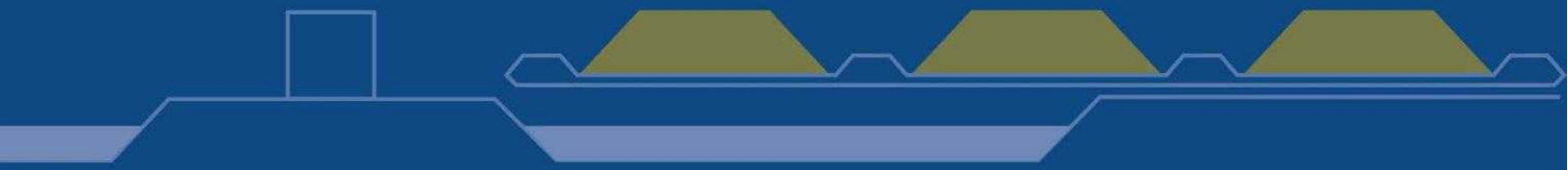
Proceedings of the Heap Leach Solutions Conference



September 22-25, 2013
Vancouver, Canada

Heap Leach
SOLUTIONS 2013

Proceedings of the Heap Leach Solutions Conference

September 22–25, 2013, Vancouver, Canada

EDITORS

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Conference organized by:

InfoMine
Mining Intelligence & Technology

in partnership with:

NORMAN B. KEEVIL
INSTITUTE OF MINING ENGINEERING
THE UNIVERSITY OF BRITISH COLUMBIA

UBC Materials Engineering

Production and editorial: Joe Goodwill, InfoMine Inc.
Cover design: Muhammad Asad, InfoMine Inc.

ISBN: 978-0-9917905-2-4

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PREFACE

Many technological, regulatory and economic developments have resulted in much more sophisticated planning for the development, operation and closure of heap leach facilities. While heap leaching remains an appropriate recovery method of metals at lower capital and operating costs in drier climates, modern heap leach operations are currently being developed and run in a much broader range of climatic conditions. Many professionals have worked extremely hard for more than 30 years to establish much more robust projects.

It is against this background that the Heap Leach Solutions 2013 conference takes place. The published papers and presentations represent a very good snapshot of the range of aspects that must be considered for successful heap leach projects.

From the outset this conference was planned to bridge the gap that may exist between professionals with different expertise, such as metallurgists and geotechnical engineers. A successful heap leach project can only be developed if diverse professionals work as a team, pooling their knowledge and developing a clear understanding and appreciation for the contributions of all team members.

This proceedings volume contains 46 fully reviewed papers. The conference program includes a further nine presentations. All presentations for which we can get permission will be available at http://www.heapleachsolutions.com following the conference.

This volume reflects a synthesis of global knowledge, research and practical experience of heap leaching. As such, it will become a very useful reference document for the development of future projects, and in the education and training of young professionals involved in heap leach practice all over the world.

ACKNOWLEDGEMENTS

The Organizing Committee acknowledges with gratitude the authors' contributions of high quality, detailed and innovative papers.

We would also like to thank the technical reviewers, the employees of InfoMine and all those involved in the organization of this conference and the preparation and production of these proceedings. The support of the committees is greatly appreciated.

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 inaugural **Heap Leach Solutions Conference**. We look forward to seeing you all again at the next conference in this international series.

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PART 1

MINERAL PROCESSING

Experience in heap leaching research work and technology development for oxidized ores from porphyry copper deposits

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Abstract

A brief summary of modern conditions of copper recovery from oxidized ores using the heap leaching method is given. It is shown that this technology implementation in Russia is promising for the treatment of ores from porphyry copper deposits. Russia has considerable reserves of copper, but they are located in the northern regions. Consequently, the development of heap leaching technology under low temperature conditions is of interest.

The results of an integrated study of oxidized ores from porphyry copper deposits performed at OMS Institute are presented in this paper. The study was carried out to gain information about the possibility of treating ore with the heap leaching method. A material composition study with the QEMSCAN system allows one to conclude that the ore technological characteristics are its high level of oxidation, a high volume of fine fractions, low hardness properties and a high level of acid-consuming minerals. Technology development for this type of ore treatment requires a special approach.

The results of laboratory testing for ore heap leaching in small columns of 1.5 m in an open cycle have shown that the optimal grind size for heap leaching is equal to –30 mm, taking into account the necessity to provide the appropriate hydro-physical properties of material. Copper dissolved into leach solution under these laboratory conditions equals to 85.6%, while sulfuric acid consumption is 82 kg per tonne of ore.

The result of testwork performed in a commercial depth column (over 70 days) in a closed cycle with solvent extraction (SE) showed that the received copper extraction was 77.8%. Copper content in the tailings was less than 0.09%. The total sulfuric acid consumption for pelletizing and leaching decreased down to 66 kg/t of ore, but still remains high, and that is connected with the presence of a significant amount of acid-consuming minerals in the ore. At the industrial level of heap leaching technology

implementation it is planned to decrease the level of sulfuric acid consumption down to 43 kg/t of ore using a step-by-step decrease of acid concentration in the leaching solution, a decrease of copper recovery targeted value, and an increase of leaching time.

Taking into account ore specific characteristics and its processing properties it was suggested to treat these minerals in heaps not higher than 4 m. Heap height is limited by the ore hardness properties, and in order to increase the heap height, the principle of blending the ore with lower and higher hardness was implemented. High moisture and low hardness of the discharged material do not allow one to organize a multi-stage heap leaching system.

The experience of heap leaching technology development under low temperature conditions is very important for the development of porphyry copper deposits in Russia.

Introduction

In 2011 the world production of refined copper by means of the leaching solvent extraction (SX) electrowinning (EW) process amounted to about 3.4 million tonnes, making up about 17% of total refined copper production, or 20% of primary copper production (ICSG, 2012; USGS, 2013). A hydrometallurgical process is normally used for the treatment of oxidized copper ores containing malachite and azurite, and for ores containing chalcosine. Leaching of copper sulfide minerals such as covelline and bornite, as well as leaching of native copper, can be performed under bacterial oxidation conditions. Primary Cu sulfide minerals, e.g. chalcopyrite, are not susceptible to standard heap leaching (Schlesinger et al., 2011). Thus, heap leaching of oxidized copper ores is the basic method of Cu hydrometallurgical production.

In Russia Cu is not actually produced by such hydrometallurgical processes as heap leaching. Since 2005 JSC ("UralGydroMed") has processed oxidized Cu ores at the Gumeshevskoe deposit using underground leaching technology. The plant production capacity is 5,000 tonnes of refined copper per year, which is 1% of the total annual refined copper production of Russia. At the Severonickel plant owned by OJSC ("Kolskaya Mining and Metallurgical Company") copper is partly produced by roasted Cu concentrate leaching, followed by electroextraction (EE).

At present CJSC ("Russian Copper Company") develops projects for the processing of oxidized Cu ores from the Mikheevskoe and Tominskoe deposits (in the Chelyabinsk region) by heap leaching. Cumulative reserves of these two porphyry copper deposits amount to 800 million tonnes of ore with an average Cu content of 0.4% (Altushkin et al., 2012); oxidized ore reserves are estimated at 10.8 million and 64.2 million tonnes for these two deposits, respectively. The production capacity of the plants that are planned to use heap leaching combined with SE and EW processes will amount up to about 10,000 tonnes of cathode copper per year.

In addition, Russia possesses considerable inferred Cu reserves, presented by porphyry copper deposits. The Peschanka deposit, which is part of the Baimskaya ore zone (the Chukotka Autonomous District) is considered to be one of the most promising deposits, with potential resources initially estimated to be about 27 million tonnes of Cu and 1,600 tonnes of Au. In view of the fact that oxidation zone thickness may reach some tens of meters, and the average Cu content in the ore body is between 0.5 and 0.7%, there is great interest in processing the oxidized ores of the deposit by heap leaching. In this context, the development of an acid heap leaching process for use in rigorous climatic conditions is of great interest.

Testwork

Research has been done on oxidized ores from southern Ural porphyry copper deposits. The studies included detailed material composition analysis using Quantitative Evaluation of Minerals by Scanning Electron Microscopy (QEMSCAN), laboratory scale testing of process ore characteristics and pilot scale tests.

Mineralogy

The tested ore contains about 0.4% of Cu, predominantly in oxidized form. The total oxidized copper portion comprises 94%, where 70% is accounted for copper in free oxidized minerals (basically malachite and azurite). This copper is readily soluble under moderate acidity conditions (pH <5) (Schlesinger et al., 2011). The sulfide minerals portion contains only 6% of Cu.

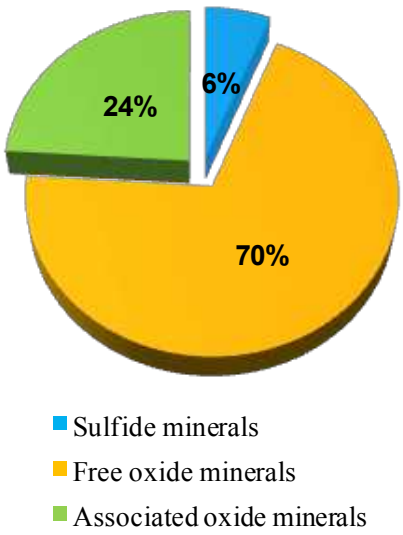

Figure 1: Cu minerals in the ore

Particular process ore characteristics include:

- an extremely high oxidation rate;
- a large amount of fine fractions material;
- low strength properties; and
- the presence of many acid-consuming minerals.

These characteristics cause low permeability of the ore pile and high acid consumption. That is why this tested ore cannot be considered as one of the classical oxidized ores of porphyry copper deposits conventionally processed by heap leaching. For successful heap leaching of this ore a special approach must be developed, taking into account the ore mineral composition and its physical properties.

A preliminary study of the physical properties of the ore showed that efficient heap leaching of the ore requires its agglomeration with binders. Sulfuric acid will be added to the ore at a rate of 15 kg/t during the agglomeration. Agglomerated ore can be piled in heaps up to 4 m high, and the density of the leach solution flow should not exceed 5 l/h*m^2.

Agitated leaching tests

Laboratory ore testing by agitated sulfuric acid leaching was carried out on material of 80% –38µm size over 120 hours, with pH level kept at 1.0. The leaching was performed in a beaker with mechanical agitation; the pulp density was 33% solids. Figure 2 shows plots of Cu dissolved into solution and sulfuric acid consumption as a function of the ore agitated leaching time.

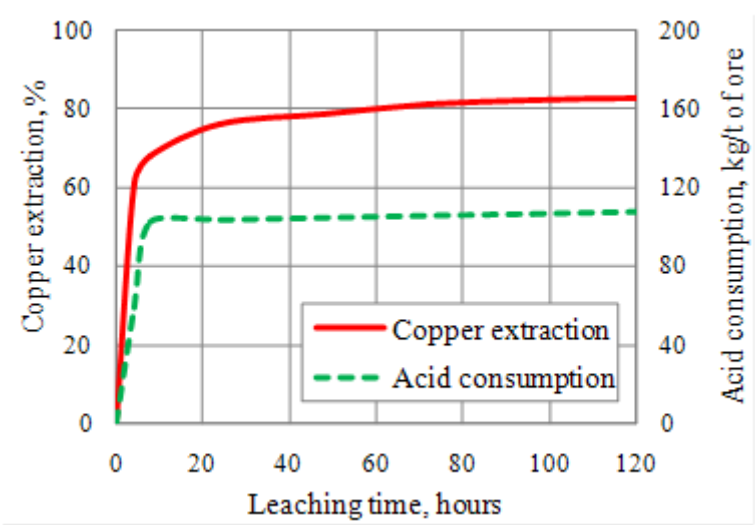

Figure 2: Cu dissolved in pregnant leach solution, and sulfuric acid consumption in the course of agitated leaching

Copper dissolved from the ore using agitated leaching totals 82−84%. The most intensive Cu dissolution occurs within the first four hours of leaching, when the majority of free oxidized Cu minerals (malachite, azurite) are being dissolved. Further leaching leads to the dissolution of the remaining free oxidized minerals and partially associated oxidized minerals. The latter are represented as inclusions in rock-forming minerals. Sulfuric acid consumption is within a range of 80−130 kg/t, and primarily related to the dissolution of gangue acid-consuming minerals, rather than to Cu dissolving.

Agitated leaching of 80% –38μm material quickly and thoroughly liberates minerals and enables a high Cu amount to pass into solution, but it cannot serve as a model for commercial level heap leaching. Thus, the obtained results should be regarded as a general evaluation of the ore leaching behavior into sulfuric acid solution.

Laboratory leaching in mini-columns

Laboratory scale heap leaching tests were carried out in mini-columns of 1.5 m height in an open cycle on the agglomerated ore of –30, –20, and –10 mm sizes. The heap leaching duration was 71 days. Table 1 presents the results of the ore leaching in columns.

Table 1: Results of the ore leaching in mini-columns

The ore crush size, mm	Cu passed into solution, %	H2SO4 consumption, kg/ton of ore
–30	85.6	82
–20	86.8	85
–10	85.9	93

The results of ore leaching in mini-columns (as shown in Table 1) demonstrate that size reduction from –30 mm to –20 mm and –10 mm does not allow any significant increase of Cu passing into solution, but at the same time leads to an increase in total consumption of sulfuric acid.

Figure 3 demonstrates curves of Cu passed into leach solution as a function of leaching time in columns. Figure 4 shows the Cu concentration changing in pregnant solutions over the course of the test duration. Figure 5 illustrates sulfuric acid consumption as a function of Cu passing from the ore into solution.

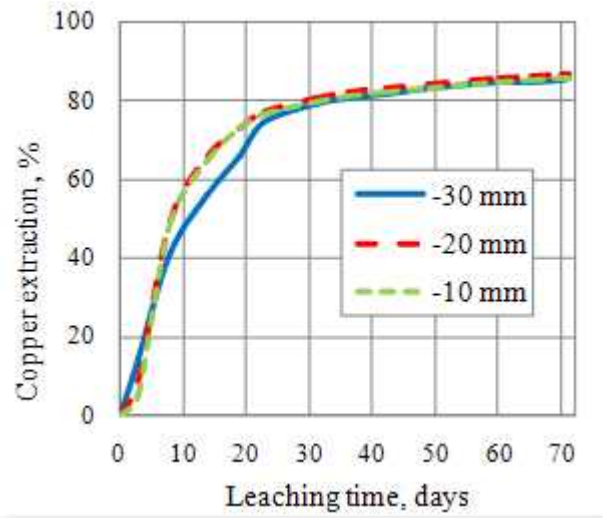

Figure 3: Cu extraction from the ore

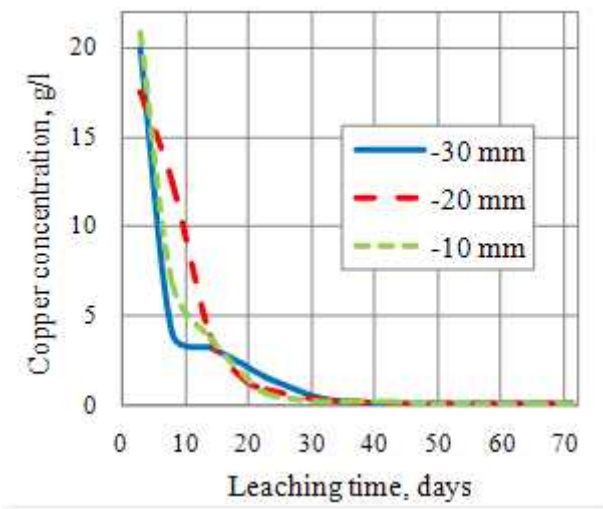

Figure 4: Cu concentration in solution

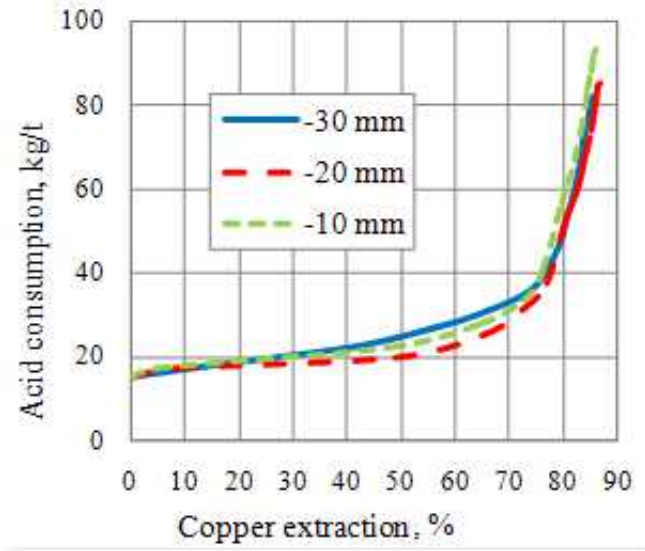

Figure 5: Sulphuric acid consumption

Based on the data obtained from the ore percolation leaching in mini-columns and taking into account the need to keep hydro-physical properties of the material at a sufficient level, it can be concluded that –30 mm is the optimal material size for heap leaching.

Laboratory testing in mini-columns carried out in an open cycle (without solutions recirculation) makes it possible to evaluate only the general level of metal solubility depending on the ore crush size. Pilot scale testing can provide more detailed data. Such pilot scale tests will be performed in a larger diameter column with depth corresponding to heap height, and in a closed cycle with Cu extraction from pregnant leach solutions.

Pilot testing

Testing in a commercial depth column was carried out in a closed cycle with SX. Figure 6 shows the testing conceptual flowsheet.

