies data into alert levels. Internal stakeholders review, provide comments, and confirm the performance of the HLP on a monthly basis using recorded KPI data. The main benefit of this approach is that deviations from the intended design or performance are quickly identified, and corrective actions are then taken before challenges develop into issues of significant risk. Historical performance can also be tracked, and general trends identified, for further action and decision making. An informed knowledge of the HLP performance is crucial in maintaining a manageable risk profile.

Freeport-McMoRan has a strong safety culture for management of large HLP facilities. This paper describes an example of how this safety culture is embodied in day-to-day work. Freeport-McMoRan's HLP KPIs focus on geotechnical stability in the near term, as well as good practices for maintaining stability over the longer term. Freeport-McMoRan site geotechnical engineers measure KPI data and compare the results to EOR-defined alert levels, resulting in easy-to-access KPI scoring that is summarized in an early indicator dashboard for managing and communicating the operational performance of HLP across its portfolio. The report is provided monthly to site leaders and corporate technical reviewers; a summary is further provided to corporate senior leadership quarterly.

The following KPIs are used in managing the safe performance of the HLPs; however, selected KPIs will be described in this paper for brevity. These KPIs are grouped into seven main categories, and each category is further divided into sub-sections.

- Phreatic surface (piezometer water elevation, piezometer system operational availability, and piezometer system operational availability – single instrument).
- Crushed ore material quality (agglomeration, P80, fines content [–200 sieve], and total clay content).

- Movement rate (inclinometers and shape arrays, surface movement InSAR, and surface movement GPS units).
- Pond management.
- Material placement (surface preparation, updated stacking plan, and adherence to stacking plan).
- Raffinate and PLS flows (raffinate application rate, PLS flow decrease/increase and monthly solution balance).
- Monitoring and inspection (site engineer inspections, EOR inspections and action items status, crushed leach stewardship action items status, and number of new action items related to seep management).

KPI scores are denoted by description of conditions and indicate trigger levels as follows:

- Within Limits – operating conditions meet design intent and good practice. Continue current operating practices.
- Approach Limits – conditions during the reporting period approach limits of design intent or expectation of good practice. Increase level of attention required and/or action plan required to move towards being within limits of operating conditions.
- Exceeded Limits – does not meet design intent or expectations of good practice. Immediate development of an action plan needed to achieve being within limits of operating conditions.

Phreatic surface

Slope movement occurs when the driving force tending to cause slope movement exceeds the resisting shear strength of the soil. A common trigger for slope movement is the buildup of pore pressure in a slope. Pore pressures reduce the shear strength of the soil along a potential failure plane by reducing the effective normal stress. There is always some uncertainty associated with the potential pore pressure that may develop and its impact on the shear strength of the crushed ore. Considering the potential safety risks associated with slope failure, as well as environmental and economic risks, it is imperative to carry out routine pore pressure monitoring to identify conditions that could lead to slope failure and to determine if the design assumptions require modification. The potential variability of the pore pressures in the stockpile is one of the factors that could reduce slope stability during its operational lifetime.

The pore pressures may be influenced by:

- Operational abnormalities such as high application rates, failure of the solution collection system, or failure of the solution distribution system.
- High volume precipitation events.
- Changes in the hydraulic flow properties of the ore materials due to loading and ore decrepitation.

Freeport-McMoRan HLP uses vibrating wire piezometers to measure pore pressure conditions/solution levels within the stockpile. These piezometers are installed in clusters to measure pore pressure at different depths. Also, regular CPT and drilling/testing campaigns are conducted to update the piezometer thresholds regularly to account for changes in the properties of the crushed ore over time. Piezometers are connected to an automatic data acquisition, transmission, and storage system (supplied by Canary).

The EOR for Freeport-McMoRan HLP uses a two-dimensional (2-D), limit-equilibrium stability, computer modelling program to conduct the slope stability analysis for each control section of the HLP. From stability analysis using material characterization based on multiple lines of evidence, the EOR assigns Alert Level elevations to each piezometer resulting from a range of factors of safety corresponding to that section. A summary table of alert level elevations assigned to each piezometer is provided to the site geotechnical engineer from the EOR after analyzing the stability of each control section of the pad, as shown in Table 1.

Table 1: Piezometers and their assigned alert level elevations and depth

Name	Cross section model	Set depth (m-bgs)	Set elevation ⁽²⁾ (m-msl)	Alert level 1			Alert level 2		
				Elevation (m-msl)	Depth (m-bgs)	Height above instrument (meter)	Elevation (m-msl)	Depth (m-bgs)	Height above instrument (meter)
PZ-3a_L3	1	19.8	1,419.1	1,420.3	18.6	1.2	1,424.3	14.6	5.2
PZ-16-L6a	1	31.1	1,422.7	1,423.9	29.9	1.2	1,430.1	23.8	7.4
PZ-16_L9a	1	57.3	1,421.5	1,423.9	56.1	1.2	1,438.6	40.1	17.1
PZ-16_L9b	1	45.7	1,433.2	1,434.4	44.5	1.2	1,438.6	40.2	5.4
PZ-21_L12	1	84.9	1,412.5	1,413.7	83.7	1.2	1,438.6	58.8	26.2
PZ-2a_L3	2	19.5	1,420.8	1,422.1	18.3	1.2	1,426.3	14.1	5.5

The site geotechnical engineer updates the piezometer tracker daily using the calculated distance to Alert Level 2; this identifies piezometers with solution level approaching Alert Level 2. A summary of piezometers with solution levels within 3 meters to Alert Level 2 and their respective KPIs are sent out to the Hydromet team daily for adjacent leaching sections to be turned off to control pore pressure in the area.

If the KPI is at 85% and increasing, the irrigation for corresponding section is recommended to be progressively turned off, as shown in Table 2.

**Table 2: Summary of piezometers within 3 meters to
Alert Level 2, their KPis and respective turn off sections**

Piezometer ID	Alert level 2 depth	Distance to AL2 (KPI)	Turn off sections (recommended)		
			1st	2nd	3rd
PZ-26-L12b	0.5	0.6 (0%)	South 31	South 32	—
PZ-118_L12c	1.61	1.2 (25%)	South 33	South 43	—
PZ-45_L12b	14.9	1.9 (88%)	South 26	South 27	—
PZ-110_L9a	3.4	1.8 (47%)	North 65	North 66	—
PZ-15_L6a	8.7	1.9 (78%)	South 038	South 039	—

The recorded piezometer data for the entire month enables the site geotechnical engineer to score piezometric elevation and availability based on the KPI criteria set shown in Table 3.

Table 3: Scoring criteria set for phreatic surface monitoring

No.	Parameter	This month's score	Scoring criteria		
			Within limits	Approach limits	Exceed limits
1.1.1	Piezometer water elevation	Within limits	Solution level is <85% of distance between action level 2 and piezometer set elevation or ≥85% and solution level is stable or decreasing.	Solution level is ≥85% of distance between Action Level 2 and Piezometer set elevation and increasing.	Solution level has reached or is above action level 2.
1.1.2	Piezometer system operational availability — system	Within limits	≥92% critical instrument availability	85% ≤ X <92% critical instrument availability	<85% critical instrument availability
1.1.3	Piezometer system operational availability — single instrument.	Within limits	Action plan developed and on schedule for all unavailable critical instruments or single critical instrument unavailable ≤4 weeks with no action plan.	Action plan is not developed to address connection with any single critical instrument down >4 weeks but ≤6 weeks. Any action plan is not anticipated to be complete per agreed schedule.	Action plan is not developed to address connection with any single instrument down >6 weeks or action plan is behind schedule to address connection with any critical instrument regardless of time down.

The scores and data are transmitted via internal digital workspace to the Crushed Leach EOR, the Leaching Manager and the Corporate HLP Point of Contact (POC) for reviews, comments, and acceptance for final distribution to management. A summary report using Power BI is generated and distributed to all

relevant stakeholders. Figures 1 and 2 show the digital workspace and Power BI respectively used to communicate.

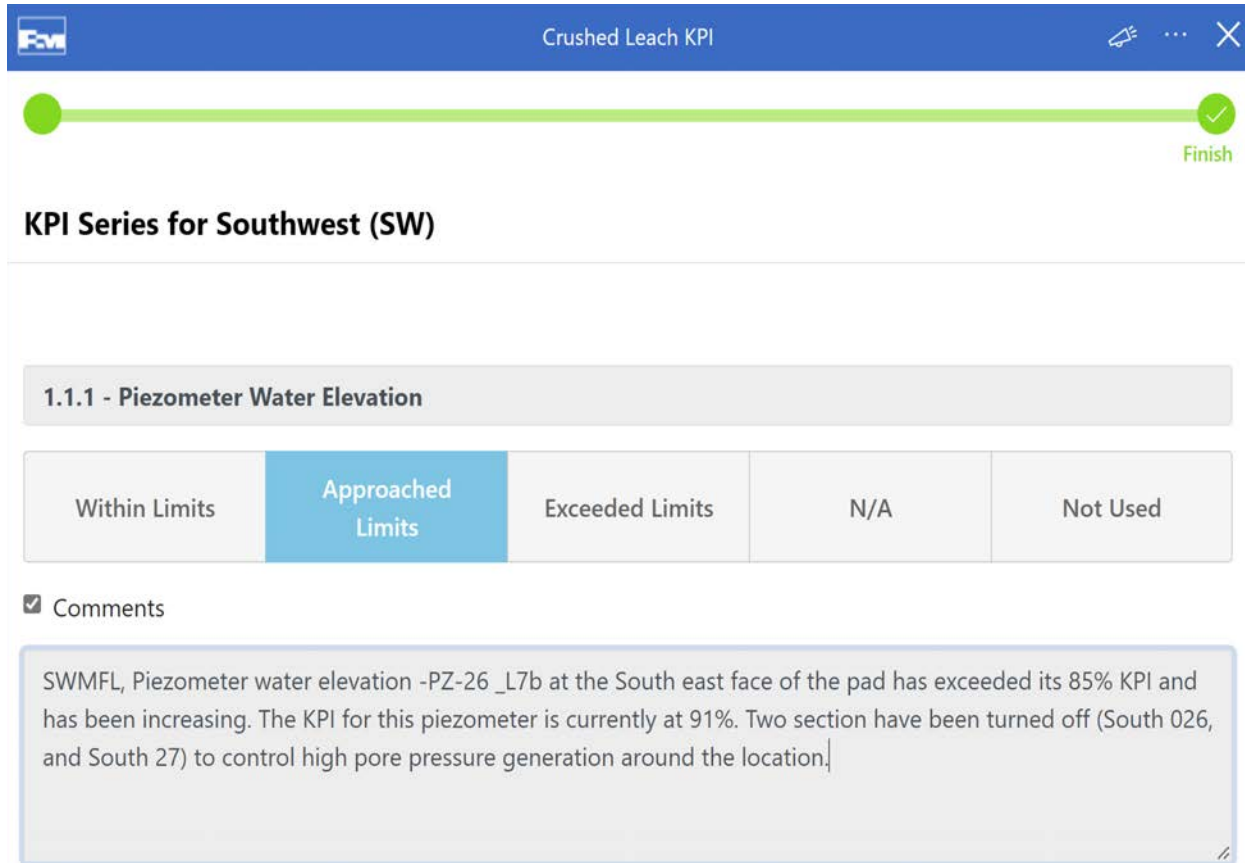


Figure 1: Digital workspace showing piezometer water elevation KPI score



Figure 2: Power BI showing a summary of phreatic surface KPI score for a particular month

Crushed ore material quality

From the hydrometallurgy point of view, smaller crushed size material will lead to larger overall particle surface areas susceptible to leaching. The bigger the surface area available for leaching, theoretically, the more metal can be recovered quickly. However, smaller particles may experience a more rapid timeline for decrepitation and strength change, which may have a negative impact on the stability of the pad. Also, a high variability of the P_{80} sizes can lead to the long-term variable permeability of the stockpile and possibly lead to a reduction in permeability due to increased fines migration. This has the tendency to create high pore water pressure within the pad, thereby reducing the shear strength of material in the pad.

The presence of a significant amount of fines content could reduce the overall voids of the crushed ore and produce a significant reduction in the drainage capacity of the HLP. In this case, the material would present a low hydraulic conductivity. Application of solution on low hydraulic conductivity material will develop ponding on the surface and potentially saturated areas with the generation of pore water pressures within the pad. The shear strength of crushed leach material in the pad is highly influenced by the amount of fine-grained material deposited as well as continuous leaching of the ore. Recovery of metal also tends to be low during the leaching process in areas of high fines content. In general terms, when the amount of fines is low and they are of low plasticity or non-plastic, the mechanical and hydraulic behaviour of the mineral is more like that of granular soils (gravels and sands); i.e.: drained behaviour when subjected to shear stresses, high effective friction angle, high stiffness, low compressibility, and high permeability. As the amount of fines increases and they are of high plasticity, the mechanical and hydraulic behaviour becomes more like that of cohesive soils (silts and clays); i.e.: undrained behaviour, low angle of effective friction, less stiffness, high compressibility, and low permeability when subjected to loading.

The presence of a significant amount of clay materials can be a source of problems in crushed leach operation. Generally, material with high clay content has low shear strength, high compressibility, a high level of volumetric change, and low hydraulic conductivity. Application of solution on low hydraulic conductivity material will develop ponding on the surface and potentially saturate areas with the generation of pore water pressures within the pad. The shear strength of crushed leach material is highly influenced by the amount of clay content in them. Recovery of metal also tends to be low during the leaching process in areas of high clay content or minerals.

The Freeport-McMoRan HLP site geotechnical engineer tracks all these parameters on a weekly basis to ensure laboratory test results do not violate the KPIS set for these parameters. If any of these test results do not meet the KPI target, actions are taken for a future slope monitoring program. Some of the actions taken include:

- The Crushed Leach Geotechnical Engineer investigates information from the crushed and convey team regarding the location (cell or section) that was stacked on the day(s) that the KPI was violated.
- The shape file of the location stacked is stored in the GIS database as reference for the routine geotechnical weekly inspection and monitoring program.
- The site geotechnical engineer will also notify the EOR and the corporate POC of the situation for future geotechnical drilling and sampling program, CPT testing, and piezometers installation. Operations also increase the density of wick drains during surface preparation for the next lift. Figure 3 depicts a shape file in ArcGIS on how surveyed location is kept for KPI violation.

In addition to these, routine drilling and sampling and CPT programs are conducted to update material geotechnical properties of previously placed lifts to account for the decrepitation process.



Figure 3: Shape (yellow), showing high clay content of material stacked on a particular day

The laboratory test results for P_{80} , total clay and fines content for the entire month enables the site geotechnical engineer to score each individual parameter based on the KPI criteria set shown in Table 4.

The scores and data are transmitted via internal Digital Workspace to the HLP EOR, the Leaching Manager and the corporate HLP POC for reviews, comments, and acceptance for final distribution to management. A summary report of Power BI is generated and distributed to all relevant stakeholders. Figures 4 and 5 show the digital workspace and Power BI used to communicate to stakeholders.

GOVERNANCE AND MANAGEMENT: USING KPIS AND EARLY INDICATOR DASHBOARD
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Table 4: Scoring criteria set for P₈₀, fines content and total clay content

No.	Parameter	This month's score	Within limits	Scoring criteria	
				Approach limits	Exceed limits
3.2.2	P ₈₀	Within limits	>12.7 mm (daily)	One day exceedance of <12.7mm but >11.4mm	One day exceedance <11.4mm
3.2.3	Fines content (passing #200 sieve)	Within limits	<10% (daily)	One day exceedance of ≥10% but < 15%	One day exceedance of ≥15%
3.2.4	Total clay content	Within limits	<14% over 24k tons average and <16% over 8k ton average	More than 14% but less than 16% over 24k ton average or more than 16% but less than 18% over 8 k tons average	More than 16% over 24 k ton average or more than 18% over 8 k ton average

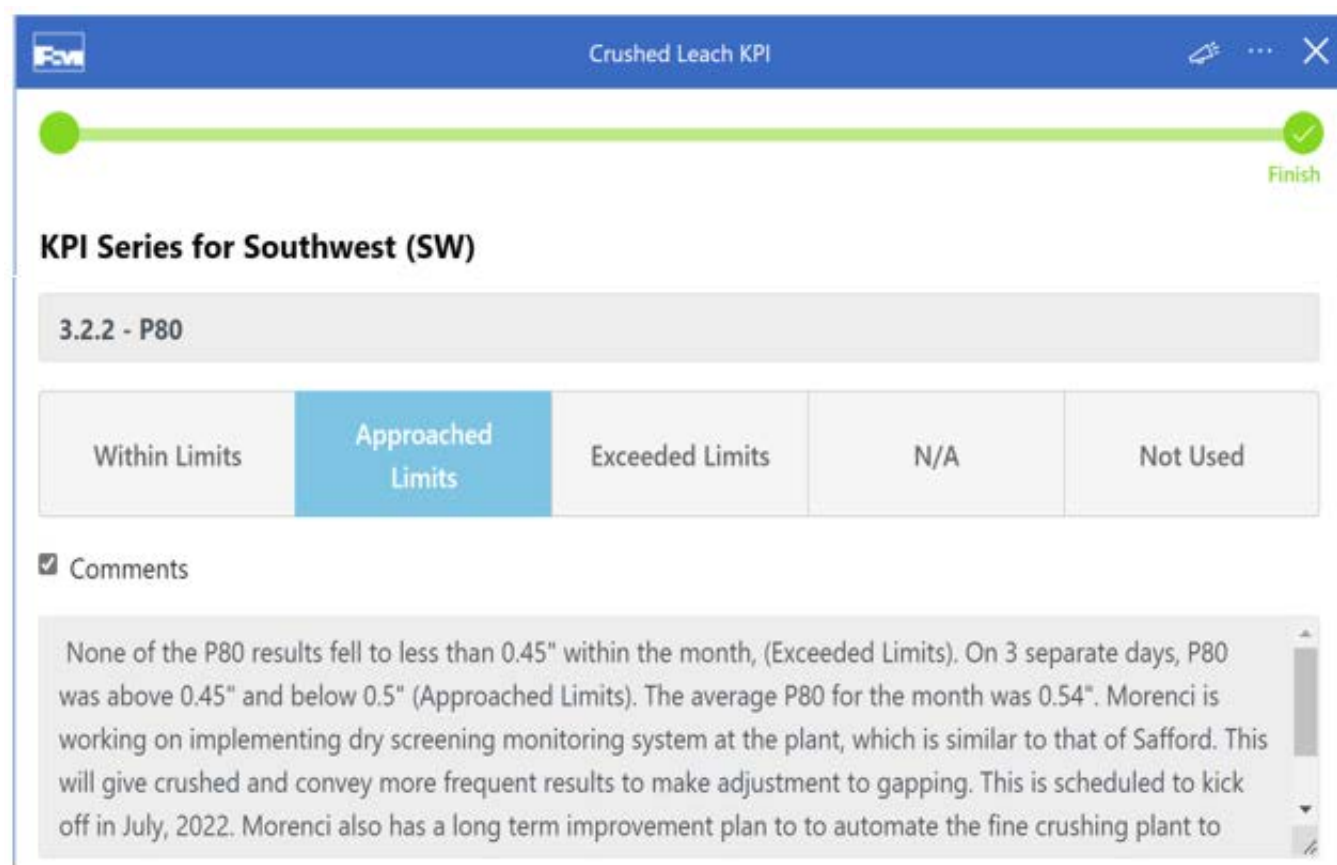


Figure 4: Digital workspace showing P₈₀ and fines content KPI score



Figure 5: Power BI showing a summary of P₈₀, fines content and total clay content KPI score

Monitoring and inspection

The purpose of these KPIs is to visually assess the performance of the HLP for early detection of physical conditions that may indicate the potential for slope instability in the HLP. On-site personnel, along with consultants, perform the inspections to find early indicators of slope instability conditions or possible problems so that they can be corrected before they impact the safety and operations of the facilities. Inspections are conducted on a weekly, quarterly, and yearly basis. The site geotechnical engineers perform inspections on a weekly basis and send out weekly inspections' findings and their respective corrective actions to be taken. The EOR, corporate POC, and leaching operations, together with site engineers, conduct inspections quarterly for findings, their corrective actions, and timelines to complete these action items.

Annual inspections and technical reviews are also conducted by the Crushed Leach Stewardship Team (CLST), which is led by a third-party consultant along with corporate experts and site participants. The third-party consultant reviews all data associated with the HLP. These include piezometers' data, drilling, and sampling test results, EOR quarterly site visit reports and recommendations, CPTu tests' results, site crushed leach geotechnical weekly inspection reports, EOR quarterly instrumentation review reports, annual report, and all construction activities performed within the year, as well as future projects. The external consultant writes a report with findings and recommended actions, including a date by which Freeport-McMoRan should complete the recommendations.

Each Freeport-McMoRan HLP has individual KPIs set for all inspections and recommendations performed by all the teams mentioned above. Table 5 summarizes the KPIs set for all inspections and

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recommendations. The site geotechnical engineer manages all action items and recommendations and scores each KPI based on its status.

No.	Parameter	This month's score	Scoring criteria		
			Within limits	Approach limits	Exceed limits
3.4.1	Site (TCLW engineer or designee) inspection	Within limits	Inspection completed on schedule	Inspection not completed on schedule for the reporting period	Two or more scheduled inspections have been missed
3.4.2	EOR inspection	Within limits	Inspection completed on schedule	Inspection not completed on schedule for the reporting period	Two or more scheduled inspections have been missed
3.4.3	Site (TCLW engineer or designee) action item status	Within limits	All actions on schedule and/or complete	At least one action which is not anticipated to be complete per agreed schedule	At least one item past due on completion
3.4.4	EOR inspection action items status	Within limits	All actions on schedule and/or complete	At least one action which is not anticipated to be complete per agreed schedule	At least one item past due on completion
3.4.5	CLST action items status	Within Limits	All actions on schedule and/or complete	At least one action past its due date by one month or less	At least one item past its due date by greater than one month
3.4.6	Number of new action items related to seep management	Within Limits	≤15 findings ¹	>15 but ≤25 findings ¹	>25 findings ¹

Table 5: Scoring criteria set for monitoring and inspections

A summary report is sent out for approval, as discussed previously. A summary report (score, comments and actions taken, and supporting data) of all KPIS listed in the introduction is sent out each end of the month to the site processing manager, director of Tailings, Crushed Leach, and Water (TCLW) as well as all other stakeholders in a Power BI format. Figure 6 presents a summary of how the final report is distributed.

¹ These criteria also consider severity of the seepage (flow rate, turbidity, piping etc.)

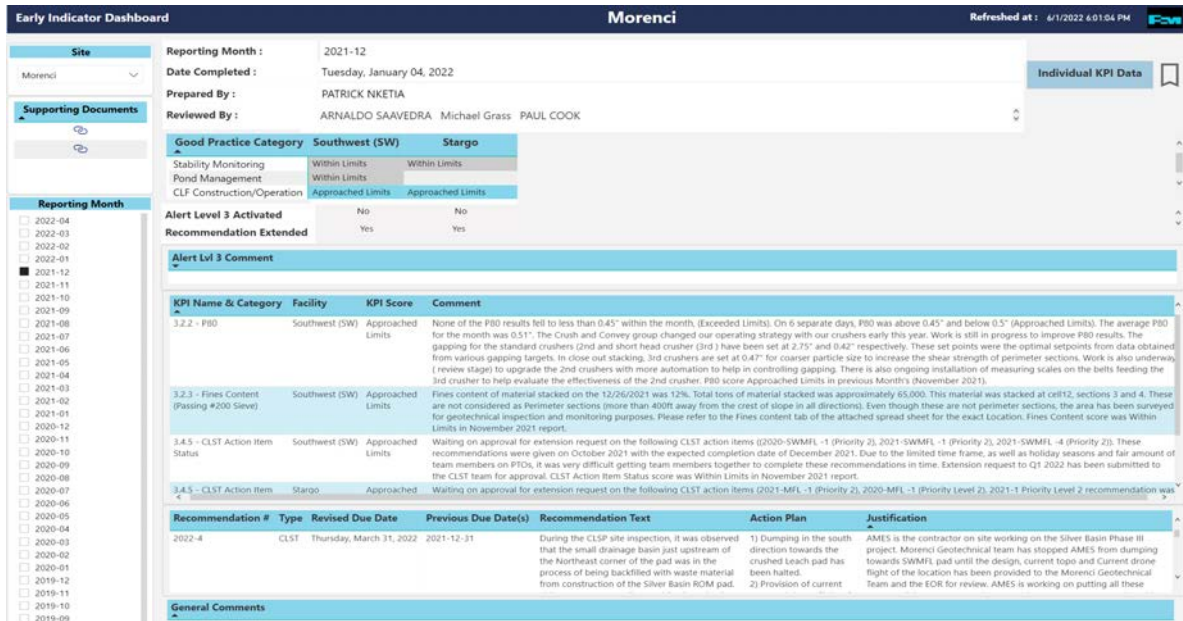


Figure 6: Power BI summary report of all KPI(s)

The criteria shown in this paper are for a snapshot in time. Criteria or Alert levels for each KPI vary during the life of the HLP based on many factors. To update the Alert Levels for a given KPI, the EOR makes a recommendation and provides justification for the change; site engineers submit the agreed recommended changes to the corporate PoC and TCLW Director for approval prior to updating alert levels.

Conclusion

Using the KPIs scores and reporting system at the HLP site allows site geotechnical engineers to analyze, review, and interpret data and take remediation actions quickly and effectively. It gives a clear and accurate picture of the performance of the HLPs. Through the KPIs scores and use of the early indicatory dashboard report, all internal stakeholders can understand the status and performance of the HLP. Data interpretation and visualization become simple for stakeholders who are not directly involved in the day-to-day management of the facilities. This also serve as a communication tool to advise on situations of concern and highlight areas requiring leadership support.

Acknowledgement

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Chapter 5:

Engineering Design, Geotechnical Track, and New Designs



On the Use of Buried Pipes in Heap Leach Facilities

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Abstract

Heap leach facilities can place very high loads on pipes that are constructed into their drainage systems. With heaps up to 150 m (500 ft) or more in height currently in operation, the integrity and long-term survivability of these pipes should be of paramount concern to heap owners, operators, and designers.

In typical civil engineering applications, buried pipes receive significant, prescribed mechanical compaction efforts. Unlike their civil engineering counterparts, drainage pipes in heap leach facilities do not normally receive this mechanical compaction effort. This paper provides some background and guidance regarding the design, construction, and survivability of pipes buried in heap leach facilities.

Heap leaching basics

The design of a heap leach facility requires several types of engineering specialists, the description of which is beyond the scope of this paper. The engineering firm that completes this design is usually termed the “design engineer”. The design engineer prepares the construction drawings and technical specifications for the project, among other things. The technical specifications are essentially instructions to a contractor, telling them exactly how they are expected to complete their work.

During construction of a heap leach facility, it is common to employ an independent firm to help assess whether the construction is being completed in accordance with the design. This service is sometimes referred to as construction quality assurance (CQA), quality assurance (QA), quality control (QC), or quality control/quality assurance (QC/QA). The design engineer’s company commonly performs this work. Sometimes the contractor is expected to provide their own QC work.

In order for the retention and recovery of the enriched solution to be possible, a system including lining and drainage is constructed at the base of the ore pile, prior to placing the ore. Following some re-contouring of the area to receive ore, a layer of material is placed to assist in retaining the solution. This layer is sometimes referred to as “clay”. The term “clay” has a very specific meaning to geoscientists such as geotechnical engineers, who commonly lead the design of heap leach facilities. The actual goal of that layer is to achieve a desired permeability. Permeability indicates the rate an aqueous (predominately water) solution will pass through the soil. Geotechnical engineers have come to refer to this clay layer as a low

hydraulic conductivity soil layer (or LHCS layer) because the layer may have soil types other than clay as its key component. Hydraulic conductivity, in this case, means the same thing as permeability. In some cases, the LHCS layer may be replaced in part or in whole by a geosynthetic product known as a geosynthetic clay liner (GCL). The design engineer must take great care when designing with a GCL, due to the potentially low internal shear strength of the GCL, which may create slope stability considerations.

Immediately above this clay layer is typically placed a geomembrane. A geomembrane is in a family of geosynthetic products. In this case, the geomembrane comes in a roll and is unrolled over the clay layer. A geomembrane is typically made of some form of specialty plastic that is manufactured to withstand the sometimes-harsh environment in which it may be deployed at a mine site. High-density polyethylene (HDPE) is currently among the most common plastics used in geomembranes for heap leach facility construction. Once unrolled, the geomembrane is “welded” together to form a tight seal, greatly limiting the amount of enriched fluid that can escape from the containment of the heap leach facility. The welding is accomplished by various means that essentially cause the localized melting of one layer of geomembrane into another layer.

To facilitate the recovery of the enriched solution from the heap leach facility, a drainage layer is constructed on top of the geomembrane. This layer is sometimes referred to as the overliner. This layer consists of highly permeable sands and gravels, and normally also has a network of slotted pipes to assist in more rapidly removing the enriched solution from the heap leach facility. The design of this layer is very specific to the project materials, but normally varies between 0.5 m (18 inches) and 0.9 m (3 feet) in thickness.

The sands and gravels used for the overliner often have to be processed in order for them to comply with the project’s technical specifications. While some projects have naturally occurring materials that comply with the project requirements, other projects will have to improve their materials, either by crushing their material into a smaller product, or screening it to remove some materials, or a combination of crushing and screening. In these cases, specialty crushing and/or screening plants are used.

Occasionally there are other layers involved in a heap leach facility. The construction of some of these components may be completed by one or more independent contractors.

Pipe installation and compaction

The practice of installing pipes in civil engineering projects differs significantly from the practice of installing pipes in heap leaching projects. Pipes installed in heap leach facilities are often buried very deeply, while those in civil engineering applications are usually quite shallow. For example, the Plastic Pipe Institute (2009) indicates that for a pipe surrounded by crushed rock and buried by dumped fill can only be buried to a maximum depth of 6.4 m (21 ft). ADS (2016) indicates that pipe can be buried up to a depth of

11.3 m (37 ft), depending on the pipe diameter and backfill characteristics. Those limitations on burial depth would never be useful in heap leaching projects. A discussion of these practices is provided to expand on this matter.

Pipe installation in civil engineering projects

The piping manufacturer company Advanced Drainage Systems (ADS, 2018) provides a good source for guidance on adequate depth for pipe burial when it is subject to trafficking. For example, they state that a 0.3 m (12-inch) cover is adequate to protect a 0.3 m (12-inch) diameter pipe. Table 1 from that ADS publication is reproduced below.

Table 1: Table of minimum pipe burial depths (from ADS, 2018)

Pipe inside diameter		Minimum cover	
mm	in	m	ft
100	4	0.3	1
150	6	0.3	1
200	8	0.3	1
250	10	0.3	1
300	12	0.3	1
375	15	0.3	1
450	18	0.3	1
600	24	0.3	1
750	30	0.3	1
900	36	0.3	1
1,050	42	0.3	1
1,200	48	0.3	1
1,350	54	0.6	2
1,500	60	0.6	2

That specification is found in the *ADS Drainage Handbook*. The handbook provides instructions on placing and compacting pipe backfill for the lower half of the pipe and for its upper half. For the lower half, ADS recommends placing the fill in 4- to 6-inch-thick lifts and compacting it in accordance with product specific guidelines listed in their appendix. ADS indicates that this zone, “is the most important since it is this layer that provides the pipe with support against the soil and traffic loadings”. For the backfill around the upper one-half of the pipe, ADS indicates, “this area of the backfill anchors the pipe and ensures that loads are distributed as evenly as possible into the haunching [lower one-half of the pipe]”. Refer to Figure 1 for illustration of the haunch of a pipe. Material placement and compaction guidelines are provided. ADS

indicates that these zones are to be compacted to a minimum of 90% of standard Proctor density, indicating that a considerable degree of compaction is needed. The Plastic Pipe Institute (2009) states, “Do not allow rocks or other hard objects in the vicinity of the buried [pipe]”.

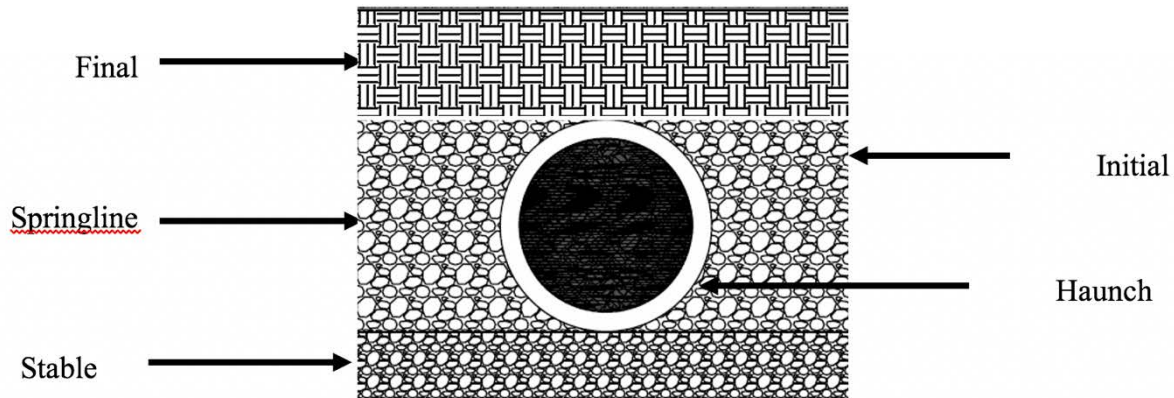


Figure 1: Illustration of pipe burial showing terminology (modified after ADS, 2016)

Pipe installation in heap leaching projects

In a heap leach facility, pipes are typically placed on the top of the lining surface (for example on the geomembrane liner). Once positioned and (potentially) stabilized in place, low ground pressure equipment is used to spread overliner onto the pipe. At this point, it is very important that an observer visually assesses whether the overliner material is flowing freely, and into the haunch area of the pipes. If the overliner is too wet, the material is unlikely to flow into this area. It is imperative that dry material is used so that the haunch is adequately supported.

The material surrounding the pipes is typically not compacted. This is because such compaction would likely damage the geomembrane, and the compaction would reduce the permeability of the overliner. Table 2 provides a summary of some of the differences between pipe placement in civil applications versus in mining applications.

Table 2: Summary of differences in pipe placement in civil versus mining applications

Civil application	Heap leaching application
Install is in a well-formed trench	Install is on an open surface
Middle one-third of bedding should be loosely placed	Bedding layer is uniformly compacted
Compact fill into haunch in prescribed layers	Ensure the fill flows beneath the haunch
Remainder of embedment zone placed and compacted in layers	Material is not compacted
Compactive effort depends on soil type	Material is not compacted

The industry practice for placement of overliner is essentially to deliver the overliner using the least amount of equipment trafficking as possible/practicable so that compaction of the overliner material is minimized. This is done by identifying delivery paths (“roads”) for the haulage equipment (trucks) to deliver the overliner to the pad. From there the material is spread into place using low ground pressure dozers. The roads are dedicated trafficking areas for trucks delivering overliner to the pad. The thickness of the overliner is increased along the length of the roads in order to better protect the liner/lining and drainage system and to reduce the amount of crushing of the overliner. Once all the overliner delivery is complete, the additional overliner along the roads is removed, and the remaining material once beneath the road is ripped, if necessary. This construction sequence is chosen to minimize damage to the liner/lining and drainage system and to reduce compaction to the overliner material.

The use of a test fill is a proof of principle in that it is a field-scale demonstration of the proposed construction method. In the case of a heap leach facility, a test pad would be used to assess the integrity of the lining/liner and drainage system when subjected to the forces applied by the contractor’s proposed equipment. It is common to document how the test is carried out, indicating the number of passes of each piece of equipment passing over the test fill, and often test fills of various thicknesses of overliner are included. It is important to know how many times each piece of equipment has passed over the test fill, as pipes and other buried items may survive a limited amount of equipment passes, but not a greater number of passes.

A test fill is especially important in understanding how the pipes will perform when they are buried in a relatively shallow state, i.e., when the overliner layer is being constructed and equipment is trafficking over the pipes.

Deeply buried pipe

As discussed, pipes installed in heap leach facilities are very often installed to be at much greater burial depths than is advised by pipe manufacturers. In the following sections we will explore this further.

The Ohio experiment

In the Ohio experiment, Sargand et al. (2001) buried HDPE and polyvinyl chloride (PVC) pipes at depths of 6.1 and 12.2 m (20 and 40 ft). Soil pressure cells were installed above the pipes at various depths. One of the soil pressure cells was installed approximately 25 mm (1 inch) above the pipes. The test results clearly demonstrated that the vertical earth pressure was normal except near the pipe, where there was a significant reduction in the pressure. Although the paper by Sargand et al. (2001) has the phrase “deeply buried thermoplastic pipe” in the title, their pipe burial depths are relatively shallow for heap leach facilities. They offer very little in the way of explaining why the vertical overburden pressure decreases close to the pipe.

Sargand and Masada (2003) provide additional analysis. They state that “the soil pressure over the pipe is influenced by the vertical settlement ratio between the pipe and the adjacent soil column at a horizontal surface above the pipe.” This concept was first explained by Marston and Anderson (1913). They indicate that “frictional resistance relieves part of the vertical pressure [to the pipe] near the sides of the ditch [walls], so that at the level of the top of the pipe the vertical pressure of the filling material is much heavier in the middle of the ditch than at the sides”. And “hence the pipe must be strong enough to carry safely the entire weight of the ditch filling materials above the top of the pipe less the friction of the filling against the sides of the ditch”. Since pipes in heap leach facilities are rarely installed into excavated trenches, this case holds no merit for our application. Marston and Anderson (1913) provide no real discussion of arching.

Terzaghi’s theory of arching

According to Terzaghi (1943), “if one part of the support of a mass of soil yield while the remainder stays in place the soil adjoining the yielding part moves out of its original position between adjacent stationary masses of soil. The relative movement within the soil is opposed by a shearing resistance within the zone of contact between the yielding and the stationary masses. Since the shearing resistance tends to keep the yielding mass in its original position, it reduces the pressure on the yielding part of the support and increases the pressure on the adjoining stationary part. This transfer of pressure from a yielding mass of soil onto adjoining stationary parts is commonly called the arching effect, and the soil is said to arch over the yielding part of the support”.

Terzaghi (1943) went on to explain this with an analogy. In the analogy, there is a layer of sand above a solid base. In the center of the solid base is a trap door (Terzaghi refers to it as a “strip”, rather than as a trap door). If the trap door is opened slightly and slowly, sand in the immediate vicinity of the trap door will subside into the newly created void. Frictional forces within the remaining sand mass resist full movement of sand into the void. The reduction of stress on the trap door is balanced by an increase in stress elsewhere on the solid base. In the case of a deeply buried pipe, if the crown of the pipe yields slightly (i.e., it acts as Terzaghi’s trap door), the stresses of the overlying soil are shed to the surrounding materials, and the vertical stresses on the pipe decrease toward zero. This is Terzaghi’s theory of arching in practice and this explains why there was a reduced vertical stress near the pipe crowns in the Ohio experiment. The crowns on those pipes buckled inward slightly, acting as Terzaghi’s trap door. Soil above was stabilized due to frictional forces, and the reduction of stress on the crown of the pipes was shed to the soil surrounding the pipes. Yielding of the crown is also referred to as ring deflection (e.g., Lupo, 2005).

It is because of the stress-relieving effect of soil arching that pipes can survive deep burial in heap leach facilities. That is, if they are otherwise well-constructed.

Pipe integrity

The theoretical performance of buried flexible pipes was published by Burns and Richard (1964) for an elastic circular pipe deeply buried in an infinite elastic medium subjected to horizontal and vertical loading. Moser (1997, in Suleiman, 2002) stated that the greatest shortcoming of the Burns and Richard solution is its assumption of double symmetry. That is, the assumption that the soil-pipe system is symmetric about horizontal and vertical axes. In this solution, no assumption was made (or needed) for the pressure distribution around the pipe. Smith (2004) reported that the Burns-Richard solution was derived from classical civil engineering applications with maximum burial depths of 5 to 20 m beneath rigid structures such as pavements or buildings. Thus, for deeply buried pipes in a heap leach facility, the Burns-Richard solution, as it stands, is of little use.

As discussed in Rosemont (2010) a series of large-scale laboratory vertical load tests was conducted on a 600 mm (24-inch) diameter corrugated plastic tubing (CPT) pipe manufactured by Advanced Drainage Systems Incorporated (ADS). The testing was conducted at the U.S. Bureau of Reclamation (USBR) in Denver, Colorado using a large-scale cross-section of a heap leach pad liner system including soil liner, geomembrane, protective soil, a drainpipe, and a drainage layer. The testing was conducted to evaluate the performance of the pipe under the loading conditions expected on a heap leach pad stacked to a height of 170 m (560 ft). The testing indicated a deformation of 130 mm (5.1 in), approximately 21%, at 140 m (460 ft) and a vertical deformation of 178 mm (7.02 in), approximately 29%, at 170 m (560 ft). As discussed by Lupo (2005), non-pressurized HDPE pipes appear to suffer serious deformation and buckling if the crown deflection exceeds 20 percent. Lupo (2005) indicates that a maximum crown deflection of 15 percent can be used for design if the reduction in flow area due to crown-deflection is tolerable. In Lupo's opinion, the degree of crown buckling observed in the USBR tests would not be advisable.

Lupo (2005) discusses the shortcomings of the Burns-Richard approach. Lupo states that the solutions presented by Burns and Richard, and as well by Höeg (1968, in Lupo, 2001) assume both the properties of the pipe and the soil are constant and that they are linear elastic. As stated by Lupo, these simplifications limit the usefulness of these equations for flexible pipe design, since both the pipe and ore materials have non-linear behaviour under compression. Lupo (2001) provides a modified version of the Burns-Richard equation to account for non-linear ore and pipe behaviour as well as arching.

As an alternative, one could utilize numerical modeling of buried pipe, as discussed by Saadeldin et al. (2015).

Chapter 8 of the Plastic Pipe Institute's manual (2009) discusses Florida's Department of Transportation 100-Year Service Life Protocol, which was "to ensure that pipes are manufactured with [of corrugated HDPE pipes manufactured with virgin] materials sufficient to prevent brittle failures and with sufficient antioxidants to prevent chemical failures, prior to 100 years of service". This protocol can be used

if one wishes to calculate the service life of a pipe; however, in the author's opinion, most pipes manufactured by reputable pipe vendors will have a service life longer than the life of a heap leach facility.

Other advice

Although it has not been discussed, so-called “single wall” corrugated pipes have little-to-no application as buried pipes in heap leach facilities. Buried pipes should either be the dual-wall variety (smooth interior, corrugated exterior) or HPDE pipes with a considerable wall thickness. Coupling of pipes should always follow the manufacturer's recommendations, and when using bell and spigot pipes, it is very important to ensure that the male pipe is inserted completely into the female pipe.

Also, it can be very beneficial to carry out camera inspections of pipes. This will tell you if the normal soil arching is occurring or whether excessive deformation is taking place. A photograph of such a camera inspection is provided in Figure 2.



Figure 2: Photograph of camera being used for pipe inspection

Another matter pertains to large-diameter pipes. There is a tendency for designers to indicate a locally thick zone of overliner being placed over larger-diameter pipes. The other, lesser chosen, option is to install these pipes in a trench. Not only does this lead to a simpler overliner placement operation, but if installed in trenches, the effects of arching are improved for these larger-diameter pipes. This concept is illustrated in Figure 3.

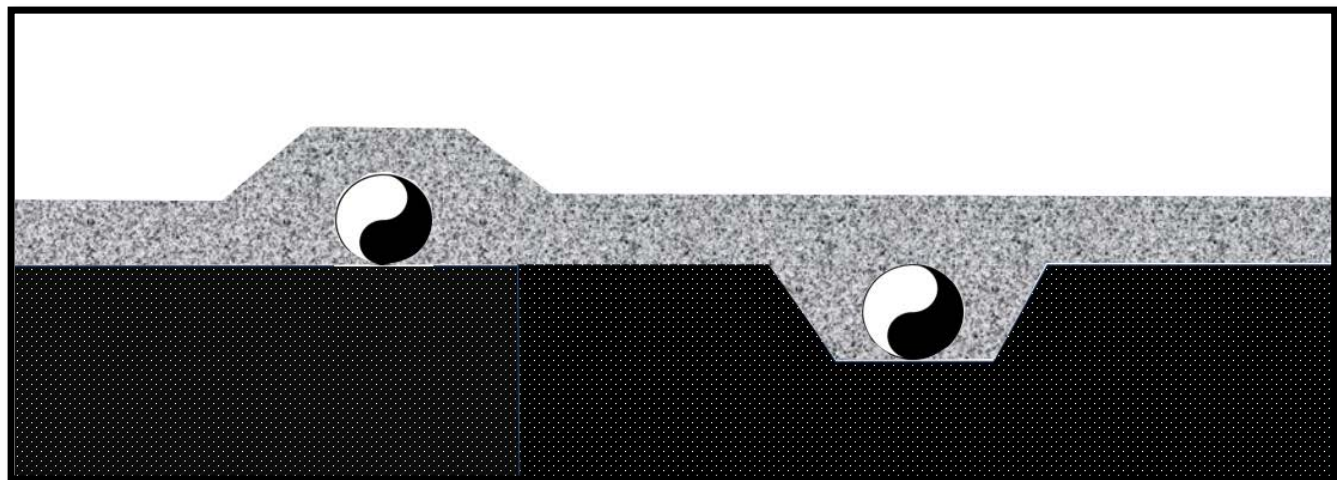


Figure 3: Illustration of large-diameter pipe placement options, mounded versus trenched

And one final word on the placement of pipes running in parallel. Such pipes should be placed far enough apart to allow the overliner to flow into the haunches of each pipe, which is at least somewhat dependent on the properties of the overliner.

Conclusion

As has been discussed, pipes installed in the drainage layers of heap leach facilities are typically buried much deeper than is advised by the pipe manufacturers, and much deeper than in most civil, non-mining applications. Probably the most important factor in constructing a pipe in a heap leach facility is to make sure the fill material (usually overliner) surrounding the pipes is dry and granular, and that it flows freely into the haunch of the pipe. This provides support for the lower half of the pipe and allows soil arching to take place. If the fill is not dry, or if the work is taking place during a precipitation event (be it rain or snow), all backfilling operations should be halted.

Since the cross-sectional area of the pipe will be reduced as arching occurs, the designer should take this into consideration when making pipe flow calculations. An area loss of around 50% is often assumed in order to be conservative.

Lastly, it can be very beneficial to carry out camera inspections of pipes. This will show whether the normal soil arching is occurring, or whether excessive deformation is taking place.