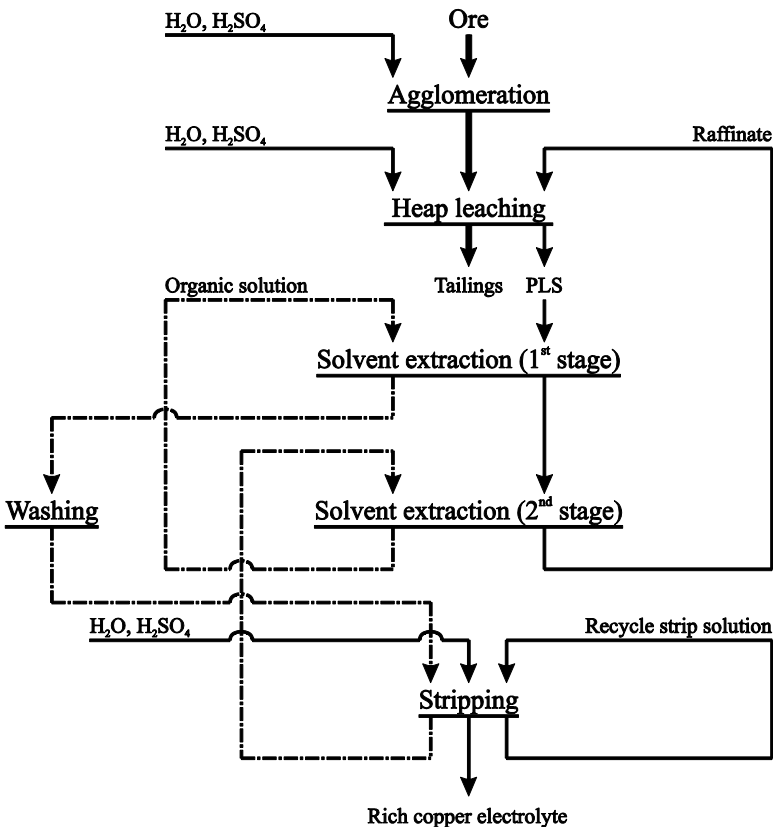

Figure 6: Pilot testing flow chart

The ore of –30 mm size was agglomerated using a laboratory pelletizing disk and adding sulfuric acid as a binder at a rate of 15 kg/t, and water for the ore wetting. To make pellets solidified and to improve their mechanical properties the material was maintained during 72 hours before charging it in the column. Agglomerated ore was loaded in the column at a height of 4 m. Leach solution (H_2SO_4) was pumped on the ore by a dosing pump. A pregnant Cu-containing solution was collected from the bottom part of the column.

The pregnant solution was subjected to a two-staged extraction with aqueous and organic phases counter flow. A mixture of LIX 984N extracting agent and ShellSolD90 diluting agent was used as an organic phase. A pregnant organic phase was reported to washing and Cu re-extraction. When the aqueous phase reached a Cu concentration of more than 30 g/l a part of the solution (copper electrolyte) was removed and stored in a separate collection tank. Then, a fresh sulfuric acid solution was added to the re-extraction solution and the latter was returned back for re-extraction. The organic phase after re-extraction was taken back to extraction.

The ore was heap leached in the column over 70 days with acidified recycling solution fed for irrigation. Then, the column was washed for three days with the addition of decoppered solution (raffinate) without acidification. After this time the solution feeding was stopped and over the course of

the following three days effluent solutions were collected until the drainage stopped. Upon completion of heap leaching the material was taken out of the column. Figure 7 shows the curves of dissolved Cu and its concentration in pregnant solution as functions of leaching time. Figure 8 illustrates sulfuric acid consumption (per tonne of ore) and the pH of pregnant leach solution depending on leaching time.

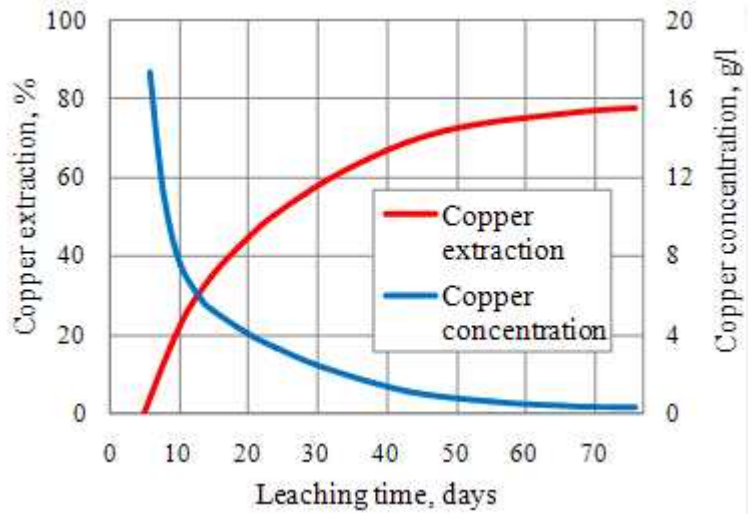

Figure 7: Cu dissolution and passing into PLS, pilot testing

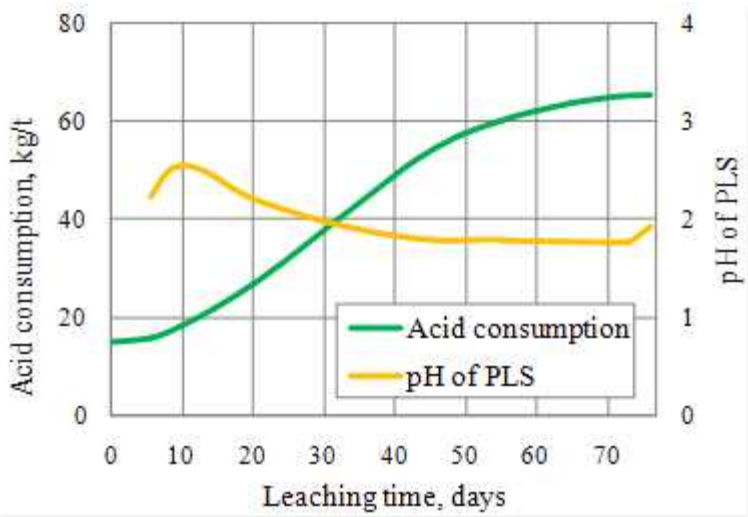

Figure 8: H_2SO_4 consumption in course of pilot testing

Pilot testing (heap leaching + SX) showed the Cu extraction into rich copper electrolyte was equal to 77.8% at less than 0.09% Cu content in tailings. It should be said that the Cu content in the ore calculated according to pilot testing balance was about 0.38%.

The total acid consumption for pelletizing and leaching decreased from 82 kg/t in a mini-column test to 66 kg/t in a commercial depth column test. Despite this, acid consumption is still high, and that is explained by the presence of a considerable amount of acid-consuming minerals in the ore.

Pilot testing was carried out with solutions in a closed cycle, as a result of which the dynamics of Cu dissolving from the ore is lower than in the tests performed in mini-columns. This is explained by the fact that leaching with recycled solutions causes an increase of the solution salt level. By the moment of leaching completion the rate of cumulative major metal impurities (Fe, Al, Mg, Mn, Si, and Ca) passing into solution was 15.4 kg/t. The summary concentration of these metals in the final pregnant solution was about 54 g/l. Salinization of solutions has a negative impact on Cu dissolving from the ore.

In addition a sharp rise of the impurities concentration in solution can lead to solution oversaturation and colmatation of the heap, especially under low temperature conditions. The maximum allowable concentration of metal impurities was determined for the pregnant leach solution. The limit of the impurities (Fe, Al, Mg, and Si) solubility in the solution at 20° Celsius amounts to 85.7 g/l; on cooling the solution down to 5° Celsius the limit of solubility goes down to 78.5 g/l. Therefore, the heap leaching processing can be carried out with pregnant solutions obtainment at a temperature of 5° C.

Processing properties

Commercial heap leaching of the ore requires consideration of the ore composition and properties as well as correction for Russian climatic conditions that are extremely different from the climate of most countries producing copper with the SX-EW process. For this reason the process technology under development has a range of specific characteristics:

1. It was proposed to process the ore in stockpiles with a height not exceeding 4 meters. Taking into account that the heap height is limited by the ore strength properties in order to obtain 4 m heap height, the blending of the ore with lower and higher hardness was implemented.
2. To increase the leach solution flow rate up to $5 l/h*m^2$ the ore is to be agglomerated with sulfuric acid addition (15 kg/t) before heap leaching.
3. For acid consumption reduction down to 43 kg/t it was offered to gradually decrease acid concentration in the leach solution concurrently with target Cu extraction value lowering and extension of leaching time against the pilot test leaching.
4. A special cycle scheme was developed to perform heap leaching with stockpiles divided into segments. This scheme allows continuous operation of leaching; SX and EW work sections both in warm and cold seasons. An advantageous characteristic of the process organization is

the resulting stable joint flow of pregnant solutions coming from the different ore piles segments.

5. To prevent pellet destruction during rainy seasons leach solutions flow for heap wetting should be reduced along with an acid concentration increase in these solutions. By contrast, in drought periods solutions feeding and acid concentration should be maintained in the range wherein pH value and an average unit performance (as per pregnant solutions and dissolved copper) are kept at an adequate level.
6. To compensate for heat losses in the cold season the Cu-containing solutions fed for extraction should be heated to above 15°C. The expected temperature drop of solutions in the course of their passing through ore heap amounts to 9−12°C.
7. Leach solutions heating is affected by heat disengaged at dilution of concentrated sulfuric acid. When the acid concentration in leach solutions is at 15 g/l and 35g/l levels these solutions will heat up to 2°C and 5.2°C respectively.
8. Working in the cold season requires implementation of warmth-keeping procedures for the main pregnant solutions collector and outdoor pipelines. The irrigation system on the pile surface is to be covered with polyethylene film as well.
9. The ore after leaching is characterized by high humidity and low strength properties, which does not allow forming of a multi-layered heap and requires a leached ore off-load after each leaching cycle.
10. In view of the cold Russian climate, dumps of leached ore will slowly lose their moisture. Rains will provoke acidic solutions flowing-off which must be neutralized in order to prevent an environmental impact.

Conclusions

Despite the seeming simplicity of heap leaching, this process faces a number of challenges that are complicated by Russian climatic conditions. To develop an efficient and reliable process technology, the following challenges must be met:

- The maximum allowed pile height should be determined, based on both initial hydro-physical properties of the material and its hydro-physical properties during and after leaching.
- The possibility of a multi-layered heap formation must be determined.
- An allowable leach solution flow rate must be determined.
- Optimal acid and impurities concentration in leaching solution must be specified.

- The required leaching time must be determined.
- A method must be found for waste reclamation under severe weather conditions (low temperatures in winter and heavy rains in summer).

The technology of heap leaching processing of ores from porphyry copper deposits is still being improved, with allowance for proper ore characteristics and its composition, as well as for particular climate conditions. However the experience of such process technology elaboration is of great importance for the development of promising copper deposits located in the territory of Russia.

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Increasing heap leaching efficiency by ore breakage in high pressure grinding rolls

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Abstract

This paper highlights the main issues with the intensification of heap leaching processes and determines a promising direction for improving this method of gold recovery from gold containing ores. It suggests the use of high pressure grinding rolls in a grinding circuit. Gold recovery in the process of heap leaching is mainly determined by the solvent access to the surface of precious metal. The head ore is subjected to crushing in order to increase its permeability, thus increasing the surface of interaction between metal and complexing agent. In addition, the fracturing of crushed material is increased by ore breakage in high pressure grinding rolls, which in turn enhances the heap permeability in the process of heap leaching due to more complete material liberation. These methods will typically increase gold leaching kinetics and permit an increase of the overall recovery level. Even a small percentage increase in the recovery level, utilizing the technology of gold recovery from mineral ores with the heap leaching method, will have a substantial economic effect. Combined with the low unit and capital costs of installation and operation of high pressure grinding rolls (in comparison with traditional crushing and grinding units), this technology can be considered to be highly efficient.

This paper describes tests which are carried out for ore grinding simulation in high pressure grinding rolls. Ore samples in the study were broken in a special unit that imitates material weakening in laboratory conditions. The pressure of breakage was equal to 10−15 N/mm^2, which corresponds to industrial parameters. After grinding the material in this unit, its material composition and technological properties were analyzed in comparison with traditional crushing products of the same size (crushing in a jaw crusher) and with material produced in an industrial unit in situ. The results of research compared

precious metals recovery level using this breakage method usage, and also indicated the accuracy of breakage process simulation in laboratory conditions.

Breakage application in high pressure grinding rolls not only enables a reduction in the number of grinding stages, thus reducing energy consumption for ore preparation, but also determines the actual selection of ore processing technology. Thus in the case of changing the material type for heap leaching technology (changing oxidized ore for blended or primary), in most cases it is necessary to construct a new processing plant in order for the project to be profitable. In some cases, implementing high pressure grinding rolls permits one not only to extend heap leaching unit life time, but also to keep this technology for all types of ore in the deposit.

Globally, there are almost a dozen ore processing plants successfully using high pressure grinding rolls. Five large plants using these units are being currently designed. The key factor which determines the preference of its use (apart from significant energy consumption reduction) is its high capacity, which can be compared with the largest SAG mills throughput.

Introduction

Traditionally, the decrease of the technological and economic parameters of gold containing ore processing from different deposits is primarily related not only to mineral resources reduction, but also to the degradation of the processed ore quality and the increase of operating costs in the ore preparation section. The technology of gold containing ore processing using heap leaching method has been studied comprehensively, but some methods for improving the efficiency of this technology require further study.

In general the main ways of improving the heap leaching process can be divided into the following categories:

1. the study and development of new cost-efficient (compared to conventional) ore preparation methods;
2. the investigation of new or updated methods of ore piling;
3. the selection of new reagents to increase the recovery of precious metals; and
4. the development of high technology controlling processes, specifically fully automated control of all technological circuit sections.

The purpose of ore preparation is to obtain ore particles with a size that enables the contact of a cyanide solution with the metal surface, maintaining ore pile stability and high filtrating capability. In general, two or three stages of crushing are used, depending on the physical and mechanical properties of the head ore, and on the heap leaching technological parameters required. One of the most promising directions for heap leaching improvement is size reduction (to size fractions of −10 mm to −3 mm, and

even smaller) of the material processed, which allows one to improve the gold recovery process. In connection with this, the use of high pressure grinding rolls in a crushing circuit is suggested by the authors.

The experience of HPGR application for heap leaching efficiency increase

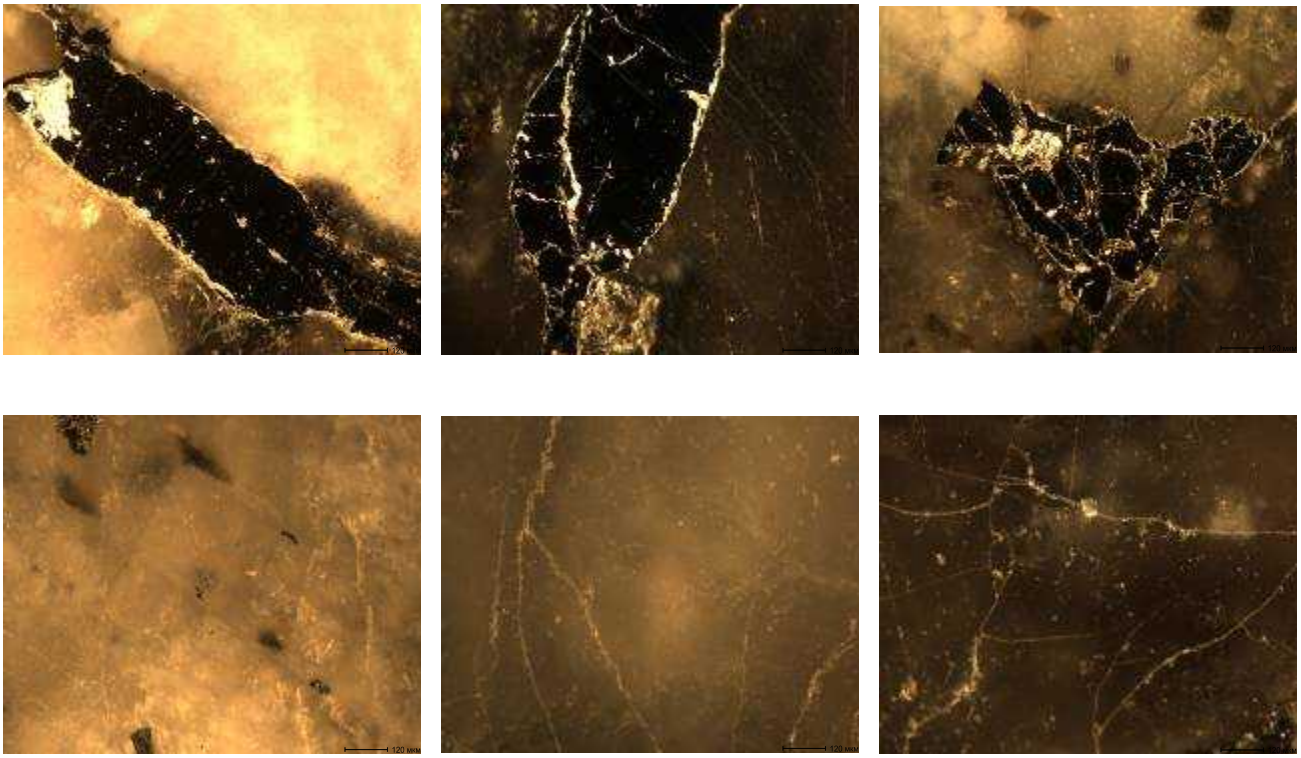
Globally there are about a dozen ore processing plants successfully applying High Pressure Grinding Rolls (HPGR). These include the "Sukhoy Log" gold processing plant in Russia. Another five large plants utilize HPGR:

- the gold processing plant Boddington in Australia, with a throughput of 35 mtpa (millions of tonnes per annum);
- the copper processing plant "Mikheevskaya" in Russia, with a throughput of 25 mtpa;
- the molybdenum plant in Ruby Creek in Canada, with a throughput of 7 mtpa;
- the copper plant Cerro Verde in Peru, with a throughput of 39.4 mtpa; and
- the copper-molybdenum plant in Spinifex Ridge in Australia, with a throughput of 20 mtpa.

The key factor that determines this HPGR usage (apart from significant energy efficiency) is a high capacity that can be compared with the throughput of the largest SAG mills. It should be noted that two plants operating in the Commonwealth of Independent States (CIS) are successfully using these units during the ore preparation stage: Vasilkovsky mining and processing plant and Nurkazgan processing plant, Kazakhstan.

In crushers of this type, a crushing method with maximum space filling between rolls is used; in this case material selective breakage and increase of crushing product fracturing takes place (see Figure 2). This enhances ore permeability, rate and metal recovery in leaching.