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Criteria to Estimate and Use Peak and Residual Shear Strength of Interfaces in Heap Leach Pads Design

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Abstract

One of the essential geotechnical aspects of heap leach pads design is related to the interface shear strength of the underlying geomembrane liner system, which isolates the leachate solution and ore from the natural ground. Conventional liner system design considers a double contention formed by a geomembrane liner (GM) over a compacted soil liner (SL) layer. The overliner (OL) and solution collection pipe systems are placed over the GM. The OL is generally granular soil that protects the GM and allows the solution drainage.

The interface shear strength between GM and SL is usually estimated through a large-scale direct shear (LSDS) test, which typically provides the peak and post-peak shear (or residual) strength parameters. The slope stability analyses are run conservatively, with the lowest interface shear strength corresponding to the residual strength. Usually, the LSDS test results show the GM-SL interfaces as having the weakest post-peak and residual shear strengths. However, this assumption could be the opposite because the recommended minimum asperity height for textured geomembrane is 0.4 mm according to GRI-GM 17 (GRI, 2021a).

The limit equilibrium method (LEM) is commonly used to evaluate the stability of a heap leach pad. In this method the interface is modeled as a continuum element (like a thin soil layer). The analysis criterion that is most widespread and accepted in practice usually assumes that mobilized shear strength reaches or exceeds the peak shear strength of the materials; thus, post-peak shear strengths are usually considered in stability analyses. However, this criterion can be very conservative in estimating the factor of safety (FS).

This paper proposes empirical formulas to estimate the GM-SL interface peak and post-peak shear strengths based on the asperity height of GM and SL classification (USSC). Also, empirical formulas for the OL-GM interface are proposed based on the gravel content of the OL. In addition, this research suggests using peak shear strength for flat zones and post-peak shear strength for stepped zones in statics slope analysis of heap leach pads.

Introduction

Among the most significant mining structures are leach piles or heaps, in which ore is dumped and irrigated with a solution that dissolves the mineral impregnated in the rocks. The solution with mineral is called the pregnant leach solution (PLS). The PLS passes through the ore in the heap via the collection and liner system located on the bottom of the heap. The liner is usually composed of clayey soil, or geosynthetic clay liner (GCL), a geomembrane (GM), granular drainage, and a protection material (overliner) that is supplemented with drainpipes at controlled spacing (Thiel and Smith, 2004).

Critical aspects of the geotechnical design of leach pads are associated with the height of the ore heap, liner system, ore saturation, site location topography, and site geology. In practice, a conservative approach is usually incorporated into slope stability analysis by using post-peak shear strength values.

Liner system design

The liner system (see Figure 1 for an illustrative example) is a containment that isolates the leaching solution from the natural terrain. The characteristics of liner systems depend on environmental regulations, site conditions, risks associated with solution leaks, operation characteristics, operating equipment, height of the heap ore, slope stability, etc. (Romo, 2015). The design of liner systems often includes a composite containment system composed of a GM over a low permeability layer (compacted clay or GCL).

In Peru, most of the leach pads are in the highlands, where underground water or lagoons are often very close to mining projects, so the current standard for liner systems design includes composite containment.

The liner system design for a leach pad depends on earthworks for grading (cut and fill), selection and characterization of geosynthetics, low permeability soil, and overliner materials. Also, the liner system design must address the operating and environmental conditions (Lupo, 2010).

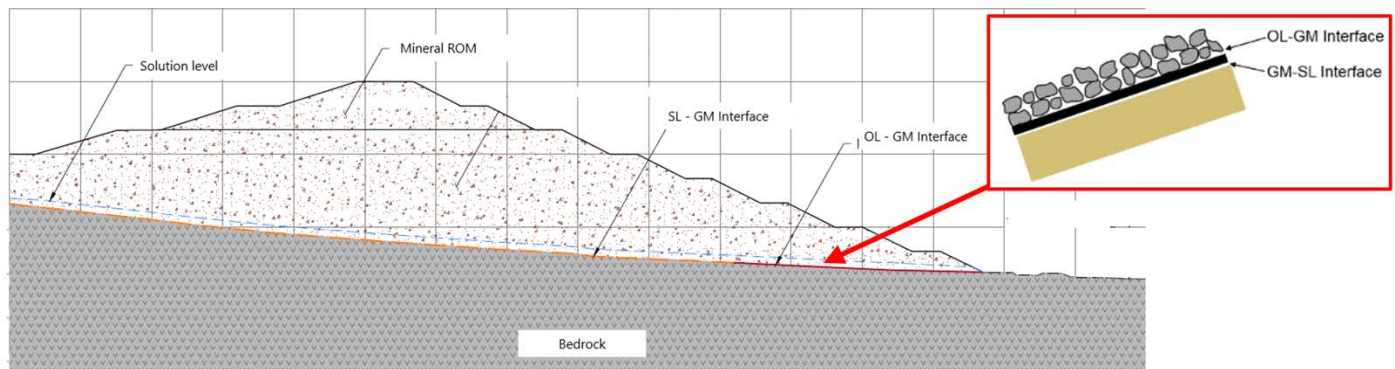


Figure 1: Heap leach pad and composite liner system

Geomembrane selection

The selection of a suitable geosynthetic liner is crucial for leach pad design because the ore is stacked over the composite liner, and the heap height is continuously increasing. The thickness of the geomembrane (GM) is generally selected based on its puncture resistance.

The extensive use of linear low density polyethylene (LLDPE) GM in leach pad operations has shown that it is suitable for containing corrosive acid drainage and metal leaching products for at least 20 years. However, there is not enough data regarding its long-term behaviour (50 to 100 years) (Renken et al., 2007). The behaviour of the GM under loaded conditions is strongly influenced by its interaction and compatibility with the components of the composite liner, so the design requires a thorough understanding of the interaction between these components, the load applied, the type of solution, and the solution level.

For mining projects, GM manufacturing is generally carried out according to the relevant Geosynthetic Research Institute (GRI) standard specifications, in addition to the recommendations of engineers and consultants specialized in heap leach facilities. GRI's standard specifications GM13 (GRI, 2021b) and GM17 (GRI, 2021a) apply to high density polyethylene (HDPE) and LLDPE geomembranes, respectively; however, some engineers may recommend more rigorous specifications.

The main restriction of the geotechnical design of heap leach pads is related to the strength of the liner system, which usually provides at least two interfaces with low shear strengths due to contact with the GM. These interfaces control the stability conditions. Some efforts have been made to adequately model and predict this kind of shear strength (e.g., Ivy, 2003; Yesiller, 2005; Blond and Elie, 2006). In previous research (Ayala and Huallanca, 2014), the authors noted that GM-SL interface shear strengths increase with GM asperity height (a), normal stress (σ_n) increment, and the increment of granular material content in the soil liner.

This paper proposes empirical formulas to estimate the shear strength of GM-SL and OL-GM interfaces.

Large-scale direct shear tests (LSDS)

The shear strength of OL-GM and GM-SL interfaces is measured by large-scale direct shear tests (LSDS) following the ASTM D5321 standard. Figure 2a shows the typical behaviour of shear strength vs. displacement. The shear stress gradually increases until the peak shear stress ($\tau_{p,1}$); it should be noted that beyond the maximum displacement of the LSDS there is no additional reduction of the shear strength, indicating that in this case, as many other observed testing results, the post-peak shear strength is the same as the residual strength. After the peak, the shear stress falls, as demonstrated in the $\tau_{p,1} - \tau_{r,1}$ path in Figure 2a. Generally, several test specimens with soil and GM are tested, varying σ_n ; for each normal stress, the shear stress vs. displacement curve is obtained, as shown in Figure 2 (a). The peak or residual shear

stress can be plotted against the corresponding σ_n , as shown in Figure 2 (b). This envelope could be defined by the Mohr-Coulomb model (angle of friction and adhesion); however, interface shear strength envelopes often show non-linear behaviour, as noted by Stark and Choi (2004).

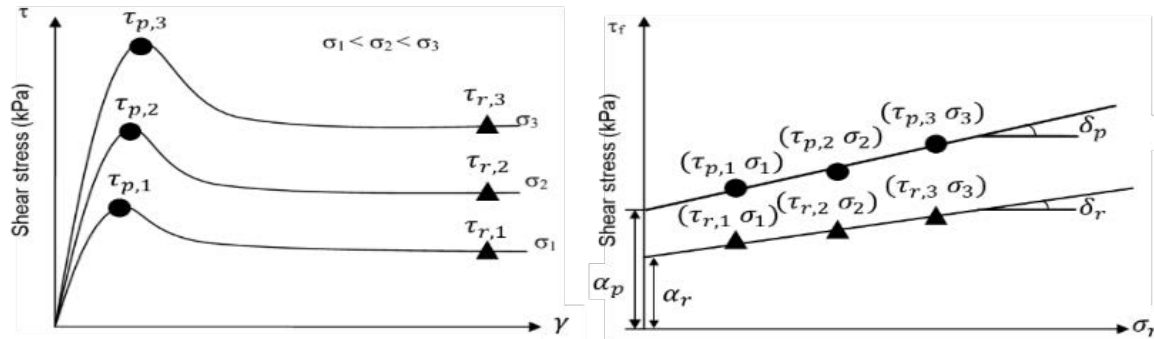


Figure 2: Typical plots from LSDS. Left (a) shear strength vs. displacement; right (b) shear vs. normal stress

Data selection

The collected data shows the typical shear stress-displacement behaviour of OL-GM and GM-SL interfaces. This research has compiled 64 LSDS tests on OL-GM interfaces with smooth GM and 388 LSDS tests on GM-SL interfaces mainly with side textured GM. Compiled LSDS tests were carried out with site-specific materials (OL and SL) of different mining projects and with GM, principally LLDPE, at each σ_n (100, 200, 400, and 800 kPa). The test data were divided into five groups based on interface type, soil classification according to the Unified System of Soil Classification (USSC), and fines content. The main features of soil samples and GM type of each group are summarized in Table 1, where the average (μ) of particle size content also is indicated.

Table 1: Summary of interface types

Interface type	Interface condition	Soil classification	Gravel content (%)	Sand content (%)	Fines content (%)	Liquid limit (%)	Plastic index (%)	Number of tests
OL-GM	Smooth	GP, GM, GM, GW and others	33 – 98 $\mu=65.5$	0 – 29 $\mu=20.6$	0 – 38 $\mu=13.9$	34.5	12.1	64
		GC	30 – 60 $\mu=41.2$	9 – 39 $\mu=29.4$	15 – 46 $\mu=33.3$	32.7	14.7	48
		SC	0 – 34 $\mu=21.7$	29 – 67 $\mu=39.3$	19 – 50 $\mu=38.8$	34.9	16.0	128
GM-SL	Textured (asperity height between 0 and 0.6 mm)	CL & CH (fines content < 65 %)	0 – 28 $\mu=12.1$	15 – 55 $\mu=32.1$	46 – 69 $\mu=55.8$	35.8	16.9	104
		CL & CH (fines content > 65 %)	0 – 20 $\mu=6.3$	3 – 40 $\mu=17.0$	54 – 97 $\mu=76.8$	48.1	24.6	108

The data regarding peak and post-peak shear strengths of the OL-GM interfaces are shown in Figures 3 (a) and 3 (b), respectively, separated by each σ_n (in different colours). It should be noted that in most of the cases the post-peak shear strength at approximately 70 to 80 mm displacement corresponds to the residual strength, because there is no further strength reduction after the maximum displacement of the LSDS. These figures also show the influence of gravel content of OL in their interface shear strength.

The data regarding the post-peak shear strength of the GM-SL interfaces vs. GM asperity height are shown in Figure 3 (c). This figure does not present the influence of soil classification or fines content. There is no clear tendency for peak shear strength of the GM-SL interface vs. asperity height.

The Mohr-Coulomb model (a failure envelope characterized by an angle of friction and cohesion or adhesion) as applied to OL-GM interface shear strengths is shown in Figure 4, in which three average lines are proposed for different gravel contents. To achieve a better relationship to estimate the OL-GM interface shear strength, expressions based on gravel content (G) are proposed in this paper.

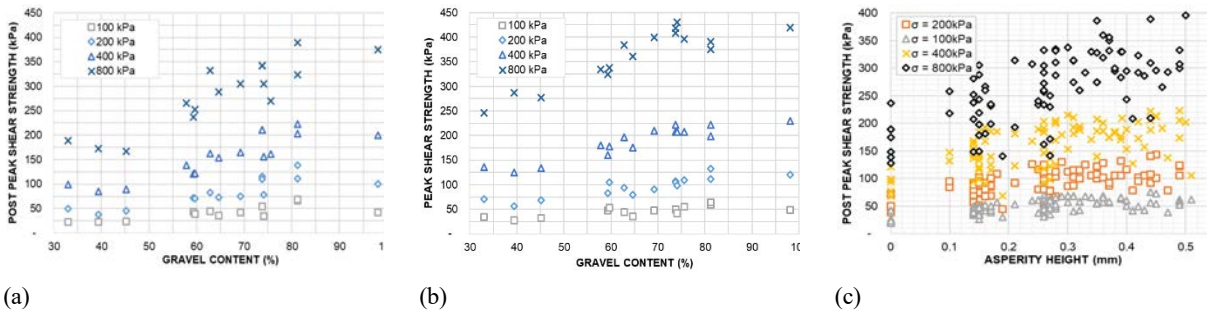


Figure 3: (a) Post-peak shear strength vs. gravel content in OL-GM; (b) peak shear strength vs. gravel content in OL-GM; (c) post-peak shear strength vs. asperity height in GM-SL interface

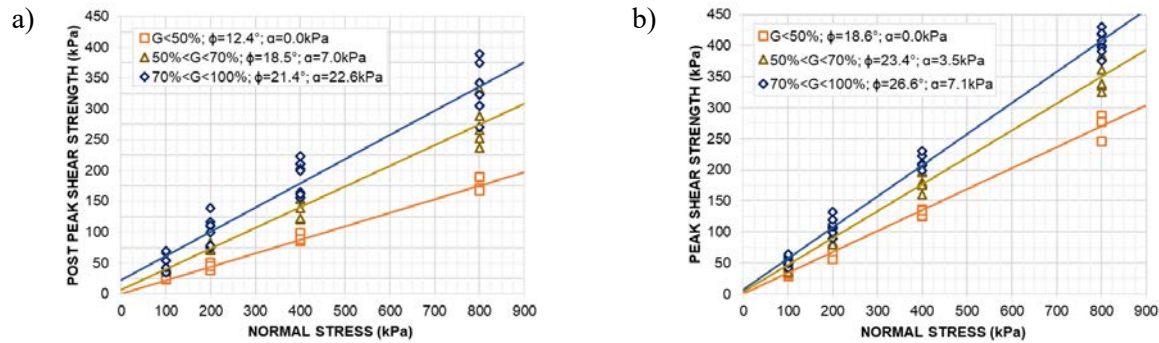


Figure 4: Average failure envelopes for different gravel content ranges: (a) post-peak shear strength vs. normal stress; (b) peak shear strength vs. normal stress

Proposed relationships for interface shear strength

The average shear strength envelope was obtained by linear and non-linear (polynomial) regressions. The “goodness of fit” was verified with the correlation coefficient (R^2); R^2 close to 1.0 means that most of the

variability in shear strength can be attributed to the considered variable (asperity height or gravel content). The fit of the average shear strength envelopes produces R^2 values greater than 0.50, which means that empirical formulas can be defined.

The interface shear strength could be greater or less than the estimated average. An overestimation of the interface shear strength can lead to inadequate physical stability analysis. Therefore, it is recommended that shear strength is defined with confidence levels, in which there is confidence with a probability (P) that the interface shear strength will be at least the estimated shear strength. Hence, a statistical treatment of data is proposed for envelopes with confidence levels. Probabilities associated with $\pm 1S$ (16 and 84% confidence) were considered. The “goodness of fit” test (Anderson and Darling, 1952) was applied to verify if the difference between the estimated and the observed values has a normal distribution.

OL – GM interface shear strength relationship

The shear strength relationship for the overliner-smooth geomembrane (OL-GM) interfaces for each σ_n was defined based on the influence of overliner gravel content. Regarding the data for this study, the following observations are noted: most of the data were obtained from LSDS tests on interfaces with LLDPE geomembrane; GM thickness has not been considered; most of the overliner corresponds to sub-angular and sub-rounded gravels; the maximum particle size of overliner was 50 mm; and the maximum σ_n was 800 kPa for all tests.

Figure 5 presents the proposed relationships between gravel content and (a) post-peak shear strength and (b) peak shear strength. Figure 6 shows the same relationships with both lower and upper confidence boundaries for one standard deviation (S). Other factors, such as the angularity of gravel particles, can also influence interface shear strength. Otherwise, the highest shear strength is not guaranteed with high gravel content. If an adequate gradation of the soil is not evaluated, the effective contact area between soil particles and GM would be lower, which reduces the interface shear strength. The evaluation of the liner system must also consider these aspects in heap leach pad design.

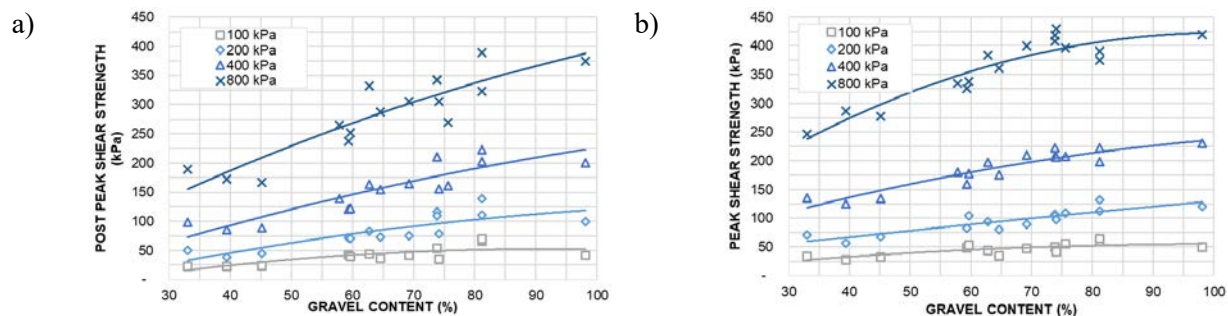


Figure 5: Proposed relationships between gravel content and (a) post-peak shear strength and (b) peak shear strength

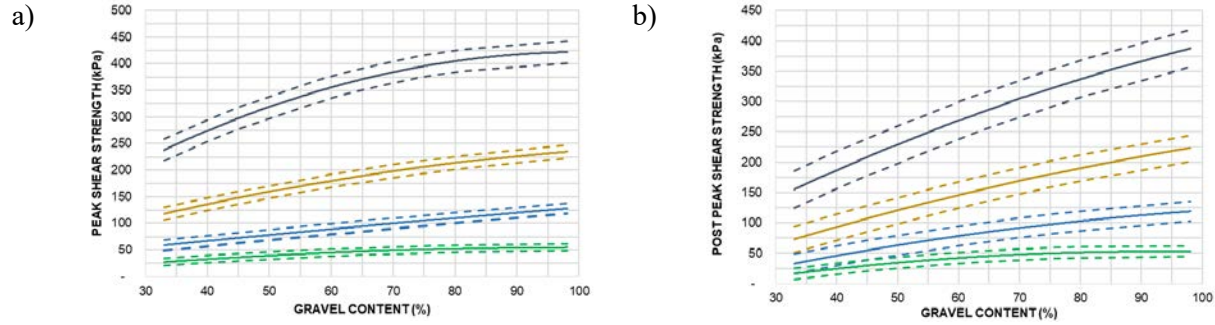


Figure 6: With confidence intervals ($\pm 1S$) based on relationships between gravel content and (a) peak shear strength and (b) shear strength (for σ_n : 100, 200, 400 and 800 kPa)

OL-GM interface shear strength discussion:

There is a clear nonlinear tendency of shear strength increment for post-peak and peak values. The standard deviation of the peak shear strength relationship is lower than the post-peak shear strength relationship; thus, there is a better correlation between peak shear strength and gravel content; for post-peak values, the S value increases for high normal stresses, as noted in Figure 6 (b).

The proposed empirical formulas based on gravel content (G) are summarized in Table 2 for post-peak (τ_r) and peak shear (τ_p) strength for different normal stresses. These relationships could be used in stability analysis where the failure occurs at the interface. The S and R^2 values obtained for each relationship are also indicated in Table 2.

Table 2: Empirical formulas for OL-GM interface shear strength based on gravel content

σ_n (kPa)	τ_r (kPa)	S (kPa)	R^2	τ_p (kPa)	S (kPa)	R^2
100	$\tau_r = -0.01G^2 + 1.92G - 35.85$	9.1	0.61	$\tau_p = -0.01G^2 + 1.17G - 4.60$	6.7	0.58
200	$\tau_r = -0.01G^2 + 2.57G - 41.70$	16.3	0.69	$\tau_p = -0.001G^2 + 1.23G + 20.05$	10.0	0.77
400	$\tau_r = -0.01G^2 + 3.68G - 37.16$	21.6	0.77	$\tau_p = -0.01G^2 + 3.48G + 17.57$	12.0	0.87
800	$\tau_r = -0.02G^2 + 5.72G - 15.82$	30.6	0.80	$\tau_p = -0.04G^2 + 8.07G + 15.55$	20.4	0.87

GM – SL interface shear strength relationship

The empirical formula for textured geomembrane–soil liner (GM-SL) interface was obtained by the influence of asperity height and the type of soil; like the OL-GM relationship, these results could be used to run a stability analysis of heap leach pads for block failure, or maybe the combination of both formulas.

Regarding the data for this study, the following observations are mentioned: most of the data were obtained from LSDS tests on interfaces with LLDPE SST geomembrane; geomembrane thickness has not been considered; and the maximum σ_n was 800 kPa for all tests and silts (ML or MH) were not evaluated due to a lack of correlation and low data availability. Figure 7 presents the proposed relationships between asperity height and post-peak shear strength for different soil types.

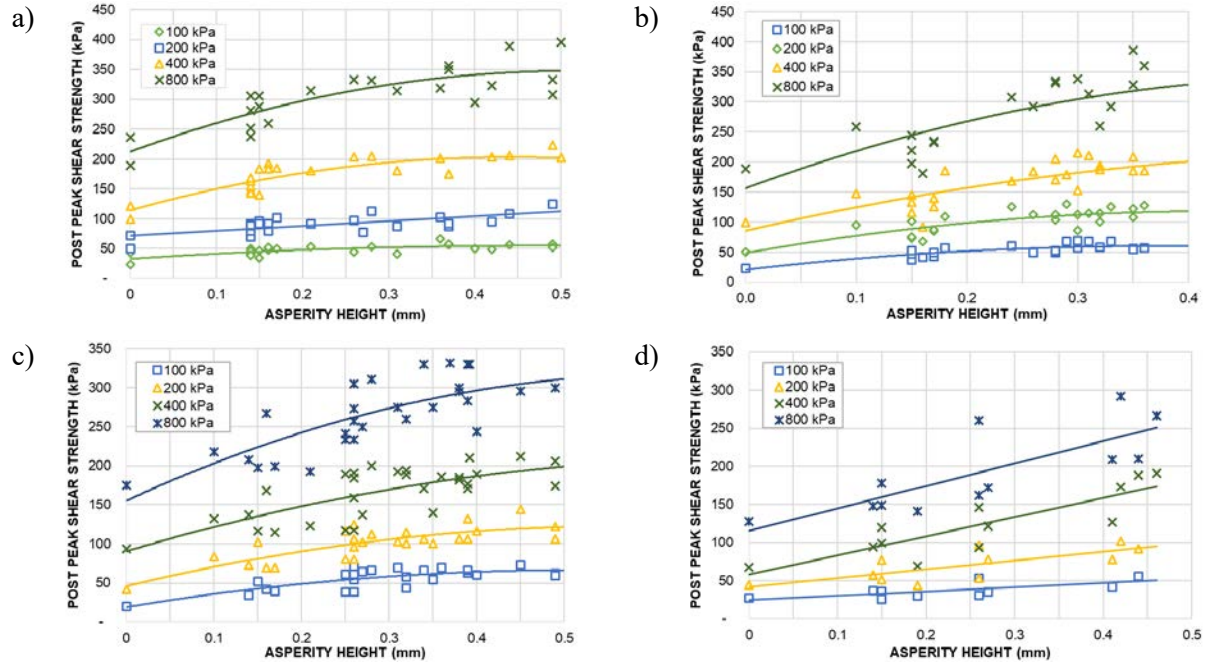


Figure 7: Relationship between asperity height and interface post-peak shear strength for the following soil types: (a) GC; (b) SC; (c) CL (F < 65 %) and (d) CL (F > 65 %)

Figure 8 shows relationships with lower and upper boundaries considering $\pm 1S$. The influence of gravel content is noted in shear strength; it can be for friction generated between gravel particles and GM.

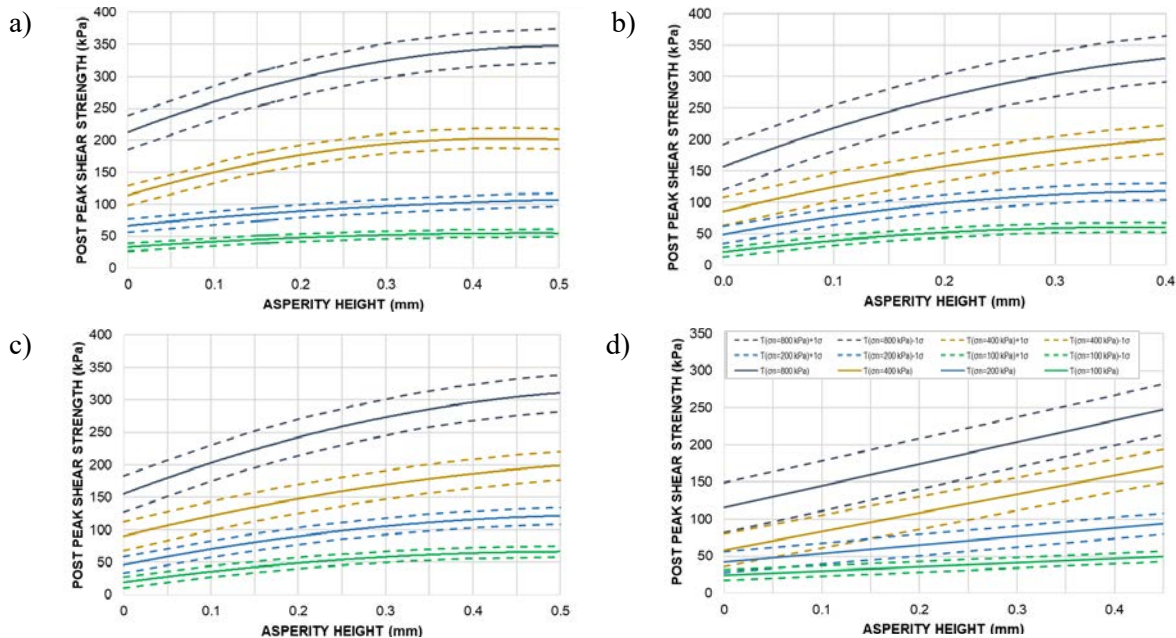


Figure 8: Graphs with confidence intervals of $\pm 1S$ based on relationships between asperity height and interface post-peak shear strength for the following soil types: (a) GC; (b) SC; (c) CL (F < 65 %) and (d) CL (F > 65 %)

GM-SL interface shear strength discussion:

There is a clear nonlinear tendency of shear strength increment for post-peak and peak values, where the behaviour tends to be asymptotic at 0.4 mm asperity height. Empirical formulas for peak shear strength were not defined due to low R^2 values and high standard deviations. The standard deviation of post-peak shear strength increases for higher normal stresses for all GM-SL types, as noted in Figure 8. The proposed empirical formulas are summarized in Table 3 for post-peak shear strength based on asperity height and considering different normal stresses. The S and R^2 values estimated also are indicated.

Table 3: Empirical formulas for GM-SL interface post-peak shear strength based on asperity height

Interface	σ_n (kPa)	τ_r (kPa)	S (kPa)	R^2
GM-SL (CL, F < 65 %)	100	$\tau_r = -179.3a^2 + 184.39 + 19.02$	8.5	0.59
	200	$\tau_r = -234.42a^2 + 269.02 + 45.88$	12.9	0.65
	400	$\tau_r = -232.17a^2 + 333.15a + 90.5$	22.2	0.55
	800	$\tau_r = -414.45a^2 + 518.37a + 155.42$	28.0	0.61
GM-SL (CL, F > 65 %)	100	$\tau_r = 55.789a + 24.634$	7.2	0.51
	200	$\tau_r = 115a + 42.195$	14.3	0.55
	400	$\tau_r = 252.07a + 57.99$	22.2	0.73
	800	$\tau_r = 294.52a + 115.57$	33.9	0.61
GM-SL (SC)	100	$\tau_r = -287.26a^2 + 212.09a + 20.82$	7.6	0.56
	200	$\tau_r = -383.19a^2 + 326.66a + 48.43$	13.6	0.58
	400	$\tau_r = -344.73a^2 + 425.5a + 85.54$	22.4	0.62
	800	$\tau_r = -623.37a^2 + 680.41a + 156.34$	36.1	0.59
GM-SL (GC)	100	$\tau_r = -100.57a^2 + 94.059a + 32.85$	6.2	0.51
	200	$\tau_r = -103.03a^2 + 132.17a + 66.74$	10.4	0.52
	400	$\tau_r = -459.25a^2 + 405.22a + 114.05$	15.5	0.73
	800	$\tau_r = -517.56a^2 + 528.9a + 212.46$	26.6	0.70

Influence of the use of peak or post-peak shear strengths

Application of interface shear strength

Gilbert and Byrne (1996) indicated that the peak shear strength mobilizes between 1 and 15 mm of displacement; also, the post-peak shear strength can be as low as 30% of the peak strength. About these findings, the current investigation found the following:

- OL-GM peak shear strength is mobilized between 12 and 22 mm of shear displacement, GM -SL peak shear strength is mobilized between 7 and 32 mm.
- OL-GM and GM-SL post-peak shear strength can be as low as 40% of the peak values.

It is logical to use post-peak shear strength of the interface in the liner system design because progressive failures across the liner mobilize the shear stress. The suggestions of different authors for the use of peak or post-peak shear strength in the design are summarized below, listed from the most to the least conservative:

- Use residual post-peak shear strength for all conditions (Stark and Poeppel, 1994).
- Use post-peak shear strength of the interface with the lowest peak shear strength (Koerner and Bowman, 2003).
- Use peak shear strength at the base of the slope (flat areas) and post-peak shear strength along the steeper side-slope (Jones et al., 2000).
- Use peak strength at the top of the slope and post-peak strength at the base (Koerner and Bowman, 2003).
- Use peak strength for all static (non-seismic) conditions (Koerner and Bowman, 2003).

This paper evaluates the use of peak shear strength for flat zones and post-peak shear strength for stepped zones of heap leach pads, and the findings compare with the results of numerical analysis.

Stability analyses of heap leach facilities

The limit equilibrium method (LEM) is commonly used to evaluate the stability of slopes by calculating a representative factor of safety (FS). In this analysis, the interface is modeled as a continuum element (like a thin layer of soil). The analysis criterion that is most widespread and accepted in practice usually assumes that mobilized shear strength reaches or exceeds the peak shear strength of the materials; thus, post-peak shear strength is usually considered in stability analysis. However, this criterion may produce conservative estimates of the FS when post peak shear strength is used.

In the present study, the mobilization of shear stresses along interfaces below and above the GM has been evaluated by numerical analysis with finite elements, estimating the relative shear stresses (τ_{rel}) in each interface. The relative shear stress indicates the proximity of the mobilized shear stress (τ_{rel}) to the shear strength of the failure envelope (τ_{max}) for a specific σ_n (see Equation 1).

$$\tau_{rel} = \frac{\tau_{mob}}{\tau_{max}} \quad (\text{Equation 1})$$

Three case studies were performed using finite element method (FEM) and LEM analyses. One of the cases considers a 90-m deep heap, which is presented in Figure 9 (a). The interface shear strength parameters were obtained from large-scale direct shear (LSDS) tests conducted for both OL-GM and GM-SL interfaces. The envelopes of post-peak and peak shear strength vs. normal stress are presented in Figure 9 (b).

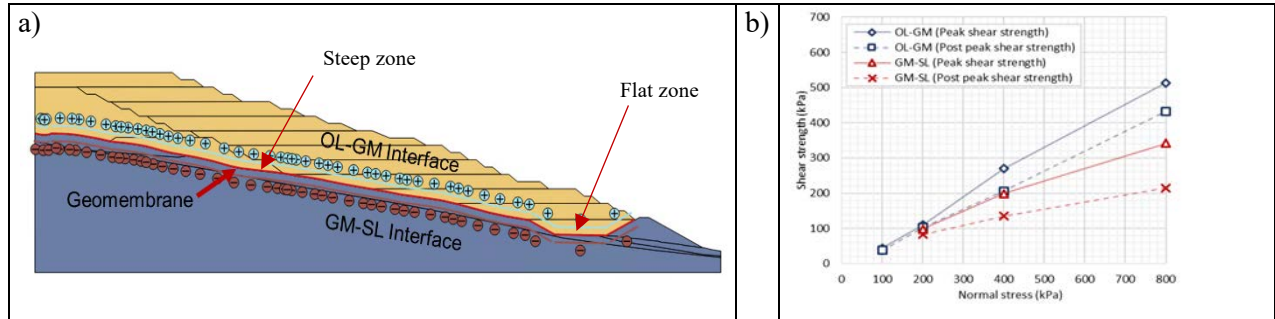


Figure 9: (a) Heap leach facility model considered in the numerical analysis with finite elements. (b) Results of LSDS tests used for stability analysis

To obtain more accurate FS, the FEM analysis were performed with the 2-D Plaxis program. The interfaces of the heap leach liner (OL-GM and SL-GM) were modeled with interface elements using the shear strength envelope calibrated with the peak shear strength. The FEM analyses verified that the shear stresses are equal in both interfaces. However, the relative shear stresses have significant differences, as seen in Figure 10. The FEM results demonstrate that interfaces at step zones are particularly more susceptible to reaching the fault limit ($\tau_{rel}=1,0$) but the peak shear strength is not mobilized. Additional failure mechanisms could cause stresses to overcome the peak strength (τ_{max}), mobilizing post-peak strengths and causing a progressive failure of the structure.

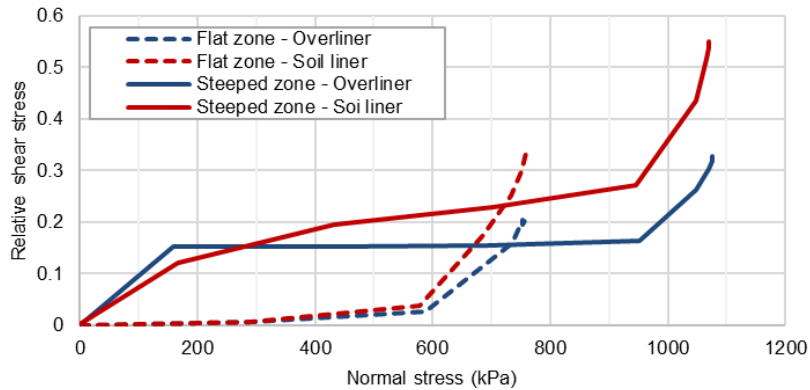


Figure 10: Relative shear stress vs. normal stress in flat and steep zones of a heap leach pad

The stability analyses using the LEM method were performed for four scenarios, considering block failures along OL-GM and GM-SL interfaces; peak and post-peak shear strength were considered in each interface. The interface has been modeled as a continuum element.

Table 5 shows that the FS obtained by the FEM is different from the FS obtained by LEM in each scenario due to different shear strength values. Likewise, post-peak GM-SL scenario trends to be more conservative than FEM results because FEM consider both interfaces (OL-GM and GM-SL) in the model, which estimate the shear strength mobilization along to the interfaces. The analysis shows that peak shear

strength is not reached to mobilize it, thus the use of the post peak shear strength along to the interfaces (flat and steep zone) could be overconservative.

Table 5: FS results with FEM and LEM

	Analysis cases	Peak shear strength			Post peak shear strength	
		FEM	Interface GM-SL τ_p	Interface OL-GM τ_p	Interface GM-SL τ_r	Interface OL-GM τ_r
Factor of safety	Generic case	1.96	1.94	2.16	1.17	1.91
	Case 1	2.05	1.98	2.84	1.35	2.53
	Case 2	2.16	2.08	2.28	1.39	2.06

Conclusions

The GM-SL interface shear strength increases based on the granular nature of the soil liner, and the asperity height is a crucial parameter for interface shear strength determination. The OL-GM interface shear strength increases based on the gravel content of the overliner with little influence by asperity height.

Analyzing progressive failures is a good approach for designing heap leach facilities. It allows for evaluating the shear stresses mobilized along both OL-GM and GM-SL interfaces. The cases analyzed show that mobilized shear stresses are less than the peak shear strength along the interface. This analysis allows the identification of the problems in the interface most likely to mobilize post-peak strength, which require greater attention in the design.

In heap leach pad design, both OL-GM and GM-SL interface shear strengths must be estimated based on laboratory tests. It is not advisable to assume that the weak interface will be above or below the GM without performing laboratory tests. Alternatively, the stability could be evaluated with a composite failure envelope, which is defined based on the lowest shear strength of both interfaces for the full range of normal stresses.

The FS obtained by LEM are more conservative than those obtained by FEM. The FEM analysis is more rigorous for stability analysis because it allows the estimation of the relative shear stresses along interfaces (under and above the liner system) and predicts zones of possible progressive failures.

Peak shear strength for static conditions for flat zones could be used in stability analyses if numerical analysis shows mobilized shear stresses are less than the peak shear strength, as long as there are data and models of higher confidence to support the FEM analysis.

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CPT-Based Liquefaction Potential Assessment of Leached Ore Exhibiting Microstructure

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Abstract

The cyclic liquefaction potential assessment is normally conducted using the cyclic stress approach, following the simplified procedure originally proposed by Seed and Idriss (1971). While calculating the cyclic resistance ratio (CRR) of leached ore under the V_s - and CPT-based approaches, some differences are obtained in their respective estimates. This paper outlines the key aspects that induce an increase in the CRR by cementation between particles forming microstructures along the ore at small strains coupling the CPT's normalized tip resistance ($Q_{tn,cs}$), which is an index of soil response at large strains. The paper proposes a methodology that includes the calculation of the normalized rigidity index (K_G) to quantitatively assess the microstructure effects on leached ore and the redefinition of the liquefaction triggering curve in the light of the threshold cyclic shear strain γ_{th} .

Results here reported indicate that leached ore with high K_G exhibit higher CRR at low levels of cyclic stresses, related to a 100-year return period, than that based on the $Q_{tn,cs}$ correlations with no microstructure. Under the proposed methodology, it is possible to avoid the conservatism of the simplified method while exploring the potential of the cyclic-strain theory in its application to heap leach assessments.

Introduction

In heap leach pads with high fines content ore, which are very common in several mining operations, the seismic behaviour is controlled by fines-dominated layers of crushed leach ore that retain leach solution, generate excess pore pressures, and undergo strength reduction. The standard procedure to assess the liquefaction triggering potential of these layers is commonly conducted according to the cyclic-stress approach, originally proposed by Seed and Idriss (1971). When this is employed on soils with microstructure, results can be implausible if shear-wave velocity (V_s) measurements are in a range where potential liquefaction is not triggered, i.e., $V_{s1} > 215$ m/s (Andrus and Stokoe, 2000; Kayen et al. 2013), are considered. The latter are usually discarded as the SCPTu allows for lower estimates of the cyclic resistance

ratio (CRR). Both tests record the soil response at the extremes of the strain range, and yet the liquefaction phenomenon manifests at large strains, hence it is necessary to review the key aspects that induce an increase in stiffness at small strains by cementation between particles forming microstructure along the ore.

The microstructure is related to the presence of cementation in the ore, which is characteristic of aged fines-dominated leached ore. In a heap leach pad, it is induced by chemical reactions induced by the acidic (mainly) or basic irrigation solution, which in some cases produces precipitation of certain elements, generating a cemented-based microstructure along the ore so that brittleness and flow liquefaction is of concern. Therefore, the study of leach ore with microstructure is important, especially when the ore has a high fines content that reduces its permeability, which in turn saturates the medium.

A liquefaction potential assessment methodology for fine-grained leached ore is proposed based on microstructure assessment under a coupled cyclic strain and cyclic stress approach to modify the liquefaction triggering curve revisiting the advantages of using the threshold shear strain.

Geotechnical investigations in leached ore

Field geotechnical investigations

In a heap leaching copper project in South America with high fines content ore, geotechnical investigations were performed to evaluate the leach pad slope stability. Six SCPTu soundings were conducted until a predetermined depth was reached for avoiding damage to the geomembrane. The V_s and V_p measurements, were both recorded at 1 m to 2 m intervals, respectively. Figure 1 (a) depicts the estimated and measured V_s profiles. In gray are presented the estimates of V_s as per the correlation proposed by Robertson (2009) as a function of the CPT tip resistance (Q_t), and in blue the measurements by the SCPTu geophones. A mismatch between these is evident and is an index of the presence of microstructure in the ore for the Q_t -based estimation is consistent in clean sands, i.e., with no microstructure, and is discordant with the V_s measurements. It can be noticed from Figure 1 (a) that for older leached ore i.e., located at the bottom the measured V_s seems to converge with the estimated one. One possible explanation is that for soils with their microstructure subjected to lower stress levels, where the difference between the estimated and measured V_s is large, brittle response is feasible to be expected; while at greater depths, despite having older leached ore, a ductile response is to be expected due to confinement pressure. Nevertheless, this should be further investigated in detail, for it falls outside the scope of this paper.

Figure 1 (b) depicts the compressional-wave (V_p) measurements. This is an interesting aspect in a heap leach pad if the $V_p=1500$ m/s is taken as a standard reference value for the definition of a steady-state flow regime forming a defined water table. This is not the case in a heap leach pad, which is designed to percolate the solution, and the fact that the $V_p<1500$ m/s confirms that the regime is a transient-state one

characterized by isolated pressure heads induced by low-permeability ore layers, i.e., fines-dominated, randomly and heterogeneously distributed, which makes assessment difficult.

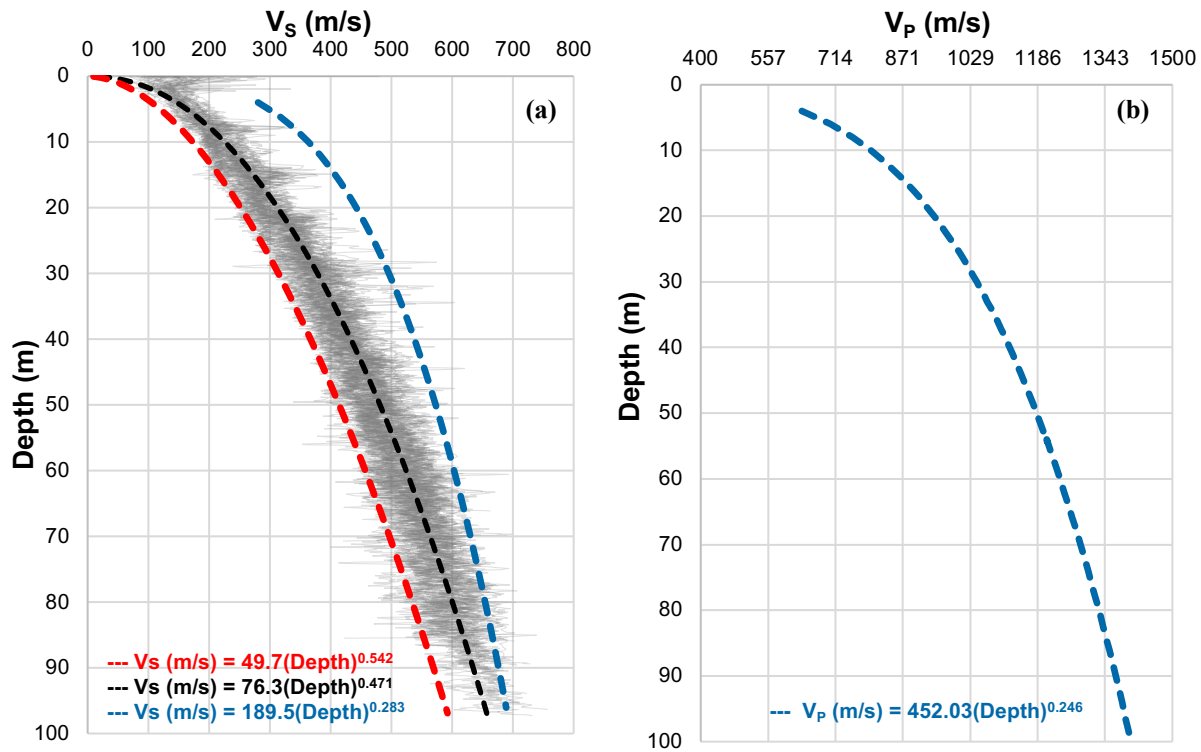


Figure 1: Measurements of (a) shear-wave (V_s) and (b) compressional wave (V_p) velocities

Microstructure and threshold cyclic shear strain analysis

In copper heap leaching, it is common for the physical characteristics of the ore to vary over time due to the effects of hydrometallurgical processes that degrade it, increasing its fines content, which in addition to its fabric form is called microstructure, which is the structure of the soil at the particle level that must be broken before considering the ore as a true particulate material in the liquefaction triggering charts.

Normalized stiffness index (K_G) based microstructure identification

It is possible to identify the existence of the microstructure with the SCPTu by the ratio of maximum shear modulus to cone tip resistance (G_o/q_n), since induced cementation increases G_o in greater proportion than soil resistance at large strains, characterized by the Q_{tn} .

Figure 2 depicts the relationships obtained between Q_{tn} and (G_o/q_n) from the SCPTu conducted for the ore analyzed in this research, where the K_G mean values obtained from the SCPTu data are shown, and which serve as the basis for the modification of the liquefaction triggering curve.