Ore breakage takes place due to material compression in the inter-particle layer; this is illustrated in Figure 2.



a b c

Figure 1: Images of crushed products:
a) ore after crushing in a jaw crusher;
b) ore crushed in a plant HPGR unit;
c) ore crushed in a laboratory unit simulating HPGR;
magnification 42×, in figure: above-sulphide inclusions, below-quartz

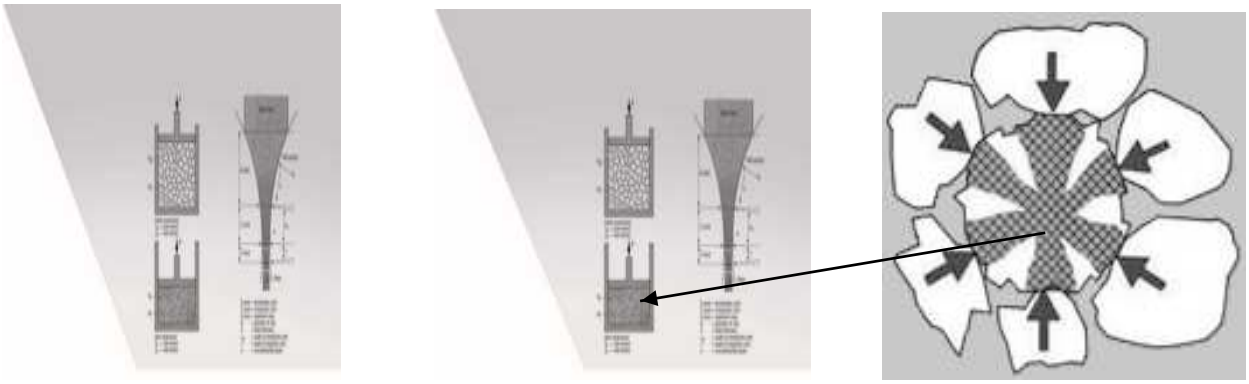


Figure 2: Material breakage mechanism in the inter-particle layer

The usage of these units enables a reduction in the consumption of the binding agent in ore pelletizing, and results in more stable pellets after drip irrigation with cyanide solutions. The free surface

of reaction increases with the size reduction of the material leached, and as a consequence ore material accessibility for cyanide and therefore gold recovery efficiency is enhanced. This increase is achieved by material crushing, which is carried out in one or several stages.

There are several main crushing methods in HPGR: crushing in one cycle; in several cycles; with further screening; and preliminary crushing prior to grinding in ball mills. Each method is used in different circuits of units depending on the technological properties of the minerals and the purposes of their further processing. In some cases it is necessary to crush the material down to a size of –2 mm, as the ores are resistant to percolation. It is difficult to make such an ore preparation circuit using conventional equipment. The usage of HPGR in this case has some indisputable advantages. First, these units (unlike conventional crushers) produce significant amounts of fine fractions (30% to 40% at –90 μm), in such a case it is possible to obtain a final fraction of –2 mm. Second, HPGR can allow one to completely or partially discard the third crushing stage; it increases gold recovery efficiency by 8% to 10%, due to an increase in fracturing and material permeability. Conventional crushing, even in up-to-date units, permits one to perform dry crushing to a minimum size of –7 mm, whereas HPGR breaks the ore down to size 80% –3 mm, and due to the fracturing and weakening of this material by high pressure, this size will correspond to –1 mm in comparison with conventional crushing.

Currently the TOMS Institute (Irkutsk) is carrying out laboratory research of processes in HPGR. For this purpose head ore samples before and after grinding in HPGR were collected from ore preparation sections of Mining and Processing plant (Kazakhstan, Vasilkovskoye deposit) and processing plant Nurkazgan (Kazakhstan, Nurkazgan deposit). Material composition of these samples was studied; particle size distribution and microscopic analyses were performed. Tests on head ore breakage simulation in HPGR were carried out. Prior to forwarding to HPGR, material representative samples were selected and subjected to breakage in a special unit simulating ore weakening in laboratory conditions. Pressure of breakage was equal to 10–15 H/mm^2, which corresponds to industrial performance. Then the ground samples were studied (particle size distribution and microscopic analyses) and were forwarded to the heap leaching unit. The tests on head ore were performed in order to compare the leaching results of these samples: head ore crushing by conventional methods (jaw crusher); and particle size distribution and microscopic analyses followed by heap leaching. From previous material composition analysis of head ore samples from these deposits it can be concluded that they are refractory to processing by heap leaching method.

Particle size distribution of the head ore, the ore crushed in a jaw crusher, the ore crushed in a plant HPGR unit and ore crushed in a laboratory unit simulating HPGR is presented in Table 1.

Table 1: Particle size distribution of the material under study

Size fraction, mm	Plant HPGR unit				Laboratory HPGR unit			
	Yield, %	Cumulative yield, %	Gold content, g/t	Gold distribu-tion, %	Yield, %	Cumu-lative yield, %	Gold content, g/t	Gold distribution, %
–40+30	16.55	100.00	1.81	12.88	–	–	–	–
–30+20	24.10	83.45	1.67	17.35	–	–	–	–
–20+15	14.71	59.35	2.75	17.44	–	–	–	–
–15+10	13.96	44.64	2.60	15.65	8.67	100.00	3.30	13.63
–10+8	5.16	30.68	3.20	7.12	10.51	91.33	1.54	7.70
–8+5	8.57	25.51	1.92	7.07	21.06	80.82	1.67	16.74
–5+2	7.41	16.95	1.90	6.06	21.08	59.76	1.67	16.76
–2+1	2.86	9.53	2.10	2.59	11.86	38.68	1.65	9.31
–1+0.5	1.61	6.68	2.80	1.94	8.41	26.83	2.50	10.02
–0.5+0.2	1.69	5.07	3.45	2.51	7.44	18.42	2.80	9.92
–0.2+0.1	0.64	3.38	3.45	0.96	4.16	10.97	2.50	4.95
–0.1+0.071	0.31	2.73	3.45	0.45	1.59	6.81	2.85	2.16
–0.071 +0.045	0.36	2.43	6.00	0.93	1.07	5.22	4.10	2.08
–0.045	2.07	2.07	7.90	7.04	4.15	4.15	3.40	6.72
–40+30	0.52	100.00	2.30	0.58	1.26	100.00	2.80	1.69
–30+20	3.02	99.48	0.72	1.06	4.43	98.74	2.15	4.56
–20+15	5.11	96.46	1.93	4.83	2.46	94.32	2.35	2.77
–15+10	6.62	91.35	2.20	7.13	4.47	91.86	1.31	2.80
–10+8	3.52	84.73	1.70	2.92	2.78	87.39	2.10	2.80
–8+5	7.96	81.21	2.08	8.11	11.66	84.61	1.84	10.26
–5+2	15.33	73.24	1.65	12.35	19.32	72.95	1.83	16.96
–2+1	8.81	57.92	1.83	7.90	14.18	53.62	1.90	12.92
–1+0.5	10.87	49.10	1.92	10.20	10.78	39.45	1.91	9.87
–0.5+0.2	11.24	38.23	2.30	12.66	12.66	28.67	2.30	13.97
–0.2+0.1	6.33	26.99	2.25	6.97	4.09	16.01	2.55	5.00
–0.1+0.071	2.58	20.67	2.20	2.78	1.66	11.92	2.40	1.91
–0.071 +0.045	3.08	18.08	2.25	3.39	2.04	10.27	2.90	2.84
–0.045	15.01	15.01	2.60	19.11	8.22	8.22	2.95	11.64
Total	**100.0**	**–**	**2.04**	**100.0**	**100.00**	**–**	**2.08**	**100.00**

The results of ore leaching in 1.5 m columns are presented in Table 2 and in Figure 3.

Table 2: Results of ore leaching in columns

Test No	Material	*Au* content, g/t		*Au* recovery to solution, %	Reagent consumption, kg/t of ore		
		in head*	in cake		NaCN		NaOH
					Pelletizing	Leaching	
1	Laboratory jaw crusher product	2.01	1.40	30.26	0.03	0.20	0.3
2		2.03	1.38	32.03	0.03	0.22	0.3
3	Plant HPGR unit	2.11	1.37	35.33	0.04	0.22	0.3
4		2.06	1.30	36.92	0.04	0.22	0.3
5	Laboratory HPGR unit	2.04	1.30	36.41	0.04	0.22	0.3
6		2.01	1.28	36.07	0.04	0.22	0.3

* Gold content in head ore samples was calculated based on the test results

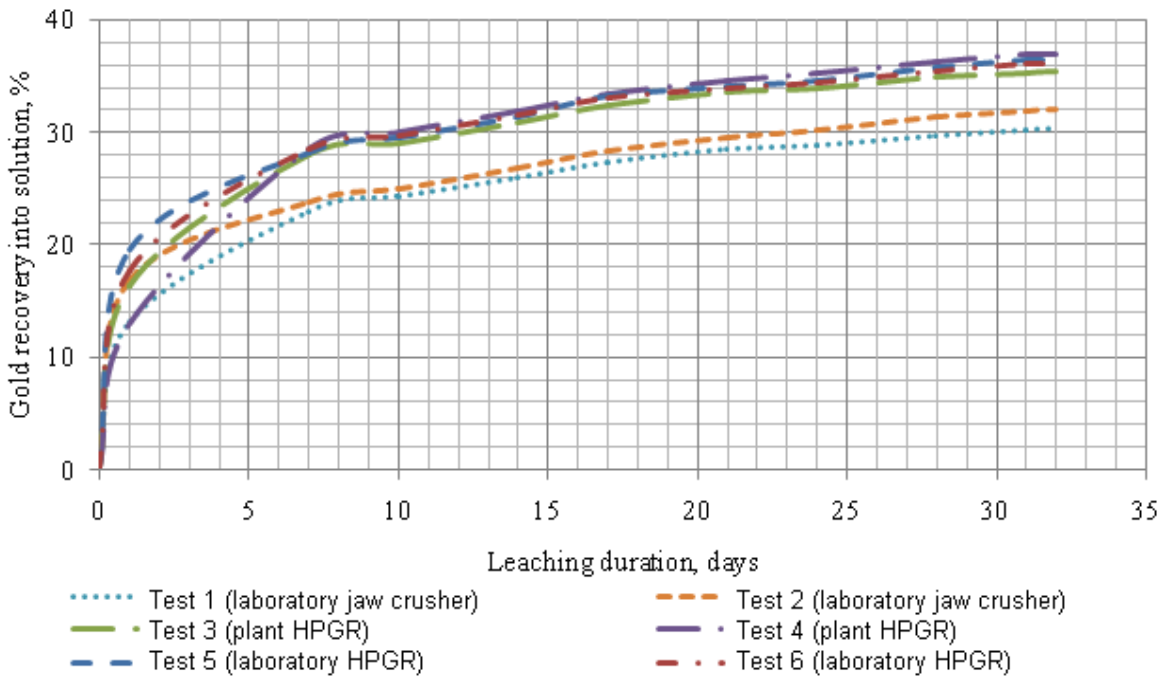


Figure 3: Gold recovery into solution versus leaching duration relationship

The leaching tests on ore crushed in different ways showed that gold recovery to solution in material leaching after grinding in HPGR on an industrial scale has the same level as material crushed in laboratory conditions simulating HPGR: 36.05% and 36.17% respectively. Gold recovery to solution from the sample of head ore crushed in a jaw crusher has an average value of 31.07%. The research results have shown that after the implementation of this method of material breakage the gold recovery in subsequent heap leaching increased by 5%. They also gave some indication of the accuracy of the breakage process simulation in laboratory conditions. As for the metal recovery level, the process is simulated accurately enough, but there are some discrepancies in particle size distribution. It is necessary

to take into account that the main aim in this study was to determine the difference between gold recovery from material prepared by different methods, and not the recovery itself.

In such a way breakage application in HPGR can permit one not only to reduce a number of crushing stages and decrease energy consumption in ore preparation, but also to determine ore processing technology selection in practice. The transfer from heap leaching technology to minerals of other types (change of oxidized ore for complex and primary one) in most cases necessitates construction of a new processing plant, or makes the project unprofitable. In some cases, using HPGR enables extension of the life cycle of the heap leaching unit, and even makes this technology applicable for all ore types in the deposit.

Conclusions

Gold recovery in heap leaching of gold containing ores is usually determined by solvent access to the surface of precious metal. Head ore is subjected to crushing in order to increase permeability; that is to say the surface for interaction between metal and the complexing agent is increased. Apart from the increase of surface in ore breakage by HPGR, fracturing of crushed material is increased, and that in its turn significantly enhances the permeability of the ore pile in heap leaching, due to extended material liberation. All of these factors will increase gold leaching kinetics and permit an increased overall recovery level. Increasing gold recovery from mineral ores by the heap leaching method by even a small amount will have a substantial economic effect, especially if it is combined with low costs for the installation and operation of HPGR (in comparison with traditional crushing and grinding units). Therefore, this technology can be considered to be highly efficient.

Gold-containing minerals laboratory testing applicable to the heap leaching process

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Abstract

This paper is about laboratory research into the processing properties of gold-containing minerals, with relevance to heap leaching technology. The information is based on worldwide gold-containing ore research experience analysis, and on the TOMS Institute's own research with gold-containing ores from different deposits in Russia. This paper shows that an overall study of the processing properties of the treated material is required before constructing a gold heap leaching industrial plant. Laboratory research is required to receive necessary input data for the subsequent industrial implementation of the relevant technology. The main stages of laboratory study are: material composition; physical and mechanical properties; hydro-physical properties; pelletizing parameters; searching study; and enlarged testing of heap leaching.

The material composition study stage includes the determination of chemical and mineral composition; grain size distribution; forms of gold occurrence; and its relation to other minerals. The main parameters required for ore preparation equipment and technological flowsheet selection before heap leaching are determined during a study of physical and mechanical properties.

In the course of a study of the hydro-physical properties three tests are suggested to ascertain if it is necessary to pelletize the material, as well as to determine the values of a maximum heap height and the required concentration of its irrigation with leaching solutions.

At the searching stage of the heap leaching process it is recommended to carry out tests for the sample agitation cyanidation and sample heap leaching in laboratory columns. Material agitation cyanidation is carried out to estimate the sorption activity of other ore components compared to gold, as well as to enable a preliminary prediction of the gold recovery level by heap leaching. Based on our research, it is recommended to carry out enlarged laboratory testing for heap leaching in a column with a

depth similar to an industrial heap. The result of the investigations is determining accurate data about the gold recovery level into solution, and reagent consumption for the process. After the research stage the authors recommend carrying out full-scale industrial testing of the developed technology, and performing deposit mapping in order to investigate the variability of ore processing properties.

In conclusion, the authors emphasize that only when a complex approach to mineral processing properties study is done, is it possible to minimize the risks connected with the impossibility of achieving the design technical and economic values, as well as to make long-term predictions for plant development and to use natural resources effectively.

Introduction

The process of heap cyanide leaching is one of the most common methods of gold recovery from minerals. The prevalence of heap leaching is due to several advantages of this method compared to some plant technologies for gold recovery. The most important of these advantages are the simplicity of technological flowsheet and instrumentation, as well as low capital and operational costs for the material treatment.

In spite of these advantages of heap leaching technology, this method of gold recovery is not versatile. The main limitations of this technology are: gold fine dissemination within enclosing rocks that are impermeable for leaching solutions; high content of clay fraction in the treated material; and sorption activity of the mineral components in respect to gold. These properties may cause failure to achieve the required technical and economic values for the mineral processing. In order to define and minimize the risks that lead to this failure it is necessary to carry out detailed laboratory research and pilot plant testing of the processing properties of ore before implementing heap leaching technology.

Over the past several years TOMS Institute has carried out laboratory research applicable to the heap leaching process for different kinds of minerals. For a period of time, the processing properties of over 50 samples of gold-containing minerals from different deposits of Russia were studied. As an input material to the heap leaching process different products were tested: oxidized, complex and primary ores; ores with high content of oxidized copper (up to 0.3%); conglomerates; gravity processing tailings; and oxidized roasted cinder from arsenic trioxide production. This paper contains the findings about the laboratory research of gold-containing minerals processing properties applicable to cyanide heap leaching. In addition, this paper describes the main determined relations of heap leaching to the chemical composition of minerals. A recommended procedure for heap leaching technology development is also included in this paper.

Research work

Analysis of worldwide gold-containing ores research and the TOMS Institute's own experience allows one to state that the development of reliable and effective heap leaching technology is possible only on the basis of an overall technological study of all aspects of ore treatment.

Laboratory research of gold-containing minerals processing properties can be divided into six major stages:

- material composition study;
- physical and mechanical properties study;
- hydro-physical properties study;
- pelletizing parameters study (if required);
- searching investigations for heap leaching; and
- enlarged testing of the heap leaching process.

Material composition study

Chemical and mineral composition, as well as the particle size distribution of the investigated sample are determined at this stage. Gold is studied (its mineral forms, grain size, grain shape, color and gold fineness); in addition, gold's association with other minerals is studied. The information gathered from material composition study helps to predict the processing properties of the investigated material and to make preliminary conclusions about the possibility of heap leaching technology implementation. It is also noted that if the content of sulfide minerals in the ore is 0.01% or lower, then gold heap leaching in solution is within the range of 70−96%. An increase of sulfide minerals content up to 4.5% leads to a decrease of gold leaching down to 45−80% for most of the studied ores. If the sulfides content in the ore is higher than 4.5%, this is an indicator that in the course of the heap leaching, gold passing into solution will likely not be higher than 45%.

Also the increase of gold leaching in solution is noted with the increase of the sulfide minerals oxidation level. But this behavior is less clearly seen, as rock-forming minerals can be the main gold bearers along with sulfide minerals, and the gold leaching level in these conditions will primarily depend on the permeability of gold-bearing rocks by solutions.

Physical and mechanical properties study

This study will help to select correctly the main crushing equipment and the most reasonable ore preparation flowsheet before heap leaching. This research includes a set of tests in order to get widely acknowledged mineral strength characteristics, such as Bond impact crushing work index, Bond abrasion index, unconfined compressive strength, bulk and specific weights, material angle of repose, and other

characteristics. Knowing the indicated physical and mechanical properties of the material allows one to select the most reasonable ore preparation flowsheet and equipment, and the dimensions of ore storages and parameters for leaching heaps (if the pelletizing stage is not applied). This knowledge also enables the determination of very important technological information about the heap leaching potential of the investigated material.

Low strength properties of the ore (unconfined compression test results up to 30 MPa) and low material abrasion (Bond abrasion index lower than 0.1 g) usually indicate that heap leaching will result in a high gold recovery level (75−96%). This relation is explained by the fact that ores with low strength properties and abrasion are commonly characterized by relatively low grain density and highly fractured rocks. These kinds of samples are easily permeable for leaching solutions, therefore high gold dissolution levels are achieved. For the same reasons it is usual for the ores with low bulk weight (up to 1.4 g/cm^3) to have high gold dissolution through heap leaching (about 80% and higher).

Hydro-physical properties study

After having studied material composition and the physical and mechanical properties of the ore, the main process research is carried out to determine its applicability for treatment with a heap leaching process. The hydro-physical properties of the material are studied at the first stage of the process research. The results of this research stage allow determining such issues as necessity for the material pelletizing, maximum concentration of the heap irrigation and its maximum height, as well as the material moisture values in the course of leaching and after solutions drainage. The study of the material hydro-physical properties should include three tests.

The first test is carried out under conditions of uniform wetting of the material within the heap. It is designed to determine the actual moisture values of the material in the course of leaching and upon the full completion of solutions drainage. Head subsample of the tested material is abundantly watered at concurrent mixing of the material until its moisture content is 3−5% higher than the material moisture content anticipated in the course of leaching. This anticipated moisture value can be obtained based on the material particle size distribution (PSD) and plots shown in Figure 1. The wetted material is placed into a column of 0.2−0.5 m height. The column diameter is to be at least three times larger than the size of the material maximum particle. The material watering in the column lasts until the total stabilization of the solutions percolation process (48 hours of watering is usually sufficient). Upon stabilization the watering is stopped. Next the solutions are fully drained. Based on the drained solutions amount and the material residual moisture the actual values of moisture content can be determined for the material during leaching and after the full solutions drainage.