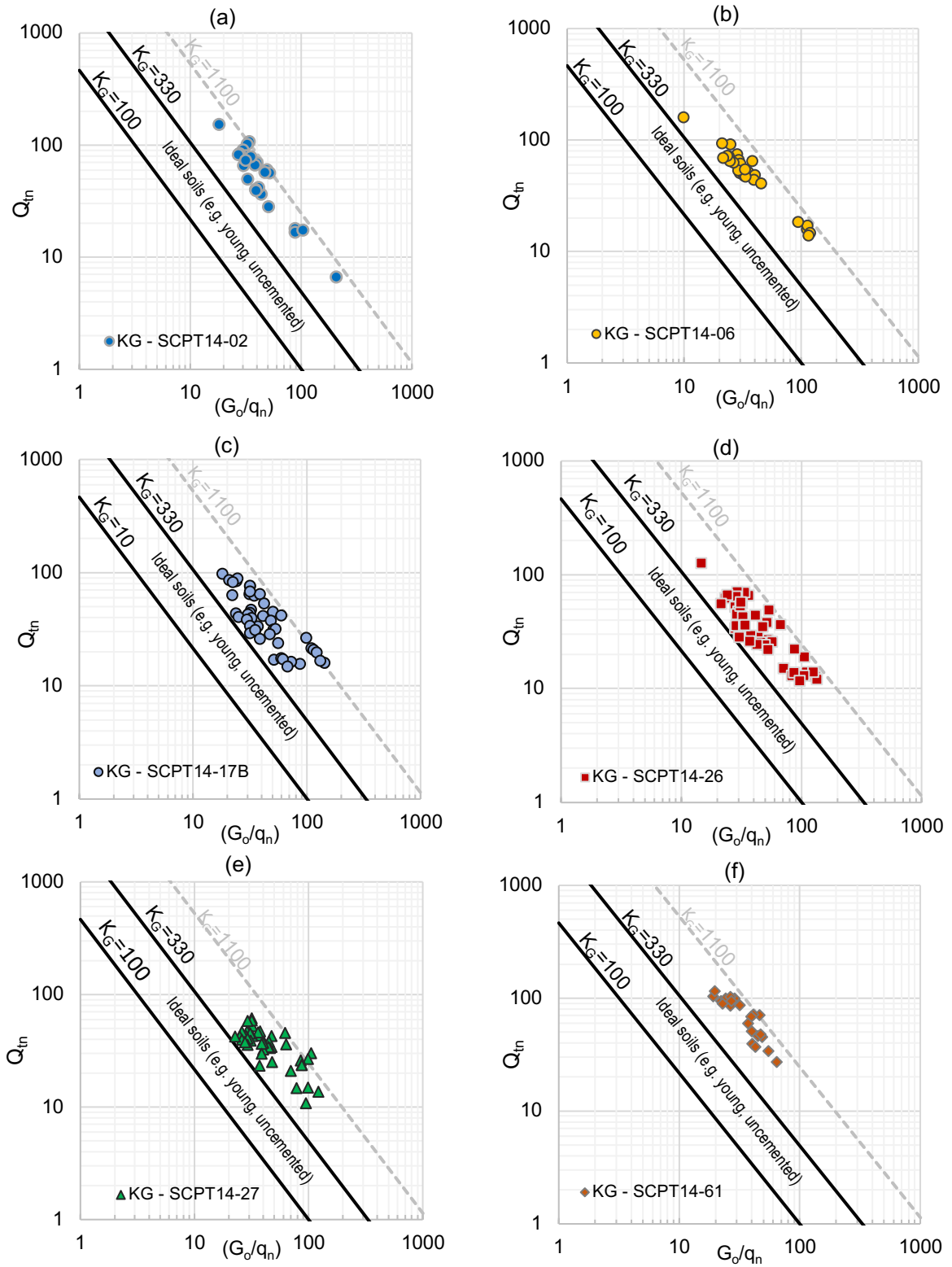


Figure 2: Q_{tn} and (G_o/q_n) relationship based on the SCPTu. (a) SCPTu14-02 - $K_{Gmean}=823$ (b) SCPT14-06 - $K_{Gmean}=684$ (c) SCPT14-17B - $K_{Gmean}=678$ (d) SCPTu14-26 - $K_{Gmean}=631$ (e) SCPT14-27 - $K_{Gmean}=651$ (f) SCPTu14-61 - $K_{Gmean}=806$

Modification of the liquefaction triggering curve

Silver and Seed (1971), Youd (1972), Pyke (1973), Stoll and Kald (1977), Dobry et al. (1982), Chung et al. (1984), Whitman (1985), Ladd et al. (1989), Hynes (1988), Hsu and Vucetic (2004), Cox (2006), Chang et al. (2007), and Hazirbaba and Rathje (2009) concluded that a threshold cyclic shear strain (γ_{th}) exists, which is approximately $1 \times 10^{-2}\%$, and is an aspect that links the onset of earthquake-induced pore pressure build-up. On the other hand, Schneider and Moss (2011) showed that the lower limit of liquefaction resistance can be defined where the earthquake-induced cyclic shear strains are less than the γ_{th} .

Equation 1 was presented by Schneider and Moss (2011) as a way to define the microstructure.

$$K_G = \frac{G_0/q_c}{q_{c1N}^{-m}} \quad \dots \text{(Equation 1)}$$

Where:

“m” is an empirical exponent typically taken as 0.75 (e.g., Rix and Stokoe, 1991).

Furthermore, G_0 can be reformulated as:

$$G_0 = K_G q_{c1N}^{-0.75} q_c \quad \dots \text{(Equation 2)}$$

To calculate γ_{th} it is required to estimate G_0/σ'_{v0} , which can be expressed in terms of τ_{th} as presented in Equation 3. The minimum value of liquefaction cyclic resistance, represented by a γ_{th} , can be related to the CRR adopting the G_0/σ'_{v0} as presented in Equation (4) and Equation (5).

$$\gamma_{th} = \frac{\tau_{th}}{G_0} = \frac{\tau_{th}/\sigma'_{v0}}{G_0/\sigma'_{v0}} = \frac{CSR}{G_0/\sigma'_{v0}} \dots \text{(Equation 3)}$$

$$CRR \geq \frac{\tau_{th}}{\sigma'_{v0}} \geq \frac{G_0 \gamma_{th}}{\sigma'_{v0}} \dots \text{(Equation 4)}$$

$$CRR \geq \frac{\tau_{th}}{\sigma'_{v0}} \geq \frac{K_G q_{c1N}^{-0.75} q_c \gamma_{th}}{\sigma'_{v0}} \dots \text{(Equation 5)}$$

As reported in Figure 2 ($631 < K_G < 901$) and the redefined relationship of the CRR as a function of γ_{th} and K_G in the preceding equations, the liquefaction triggering curve can be modified as follows:

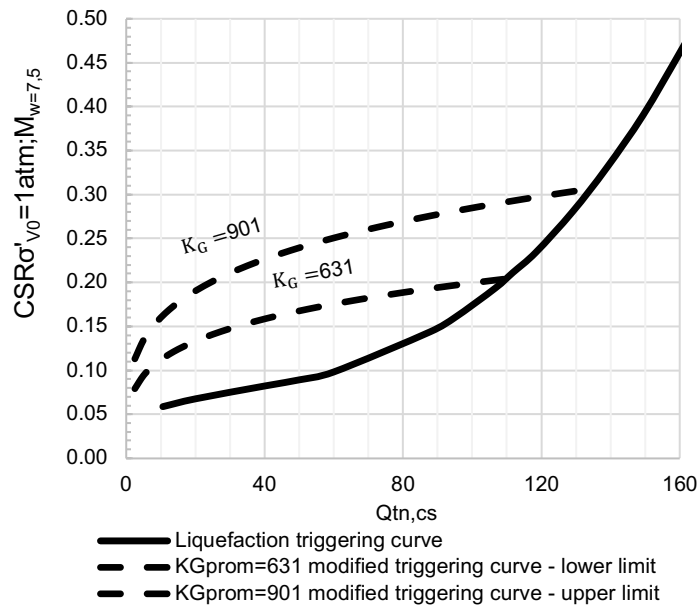


Figure 3: Modification proposed for the liquefaction triggering curve due to the presence of microstructure in leached ore

Liquefaction potential assessment for microstructured leached ore

Non-linear seismic response analysis

One-dimensional seismic response analysis was conducted in terms of total stresses considering that the soil's seismic drained response is approximately equivalent to its undrained one before liquefaction triggering. The six time-history series were spectrally matched in the time domain to the site uniform hazard spectrum using the wavelet algorithm proposed by Abrahamson (1992), later updated by Hancock et al. (2006), implemented in SeismoMatch software for the 100- and 475-year return period events.

It is common to adopt in the design criteria that the exposure time of the heap to an earthquake is 50 years with a level of exceedance of 10%, i.e. $T_R=475$ years. Yet, for this particular case, exposure time is less than 10 years. Assuming the same level of exceedance in accordance with a Low classification by the consequences of failure CDA's criterion, the seismic analysis can adopt a $T_R=94.9$ years (100-year return period in practice). If a risk-based criterion involving Significant consequences of failure class is incorporated, the level of exceedance can be less than 10%, i.e., requiring at least 475- or 1000-year return period events.

Figure 4 depicts the calculated profiles of the maximum shear stress to the vertical effective stress ratio ($CSR_{max} = \tau_{max}/\sigma'_{v0}$) obtained via seismic response analysis for the 100-year (Figure 4.a) and 475-year (Figure 4.b) return period events for the six time-history series; the mean CSR_{max} is depicted in black.

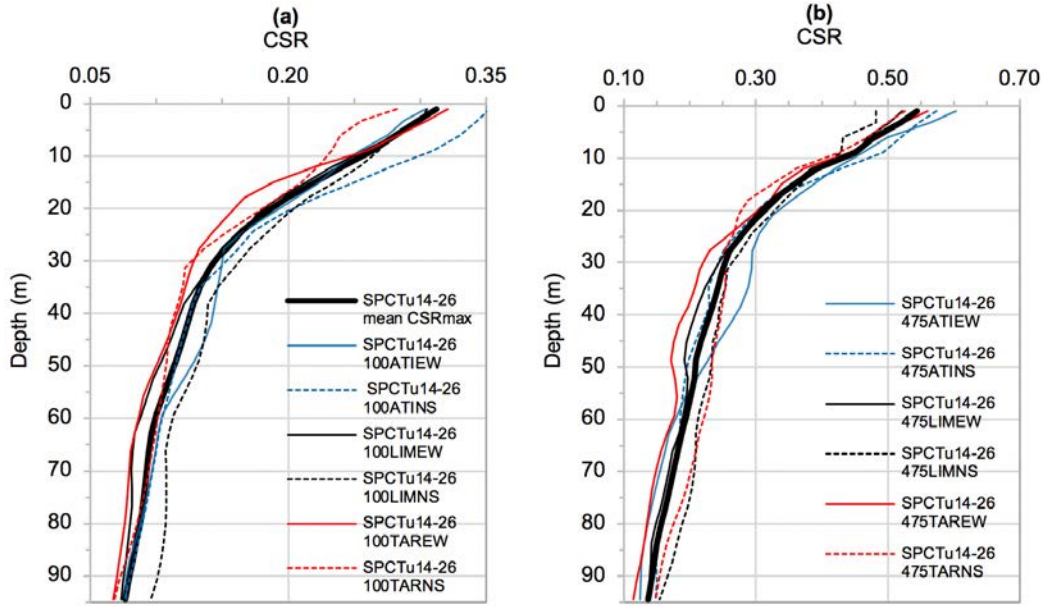


Figure 4: CSRmax profiles obtained for the deepest CPT for six different time-history series used for the (a) 100-year and the (b) 475-year return period events

Figure 5 depicts the calculated CSR_{max} factored by 0.65 as required by the simplified method ($0.65CSR_{max}$) and the one estimated by the simplified formulation, in terms of PGA, which was calculated by the 1D seismic response analysis, and r_d , in red ($CSR_{f(PGA,rd)}$). It is shown that the $CSR_{f(PGA,rd)}$ overestimates the calculated $0.65CSR_{max}$ profile in the layers of interest for a leach pad (>10 m in depth); hence, it is required to fit the PGA such that the $CSR_{f(PGA,rd)}$ best fits the $0.65CSR_{max}$, resulting in PGA values of 0.30g and 0.50g, for the 100- and 475-year return period events, respectively.

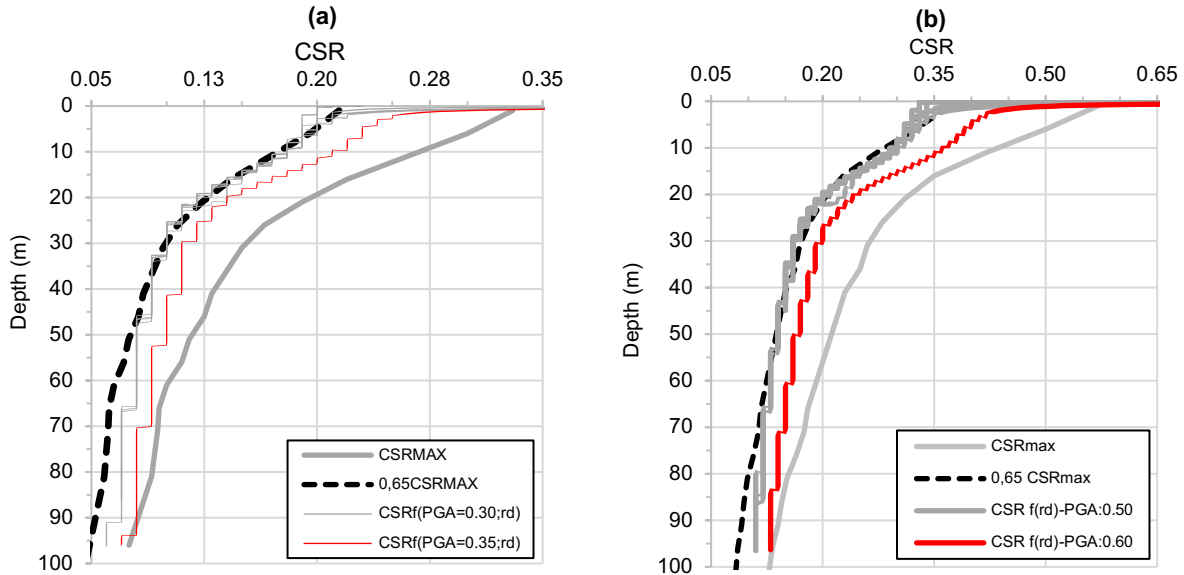


Figure 5: CSR_{max} and $0.65CSR_{max}$ obtained from the seismic response analysis and the ones estimated by the mass participation factor (r_d) for use in the simplified method for the (a) 100-year and (b) 475-year return period events

Liquefaction potential assessment according to the modified triggering curve

The procedure proposed by Robertson (2010) has been considered to include the estimation of the CRR for clay-like soils, which is common in heaps composed of fine-grained ore. To calculate the equivalent $CRR_{M7.5}$ the magnitude scaling factor (MSF), as proposed by Youd et al. (2001) and Robertson and Wride (1998), was adopted.

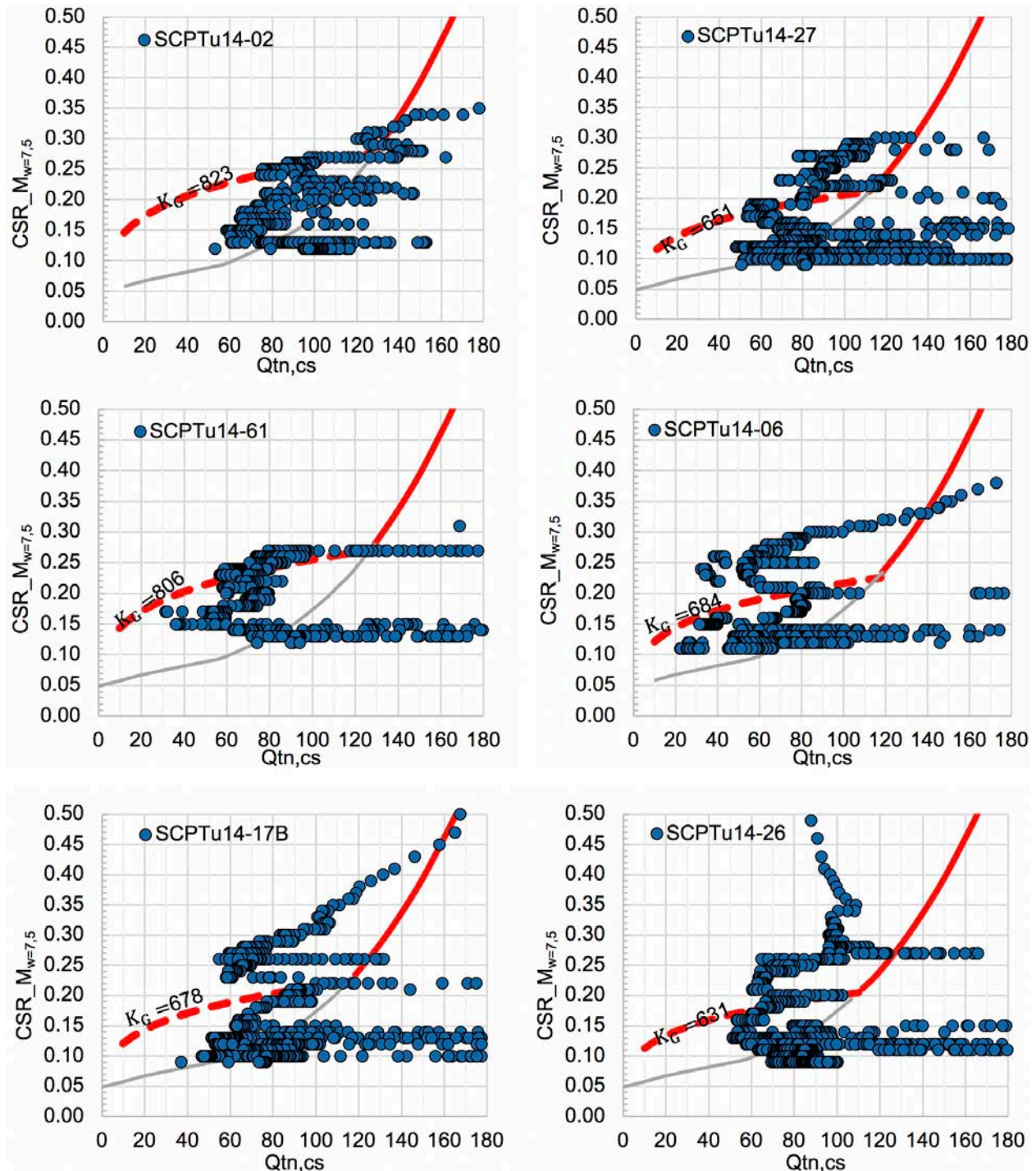


Figure 6: Liquefaction potential assessment results for a 100-year return period event

Results for the 100-year return period event

Figure 6 shows that the ore layers that will not trigger flow liquefaction ($Q_{tn,cs} < \sim 70$) or cyclic liquefaction ($\sim 70 < Q_{tn,cs} < 120$) are greater than those that would liquefy if the ore was a soil with no microstructure. It should be noted that for the SCPTu14-63 no layers would liquefy under the modified triggering curve. There are at the same time layers that, by the action of the estimated CSR, exceed the corrected CRR, which is an indicator of breaking microstructure and returning to a state where the ore behaves like a contractive strain-softening clean sand-like soil.

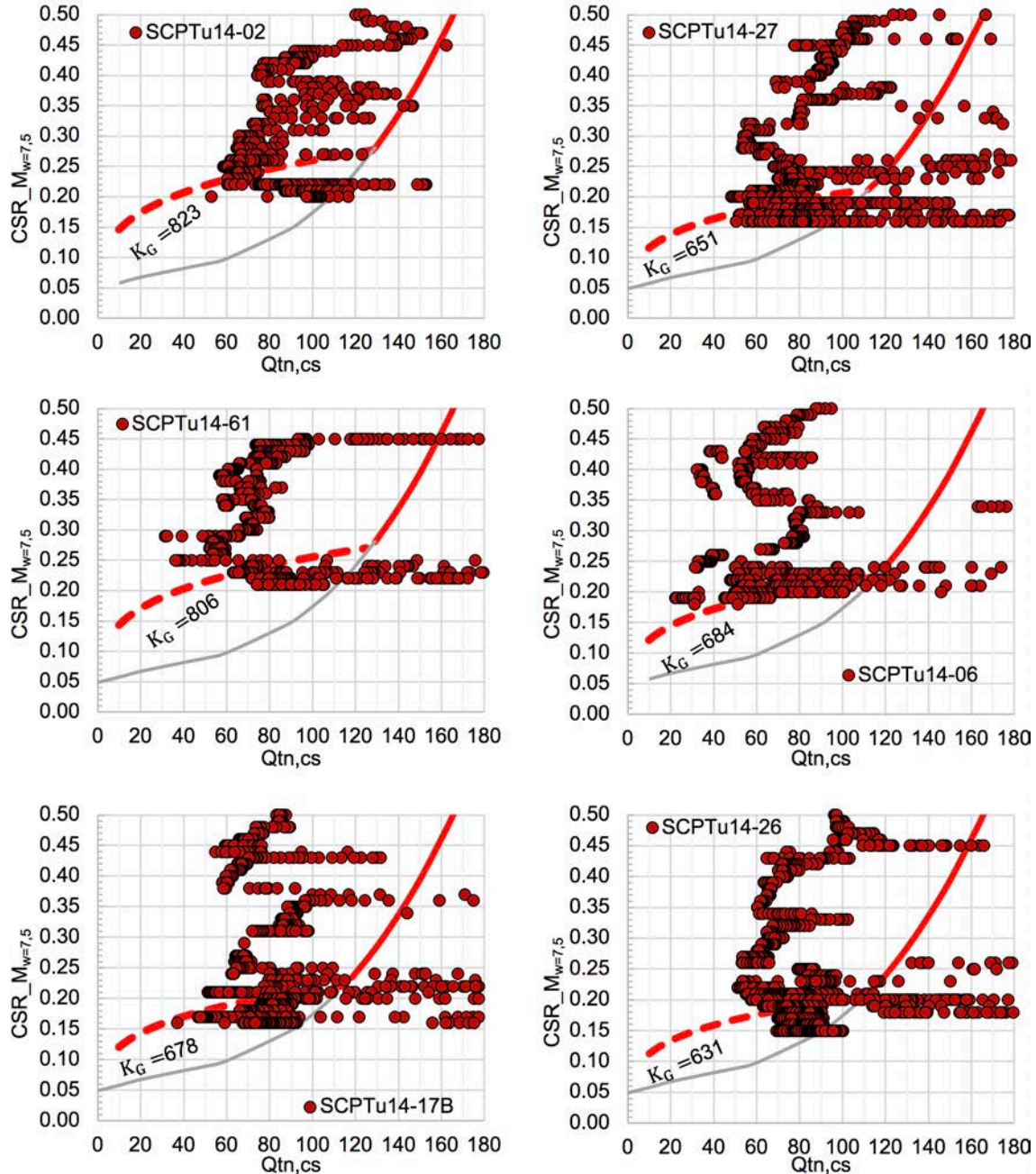


Figure 7: Liquefaction potential assessment results for a 475-year return period event

Results for the 475-year return period event

Figure 7 depicts that the microstructure effects are not sufficient to maintain the ore cyclic strength, and in fact almost the entire profile would trigger liquefaction in accordance with the analysis assumptions for the simplified method. These results indicate that the microstructure assessment is unnecessary since the leached ore will still behave as a Holocene soil in which the microstructure is broken by the effect of the mean equivalent cyclic loading, which is representative of the 475-year return period event, characterized by a PGA of 0.50 g and a $M_w=8.5$.

Conclusions

- The heap leaching process may generate microstructures in the ore that influence the in-situ ore behaviour, and therefore, its effects must be analyzed and assessed.
- With the SCPTu it is possible to identify the existence of microstructures by means of the ratio of the maximum shear modulus to the cone tip resistance (G_0/q_n).
- The cyclic strain theory (Dobry et al., 1982) provides a lower limit for liquefaction resistance of soils with microstructure, in this case a leached ore, based on the adoption of the γ_{th} for coupling with the cyclic-stress approach to avoid underestimation of the CRR in the liquefaction potential assessment in heap leach pads.
- The first scenario adopted a 100-year return period earthquake that may be associated with a short-term mining operation, where the exposure time is about 10 years and the level of exceedance is 10%. The second adopted a 475-year return period earthquake associated with a longer exposure time or a level of exceedance less than 10%, which is common when assessing the consequences of failure.
- The PGA is not what is fundamental in the estimation of the liquefaction potential, but rather the complete acceleration profile, which should be fitted by means of the simplified formulation modifying the PGA until getting the best fit for the calculated CSR profile, instead of only fitting the PGA, since doing it that way may overestimate the CSR profile. Hence, a 1-D seismic response analysis is required.
- The coupled cyclic stress and cyclic strain-based approach assessment indicates that soils with high normalized stiffness indices (K_G) exhibit a liquefaction resistance at low cyclic loadings, characteristic of a 100-year return period event or of a Low classification structure based on its consequences of failure, that is greater than the liquefaction resistance based on cone tip pressure (q_c) correlations of Holocene sands.

- The results obtained indicate that the leached ore with microstructure for a 475-year return period event behaves like a Holocene soil. Although the loss of the strength benefits of the microstructure is an important indicator to know in advance for which design earthquake, the benefits of featuring microstructure cannot be used.

The finding derived from combining the cyclic shear stress and cyclic shear strain approaches is promising regarding the leached ore performance, since it is shown that the ore will exhibit a greater elastic range. It is recommended to investigate the γ_{th} to begin to explore scenarios of greater return periods and thus cover a wider range of possibilities within a heap leach pad operation.

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First Radiographic Characterization of an Active Leaching Heap

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Abstract

This paper reports on the first in-situ, fully volumetric measurement of the bulk density in an active leaching heap. This was obtained via so-called muon density radiography, a well-established technique for the near real-time characterization and digital mapping of bulk density across large objects. Muon radiography and/or tomography has already been successfully utilized for the characterization of the interior of volcanoes, ancient pyramids, and cargo containers.

Muon density measurements are close analogues of the more widely used medical or industrial X-ray. The key difference is that in place of a conventional nuclear source or X-ray tube, muon radiography is entirely passive as it utilizes the naturally occurring flux of so-called cosmic ray particles (aka muons) originating in the earth's upper atmosphere. This benign background radiation source is not only free and intrinsically safe, but also well-characterized and continuously available everywhere on earth. Muon particles are also highly penetrating and maintain directionality even in the subsurface. This allows muon radiographers to analyze and map the interior of dense structures up to thicknesses of at least a few hundred metres. In the case of leaching heaps, 2-D (radiographic) or 3-D (tomographic) density profiles can be reconstructed with a high precision ($\sim 0.01\text{g/cc}$ relative, $<0.1\text{g/cc}$ absolute) and good spatial resolution (a few m^2) within time intervals typically ranging from a few days to a few weeks. When these data are compared with the initial ("dry") conditions of the heap material, or as a function of time, apparent density changes can be readily interpreted, with minimal systematic uncertainty, as the true fluid content (in litres/m^3) present in the heap. Thus, muon radiography is a promising diagnostic tool for the characterization of leaching heaps and other partially saturated media.

Here we report on the performance of a novel muon radiography sensor that was installed and remotely operated under an active leaching heap in the spring of 2022. We then highlight the applicability of muon density measurements as a new diagnostic tool for managing leaching heaps.

Introduction

Density measurements are a primary method for the characterization of any physical object. So-called nucleonic density (i.e., nuclear bulk density) is widely recognized as being one of the most robust and quantitatively accurate density measurements available, with broad applications ranging from medical imaging to industrial non-destructive testing (X-ray, CT, line inspection, cargo screening, etc.). In oil and gas exploration, borehole nuclear density services are worth well in excess of several billions of dollars per year in annual sales, and have been utilized since the 1960s as the primary method for determining the porosity of reservoir rocks, and ultimately to book proven reserves. In this paper we present our first results on the characterization of an open-air, gold-bearing leaching heap in the field.

Nuclear density measurements are an obvious choice to characterize granular or unconsolidated materials, determine their compaction, and quantify the presence of fluids within their pore space. The most desirable features of the nuclear density measurement are that it is independent of i) chemical composition, ii) chemical form and iii) temperature of the object under study. All nuclear density methods measure the attenuation of a known flux of penetrating radiation. For reasons of practicality, most of the nuclear density applications use γ -rays from a radionuclide source, but the method works just as well with charged particle radiation, such as electrons, protons, and other sub-atomic particles.

So-called cosmic-ray muons (aka atmospheric muons) are one such type of radiation and are particularly well suited for nuclear density measurement at large scales. Atmospheric muons are subatomic particles generated in the upper layers of the atmosphere by interaction with a primary flux of nuclear radiation generated by distant supernova explosions (mostly protons and light nuclei). This effect has been known for more than 100 years (Kaiser, 2018), and the muon flux on the surface of the earth has been very well characterized (Beringer et al., 2012). Indeed, so-called muon radiography has been utilized for looking inside of ancient pyramids (Alvarez et al., 1970), volcanoes (Lesparre et al., 2012), inspecting nuclear fuel casks (Gilboy et al., 2007) and tunnels (Guardincerri et al., 2017) and in mine exploration (Schouten and Ledru, 2018). The author of this paper was the first to build and field test a modern borehole detector for muon density measurement for an oil and gas application operating on a wireline in an actual oil-well (Botto et al., 2014).

Utilizing atmospheric muons is intrinsically safe as no dangerous nuclear source is required. Atmospheric muons come in all directions, and over a wide range of ultra-relativistic energies. This allows them to penetrate deeply into sub-surface while maintaining directionality, which is a key requirement for generating accurate radiographic images. Indeed, atmospheric muons have been recorded in very deep mines and underground laboratories at effective depths of >3 km.

Methodology

The concept of muon radiography applied to a leaching heap is schematically illustrated in Figure 1.

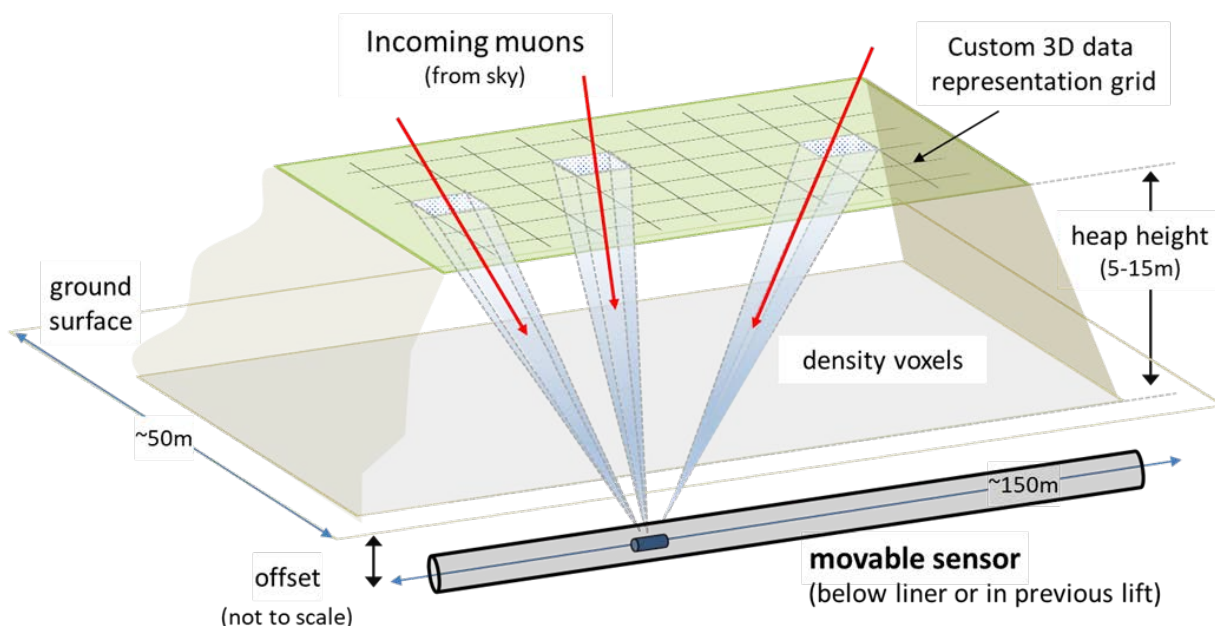


Figure 1: Muon radiography concept applied to a leaching heap. A detector buried under the heap reconstructs muon trajectories over a wide range of angles. These are grouped into inverted pyramidal voxels that are independently analyzed in terms of their average bulk density

The actual Muon Vision prototype sensor used in this work was contained in cylindrical housing with an overall tool length of 1.91 m and a 176 mm maximum diameter, and weighed a total of ~40 kg. Tool installation proceeded as follows: first, seven sections of standard smooth-wall HDPE pipe (280 mm PN16 PE100, high-density polyethylene), for an overall length of 84 m, were assembled on site. This was done in an empty, i.e. yet-to-be-filled, area of a standard leaching pad near Santiago, Chile (see Figure 2). The HDPE pipe was sealed on the far end with a custom blind section that also had an inner return pulley. The Muon Vision tool was then inserted into the HDPE pipe such that it could be moved up and down the pipe by leveraging a simple two-cable conveyance system and the return pulley at the far end. Note that the tool housing also featured two sets of spring-loaded external wheels to facilitate sensor movement and allow smooth travel over the ridge present in between joined pipe section.

The Muon Vision prototype tool has two feedthroughs to allow for data and power connections to the outside world. Tool operation (including data connection) requires ~5 W of power. For convenience, this was provided by a standard 220V AC line available on site that fed into a large electrical battery. Thanks to an on-board voltage regulator, input power supplied to the tool can be anywhere in the range of 12 to

30VDC, which is more than enough to compensate for voltage drops along the cables of different lengths. Future installations may use standalone power provided, e.g., by a solar panel.



Figure 2: Clockwise from top left:
i) the Muon Vision team on site, with the prototype sensor, prior to installation;
ii) the Muon Vision sensor inside the HDPE pipe during tests;
iii) the remote cabinet for managing power and data connection on site; and
iv) overview of the installation pipe, next to an already leached section of the heap, and prior to construction of the heap module measured

The data connection to and from the tool was realized via a standard optical fiber, which connected to a ruggedized field PC. The field PC also ran the data acquisition software and stored the data coming from the sensor. The field PC could be accessed over the internet utilizing a standard consumer-grade data plan. While muon radiography measurements are, by nature, continuous, data readout proceeded in buffered form and data taking was broken down in a series of successive runs, each with a fixed two hours' length. This approach helps minimize any potential data loss. Data is transmitted in binary form, which further minimizes disk space requirement to a mere ~40 Mbytes/day. Binary files were then transferred remotely

for offline processing of the data. Data and power connection were managed via a custom control cabinet that was also installed on site, as shown in Figure 2.

Results

As the construction of the leaching pad progressed, the sensor was initially parked at the entrance of its installation pipe, at an effective depth of just 0.4 m. During that time the tool could essentially only monitor the average surface rate of cosmic ray muons at the given location ($32^{\circ}27'S$, $71^{\circ}14'W$). As soon as a leaching module was built on top of the installation pipe, the tool was advanced and the slant section of the heap scanned across different nominal heights, as shown in Figure 3.

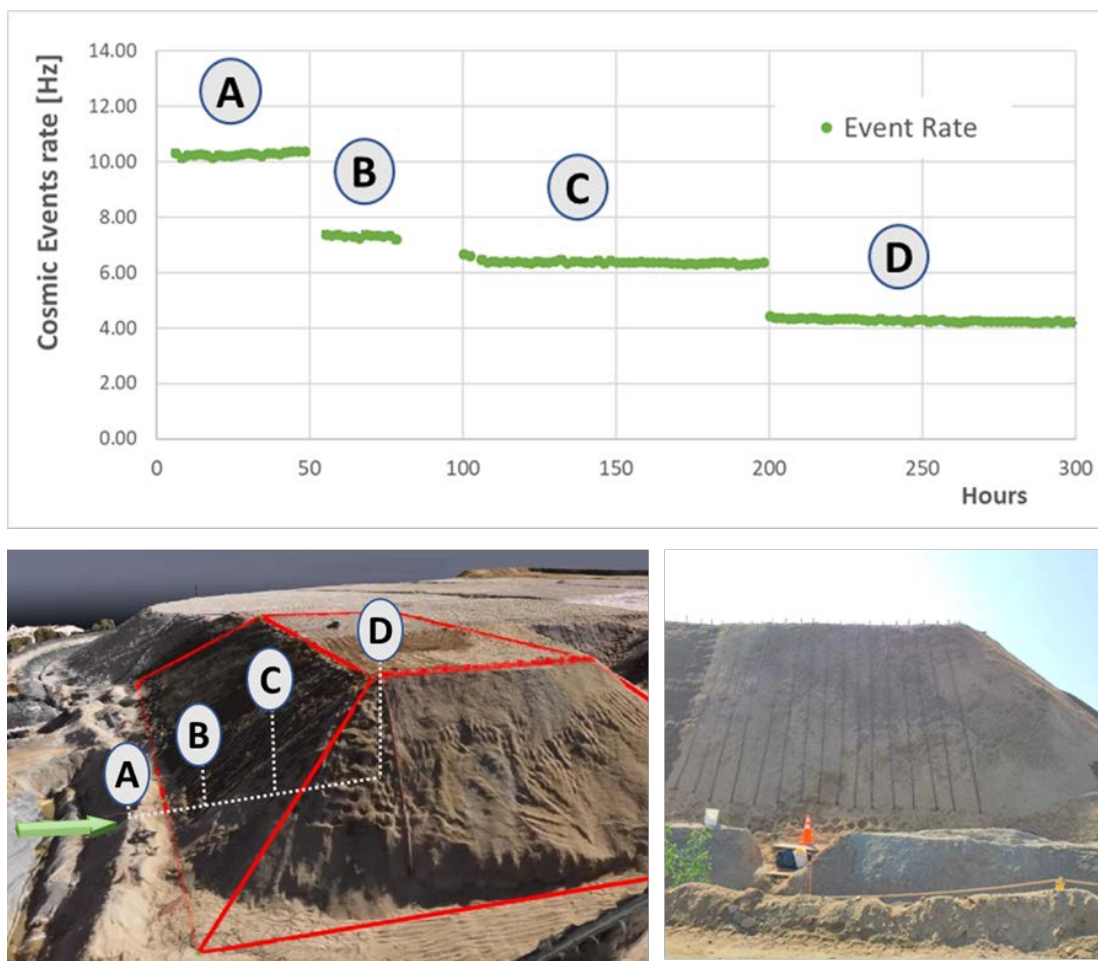


Figure 3: Top panel: measured muon count rate while scanning the slant section of the heap module. Bottom left: the green arrow indicates the entrance to the installation pipe and the red outline a simplified geometrical model that was later used in the data interpretation via Monte Carlo. Bottom right: side view of the heap and installation pipe (under the orange cone)

Cosmic ray muons are indeed individual physical particles arriving at any point on earth independently over a vast range of energies and azimuthal angles. Muons that have been generated at lower energies

generally decay faster and may not even reach the surface of the earth. Higher energy muons on the other hand, will travel several kilometres through the earth's atmosphere and well into the subsurface. Generally, the muon flux is peaked along the vertical direction, which is also the direction of the shortest path to their source point that, conventionally, is taken to be 20 to 30 km above the earth's surface. Regardless of how and where muons are generated, and as with all other nuclear probes, their measured flux at the detector is related to the density of all materials between the source and the detector. The muon particle flux measured at a detector at a depth z and within a certain solid angle is formally given by:

$$I_{\mu}(z) = \iint_{E_0}^{\infty} I_0(E_{\mu}, \theta_{\mu}) e^{-\alpha(E_{\mu})\mathcal{O}} dE_{\mu} d\theta_{\mu} \quad (\text{Equation 1})$$

Where:

the integral runs over all possible muon arrival angles (θ_{μ}) and energies (E_{μ}) above a minimum threshold ($E_{\mu} > E_0$).

Here $I_0(E_{\mu}, \theta_{\mu})$ is the intensity of the muon flux at surface and $\alpha(E_{\mu})$ the muon absorption coefficient in matter, both of which very well-known from nuclear data tables, while $\mathcal{O}$ is the so-called integrated thickness or opacity, i.e., the line integral of the density of all materials traversed in the muon path to the detector:

$$\mathcal{O} = \int_{path} \rho(x, y, z) d\vec{l} \quad (\text{Equation 2})$$

In this sense a measurement under a leaching heap of the muon flux continuously samples the local density in the heap $\rho = \rho(x, y, z)$ over the full path-length L of each arriving muon with sub-atomic precision and simultaneously over all possible directions. The number of muons arriving at the detector (i.e., the count rate) is a strong function of the threshold energy E_0 , which in turn is essentially a function of the depth of the sensor.

This general behaviour is clearly illustrated in Figure 3 (on previous page). The observed muon count rate closely tracks the heap profile. As the sensor is moved further in along the pipe, it finds more material above it, which means a higher threshold energy is required for muons to reach the detector. This results in an increased attenuation of the flux, and indeed the number of observed cosmic ray events drops from ~10 Hz at a nominal depth of 0.4 m (position A) all the way to ~4 Hz at a depth of 6.65 m (position D). Note that this data was taken as the heap was still being built and, in the plot, it has been staggered in time for clarity.

Right after our sensor reached position D, the operator started irrigating the heap material. This partially fills the porosity in the heap with the leaching fluid and thus changes the overall density of the heap. This is neatly contained in the basic volumetric equation for the bulk density of any porous material, which is commonly written as:

$$\rho_{bulk} = (1 - \Phi)\rho_{matrix} + \Phi S \rho_{fluid} \quad (\text{Equation 3})$$

where ρ_{matrix} is the matrix density of a rock formation with porosity Φ .

In a leaching heap, which is made of unconsolidated material, ρ_{matrix} is the (average) density of the crushed rock grains. When fluid, at a saturation S relative to the pore space, is added to the system then one can also write:

$$\rho_{bulk} = \rho_{dry} + \Phi S \rho_{fluid} \quad (\text{Equation 4})$$

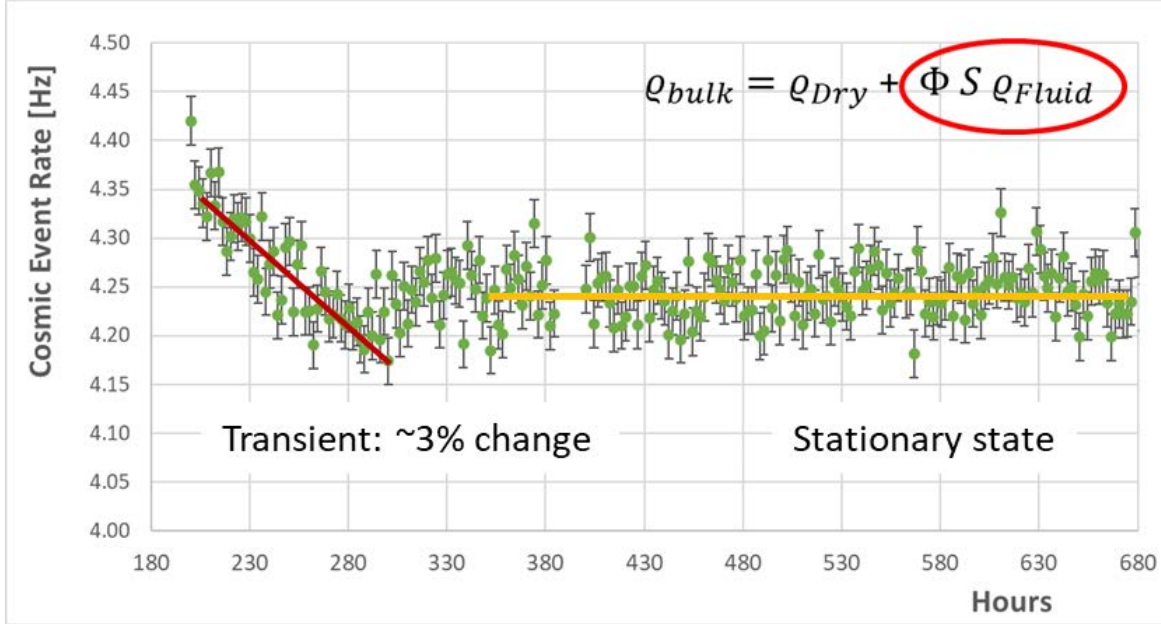


Figure 4: Observed rate of cosmic-ray muons with detector at position D. Irrigation starts at $t = 180$

Thus, given the material's dry density, its porosity and the fluid density (all of which are generally known or could be determined in-situ), a measurement of ρ_{bulk} provides a unique, nearly model independent determination of the total amount of fluid in the heap, i.e., the saturation S in Equation (3). A higher density, for a fixed heap height, also means a higher energy threshold for cosmic ray muons to reach the detector, and thus a reduction in the observed count rate. This is clearly illustrated in Figure 4, showing the data taken at position D.

Right after reaching the D position in Figure 3, which is also the moment heap irrigation was started, we observed a clear, highly statistically significant drop in the event rate from 4.35 to 4.2 Hz. This is shown in Figure 4. This ~3% change in the count rate implied that overall, the bulk density in the heap had changed by ~5%. This is well within expectations for a material with an initial density of $\rho_{dry} = 1.7$ g/cc and a 36% porosity ($\rho_{matrix} = 2.65$ g/cc, $\Phi = 0.36$) saturated up to ~21% with a fluid. For simplicity we just assumed $\rho_{fluid} = 1.0$ g/cc. A higher fluid density would have simply resulted in a lower estimation of the saturation.

It is important to emphasize that neither the chemical composition of the heap nor the fluid matters here. Also, having started with (nominal) dry parameters, what we present is the total fluid content, with no consideration for the fact that the material was pre-wet in an agglomeration stage up to a nominal 11% saturation. If one starts from the density of the agglomerated material (i.e., $\rho_{dry} = \rho_{agglomerate}$), then our method would determine only the amount of fluid added in the heap leaching stage. Ultimately, proper values for ρ_{dry} and other key parameters can be determined empirically with a proper sampling campaign, as well as in-situ by characterizing the heap prior to irrigation.

Another important and also highly statistically significant observation that can be made from the data in Figure 4, is that after an initial ~ 100 hours of transient behaviour in the 6.65 m tall heap, the muon count rate reaches a stationary state, which was then maintained over several weeks. During this stationary state the density in the material does not change any more, and thus we conclude that the fluid content in the heap has also found its equilibrium value.

Note that the stationary state points to a fluid content that is just a bit lower than the maximum fluid content reached at the end of the transient phase, even though the flow rate was not changed by the operator. This indicates that, possibly, preferential flow channels were created in the initial transient phase, through which the fluid could quickly escape, and therefore that a fraction of the fluid never really contributes to the leaching process. We believe this should be accounted for in the overall balance and analysis of leaching performance. Also, we believe that our technology is uniquely posed to diagnose and potentially help remediate this effect by, for example, optimizing the stacking and agglomeration process in view of the actual hydraulic performance of the heap.