Correlation of the obtained data with the ore strength properties study results (Figure 2) indicates that with decrease of the ore compression strength the material capillary moisture increases. This function shows that decrease of the ore strength leads to more intensive fracturing of the material and as a consequence to an increase in its permeability.

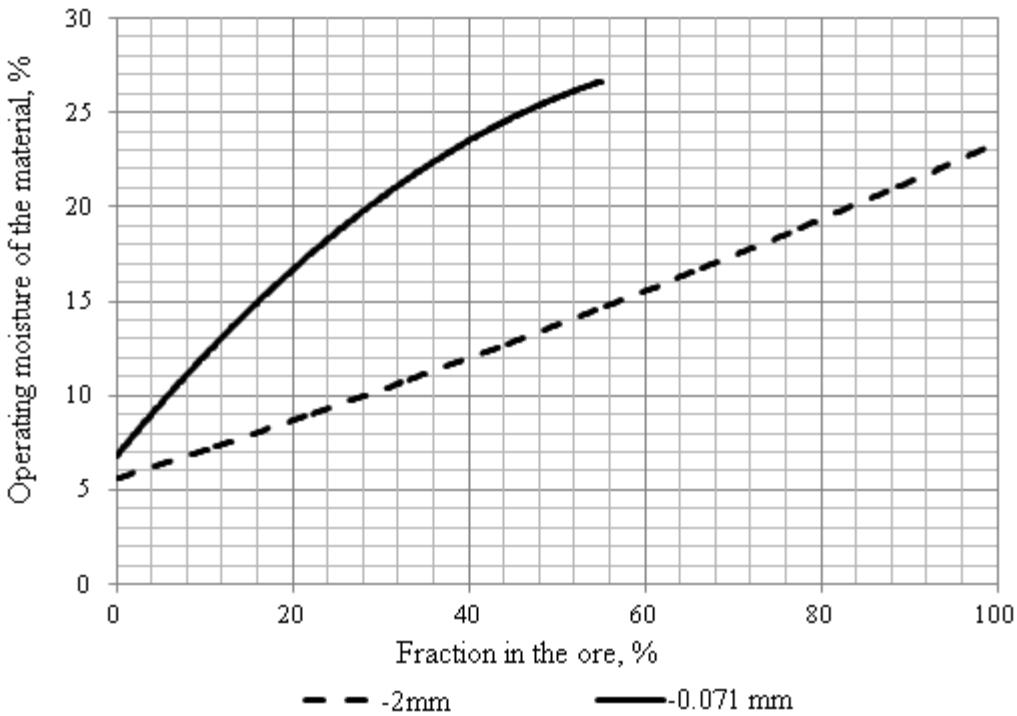

Figure 1: Anticipated operating moisture as a function of fraction content in the ore

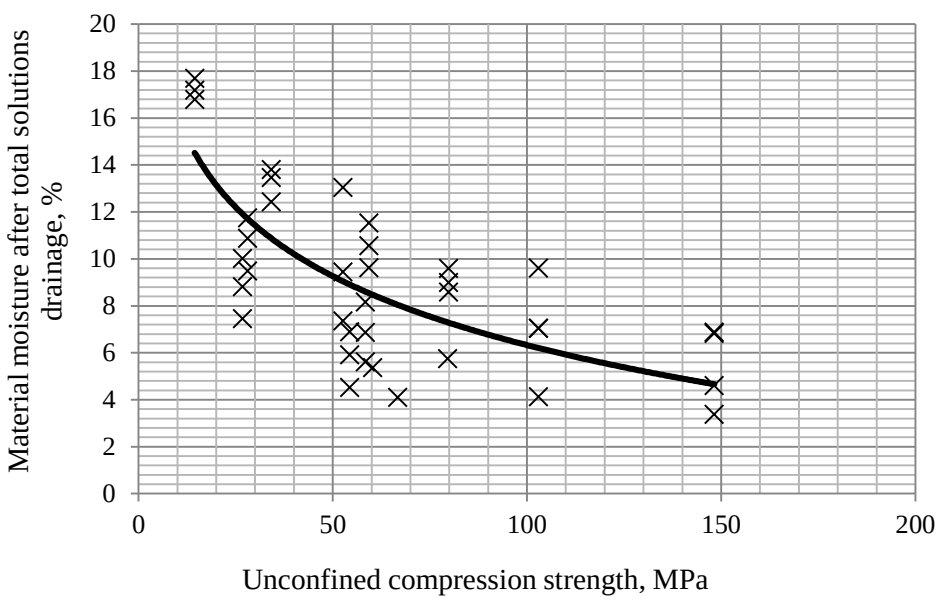

Figure 2: The material moisture after solutions drainage as a function of the ore strength properties

The second test serves to estimate hydro-physical properties of the material under conditions simulating an actual heap leaching process. The ore subsample having natural moisture content is placed in a column of 1.5−2.0 m height and with a diameter at least three times larger than the material maximum particle size. Then the material in the column is watered at different concentrations within 100 – 1000 l/ (m^2*days) range. The material moisture values are determined for every water concentration level in the course of leaching and after the total solutions drainage. The obtained values are compared with the actual moisture values (as determined during the previous test). The material is assumed to be evenly permeable for solutions if there is a difference between the obtained moisture values and the actual moisture values do not exceed 0.5−1.0%. If at irrigation concentration less than 500 l/ (m^2*days) the uniform solutions percolation through the material fails, then pelletizing of the material before its heap leaching is required. Otherwise, the necessity of pelletizing is determined according to the third test results.

The third test determines solutions percolation through the material layer in a compression device. Such a compression device represents a laboratory compression column. The top part of the column is equipped with a special mechanism imposing load on the material and simulating different height levels of the heap. The testing should determine the maximum heap height and concentration of the material irrigation which provides maintenance of uniform solutions percolation through the ore layer. For this purpose load on the material and rate of water flow in the column were changed gradually. In addition, the material moisture value was measured for every mode in the course of irrigation.

If the test results do not allow obtaining acceptable heap height values, or the material irrigation concentration values, this material must be pelletized before it is stacking into a pile. Moreover, experience obtained during the compression test performance enables one to establish dependency between maximum acceptable heap height and content of –2 mm and –0.071 mm fractions in head material (see Figure 3).

Data presented in Figure 3 indicate that if –0.071 mm and –2 mm fractions content in the ore exceeds 5.2% and 26% values respectively then the maximum acceptable height of heap leaching pile will not be higher than 10 m. It is recommended to pelletize these ores before stacking them into a pile. Maximum acceptable content of –0.071 mm and –2 mm fractions when the ore pile of 10 m height can still be formed without the material pelletizing was 8.9% and 37.2% respectively.

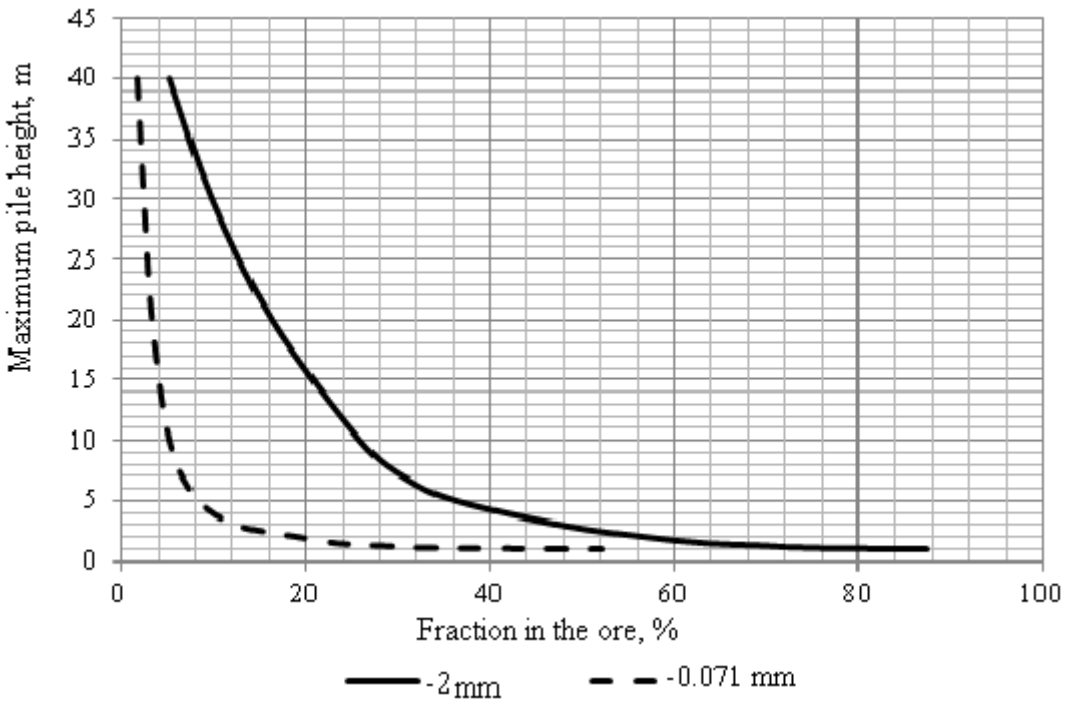

Figure 3: Maximum acceptable heap height as an average function of separate fractions content in the material

Pelletizing parameters study

If ore pelletizing is required before stacking it in piles, testing should be performed to choose the optimal process parameters such as pelletizing time, material moisture, binder and alkalizing reagents addition.

The Au leaching process is carried out using sodium cyanide solution in order to prevent this solution being hydrolyzed with toxic hydrocyanic acid generation. As a result it is necessary to keep the pH level of the leaching solution above 10.5. To do this different alkalizing agents (CaO is one of the most common) are added to the ore before pelletizing. The optimal lime addition is determined at the first stage of the pelletizing process study in the following manner. The ore subsamples are mixed with different lime amounts and placed in laboratory columns. The material is watered from above and then the pH levels of the solutions are measured as they leave the columns. The optimal lime addition is selected based on a stable (more than 10.5) pH level of draining solutions. Usually lime addition at 2.0 kg/t rate is sufficient to obtain the required pH level of the solutions. The maximum lime addition fixed in the course of the testing at pelletizing stage was 80 kg/t. This value was obtained from the study of pelletizing parameters of old Au-containing roasting cinder from arsenic production.

At the second stage of the pelletizing parameters study the optimal material moisture and pelletizing time are selected. These parameters are determined visually upon the full formation of stable firm pellets of the required size in a pelletizer. The material optimal moisture and pelletizing time depend

on the feed PSD, the amount of the coarse fraction providing pellets formation centers, and the fine fraction amount which is to be pelletized. Usually the optimal pelletizing time is within the 0.5–3.0 minutes range. Figure 4 shows the material optimal moisture at pelletizing stage as an average function of the –2 mm and –0.071 mm fractions content in the ore.

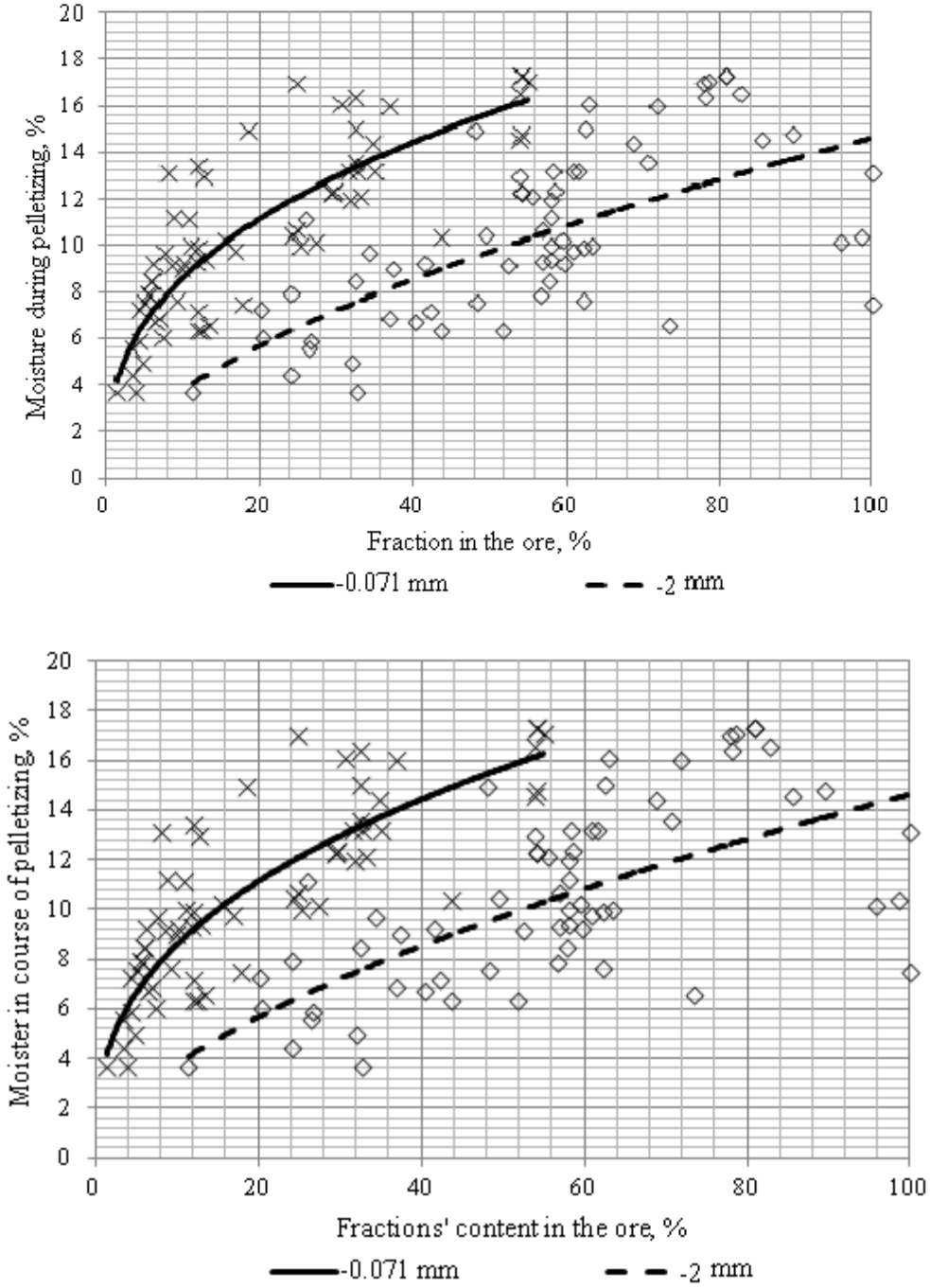

Figure 4: The material optimal moisture at pelletizing as an average function of –2 mm and –0.071 mm fractions content in the ore

The concluding stage of the pelletizing process study presupposes the selection of an optimal addition of binder providing the required mechanical strength of the pellets. Portland cement is usually used as a binder. The material subsamples are pelletized in the optimal mode at different binder additions. Then various methods are used to evaluate the mechanical strength of the pellets. We believe that the most reliable results of the pellet strength evaluation are obtained with the simultaneous use of the following five methods:

1. Test to evaluate the mechanical strength of the pelletized material during handling. Some pellets are selected from the subsamples of freshly pelletized material and their strength is determined by a free fall method from a height of 1 m on the conveyer belt surface. It is assumed that the pellets have the required strength if they do not break.
2. Unconfined compressive strength test. This and the following tests are performed on the pelletized material retained for no less than 72 hours so that the pellets solidify. Some pellets are selected from the subsamples of pelletized material, then their sizes are measured and the unconfined compression strength is determined. The obtained strength limit values for each pellet are applied to calculate the maximum pile height which the heap can bear without collapsing under the material weight. The test is assumed as successfully done if the obtained value of the maximum pile height is higher than the height required for the project under development.
3. Immersion test. The pelletized material is placed on a laboratory sieve. Then the sieve is slowly submerged in a 2/3 water-filled vessel until water has fully covered the material surface. Once the material is covered it is quickly taken out of the vessel. This operation is repeated several times. The mass reduction on the sieve allows evaluation of the strength of the pellets, and enables conclusions on the applicability of heap leaching for treatment of this pelletized material.
4. Permeability test. Pelletized material subsamples are loaded in test columns. Then the columns are filled with water from their bottom parts up to 50 mm higher than the material level. Upon reaching this level the water supply is stopped, and the material is kept in this condition for two hours. After this, in order to line the material depth at a stable level, the column sides are tapped. Then, the drainage hole on the bottom of the column is opened and the solution drainage rate is determined while keeping the water in the column at a constant level. In accordance with the test findings the value of the material shrinkage in the column and the rate of water percolation through the pelletized material are calculated, and a conclusion is made about the implementation of heap leaching to process the pelletized material.

The above -mentioned methods of evaluating pellet strength do not take a lot of time and therefore they are used to select an optimal binder addition at the pelletizing stage. After that the 5th test is carried out in the determined optimal pelletizing mode to evaluate the maximum possible height of the heap and concentration of the material irrigation.

5. Test in a compression device. This test is similar to the compression test for the study of hydro-physical properties of the head material.

Preliminary study on heap leaching

In order to estimate the level of gold dissolution the research on cyanide leaching is carried out. Gold cyanide leaching from the product under investigation is carried out in two stages. Agitation leaching of material ground to different sizes is carried out at the first stage. Within a short period of time this research allows one to receive a preliminary estimation of the level of gold recovery into solution and the suitability of the material to the process of heap leaching. At the second stage of research the tests for cyanide leaching are carried out, simulating the process of heap leaching. The second stage of research is carried out as follows: the crushed ore is loaded to the column and is irrigated with sodium cyanide solution. In the process of solution penetration through the layer of ore the process of gold dissolution occurs; the drainage solutions are collected and analyzed for gold content; according to the amount of gold transferred into solution, the suitability of material to the process of heap leaching is assessed. In order to perform the second stage of research, the results of hydro-physical properties study and pelletizing test results are required. If pelletization of material is required, the tests for cyanidation of material in column are carried out using pelletized material.

It is recommended that agitated cyanide leaching testing should be carried out on the material of at least two fractions: –2 mm and 95% –0.071 mm. Moreover, for 95% –0.071 mm fraction cyanidation is to be performed both without synthetic sorbents and in CIL/RIL (carbon-in-leach/resin-in-leach) mode. Comparison of the agitated cyanidation results obtained without synthetic sorbents and in CIL/RIL mode allows conclusions about the material sorption activity towards gold. Au recovery difference between 95% –0.071 mm and –2 mm fractions provides a possibility to predict Au recovery level by heap leaching.