The data in Figures 3 and 4 also indicates that although the muon flux at surface is expected to have variations of 5 to 10% over long periods of time, it can be considered constant on time scales of a few weeks, which in most cases are sufficient for a heap radiography. Variations of the muon flux are mostly associated with small seasonal changes in the temperature and pressure density across the full 30 km length of the earth's atmospheric column (which slightly changes its overall density), as well as to changes in the earth's magnetosphere due to effects from the 11-year solar sunspot cycle. Both of these effects are dependent on the muon energy and impact a muon detector differently according to its depth which, as discussed, determines the muon energy threshold. However, we note that seasonal changes in the muon flux have different signs at different ends of the spectrum, and indeed expected to have a zero-crossing for depths between 5 to 50 m. This is well within the range of our application, which we think is particularly well suited for muon radiography investigations compared to surface or deeper measurements.

The data presented so far has essentially been count rate data in the aggregate, with no position information. One attractive feature of muon radiography is the potential to generate 2-D or 3-D density maps. With data from a single location, as in our case, one can only generate maps analogous to

conventional 2-D radiographic X-ray images. This requires segmentation of the data along different muon directions. Since each pixel in the image will have fewer counts the total aggregate measurement, the statistical accuracy, and thus the resolving power for density differences, will be lower. Although the resulting image is 2-D, the heap is sampled along its full thickness, and as such even a 2-D projection contains bulk density data averaged in a 3-D pyramidal voxel, as shown in Figure 1. A true 3-D density map requires similar data segmentation but on multiple planes, e.g., parallel to the heap surface. Depending on the inversion algorithm this does not necessarily introduce additional statistical penalties. However, it is still true that for any true 3-D inversion, the volume under observation must be viewed from multiple view angles, and this is best done when multiple sensors are installed or moved to different locations (Guardincerri et al., 2017).

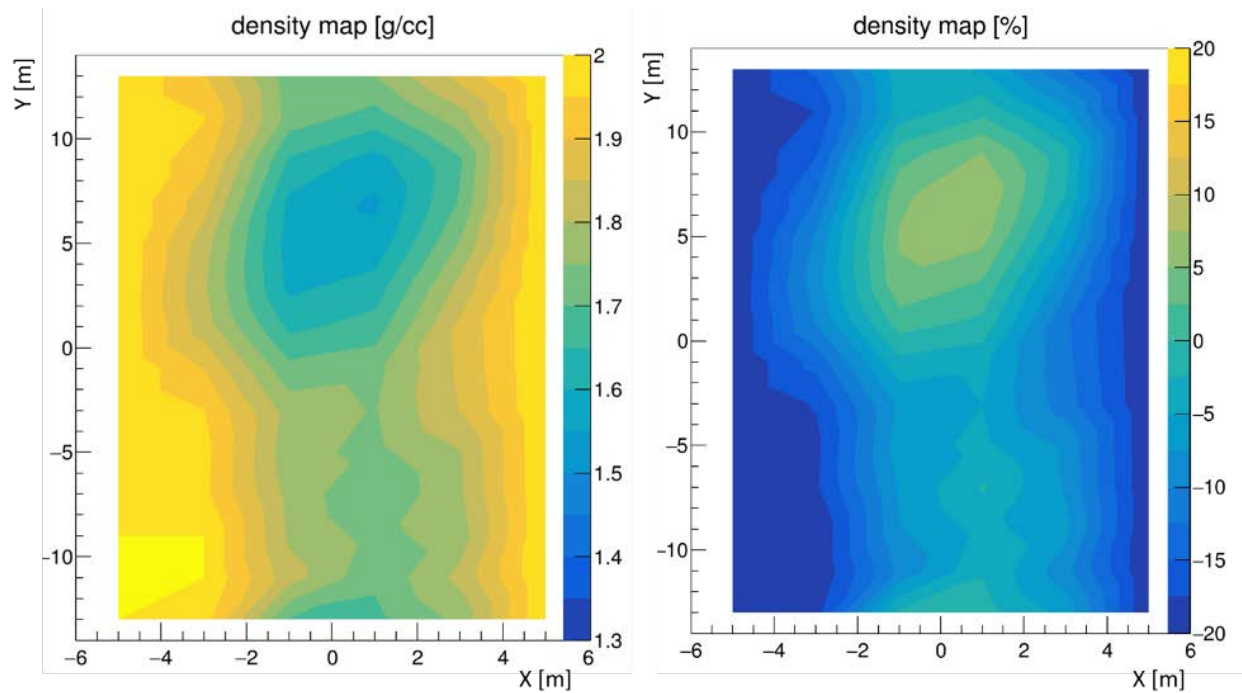


Figure 5: Absolute (left) and relative (right) density maps projected on the surface of the heap.
See text for more explanation

Our first density map is shown in Figure 5. This is a 2-D map (in the sense given above) corresponding to data collected over a period of two weeks. To arrive at the density value, we compared the measured count rate with the expected count rate from simulations, assuming a nominal muon flux. The simulation was based on a well-known, highly sophisticated, and very high-fidelity open-source toolkit for particle transport from the CERN particle physics laboratory in Switzerland called Geant4 (Agostinelli et al., 2003). The same toolkit is also used to determine treatment dose in radiotherapy applications. Geant4 takes care of the generation and travel of muons from the sky source, through the heap and to the detector, for a

nominal flux intensity. The simulation also included all the details of the detector and its response. For simplicity, we resorted to using a simple trapezoidal model of the heap (shown in Figure 6), which given the irregular shape of the heap, could not reproduce all the features in our data. This essentially limited our coverage, particularly in the region in front of the tool, where a big gap was present in the heap during data taking, which skewed the comparison with the data. All together our coverage was $\pm 60^\circ$ in the longitudinal direction (i.e., along the installation pipe) and only $\pm 45^\circ$ in the transverse direction (due to unexpected issues with the prototype after it was installed).

In Figure 5 we present both a relative (in %) and an absolute density map (in g/cc). The latter was obtained by applying a normalization factor that takes care of detector efficiencies and the actual value of the muon flux intensity in-situ. This normalization factor is indeed model dependent. In the future we plan to eliminate this dependency by bringing on site a second “witness” detector that will be placed outside of the heap and devoted to measuring the muon flux in-situ, rather than density changes in the heap. This additional measurement will provide the desired absolute normalization.

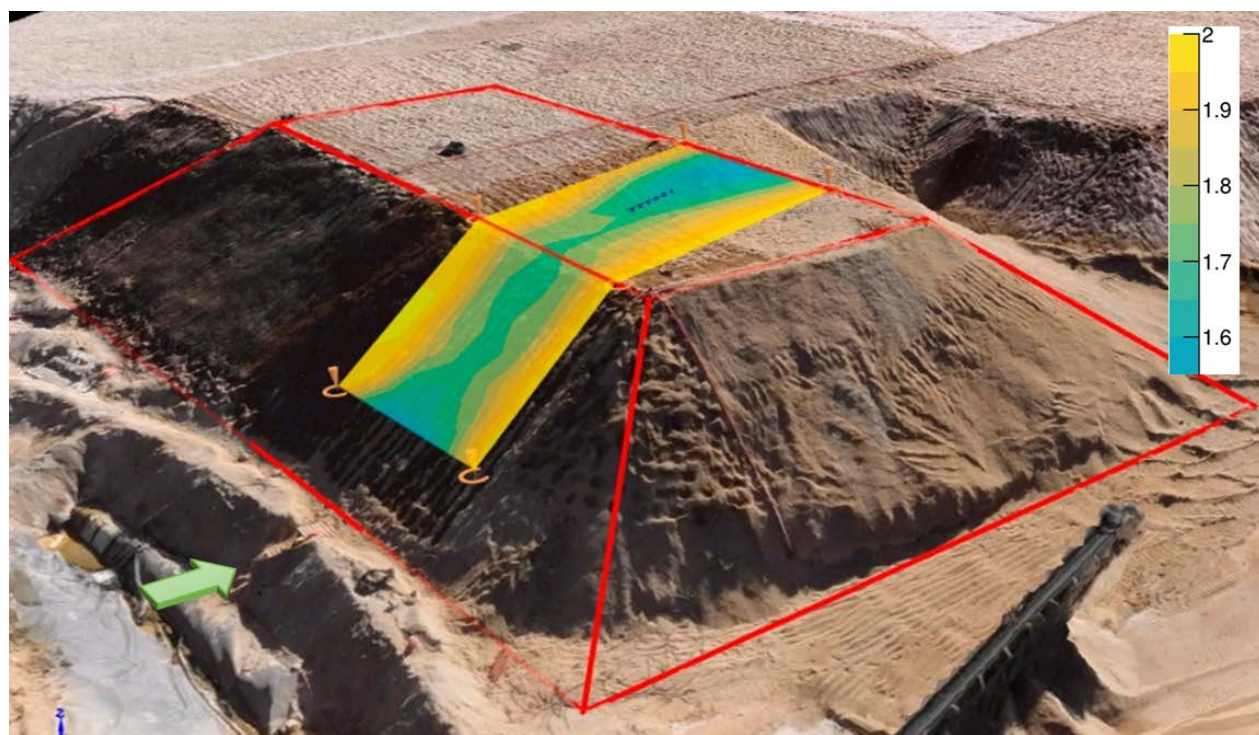


Figure 6: 3-D rendering of the heap based on measurements off a drone with our density map overlayed. The red lines are the outline of our geometrical model (a regular trapezoid) discussed in the text. The sensor position (under the heap) is indicated by the dashed line on top of the heap

In contrast, the relative density data in Figure 5 is free of this error and shows that the average bulk density in the heap was mostly within $\pm 5\%$. This is still a preliminary result as we are still in the process

of calibrating the efficiency of the detector for angular reconstruction, and we also need to account for the effect due to different heap heights at different locations. Indeed, our test heap was built with an excavator and does not have a flat top. Varying heap heights or sloping heap surfaces can be measured accurately with a drone, as was done here, and results from such scans rather straightforwardly incorporated in our inversion model and analysis.

Future improvements will also include a better characterization of the orientation of the installation pipe (here assumed to be straight) and of the tool position inside the pipe (here nominally taken to be 15 m). Finally, the data in Figure 5 was smoothed with a kernel algorithm similar to those used in image rasterization. Notwithstanding the present limitation, our smoothed data are consistent with a small density anomaly at $X=0$, $Y=+5$ in Figure 5. While we believe this is likely to be a true effect, we cannot confirm it until we improve on our analysis.

Conclusions

We have presented preliminary results on the first radiographic analysis of an active leaching heap. This was done by analyzing cosmic ray muons tracks arriving at a small, remotely operated prototype sensor (0.2 m diameter) placed under the heap. The measured cosmic ray flux was well within expectations and clearly showed an accurate correlation with the amount of fluid added to the heap. Both absolute and relative density maps covering an area $\sim 10 \times 20 \text{ m}^2$ above the detector were obtained. Our preliminary results open up the possibility of utilizing cosmic ray muons as an effective tool to quantitatively determine fluid saturation, a key parameter for the performance and safety of all leaching heaps. There are potentially many benefits to be had with such an approach. These include:

- Increased metal recoveries and reduced lixiviant consumption, via a data-driven optimization of the placement of irrigation lines and tailored irrigation rates.
- Improved NPV determination, via the localization and quantification of by-passed ore, leading to improved financial visibility and informed decision making on potential reprocessing decisions.
- Reduced GHG and water footprint, per tonne of metal produced.
- Increased safety, by reducing the need for inspection personnel on the top of the heap and by reducing the uncertainty in the geotechnical stability model.
- Mine operational improvements via advanced analytics and feedback loops correlating operational parameters (e.g., agglomerator, ore composition) with actual percolation performance in the heap.
- Near real-time continuous monitoring and validation of operational setpoints, and the prompt identification of trends and non-standard conditions.
- Support for the implementation of realistic digital twins and 3-D heap models.

- Improved ore removal at the end of the cycle based on actual soil moisture content (e.g., avoids trapping of equipment or sticky ore that is difficult to remove from conveyor belts).

While this was, we believe, its first application to a leaching heap, muon radiography is nonetheless a well-established geophysical technique and a mature technology. In our view, what remains to be determined is not the validity of measurement per se, but rather how embedding muon density maps into operational leaching workflows will lead to achieving some or all of the above mentioned potential benefits. This knowledge will only come through new deployments and service offerings in collaboration with the leaching industry. Muon Vision is eagerly looking forward to its next opportunities.

Acknowledgements

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IXOS® Technology – Case Study Details Excellent Results for Hydrometallurgical Precious Metals and Base Metals Recovery

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Abstract

Today's mining companies must focus on technologies that provide improved recovery, low cost of operation, and environmental benefits that have the potential to enhance the overall rating of the company's environmental, social and governance (ESG) posture. This paper provides an overview of the projected benefits of using IXOS® Technology for gold and silver recovery from heap leaching with a throughput of 7.2M tons per annum @ 0.49g Au/t of ore. Based on recent test work at a client's site, data analysis confirmed higher adsorption recoveries compared to activated carbon in the laboratory for single pass, estimated reduction of 1,164.5 tons of CO₂/year to carbon emission (potential positive ESG impact), an operational cost expenditure cost savings of US\$0.61M/year due to no thermal regeneration requirement, and operational cost expenditure savings due to reduced fuel usage, an improved energy cost savings of US\$0.05M / year due to faster stripping kinetics and efficiency, and cost savings of US\$0.22M /year due to lack of carbon fine losses. Thus, an estimated annual savings of \$0.88M/year or US\$0.122/ton of ore heaped.

IXOS® Technology uses a unique application of a branch of nanotechnology called "MIPS" – Molecularly Imprinted Polymers. IXOS® attributes include preferential adsorption of gold and silver cyanide complexes from mixed metal cyanide solution, mercury remediation. And gold recovery is not impacted by high silver or copper. IXOS® beads can be used for the recovery of copper cyanide after all the gold and silver have been recovered from the mixed metal cyanide complex solution. IXOS® beads are non-reactive with kerosine when used in gold and silver recovery, where kerosine is used to blind the carbonaceous ore during gold leach. Stripping of the adsorbed precious metals is conducted under ambient pressure and low temperatures (60 degrees C) at a significantly reduced strip time of three hours per batch. There is also an opportunity for additional cost savings without the thermal regeneration required with activated carbon. This yields another potential ESG benefit.

Introduction

This paper presents information regarding metal recovery, lower cost of operation and potential overall environmental, social and governance value enhancement following laboratory test work, field test work, and a detailed review of the monetary and environmental savings utilizing IXOS® beads instead of activated carbon for the field trial at a mine in Copiapo, Chile in the Atacama Desert.

Sixth Wave Innovation Inc.'s Molecularly Imprinted Polymer IXOS® beads have a profound use case in the mining sector for metal recovery. The metal selectivity coupled with the demonstrated high-loadings and recovery of gold compared to activated carbon advances IXOS® as the technology of choice when dealing with recovery of gold from a complex metal cyanide solution typical of complex ore types.

This paper provides brief details of laboratory test-work results for a single column 24-hour run for IXOS® beads and activated carbon C-60 respectively. Further, a predicted model for a five-column adsorption circuit effectively demonstrates the superior performance of IXOS® beads compared to activated carbon.

The paper provides data from a field trial to further evaluate the efficacy of IXOS® technology and to simulate real-time operation using solution from ore-heap-leach columns and a multi-column bench-scale setup for IXOS® adsorption that mimics a gravity fed counter current adsorption train and a single column strip circuit. The IXOS® adsorption performance is highlighted with a model of the adsorption performance during the first twelve hours as well as the last eight hours to confirm that the field cumulative adsorption recoveries agreed with the predicted laboratory cumulative adsorption recoveries by $\pm 0.5\%$.

The strip performance data is also presented, and indicates that $>95\%$ of the Au on the loaded IXOS® beads will strip within two hours at 60 degrees C and ambient pressure as opposed to the high temperature and pressure stripping process for gold elution.

The overall OPEX cost savings associated with using IXOS® was calculated based solely on direct energy savings, since there is no thermal regeneration of the IXOS® beads after strip as required for activated carbon. The paper also demonstrates additional benefits of using the IXOS® technology (based on a throughput of 7.2M tons per annum @ 0.49g Au/t of ore) to include carbon emission reduction of 1164.5 tons CO₂/year; OPEX cost savings of US\$0.61M/year without thermal regeneration. There are additional OPEX savings due to reduced fuel usage and improved stripping kinetics of US\$0.05M/year, US\$0.22M/year savings due to elimination of fine carbon losses and an estimated annual savings of \$0.88M/year or US\$0.122/ton of ore heaped.

Laboratory test work

Ore leach and single-pass cumulative percentage adsorption recovery

Column adsorption was conducted in the laboratory using pregnant leach solution (PLS) obtained from

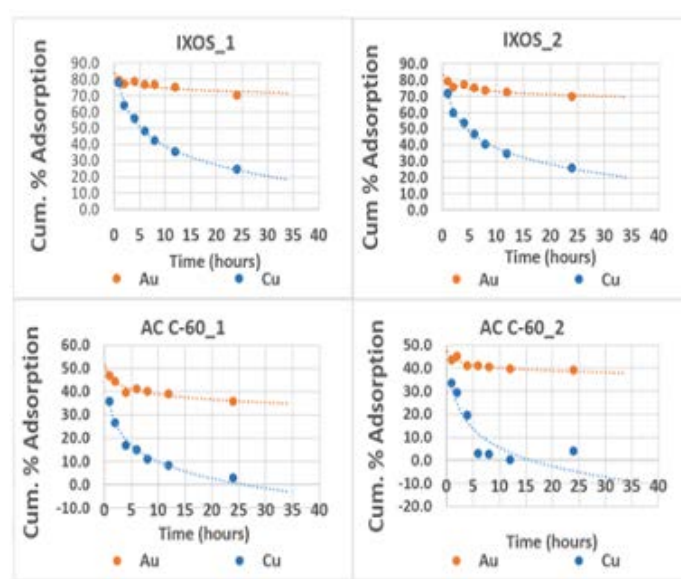
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bottle-roll leaching of client ore sample for 48 hours and collecting the filtered pregnant leach solution (PLS) with the metal cyanide complexes in solution. The ore sample leached was of average grade 0.426 g Au/t of ore, and 170.28 Cu/t of ore. The pregnant leach solution (PLS) passed through the adsorption columns was of grade 0.147 ppm Au and 1.361 ppm Cu.

The primary focus of the laboratory adsorption test was to evaluate the efficacy of IXOS® beads as a replacement for activated carbon in a proposed adsorption, desorption, and recovery (ADR) plant, with the following challenges:

- High copper concentration in the ore leached, meaning high copper in discharge which could influence adsorption kinetics.
- Low overall Au grade (~0.444 g Au/t of ore) leached.
- Process water used for all aspects of the ADR have a high salt concentration of ~207ppm of NaCl.

Test summary
Throughput 1 bv/min
Parallel flow through two column beds of IXOS® beads and activated carbon AC-60
Solution samples taken 2, 4, 6, 8, 12 and 24 hours.
Cumulative % adsorbed gold on IXOS® and AC-60 were 69.9% and 37.3% respectively
Cumulative % adsorbed copper on IXOS® and AC-60 were 25.0% and 3.4% respectively



IXOS® Beads vs. AC-60

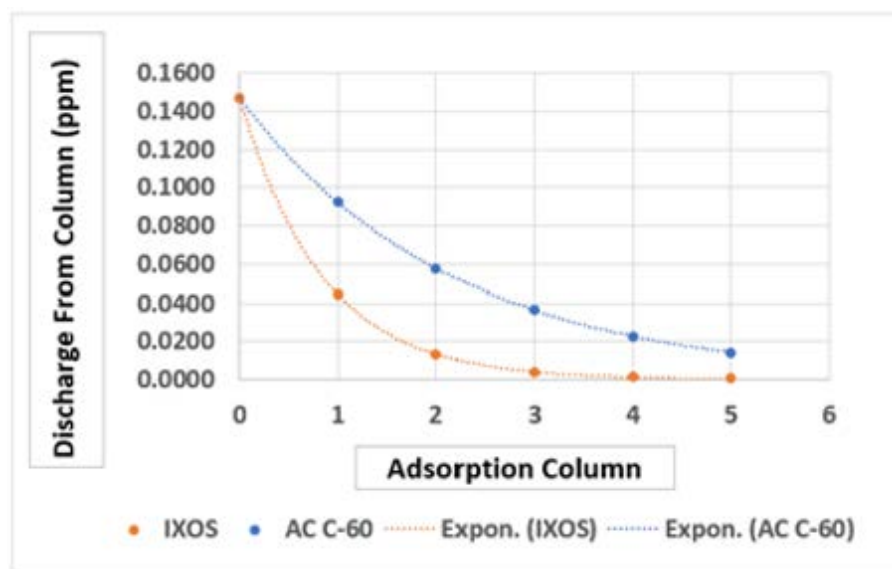
Figure 1: IXOS® beads vs. AC-60

Five-column adsorption circuit cumulative percentage adsorption recovery predictions

Using the single-pass cumulative gold adsorption recoveries for IXOS® and activated carbon AC-60 of 69.9% and 37.3%, extrapolation curves for cumulative circuit adsorption recovery were predicted for five columns in series for each train of IXOS® beads and activated carbon C-60. Figure 2 shows data and curves for predicted cumulative percentage gold adsorption for both IXOS® beads and activated carbon C-60 of 99.75% and 90.34% respectively.

Adsorption Column	Au					
	IXOS			AC C-60		
	Feed ppm	Barren ppm	Recovery %	Feed ppm	Barren ppm	Recovery %
0	0.1472	0.1472	-	0.1472	0.1472	-
1	0.1472	0.0443	69.9	0.1472	0.0922	37.3
2	0.0443	0.0133	69.9	0.0922	0.0578	37.3
3	0.0133	0.0040	69.9	0.0578	0.0362	37.3
4	0.0040	0.0012	69.9	0.0362	0.0227	37.3
5	0.0012	0.0004	69.9	0.0227	0.0142	37.3
Overall Adsorption Efficiency	99.75%			90.34%		

24-hr 1bv/min extrapolation data



Projected recovery for 5 columns in series

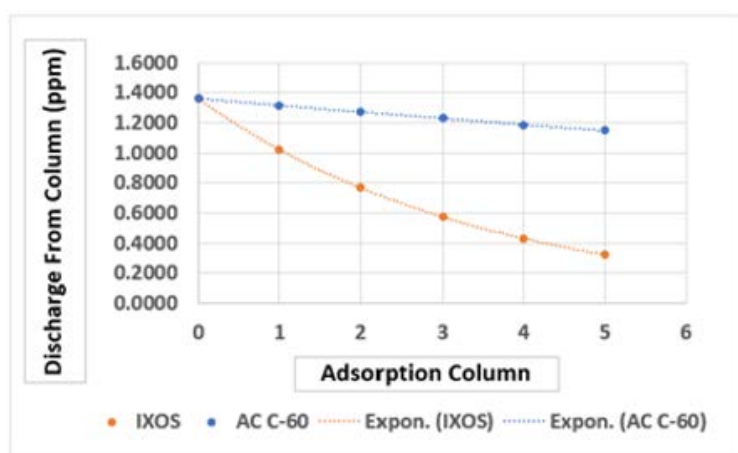
Figure 2: Predicted cumulative circuit gold adsorption percentage recovery

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Figure 3 shows data and curves for predicted cumulative percentage copper adsorption for both IXOS® beads and activated carbon C-60 of 76.21% and 15.69% respectively.