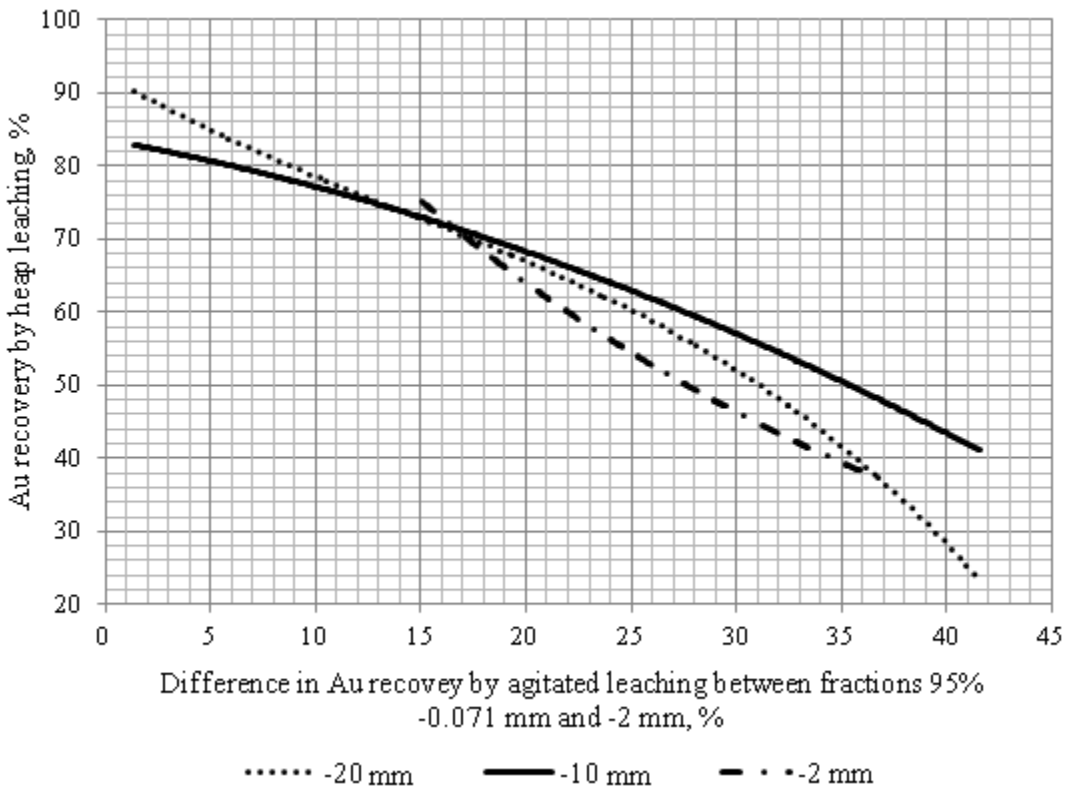

Figure 5: Average dependence of Au recovery by heap leaching on decrease of Au recovery by agitated leaching in the course of material size increase from 95% minus 0.071 mm up to minus 2 mm

It is more feasible to conduct a searching study on the ore heap leaching in columns of 1.5–2.0 m depth and with a diameter three times larger than the material maximum particle size. The searching study stage includes selection of optimal values of material size and cyanide concentration. Head ore subsamples are loaded in columns and irrigated with sodium cyanide leaching solution. Drained solutions are collected from the column bottom part, and Au and reagents concentrations are determined in the solutions. Upon the process completion (almost complete cessation of Au dissolution) the leaching solution supply is stopped and the complete drainage of solutions is awaited. After that the material is unloaded from the column and dried, and the residual Au content is determined. The obtained data allows calculation of the Au recovery level and reagents consumption, and also indicates the optimal heap leaching mode for the particular ore.

Column leach testing

Enlarged laboratory testing of the heap leaching process is performed basing on the searching study results. The enlarged heap leaching tests are carried out in a column with a height close to a commercial ore pile height, using solution recycle at leaching stage. Cementation or sorption methods are used to

extract gold from drained solutions. Demetallized solutions are replenished with reagents and returned into the leaching process. Experience has shown that the Au recovery rate in the course of enlarged testing is often very close to the Au recovery value obtained at the preliminary study stage. However, in some cases there is an observed Au recovery decrease of 4.5−8% versus the preliminary study stage results.

Upon completion of the all of the testing regarding applicability of heap leaching to the minerals processing, it is recommended to test this process on a commercial scale.

Moreover, it should be taken into account that minerals material composition and processing properties can vary widely within one deposit. As a typical practical example, the dependence of the minerals processing properties on the ore occurrence depth can be noticed. Thus, passing from the surface down to deeper horizons it is often observed that sulfides content increases, their oxidation level decreases, and the ore strength properties improve. The ore lithological composition also undergoes changes resulting in reduction of fine fractions yield. All these factors lead to decrease of the material porosity and fracturing. As a consequence of this the proportion of gold accessible for contact with leaching solution in the ore reduces and a decrease of Au passing into solution is observed. Variability of processing properties of ores within a deposit should be studied at a searching stage in order to avoid these negative impacts in the course of material industrial treatment. Samples for the ore processing properties variability study should be collected along the whole length, width and depth of ore occurrence. The variability study will allow one to predict the ore processing efficiency by heap leaching at different stages of the deposit development, as well as to correct the process flowsheet and parameters of equipment operation in the case of minerals properties change.

Comprehensive laboratory study, deposit mapping in terms of the minerals processing properties, and carrying out of full-scale commercial testing are integral parts of the successful commercial introduction of an Au heap leaching process. This integrated approach to minerals study before its implementation in industrial processing will allow minimizing risks associated with failure to achieve the designed technical and economic performance, as well as optimizing the mode of ore preparation and material leaching. Process-oriented mapping of the deposit enables reliable planning for long-term deposit development and plant progress.

Conclusions

Only an integrated approach to minerals processing properties study enables successful commercial introduction of Au heap leaching process. This approach includes laboratory testing, study of minerals processing properties variability within a deposit, and full-scale commercial testing of the developed technology.

Laboratory testing of Au-containing minerals' processing properties should include the following stages:

- Material composition study. This stage allows one to obtain general information on processed minerals and make a preliminary prediction about the efficiency of heap leaching technology for the particular minerals treatment.
- Physical and mechanical properties study. All basic parameters necessary to select the equipment and prepare the ore preparation flowsheet are determined at this stage.
- Hydro-physical properties study. This study enables one to evaluate the need for material pelletizing before it is stacked in a pile, as well as to determine the maximum heap height and concentration of its irrigation with leaching solutions.
- Pelletizing parameters study. This study stage is performed when the material in question has unfavorable hydro-physical properties. In the course of this testing basic process parameters of pelletizing (material moisture, pelletizing time, type and load of binder) as well as maximum heap height and irrigation concentration values are determined.
- Searching study on heap leaching. Results of a searching study of heap leaching serve as a basis for the selection of optimal material size and leaching reagent mode. They also determine preliminary values of Au passing in solution and reagents consumption.
- Enlarged testing of heap leaching process. This testing is aimed at verifying results obtained at the searching study stage. The enlarged testing provides all initial data required to carry out full-scale commercial testwork.

Comprehensive laboratory testwork along with process-oriented mapping of a deposit and pilot testing can minimize risks associated with failure to achieve designed technical and economic performance, as well as facilitate long-term prediction of plant progress and the efficient use of natural resources.

Mobile heap leach stacking conveyor technology – higher capacity applications and adapting to IPCC and waste handling

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Abstract

Mobile stacking conveyor systems have been successfully used in heap leach operations for copper, gold, and, relatively recently, uranium. For fixed-pad, multiple-lift operations, portable conveyor or "grasshopper" type mobile stacking conveyor systems are commonly employed for throughputs up to approximately 3,000 tonnes/hour (t/h), using conveyors ranging from 750 to 1,200 mm wide. As throughput increases, the frequency of portable conveyor movement increases as the stacking system must retreat or advance across the heap leach pad. Furthermore, higher capacity requires wider conveyor belts and heavier mechanical and electrical components on-board the portable conveyors, increasing their size and weight. Traditional grasshopper conveyors, typically 30 to 40 m long, are normally moved and positioned on the leach pad using forklifts or loaders. Advances in portable conveyor technology have now enabled operations to achieve throughputs nearing 10,000 t/h, with 1,400 to 1,800 mm wide belts. This is achieved by replacing the traditional non-propelled grasshopper conveyor chain with a series of self-propelled (via crawler tracks) mobile conveyors, each typically twice as long as a traditional grasshopper. The longer length allows the conveyor to remain operating in a given position for a longer duration before shutting down to index (retreat or advance) the stacking system, which effectively increases system availability. The self-propulsion design enables the larger, heavier mobile conveyors to be positioned without the need for ancillary mobile equipment and associated labor and operating costs. The design features and operating principles of this technology, along with current operating examples, will be presented.

Mobile conveyor technology pioneered for heap leach applications is being applied to waste handling and IPCC (in-pit crushing and conveying). The increasing costs of operating and maintaining truck fleets are making mobile conveyor systems more attractive. In a mobile heap leach stacking conveyor system, a chain of portable conveyors delivers ore to an indexing conveyor unit and a stacker. As the heap is stacked, the stacking system retreats or advances (depending on application). Using the

same principles, a portable conveyor system can convey and stack ripios (leached ore), waste or overburden to a waste dump, or, in the case of IPCC, the system would convey ROM (run-of-mine) or primary crushed ore or waste away from the mining face to a stockpile or waste dump. The flexibility of self-propelled mobile conveyors in a pit or bench operation allows the conveyors to be easily positioned to suit varying terrain and mining conditions. System concepts, current developments and operating examples will be presented.

The focus of this paper is the grasshopper type system, in particular the advancements in its technology and flexibility for higher throughput operations, as well as its adaption to complementary mining operations such as IPCC and waste handling.

Introduction to mobile heap leach stacking systems

Mobile belt conveyors have been used for the transport and placement of ore in heap leach operations worldwide for some 30 years. Copper and gold comprise the majority of heap leach operations, with uranium leaching recently becoming technically and economically viable. Mobile stacking conveyor systems have been successfully implemented for both permanent, multiple-lift heap leach pads, as well as dynamic or "on-off" leach pads.

A multiple-lift heap leach pad involves the placement of ore in several layers, or lifts, with each lift being leached with acid solution prior to placement of the next lift. The heap leach pad is stacked in several lifts until its maximum height is reached, normally determined by mine ore volume, heap leach pad geotechnical stability and/or permitted height. A dynamic − or "on-off" − heap leach pad involves the placement of ore in one lift over the footprint of the pad. It is then leached with acid solution and subsequently reclaimed from the pad, then transported and deposited in a spent ore pad, commonly called a ripios dump. Placement of new ore follows the reclamation of the spent ore, and the cycle repeats.

Mobile conveyor systems can be used for both applications.

Typical system configurations

Typical mobile conveyor systems for heap leach applications can generally be grouped into one of two configurations:

1. A series of cascading mobile conveyors or "grasshopper" type conveyors with a radial stacker, in which the grasshopper conveyors are intermittently re-arranged to stack ore on a leach pad evolving in both footprint and height.
2. A "bridge" conveyor with spreader, in which a mobile conveyor, usually spanning the leach pad width, traverses and stacks across the length of the leach pad. While both types of systems can be configured to operate in multiple-lift or dynamic pad operations, the majority of

multiple-lift leach pad operations tend to utilize grasshopper type systems, while many dynamic leach pad operations utilize the bridge configuration.

Mobile conveyor ("grasshopper") and stacker systems

These systems normally include a conveying system to deliver ore (normally crushed to 6 to 25 mm) from the plant to the leach pad, where it is transferred to a fleet of cascading mobile conveyors, or "grasshoppers", which feed an indexing conveyor and radial stacker. The stacker places the ore on the leach pad in a semi-circular arc as the stacker pivots at the tail end and moves in a radial motion (as shown in Figure 1).

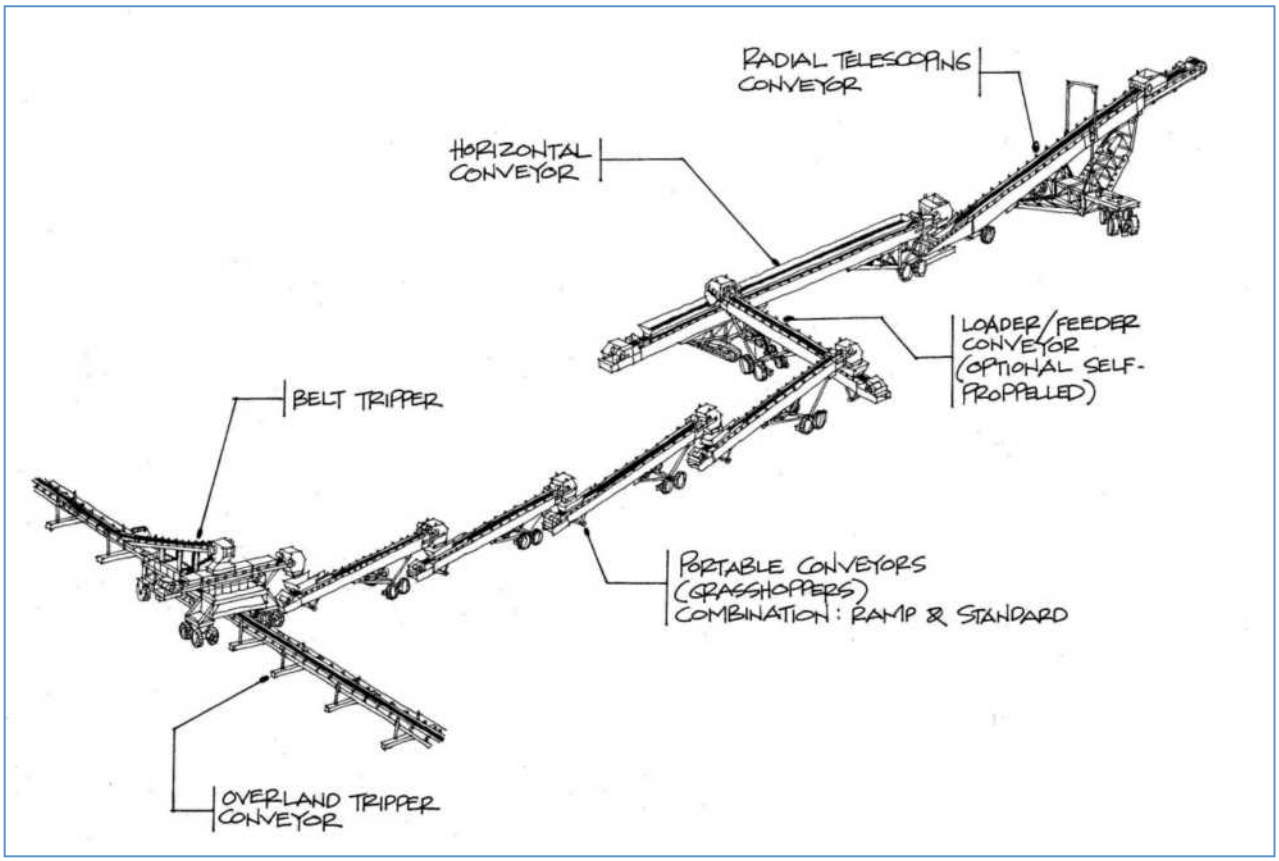

Figure 1: Grasshopper type mobile conveyor stacking system (©Terra Nova Technologies, Inc.)

With each radial pass, the stacker discharge will need to either move backward (for "retreat" stacking) or forward (for "advance" stacking) in order to maintain a constant stack height. This is commonly achieved with a combination of a telescoping discharge chute on the stacker that provides a retreat/advance range of approximately 6 to 8 m, along with an indexing conveyor unit upstream of the

stacker which allows the stacker to retreat or advance without having to shut the system down to remove (in retreat stacking) or insert (in advance stacking) a grasshopper (see Figure 2).

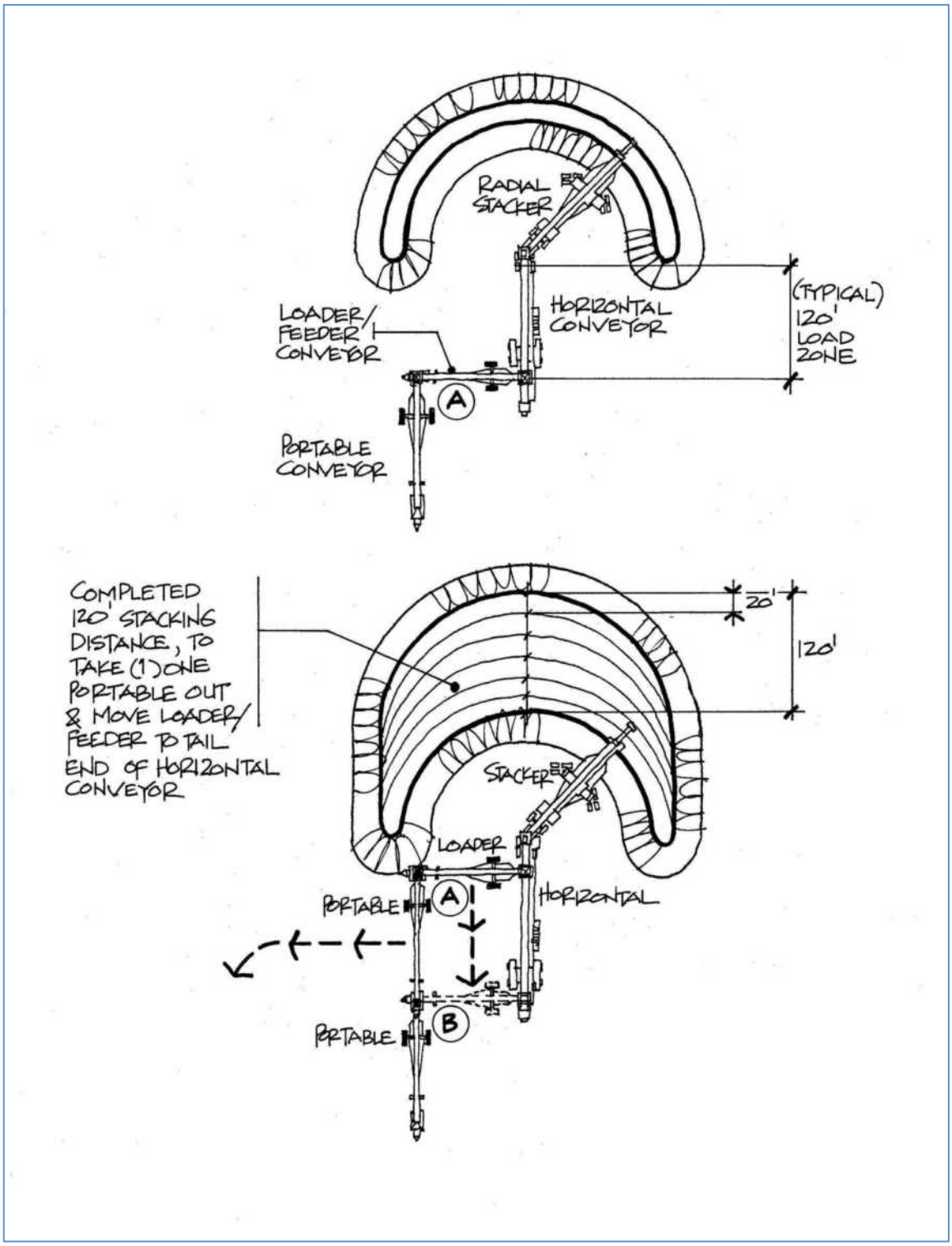

Figure 2: Radial stacker and indexing operation (©Terra Nova Technologies, Inc.)