Adsorption Column	Cu					
	IXOS			AC C-60		
	Feed ppm	Barren ppm	Recovery %	Feed ppm	Barren ppm	Recovery %
0	1.3615	1.3615	-	1.3615	1.3615	-
1	1.3615	1.0216	25.0	1.3615	1.3158	3.4
2	1.0216	0.7666	25.0	1.3158	1.2717	3.4
3	0.7666	0.5752	25.0	1.2717	1.2290	3.4
4	0.5752	0.4317	25.0	1.2290	1.1878	3.4
5	0.4317	0.3239	25.0	1.1878	1.1479	3.4
Overall Adsorption Efficiency						
	76.21%			15.69%		

24-hr 1bv/min extrapolation data



Projected recovery for 5 columns in series

Figure 3: Predicted cumulative circuit copper adsorption percentage recovery

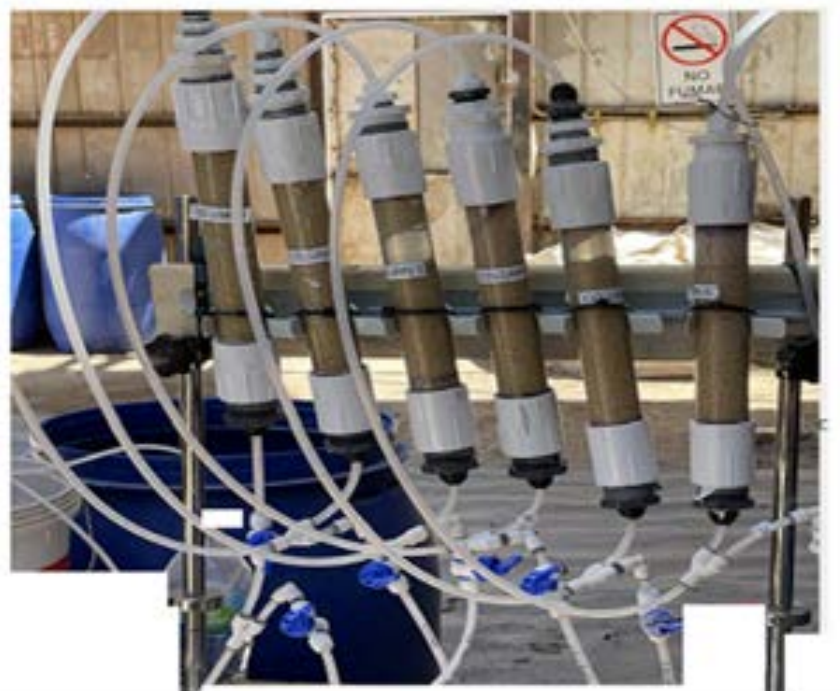
Field test work

Following compelling results obtained in the laboratory, the client contracted field testing using the same ore sample on a larger scale and at the mine site. Figure 4 contains pictures of the heap-leach columns and IXOS® adsorption column setup used. The IXOS® adsorption columns are scaled up from the laboratory configuration and representative of a typical carbon or resin train, and the scale-up parameters can be used for a full-scale operation with higher throughput.

A strip circuit setup with solution made up of 0.5% sulfuric acid and 1% thiourea was employed for stripping. Figure 5 shows the IXOS® Strip (Desorption/Elution) schematic process used.



Heap leach column set-up



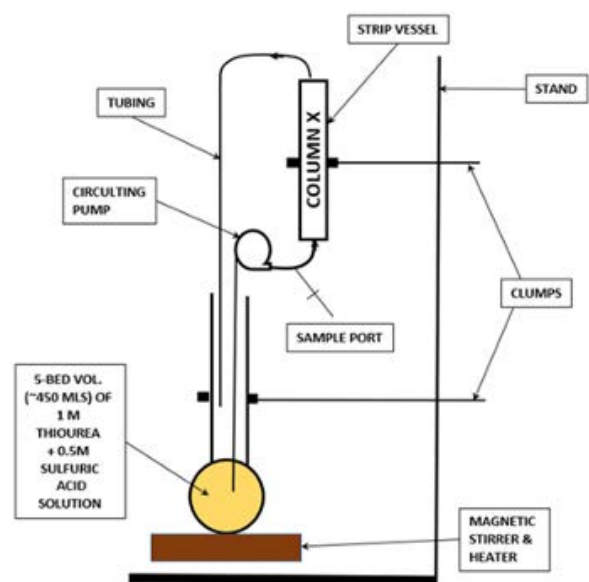
IXOS® adsorption column set-up

Figure 4: Field setup for heap-leach columns and IXOS® adsorption columns

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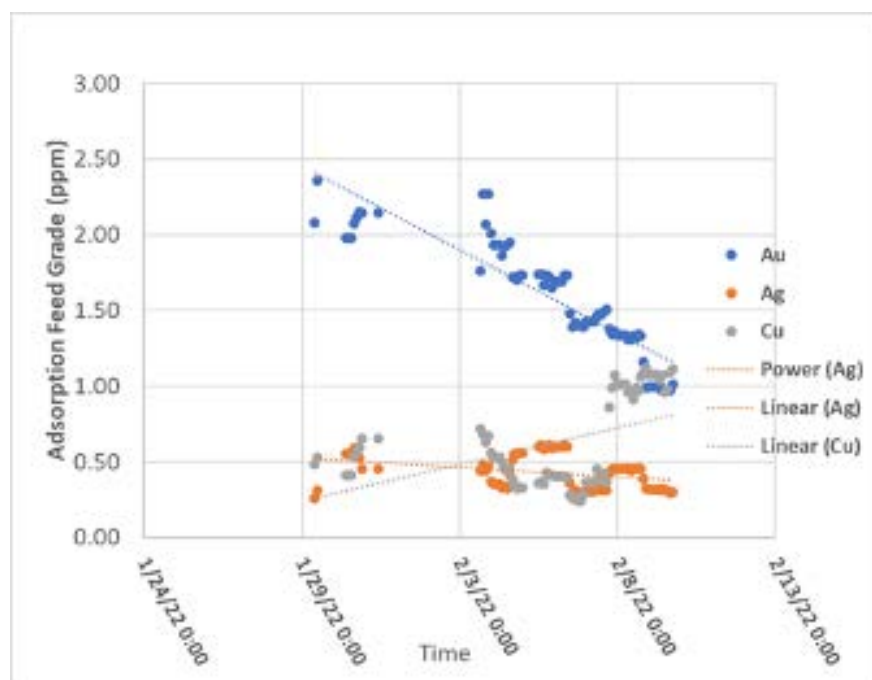
Site (strip/desorption/elution) circuit



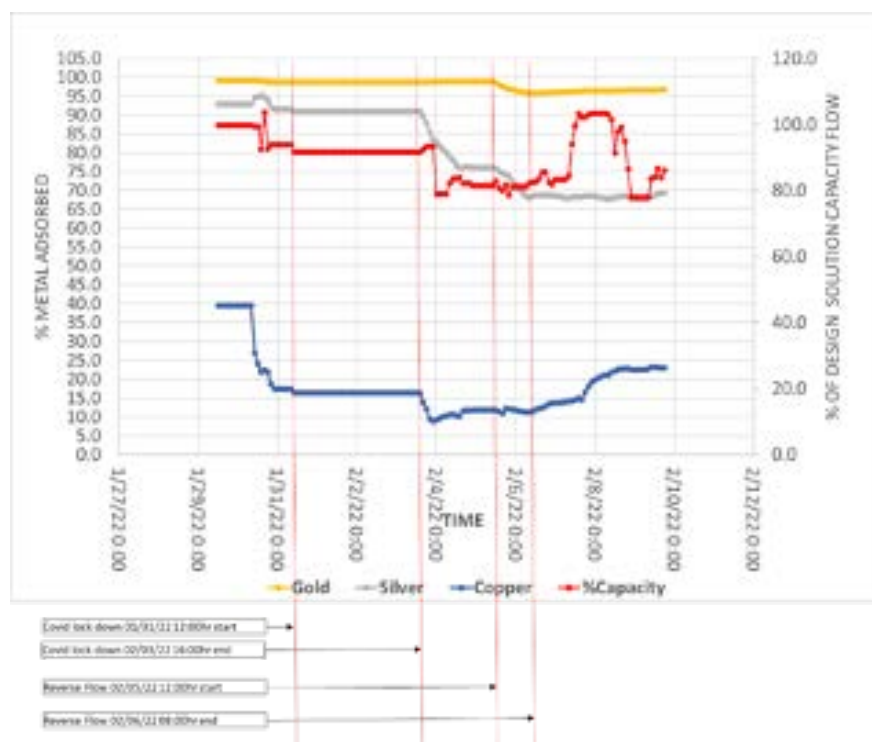
Schematic strip (desorption/elution) circuit

Figure 5: Strip circuit – site and schematic strip (desorption/elution)

Run hours for the entire test program were 145.92 hours. During this time, 693.20 L of PLS was processed through the adsorption circuit. IXOS® bed volume per column was average 90 ml, and average solution flowrate was 90 ml/min (i.e., 1 bv/min). A metallurgical accounting sheet detailing results of solution samples was generated based on field data and solution samples submitted to the client's independent laboratory for analysis. Figure 6 provides PLS grades to adsorption and cumulative metal adsorbed.



PLS grades to adsorption circuit

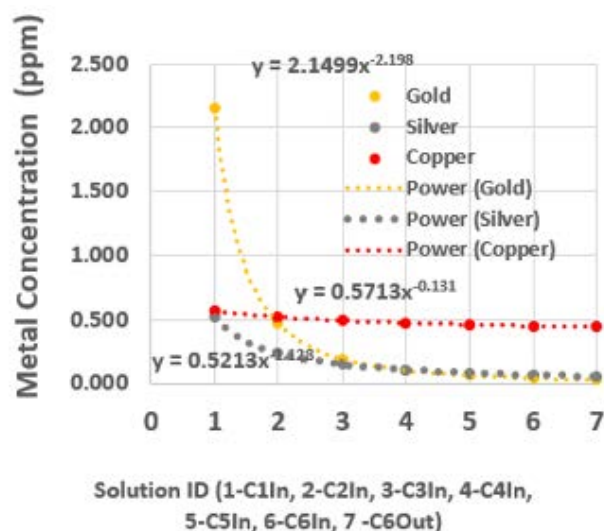


Cumulative metal adsorbed

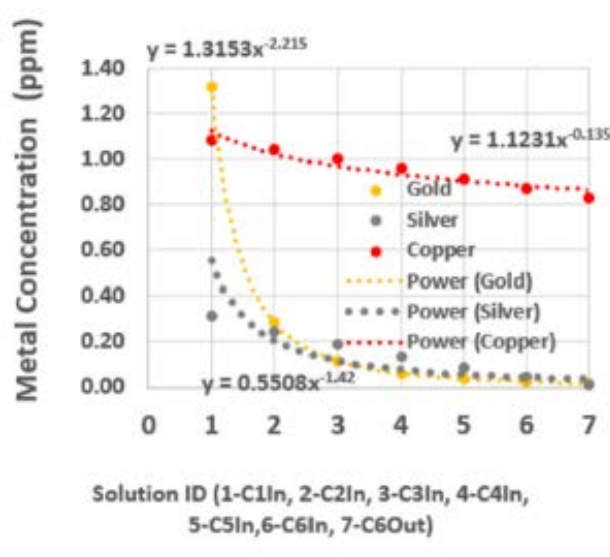
Figure 6: Pregnant leach solution (PLS) and cumulative metal adsorbed during test run

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Column rotation was established to remove batches of loaded IXOS® beads from the lead column, advance succeeding columns, and replace the last columns in each rotation with freshly stripped IXOS® beads consistent with a counter-current flow. With the data obtained, model derivatives were established for the first 12 hours at the start of testing and the last 8 hours before the test program came to an end.



First twelve hours of field test run



Last eight hours of field test run

Figure 7: Cum. ads models – first twelve hours and last eight hours

Figure 8 shows most of the gold being stripped within the first 3-4 hours. The low temperature strip and shorter strip time invariably will lead to energy savings which may reflect in the company's ESG performance.

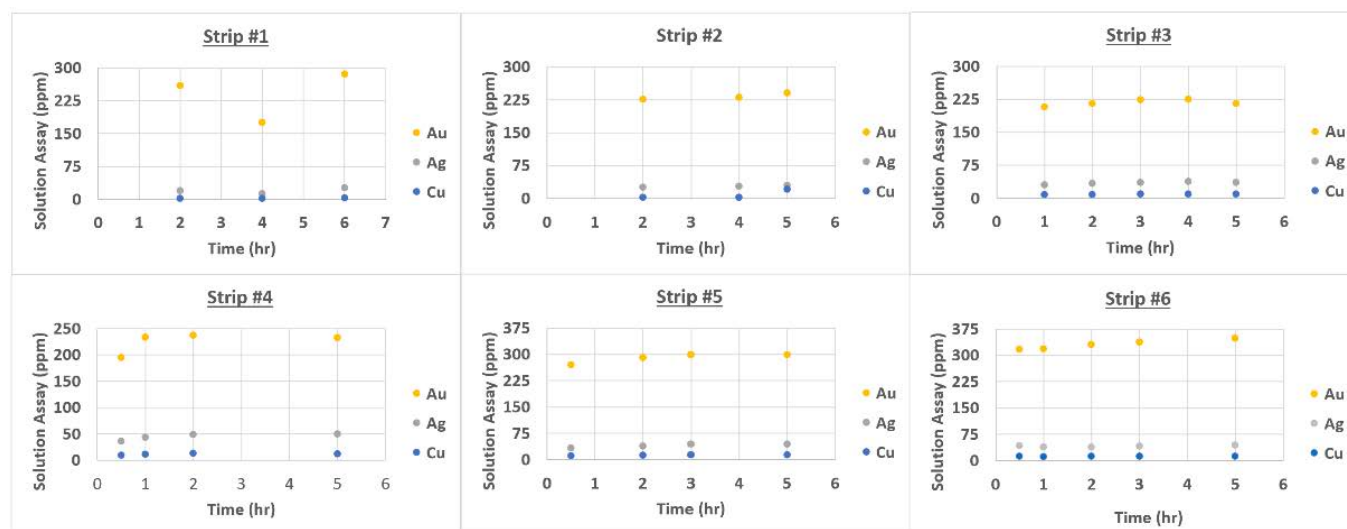


Figure 8: Strip solution curves for each batch of loaded IXOS® beads

Low cost of operation

AREA	Carbon-In-Solution Circuit			IXOS® -In-Solution Circuit		
	Installed Power (kw)	Maximum Demand (kw)	Energy Consumption (kwh/month)	Installed Power (kw)	Maximum Demand (kw)	Energy Consumption (kwh/month)
Facilities	37.3	22.4	14502.2	37.3	22.4	14502.2
Handling of Ragents	106.8	25.8	8651.0	106.8	25.8	6440.5
Adsorption	64.3	26.0	15971.1	65.1	26.4	17131.3
Desorption and Electrodeposition	579.4	303.1	130942.4	579.4	303.1	130942.4
Acid Wash	11.2	4.0	2175.3	11.2	4.0	2175.3
Thermal Regeneration	685.2	411.1	222001.6	0.0	0.0	0.0
Carbon Handling	34.5	8.1	3054.8	34.5	8.1	3054.8
Foundry	360.8	178.7	57886.9	360.8	178.7	32159.4
Complimentary Services	153.0	85.8	55604.2	153.0	85.8	55604.2
Total ADR Plant	2032.4	1065.0	510789.6	1347.9	654.3	262010.1
Cost \$US/ kWh	0.24			0.24		
US cost			122,589			62,882
Mineral tonnage/month	600,000			600,000		
Cost US/t			0.20			0.10

Cost Savings US/t			0.10
Annual throughput	7,200,000		
Cost Savings in millions US			0.72

Figure 9: Energy requirement carbon-in-solution vs. IXOS®-in-solution

With the excellent field data obtained, a review of the comparative operating cost was undertaken. This was conducted with the full vetting/approval of the client's subject matter expert. IXOS® beads do not require thermal regeneration after strip. This is because IXOS® beads are not organic and therefore are not

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FOR HYDROMETALLURGICAL PRECIOUS METALS AND BASE METALS RECOVERY

subject to organic fouling. Inorganic materials like scaling are primarily removed during the IXOS® strip or desorption process. As can be seen in Figure 9, savings of \$0.72M/ annum is expected in energy savings simply by eliminating the thermal regeneration equipment.

Per the Vertical Minfurn vendor manual, and depending on throughput, Figure 10 provides a summary of the Kw.h/ year requirement for a carbon batch of 10.375 tons per column going to strip. Equipment selection provided for a throughput of 432 kg carbon/hour, which will allow 10,375 kg of carbon to be regenerated in 24 hours. Thus, the Vertical Minfurn of 470 kw capacity is selected from the vendor specification as per Figure 10.

	Power								
	Power Kw	Kwh /Year	Kwh /day	Kg Carbon /hr	Kg Carbon /day	Kwh/ Kg	Kw.h /day	Kw.h /month	Kw.h / year
Vertical Minfurn	136	1191360	3264	125	3000	1.09	3264	97920	1175040
Vertical Minfurn	470	4120120	11288	432	10375	1.09	11288	338640	4063680
Carbon in Circuit	83	tons							
Carbon per tank	10.375	tons							

Figure10: Heat energy requirement – vertical Minfurn carbon regeneration

Description	Unit	Thermal Regeneration	Stripping (50% of combustible fuel for stripping loaded carbon)	Total Savings
	kwh/ month saved	338640.0	-	-
	months	12.0	-	-
	kwh/year	4063680.0		4,063,680.0
wt. of 1 liter of diesel	g	835.0	835.0	835.0
%C for 1 liter of diesel		0.9	0.9	0.9
C per liter of diesel	g	719.8	719.8	719.8
Oxygen required	g	1919.4	1919.4	1,919.4
CO2 from diesel	g/liter	2639.2	2639.2	2,639.2
1 liter of diesel produces approx.	MJ	38.0	38.0	38.0
1 liter of diesel produces approx.	kwh	10.0	10.0	10.0
Energy required for Vertical Minfurn	MkWh /year	4.1	0.0	4.1
Fuel required	liters / year	406368.0	34856.0	441,224.0
cost of diesel	US \$/liter	1.5	1.5	1.5
Cost savings	US \$M/ year	0.61	0.05	0.66
Reduced CO ₂ emission	tons/year	1072.5	92.0	1,164.5
Tonnes of ore stacked	tons/year	7200000.0	7200000.0	7,200,000.0
Cost Savings per tonne of ore	US \$/ton	0.08	0.01	0.09
Cost savings	US\$/month	50796.0	4357.0	55,153.0
Carbon losses	kg/yr			72,000.0
Price of carbon	US \$/kg			3.0
Carbon losses	US\$/yr			216,000.0
Total Savings	US\$/yr			877,836.0
Total Savings	US\$/month			73,153.0
Cost Savings per tonne of ore	US \$/ton			0.1

Figure 11: Savings due to using IXOS® beads

Figure 11 shows the combined energy savings using IXOS®-In-Solution for gold recovery for eliminating thermal regeneration, (US\$0.61M / year). Reduction in the use of the combustible fuel (~50% of typical loaded carbon strip circuit) during strip due to shorter strip times and no heating above 60 degrees provides energy savings of (US\$0.05M per year). Combined total savings of (US\$0.66M per year) is expected. Savings from carbon losses of US\$ 0.22 M per year with no carbon usage. Total of US\$ 0.88 M per year with the use of the IXOS®-In-Solution Technology.

Potential enhancement of ESG values

The client case study budget figures were confirmed by an independent consultant familiar with similar mining properties in South America, indicating an ADR plant for handling gold recovery with ore throughput of 7.2M tons/year will require an average 69,712 L per annum for maintaining heat energy combustion. Using IXOS® it is anticipated that only 50% of fuel will be required to keep the strip at 60 degrees C since there is no need for stripping eight hours, as is the case with loaded activated carbon. Subsequently, the amount of fuel saved will be 34,856 L per year. With 2,639.2 g, CO₂ generated per litre of diesel used, the reduced emission resulting from stripping IXOS® beads is on the order of 91.99 tons reduction per year.

Similarly, without thermal regeneration of activated carbon, the client's consultant determined the fuel reduction is 406,368 L per year (Figure 11). Thus, reduced CO₂ emission per year is estimated to be 1072.47 tons per year per as also calculated in Figure 11.

The total reduction in CO₂ emission for using IXOS® compared to activated carbon is 1164.46 tons per year just from these usage sources. Additional indirect CO₂ savings can be quantified for fuel deliveries without the use of IXOS® for a functional activated carbon ADR plant at the mine site at high elevation in the Atacama Desert but are not presented for this analysis.

Results and discussion

The laboratory test work from Figure 1 and Figure 2 details:

- Better gold adsorption performance of IXOS® compared to activated carbon. The predicted 99.75% cumulative gold recovered using IXOS® beads for a 5-column adsorption circuit suggested that a 6-column adsorption circuit was optimum for a field test in combination with the strip.
- Removal of copper from solution using the predicting model suggests 4 times higher for IXOS® compared to activated carbon. However, it is well known from lab work that gold is preferentially adsorbed to copper in succeeding adsorption columns in a circuit. It should be noted that the laboratory test was data derived from a 24-hour single pass and therefore the field test was

warranted to investigate and ascertain the adsorption kinetics over a longer period at more operationally relevant conditions.

Furthermore, the field test results found:

- Per the model, derivatives in Figure 7, show 98.61% Au, 88.86% Ag and 22.46% Cu adsorption efficiencies respectively within the first twelve hours of the bench-scale adsorption testing.
- Per the model, derivatives in Figure 7, show 98.7% Au, 96.6% Ag and 23.7% Cu adsorption efficiencies in the last 8-hours. Note: Gold values in PLS to adsorption circuit were low, copper values high and five column adsorption rotations, and two batches of stripped IXOS® beads transferred to two adsorption columns in the adsorption circuit.
- Achieved excellent performance, >98% gold adsorption recovery, at a nominal flowrate of >80% of design capacity. In addition, lower copper adsorption makes IXOS® beads a compelling adsorbent.
- Per Figure 8, strip times of ~2 hours are achievable without compromising complete metal removal. The benefits of 100% efficiency cannot be over-emphasized, as it allowed stripped IXOS® beads to return to the adsorption circuit without loss of adsorption recovery capacity, thereby mitigating gold losses in circuit performance. Moreover, it allows for a comparatively smaller strip circuit with lower energy and reagent usage

Conclusion

Use of IXOS® technology at heap leach mines can result in:

- Improved gold adsorption compared to activated carbon (lower Au/Ag in tailings).
- No losses to carbon fines.
- Reduced carbon dioxide emission or carbon footprint worth ~ \$.116M annually in tradable carbon credits.
- Potential enhancement of ESG position.
- Reduced energy consumption resulting in US\$ 0.88M per annum for the proposed 7.2 M tons of ore to heap per annum.

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The Piauí Nickel Heap Leach Commercial Plant

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Abstract

Located in north-eastern Brazil, the Piauí nickel project will be the first full-scale commercial nickel and cobalt heap leach facility in the world. The first stage of the commercial plant was commissioned in the spring of 2022 and the first metal production will occur in June of 2022. Piauí will produce intermediate nickel and cobalt products specifically designed for the electric vehicle battery market. Some advantages of heap leaching for EV metals include lower capital intensity, lower operating costs, smaller environmental foot print, and reduced CO₂ emissions. Regarding CO₂ emissions, Piauí will produce one of the lowest carbon-intensity products in the nickel industry, and research is currently underway to make the product carbon neutral, or possibly carbon negative. In parallel with commissioning this facility, Brazilian Nickel is completing a feasibility study for the ultimate plant.

This presentation will present the results from the commissioning and first few months of operations of the Piauí facility, along with projections for full ramp-up and the implications for the larger facility.

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