The line of grasshoppers and stacker continues to stack a "cell" along the width of the leach pad, with the cell width normally defined by approximately twice the stacker length, effectively the radius of the cell arc.

Once a cell is completed, the mobile conveyor system is repositioned in the adjacent cell and the process repeats until the entire lift is completed. The length and number of cells per lift is determined by the heap leach pad footprint and cell width. For multiple-lift leach pad applications, once a lift is complete, the mobile conveyor system is repositioned to the first cell of the next lift, on top of the prior lift which has, by that time, been sufficiently leached. In dynamic leach pad applications, the system would be repositioned at the start of the leach pad, which by that time would have been reclaimed and the leach pad area would be ready to receive new ore.

Grasshopper type stacking systems are manufactured by several companies, including Terra Nova Technologies, Inc., FLSmidth, Senet and Superior.

Bridge and spreader systems

These systems typically consist of a single mobile conveyor unit, or "bridge", as wide as the leach pad, self-propelled via several crawler-type tracks and traversing the entire length of a leach pad. The bridge is oriented perpendicular to the direction of travel. Fed by a mobile tripper running the length of the leach pad along an overland conveyor, the bridge incorporates a linear-traveling spreader boom on the bridge which places the ore behind the bridge, along its full length (as shown in Figure 3). Once the spreader completes stacking along the full length of the bridge, the bridge advances forward and the spreader repeats the cycle. Once the leach pad is stacked, the bridge would be repositioned at the start of the leach pad, which by that time would have been reclaimed and the leach pad area would be ready to receive new ore. In multiple-lift applications, the bridge would then be relocated to the top of the lift for stacking the next left. As the leach pad width decreases with each lift (due to the pad side slope), the bridge unit is shortened by removing sections.

Figure 3: Bridge and spreader (© FLSmidth)

Bridge and spreader type systems are manufactured by several companies, including FLSmidth, FAM and Takraf.

Increasing capacity of portable conveyor and stacker systems

Traditional mobile conveyor or "grasshopper" type mobile stacking conveyor systems are commonly employed for throughputs from about 200 t/h to 3,000 t/h, using conveyors ranging from 750 to 1,200 mm wide. Each grasshopper conveyor, typically 30 to 40 m long, consists of a belt conveyor mounted on a steel truss structure supported on wheels, and is moved and positioned on the leach pad using forklifts or loaders, as illustrated in Figure 4. The radial stacker and indexing conveyors, however, are normally self-propelled via on-board diesel generators.

Figure 4: Typical 38 m long "grasshopper" conveyors

As throughput increases for a given stack height (which is normally a function of metallurgical recovery), the rate of retreat or advance of the radial stacker will increase for a given set of stacking system parameters: cell-width, grasshopper length, index conveyor length. If this rate increases, the frequency of system shutdown will also increase, in order to remove (for retreat stacking) or insert (for advance stacking) grasshoppers, which clearly will adversely affect system availability.

It follows then that if the retreat/advance rate can be slowed, higher throughputs can be achieved with acceptable system availability.

Stacker length and cell width/height

Clearly, the longer the stacker − which determines the cell arc radius − the wider the cell. At a given throughput, stack height and stacker slewing rate, a wider cell will require more time to stack, which effectively slows the stacker retreat/advance rate. A longer stacker, however, is heavier and must be supported on a slewing undercarriage which carries and propels the majority of the stacker weight. Excessive ground bearing pressure from mobile equipment on leach pads can be detrimental to the ore and can cause mobility problems, especially if the ore has high moisture content from either acid or rain. In addition, the stacker must be of a manageable length to allow easy movement on and around the leach pad. Finally, a longer stacker has a higher capital cost than a shorter one for the same throughput. It is therefore necessary to evaluate the capital and operating costs in order to determine the optimal radial stacker length for a heap leach application. Generally, all other parameters being equal, a longer stacker can prove beneficial over the life of the operation.

Many radial stackers in the 200 to 3,000 t/h range are 30 to 45 m long. Advancements in computer modeling of radial stacker structures (to increase strength/weight ratio), undercarriage track or wheel design, and propulsion systems have made longer stackers (e.g. 50 to 70 m) technically viable in terms of ground bearing pressure and operability. Several stackers of this size are in operation worldwide at heap leach facilities, including Cerro Verde (Peru), El Abra (Chile), Cerro Colorado (Chile) and Safford (USA). These longer stackers significantly increase cell width and, along with other system features, enable them to achieve throughputs from 2,000 to 10,000 t/h. Even with lower throughputs (e.g. 2,000 t/h), the longer (and thus wider) cell will increase system efficiency. See Figure 5.

Figure 5: 64 m long radial stacker at Cerro Verde (Peru)

Mobile conveyor length and index conveyor length

Traditional grasshoppers of 200 to 3,000 t/h capacity range in length from 30 to 45 m. Similar to the cell-width argument for the radial stacker length, a longer grasshopper also has benefits:

1. fewer units required for a given heap leach pad size; and
2. less frequent shutdowns to remove/insert a unit (provided that the index conveyor length is matched to the grasshopper length).

Most grasshopper type mobile stacking systems have an index conveyor as long as one grasshopper, which matches the indexing distance with the removal or insertion of a grasshopper. With all other parameters equal, it is possible to decrease by half the frequency of system shutdowns for removal or insertion by doubling the length of the index conveyor, so that upon completion of a complete index, two grasshoppers are removed or inserted instead of one. This type of system has been in operation at Cerro Verde since 2005, which now uses a total of 39 grasshoppers, each 38 m long, and a 76 m long index conveyor. One of the largest grasshopper systems in the world in terms of grasshopper quantity, this operation achieves relatively high utilization with the double-length indexing conveyor.

Mobile conveyor length and self-propulsion

Traditional grasshopper units are usually positioned on the leach pad with forklifts, loaders or dozers, as they are not self-propelled. Traditional grasshoppers of 30 to 45 m length have become common in industry due to their practical size and weight which make them relatively easy and flexible to position around a leach pad.

To achieve higher throughputs, it becomes necessary to consider significantly longer and larger grasshopper units due to the conveyor system equipment requirements (i.e. belt width, drive power, electrical equipment, etc.). In addition, the length and on-board equipment combine to increase weight to a point at which self-propulsion becomes necessary for more efficient and reliable operation.

Advancements in portable conveyor technology have yielded significantly greater throughputs by increasing length and adding self-propulsion. With the main conveyor structure supported on crawler tracks at both ends, longer and heavier conveyors can be accommodated while maintaining acceptable ground bearing pressure and maneuverability (see Figure 6).

Figure 6: 76 m long self-propelled portable conveyor, 1,800 mm belt width, 8,500 t/h

Technological advancements in mechanical, structural, electrical and control engineering and equipment design have made much higher throughputs possible on these mobile conveyors. Advanced structural computer modeling enables relatively light-weight but strong and resilient structures, suitable for crawler-track type propulsion over undulating terrain while supporting high capacity conveyor systems. Advanced hydraulics systems are used for several operating functions, including self-leveling of the conveyor structure to allow for operation on side-slopes, automated conveyor belt tensioning, and luffing of head and tail end structures to provide accurate positioning capability of conveyors relative to each other. Complete on-board electrical power distribution systems, including diesel generators for self-propulsion, provide the ability to safely power the units with high voltage (e.g. 13 kV, 23 kV), thereby allowing multiple interconnected units to operate with an acceptable voltage drop. Variable-frequency-drives (VFDs) are used to control the crawler track units for ease of mobility and positioning on the leach pad. Finally, advanced on-board PLCs are used to monitor and control all conveyor and propulsion functions, while interfacing with upstream and downstream units.

This advanced mobile conveyor design has been used to achieve significantly higher throughputs at several copper leach operations.

- El Abra (Chile): 23 units, 76 m long, 1,800 mm belt width, 8,600 t/h (copper leach. See Figure 7.

Figure 7: Self-propelled portable conveyor type heap leach stacking system at El Abra (Chile), 8,600 t/h

- Safford (USA): 10 units, 76 m long, 1,800 mm belt width, 6,500 t/h (copper leach). See Figure 8.

Figure 8: Self-propelled portable conveyor type heap leach stacking system at Safford (USA), 6,500 t/h

- Morenci (USA): 5 units, 76 m long, 1,400 mm and 1,800 mm belt widths, 3,000 and 5,000 t/h (copper leach).

Adaptation of mobile conveyor and stacking technology to overburden, waste and IPCC applications

As the cost of transporting material – overburden, ore, waste – by truck increases with the cost of oil and labor, along with associated emissions, mobile conveyors offer an increasingly attractive alternative. In addition, use restrictions or availability of water increasingly necessitate filtered or "dry-stack" tailings operations at several new and expanding mines in lieu of typical slurry deposition systems. Of substantial industry implication, within the last decade, the use of mobile conveyors for IPCC technology has been studied extensively by the world's largest mining companies and system manufacturers. The authors' experience in this industry suggests extensive pilot scale testing within the next two to three years and full-scale implementation (i.e. 100% mobile conveying and not a hybrid truck-and-conveying system) within 8 to10 years.

While the initial capital costs of mobile conveying systems are typically higher than that of a truck fleet, the long-term operating costs and thus NPV of the material handling operation can be significantly lower, depending on the operation (i.e. pit/bench configuration, electric power cost, mining plan, transport

distances, etc.). When considering such tradeoff studies, it is necessary to consider all aspects of the material handling operation to accurately compare the costs. For example, the estimated cost of a truck fleet must also include the truck maintenance shop and its labor requirements, road building and maintenance, training, safety, and an NPV (Net Present Value) analysis must also consider a sensitivity analysis of labor and oil prices, as the latter directly influence the cost of fuel and tires. Similarly, the cost of a conveyor system must include capital spares (e.g. belting, drives, etc.) and sensitivity analysis of electric power costs.

A combination of fixed, shiftable and mobile conveyors and spreaders have been used to transport and deposit overburden, waste and ripios for decades. Within the last five years, portable conveying and stacking systems have been studied extensively, and recently implemented, in filtered tailings, waste handling and IPCC applications.

Filtered tailings applications

The deposition of filtered tailings with mobile conveyors is often made challenging by the physical properties of the tailings material, namely its bearing strength once placed and moisture content. As in a multiple-lift heap leach operation, the mobile conveyors operate on top of the stacked material, which therefore must have the inherent strength and stability to support the mobile conveyors. Filtered tailings will typically have relatively high moisture content (e.g. greater than 15%) and low cohesive strength.

The advancements in mobile conveyor technology described above enabled mobile conveyors light enough, in terms of ground bearing pressure, to be used for the stacking of some filtered tailings. For example, the Ma'aden Phosphate Company in Saudi Arabia operates a self-propelled grasshopper type mobile conveying and stacking system to stack filtered phospho-gypsum waste in a multiple-lift waste facility at its phosphoric acid production plant. The waste product contains at least 25% moisture. Its physical properties were studied extensively in order to optimize the stacking lift height and compaction behavior prior to selecting the stacking system.

Figure 9: Filtered tailings (phospho-gypsum) stacking – Ma'aden Phosphate Co. (Saudi Arabia)

The stacking system, supplied by Terra Nova Technologies, is a customized version of a copper leach stacking system, showing how mobile stacking conveying technology can be adapted to filtered tailings application.

Waste stacking applications

The use of grasshopper type mobile conveying and stacking systems for stacking mine waste can be best visualized as a typical retreat heap leach stacking system operated in reverse. While a typical heap leach stacking system stacks ore in a semi-circular arc and retreats backward as it stacks, the same system could be used to stack waste or overburden as it moves forward, with the stacking system operating on top of freshly-stacked material.

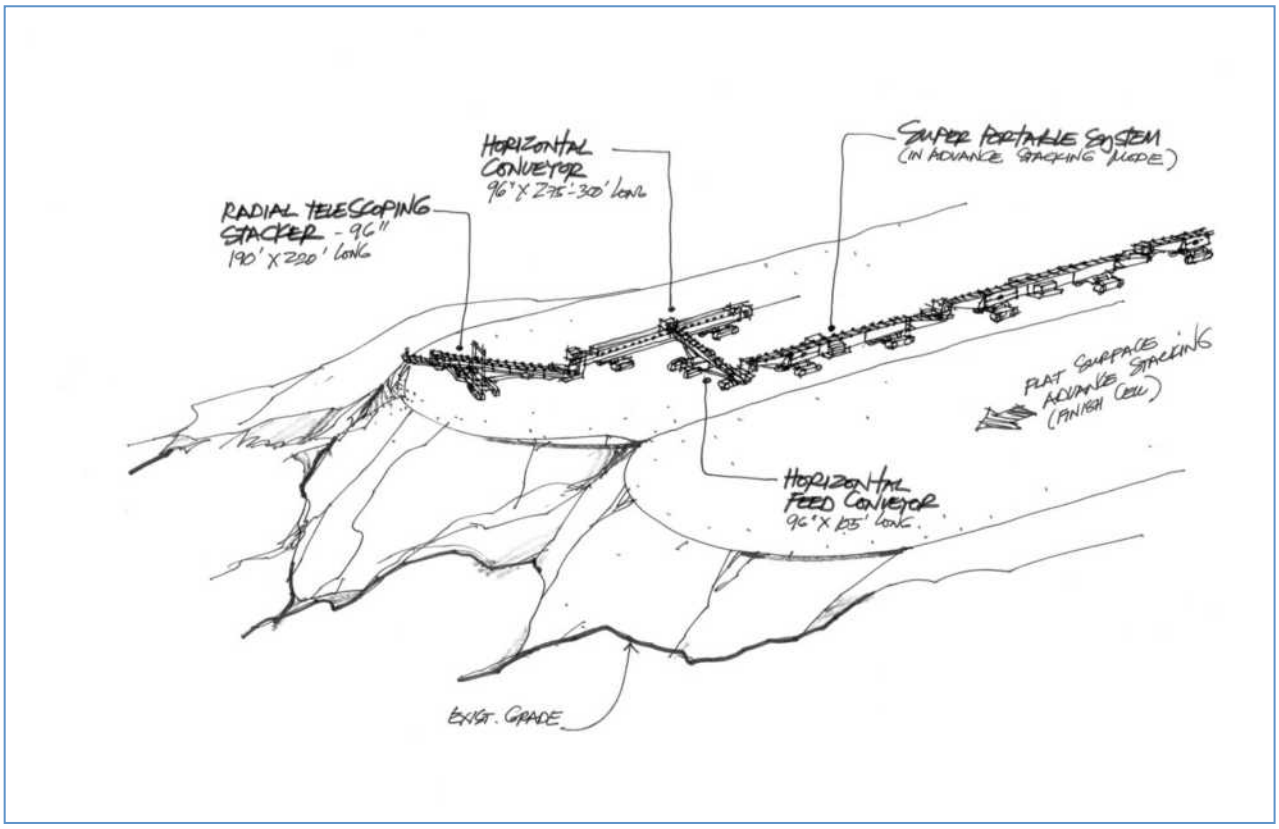

Figure 1: IPCC application: Self-propelled portable conveyors and radial stacker for overburden/waste stacking

With either a single-lift or multiple-lift waste facility, the stacking system operation – with the periodic insertion of a grasshopper unit into the conveyor line and stacker-and-indexing – is the same. Waste handling operations are typically of relatively higher throughputs (e.g. 5,000 to 15,000 t/h), thereby making the larger, self-propelled conveyors more appropriate. As discussed above, the "cell" width and height have a significant influence on efficiency.

IPCC applications

The adaption of self-propelled portable conveying units for IPCC applications has been in development by some manufacturers for several years and studied extensively by several mining companies since around 2010.

Typical ore and waste handling rates of IPCC applications range from 5,000 to 20,000 t/h, necessitating larger, self-propelled conveyor units with belt widths in the 1,800 to 2,400 mm range and on-board conveyor drive units of over 1,000 kW.

The function of these mobile conveyor units in an IPCC application is fundamentally the same as in a heap leach stacking operation: the periodic, scheduled removal or insertion of a mobile conveyor to retract or extend a conveyor system to accommodate an ever-changing operating position (i.e. a shovel, excavating or stacking position). In addition, these mobile conveyors can provide flexible links between the primary operating equipment, such as shovels and mobile crushers, and the transport system which can include a combination of mobile, shiftable and fixed conveyors. The incorporation of mobile conveyor units into IPCC applications is conceptualized in Figures 11 and 12.

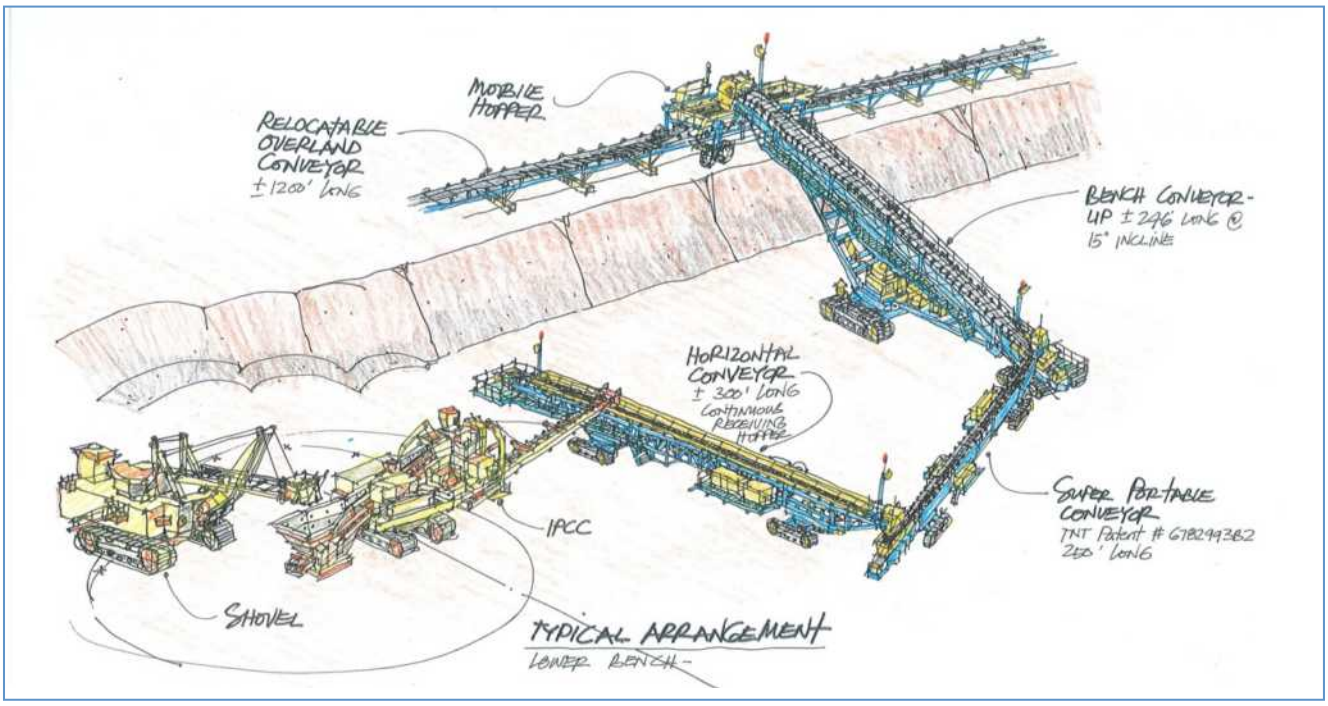

Figure 11: IPCC application: self-propelled portable conveyors and shovel/crusher operation (©Terra Nova Technologies, Inc.)

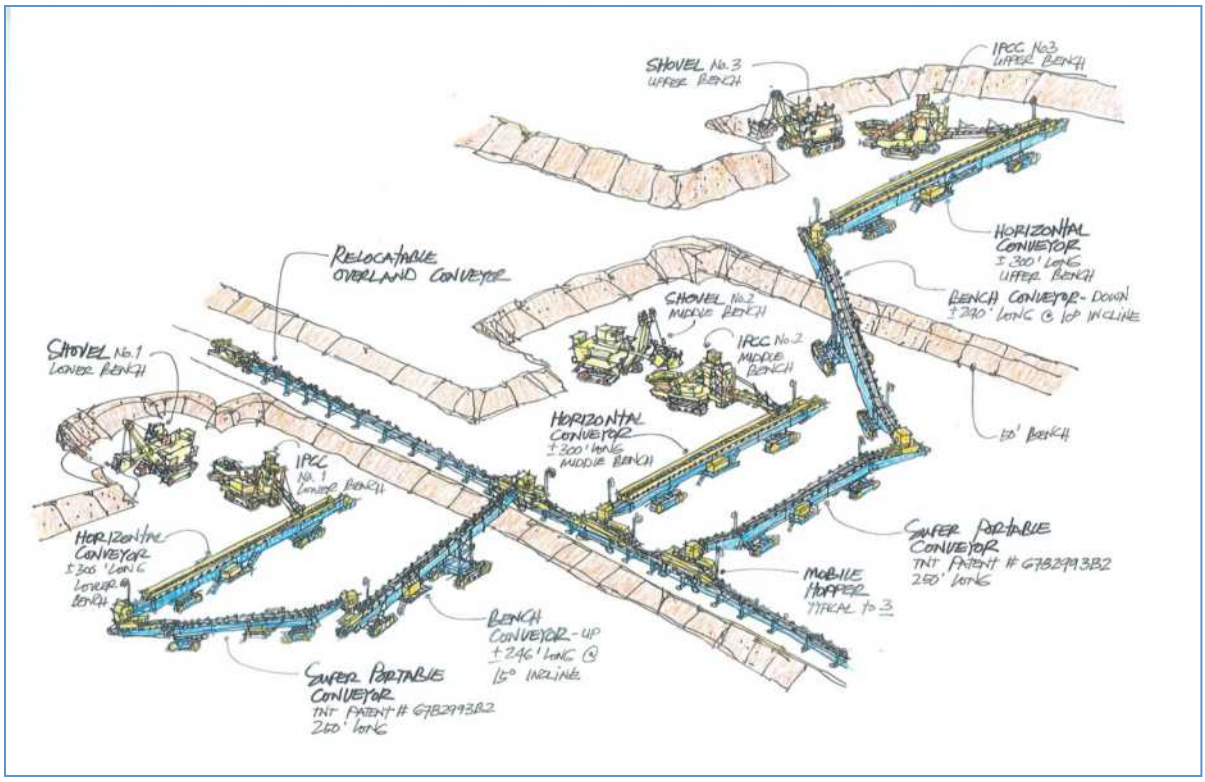

Figure 12: IPCC application: self-propelled portable conveyors in multi-bench mining operation

Mobile conveyors used for heap leach stacking applications are optimized for handling crushed and sized ore, normally less than 25 mm, and for operating on heap leach pads. This relates primarily to the design of transfer chutes, undercarriages (i.e. for ground bearing pressure considerations) and acid-resistance, among other parameters specific to the application (i.e. ambient conditions, altitude, power supply, etc.). For adaptation of these systems to filtered tailings and IPCC applications, several areas clearly require custom tailoring in terms of design and equipment specifications for the specific application.

For filtered tailings, with high moisture content, transfer chutes must incorporate low friction liner material, such as ultra-high-molecular-weight polyethylene (UHMW) or high density polyethylene (HDPE). They must also minimize any sloped impact chute surfaces. Use of tubular steel, as opposed to angle-iron and rolled-shapes, can also reduce spillage collection areas. Low ground pressure criteria can also require a larger equipment footprint area (e.g. larger crawler tracks or high-flotation tires) for a given machine weight.

Overburden and IPCC applications will normally involve ROM or primary crushed rock, ranging from 150 to 300 mm or more. This requires special attention to transfer chute design and impact zone design in order to minimize impact energy from large rocks onto the mobile conveyor structure, idlers and conveyor belt. For example, receiving chutes on mobile conveyors should include inclined “grizzly” bars to intercept larger rocks from directly impacting the conveyor, and directing them onto the belt with a

horizontal velocity component to prevent direct vertical impact with the belt. With this design, finer material can fall between the bars and provide a bed of fine rock onto which the larger rocks can fall. Impact zones on the conveyor require extra-heavy duty, live-shaft impact idlers or impact beds, while belting should include thicker, gouge and abrasion resistance cover compounds. For high capacity, large rock size applications, computer modeling of transfer chute design is highly recommended.

In-pit and overburden handling operations normally do not have the same ground pressure limitations that heap leach and filtered tailings applications do. Therefore, relatively speaking, the footprint area of these machines need not necessarily be of prime concern. Mobility is very important for any IPCC system to work well and there must be freedom of movement both on the loading and the stacking sides. Self-propelled portable conveyors and stacking/indexing systems provide the movability that will make full IPCC a reality.

Conclusion

Advances in grasshopper-style mobile conveying technology have enabled heap leach stacking systems to achieve significantly higher throughputs, with systems now achieving upwards of 10,000 t/h, and technology available to handle even more.

Self-propelled portable conveyors increasingly offer economically attractive alternatives to truck haulage for overburden, waste and ore handling.

Mobile heap leach stacking conveyor technology has been adapted to filtered tailings, waste and overburden. Full scale IPCC applications with specialized and custom modifications have been designed and are currently undergoing testing and pilot studies, utilizing the same fundamentals of proven equipment design and operation.

Applying distributed temperature sensing to the heap leach industry

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Abstract

Heap leaching of ore relies on a number of temperature dependent processes. Chemical rates of reaction, chemical solubility, solution viscosity and surface tension, microbial activity, and the development of ice wedges are all temperature dependent. Temperature is not a commonly measured variable in the heap leach industry, despite its importance in myriad reactions and processes. A novel technology known as distributed temperature sensing (DTS) has been developed for the oil and gas industry and has recently been applied in the hydrology and geotechnical sectors. A high power laser reader is used to interrogate a continuous optical fiber in what becomes analogous to a 5,000 m thermometer. A DTS system allows for precise temperature measurements at spatial and temporal resolutions as low as every 1 m, every 30 s, respectively. By installing a DTS system at multiple depths across the entire spatial extent of the heap, a very clear picture of the inner workings of the system can be determined. This paper details a case study of two DTS installations in reclamation soil cover systems. Although designed to act as one-dimensional, homogeneous systems, the interaction of surface and sub-surface heat transfer processes led to spatial variations in temperature distributions. The importance of intensively monitoring spatial distributions of temperature in the heap leach industry is highlighted.

Introduction

Temperature is a critical state variable governing the efficiency of heap leach programs. Chemical reactions in the leaching process can be accelerated at increased temperatures, possibly increasing recovery rates (Trexler et al., 1987). Biologically mediated mineral recovery is most often facilitated by temperature mesophiles that are metabolically active in the range of 20 to 45°C (Petersen and Dixon,

2002). Some sulfide minerals such as chalcopyrite are best leached using high temperature thermophiles that require temperatures between 65 to 75°C in order to be active (Petersen and Dixon, 2002). Temperatures throughout the entire heap must be ideal in order to optimize mineral recovery, and have been shown to be both spatially and temporally variable at the laboratory scale (Sampson et al., 2005). Unfortunately, there is a paucity of data on the spatial distribution of temperatures within constructed heaps at the field scale.

Typical methods for monitoring temperature at the spatial resolutions required for an operational heap leach system are not suited to the extensive pad areas and deep lifts that are becoming common within the industry. To instrument even a small portion of a heap pad using thermistors connected to a datalogger as with standard temperature monitoring would require hundreds of sensors at what is likely a prohibitive capital cost. In order to adequately monitor temperatures within a heap, a system must be robust and capable of continuously measuring at high spatial resolutions over large spatial extents.

DTS is a technology developed for the oil and gas industry to measure temperature at high spatial and temporal resolutions. A DTS system uses a high powered laser reader to propagate a laser pulse along an optical fiber. The backscatter of the light returning to the reader is composed of three spectral components: Rayleigh, Brillouin, and Raman scattering. Wavelengths of the return Rayleigh and Brillouin scatter are the same as the initial light pulse. Raman spectra, however, return at wavelengths and intensities slightly above and slightly below the initial pulse, called the Stokes and Anti-Stokes component. The ratio of the Stokes and Anti-Stokes component is temperature dependent. Given that the velocity of light in the fiber is known, the exact position of a temperature change can be determined. The optical fiber can be thought of as a very long continuous thermometer, while the laser reader interrogates the fiber to determine the temperature. The reader is directed to Tyler et al. (2009) for a comprehensive DTS review.

Due to the efficiency of propagating a high intensity laser pulse down a very clear fiber, cables up to 10,000 m in length can be deployed in the field. For most DTS readers, a 1 m measurement resolution is possible, meaning that a single 10 km cable will have 10,000 independent temperature readings. Temporal resolution is set at the discretion of the user and can be as frequent as every 30 s, although precision increases with increased signal integration time.

Distributed temperature sensing technology has found wide application recently. For example, DTS systems have been deployed to monitor stream stream temperatures and hydrology (Selker et al., 2006a, b). In soil science applications, DTS systems have been used to infer near surface soil energy balances (Rutten et al., 2010), and soil water content (Sayde et al., 2010; Steel-Dunne et al., 2010; Krzeminska et al., 2011). With respect to waste storage, DTS systems have been applied to detect the percolation of water into municipal landfills (Weiss, 2003). Although DTS systems are well suited to taking high

precision and accuracy temperature measurements at unprecedented spatial resolutions, the technology has yet to be deployed in the mining sector.

Accurately determining the distribution of temperatures within heap leach piles will allow for fully optimized, and therefore, economically efficient operations. While the importance of temperature in determining rates of reaction and recovery of metals is well recognized, there remains a critical knowledge gap in the literature. The objective of this paper is to document the implementation of a DTS system in a mining application and demonstrate the potential benefits in applying the technology to heap leaching. Examples from reclamation cover systems will be used to demonstrate the utility of a DTS system.

Methodology

Two case studies involving reclamation cover systems will be presented to demonstrate the utility of a DTS system in measuring the spatial distribution of temperatures. The first case study will document the implementation of a DTS system in a linear transect at multiple depths, while the second will document the implementation of a DTS system in a grid pattern at two separate depths.

Study area

The first field experiment (Site #1) was conducted at a mine in northern Saskatchewan, Canada. The study area was a 1 m thick cover system over a waste rock dump. The cover system was configured as a 1 ha plateau sloping at a south-facing 2%, and a 0.5 ha north-facing 25% slope. The cover system was constructed using fine sand salvaged from a local drumlin. Average sand, silt, and clay content were 94, 5, and 1%, respectively, with an average bulk density of 1.55 g cm^{-3}.

The second field experiment (Site #2) was conducted at a mine in the western United States. The study area was a waste rock dump on which a cover system comprised of compacted local material was constructed. The study area was a west-facing slope at 3H:1V. Average gravel, sand, and silt / clay content were 33.5, 18.75, and 47.75%, respectively. Bulk density of the compacted material was 1.83 g cm^{-3}.

Climate

The climate at Site #1 was classified as continental and had mean January and July air temperatures of –23 and +16°C, respectively. Thirty year mean annual precipitation was 481 mm.

The climate at Site #2 was classified as hot summer continental. Thirty year climate normals for the site indicated minimum and maximum average air temperatures of –3 and +13°C, respectively. Average rainfall for Site #2 was 440 mm yr^{-1}.

DTS system

The DTS system used for the field experiment was an Oryx DTS-SR (Sensornet, UK). The Oryx was a robust field unit with a measurement range up to 5,000 m at 1 m measurement resolution. Temperature resolution can be as fine as 0.01°C depending on integration time. Measurements were calibrated by passing a reserved length of cable through an area of known temperature. In the case of the present case studies the known temperature area was a standard beverage cooler filled with ice. A calibration coil 50 m in length was submerged in the ice bath where temperatures were independently verified using a pair of PT-100 platinum thermistors.

Power draw for the Oryx was rated at 30 W during active measurement and 0.5 W when idle. Power for the system was supplied by deep cycle batteries that were charged with a solar panel array. The solar system also powered a peripheral laptop for data logging.

The cable used at both sites was the Sensornet DamSense cable (Sensornet, UK), rated for accurate temperature measurements from –55 to +85°C. An example of a field installation of a DTS system for Site #1 is shown in Figure 1.

Figure 1: DTS installation at Site #1 showing (from top to bottom): solar panel array, deep cycle batteries (in coolers), calibration bath (blue and red coolers) and Oryx DTS reader (silver enclosure)

Cable installation

For Site #1, 500 m of fiber optic cable was installed into a 100 m linear transect at depths of 90, 50, 10, and 0 cm. The 0 cm cable was laid directly on the surface and covered only with sufficient material as to prevent the cable from being directly exposed to the atmosphere.

The cable was installed by excavating a trench to the prescribed depth. The cable was then laid out was laid out by hand and protected with a layer of the *in situ* sand. The trench was then backfilled until the next depth was reached and the process was repeated. In this way, the cable was extended in one continuous length and allowed for full spatial coverage of all depths. An example of how the cable was installed at Site #1 is provided in Figure 2.

Figure 2: Fiber optic cable installation into a cover system at Site #1. Note the yellow fiber optic cable

The fiber optic cable at Site #2 was installed in a grid pattern at 100 and 130 cm, similar to what is shown in Figure 3. The cable was laid down in a grid pattern and protected with in situ cover soil and then buried when the next lift of material was placed. By installing the cable in a grid pattern a two-dimensional visualization of temperature distribution can be determined.

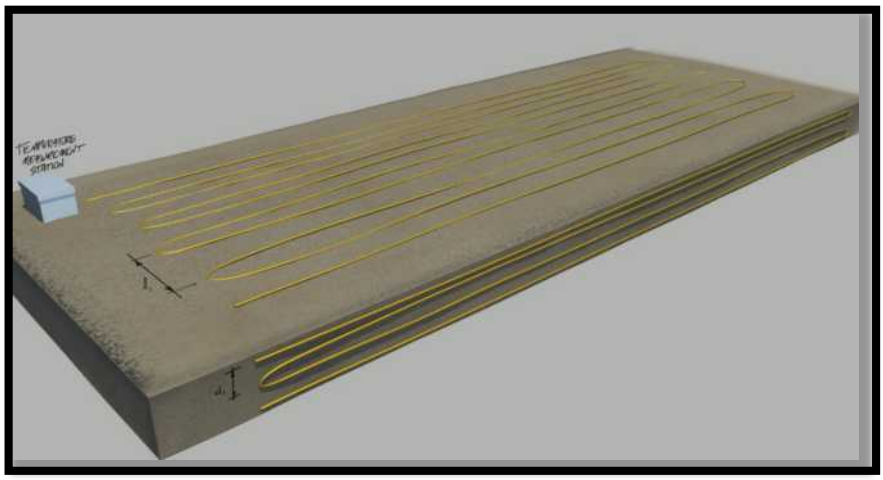

Figure 3: Schematic of fiber optic cable installation at Site #2 in a grid pattern at multiple depths

Data collection

Data were collected every 20 minutes using a 60 s signal integration time. At both sites an automated weather station was installed to collect complementary meteorological data. The meteorological stations measured air temperature and humidity, wind speed and direction, and net radiation.

Results

Site #1 data

The field experiment at Site #1 occurred in late August as air temperatures were declining from their summer maxima. Distributed temperatures at the soil surface corresponded closely to air temperatures (Figure 4). Two distinct temperature zones are seen that correspond to plateau and slope areas of the cover system. The slope was north facing and was subject to lower incoming radiation.

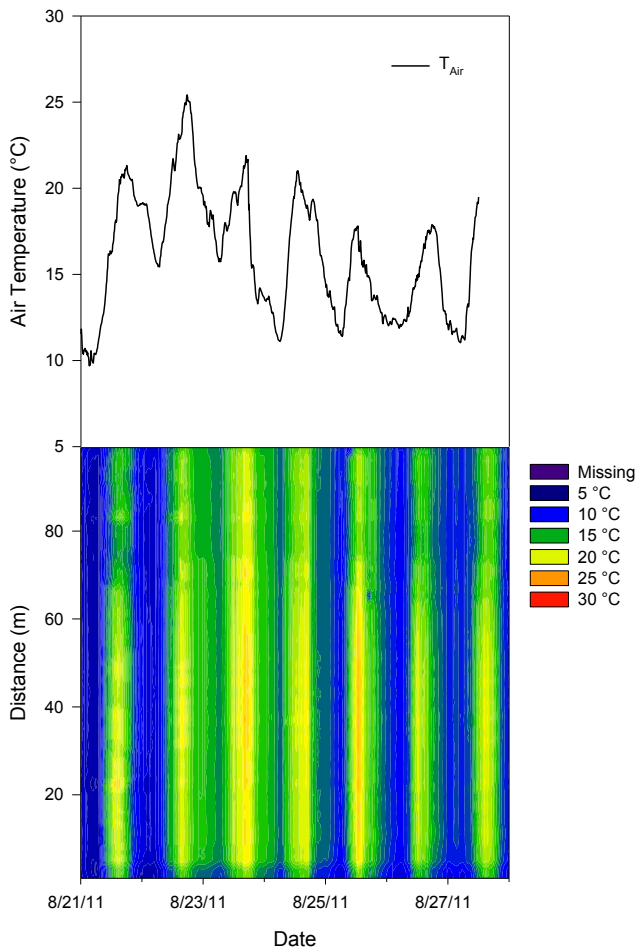

Figure 4: Air temperature and 0 cm DTS cover system temperatures as a function of time at Site #1. Note that the plateau extends from 0 to 65 m and slope extends from 65 to 100 m

Cover system temperatures decreased with increasing depth at Site #1 (Figure 5). Regardless of depth, however, a similar pattern was seen where a distinct separation was found between the plateau (0 to 65 cm) and the slope (65 to 100 m).

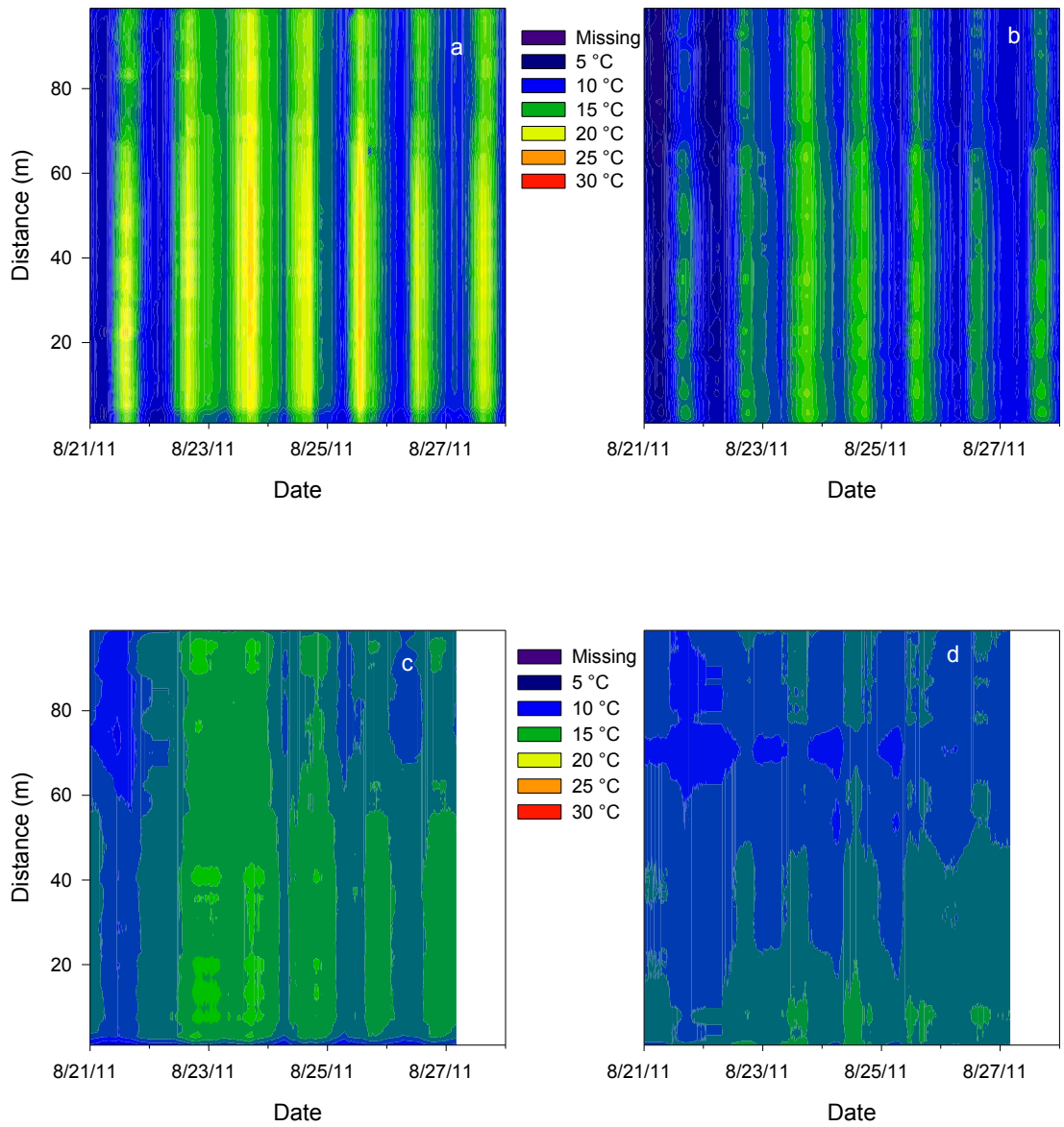

Figure 5: Cover system distributed temperatures along the linear transect as a function of time at Site #1. Depths are a) 0 cm, b) 10 cm, c) 50 cm, and d) 90 cm

Site #2 data

Spatial distributions of cover system temperatures demonstrated a gradual release of heat at the onset of winter (Figure 6). Cover system temperatures remained higher at the 130 cm depth (Fig. 6a) as compared

to the 100 cm depth (Fig. 6c), indicating that the annual summer high temperature wave had propagated below this depth and was now being released to the surface. The effect was much more pronounced when comparing between days, as an appreciable amount of heat was still retained at the 130 cm depth on November 25th (Fig. 6d) when compared to the same time at the 100 cm depth (Fig. 6b).

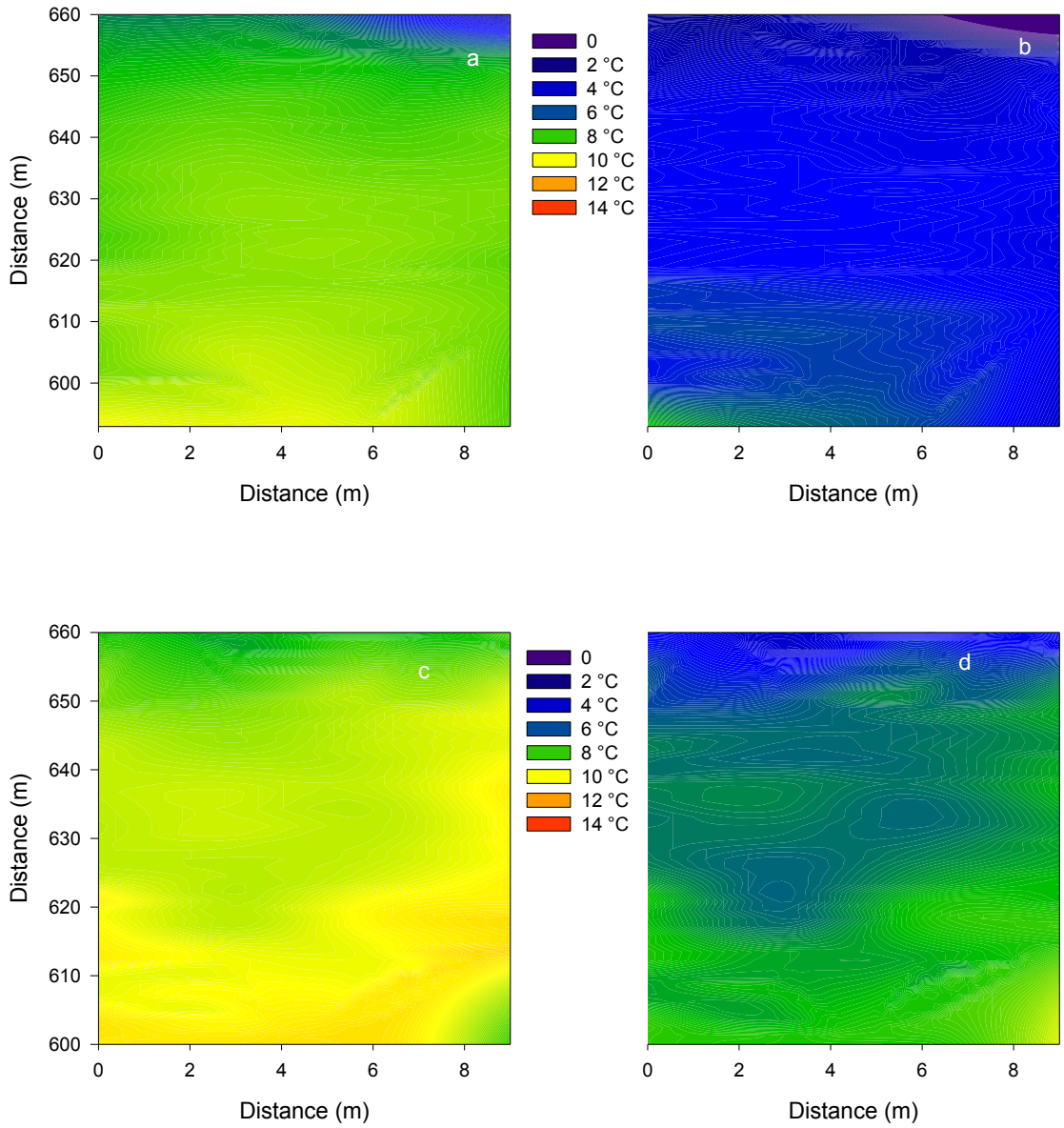

Figure 6. Spatial temperature contours as a function of space measured at Site #2 demonstrating how the cover system cools in early winter. Two depths were measured: a) 100 cm on November 8, 2012; b) 100 cm on November 25, 2012; and c) 130 cm on November 8, 2012; d) 130 cm on November 25, 2012

When spatial temperatures are compared within the same day, elevated temperatures are seen at the toe of the cover system slope (600 to 620 m) as compared to areas nearer the crest (640 to 660 m). The cover system was constructed on a 30 m waste rock dump. Increased temperatures near the cover system base are a result of a combination of increased mass available for ground heating and increased insolation at lower slope positions.

Isolating temperature readings from a single transect extending from the cover system toe to crest allows for an examination of the temporal evolution of temperatures along the slope (Figure 7). Both the 100 cm and 130 cm depths responded to a sharp drop in air temperature to less than –10°C. Although air temperatures returned to positive values in the second half of the field experiment the ground heat flux at the surface was insufficient to increase temperatures at depth. Temperature evolution seen in Figure 7 compares well with Figure 6 in that lower slope areas from 600 to 620 m cooled more slowly when compared to areas between 620 and 680 m.

Discussion

The spatial distribution of temperatures detailed above provides interesting parallels to the heap leach industry. Both cover systems were constructed as homogenous systems with no variation in physical properties. Nevertheless, spatial differences in temperatures were seen, suggesting that physical homogeneity does not preclude the possibility of spatial variations in temperatures. The propagation of temperature within the subsurface depends on both the physical and thermal properties of the material, as well as highly non-linear heat transfer processes taking place at the interface between the soil and the atmosphere (Hillel, 1998). The addition of biological processes introduces further non-linear processes that can affect the distribution of temperature in a heap.

The DTS system was able to discern very fine temperature differences at high spatial and temporal resolutions. A comparison of Figures 6 and 7 demonstrates that a grid type pattern giving three-dimensional type data provides a further refinement of the dataset. From Figure 7, which is a two-dimensional type dataset, the elevated temperatures at the toe of the slope were seen. However, it is clear from Figure 6 that the distribution of temperatures in the lower slope was not uniform, despite the homogeneity of the material used to construct the cover system. With regards to the heap leach industry, areas of differential thermal behavior could represent decreased recovery rates due to depressed temperatures. A DTS system deployed to maximize spatial coverage could alert the operator to areas where recovery is potentially decreased, allowing for immediate corrective action.

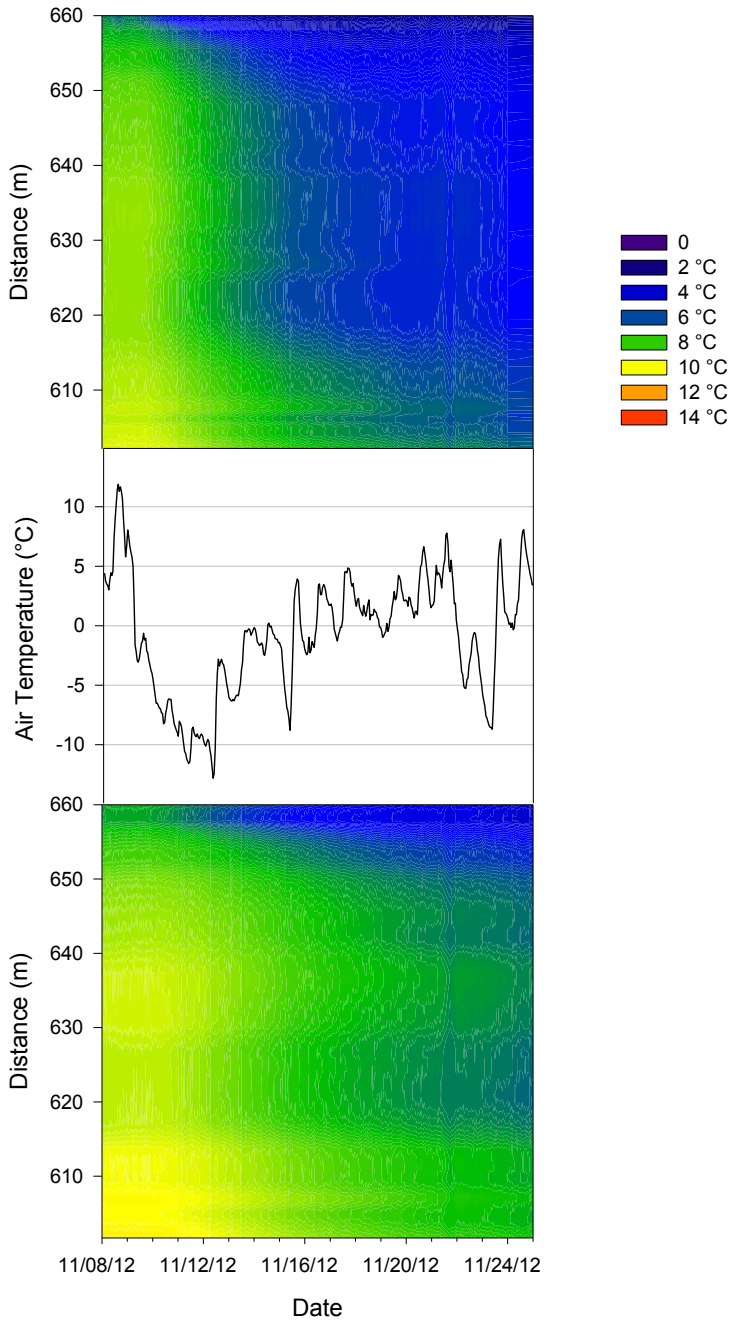

Figure 7: Temporal evolution of cover system heat release at 100 (top) and 130 cm (bottom) in relation to air temperature (middle). Temperature was measured as a function of time along a single transect extending from cover system crest to toe

Conclusion

Implications for heap leach

Chemical and biological processes at work in heap leach operations are temperature dependent. Areas of below-optimal temperatures represent decreased recovery efficiency. Unless these areas are identified through intensive monitoring with a system such as the one detailed in the current manuscript, the decreased efficiency will inevitably lead to sub-optimal economics. A DTS system provides a simple and cost effective way for intensively monitoring temperatures over large areas. By actively monitoring heap temperatures the mine operator is able to maximize recovery of minerals, leading to optimal economics.

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Optimum metal recovery from solutions: Design considerations

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Abstract

There is a good knowledge base for leaching in heaps. The metal recovery from solution is generally straightforward, and for gold projects, CIC (carbon in columns) circuits are often used, and this technology has not changed very much over the years. There are components such as vessel dimensions, solution distribution plates, and pipe sizes that can result in problems such as pressure drop, and poor fluidization that can lead to poor circuit performance. There are also parameters such as pH, temperature, and reagent addition locations that affect the loading of metal onto the carbon. Metal loadings on carbon in solution circuits are generally higher than in slurry circuits and these are compared, with an assessment of the differences.

Carousel configurations have been effective for gold leaching in CIP/CIL (carbon in pulp/carbon in leach) circuits where the carbon remains in each tank, preventing backmixing during transfer. This results in a steeper loading profile, which in turn contributes to higher loadings and decreased solution losses. The higher loadings can result in reduced capital and operating costs for the downstream elution process. Carousel CICs have only been used in a few small circuits globally. CIC circuits should be better suited to carousel arrangements than CIP/CIL circuits since there is no slurry to deal with when collecting a batch of loaded carbon for treatment. For larger throughput operations there are a number of design considerations that should be considered that contribute to the effective operation of a carousel CIC. These include flexibility of feedrate, minimization of carbon attrition, retention time of carbon in the circuit, number of tank stages, maximizing the loading, and ease of transfer to the elution circuit. An assessment of these factors and design considerations is presented.

Introduction

Many high grade gold deposits have already been mined, so now there is a shift to exploration of lower grade deposits. For many of these projects, heap leaching with the associated lower capital and power costs can be beneficial.

In most heap leach applications, the solution treatment facility is given little attention. Gold heap leach solutions typically flow to a pregnant solution pond, and are then pumped to a few CICs. Gold tends to load fairly well onto these, and with clear solutions there is very little supplementary processing equipment required. Solution recovery is not generally critical, since the solutions are often kept in closed circuit with the heap, and most of this is recovered when the solution next passes through the CICs. There is some recovery loss that results from entrained solution in the heap.

Data and discussion

CIC circuit design and operation

There are a few variables affecting the performance of CIC circuits. These include:

- solution upflow velocity;
- carbon loading;
- number of stages;
- carbon circuit retention time;
- carbon transfer;
- carbon batch size;
- carbon attrition;
- carbon elution; and
- carbon regeneration.

Solution upflow velocity

A key design parameter is the upflow velocity in the CIC columns. It is important to fluidize the carbon so that there is sufficient contact between the solution and the carbon. Figure 1 indicates how much the different carbons expand when the solution flows upwards through a carbon bed. One of the key design parameters is the solution flow required by the overall water balance of the heap. The upflow should be calculated for the design flow that will be somewhat higher than the average flow. The flow should remain fairly constant, and the circuit can be stopped when the pregnant solution pond gets low. An alternative is to recirculate barren to maintain steady fluidization, but this is not good for loading since the solution feed grade decreases.

The carbon batch size in a column is generally matched with the batch size of the elution circuit, but this is not essential. The upflow velocity of the solution will determine the required cross sectional area of the CIC columns, and the height of the columns will be determined by the expansion with an allowance for freeboard.

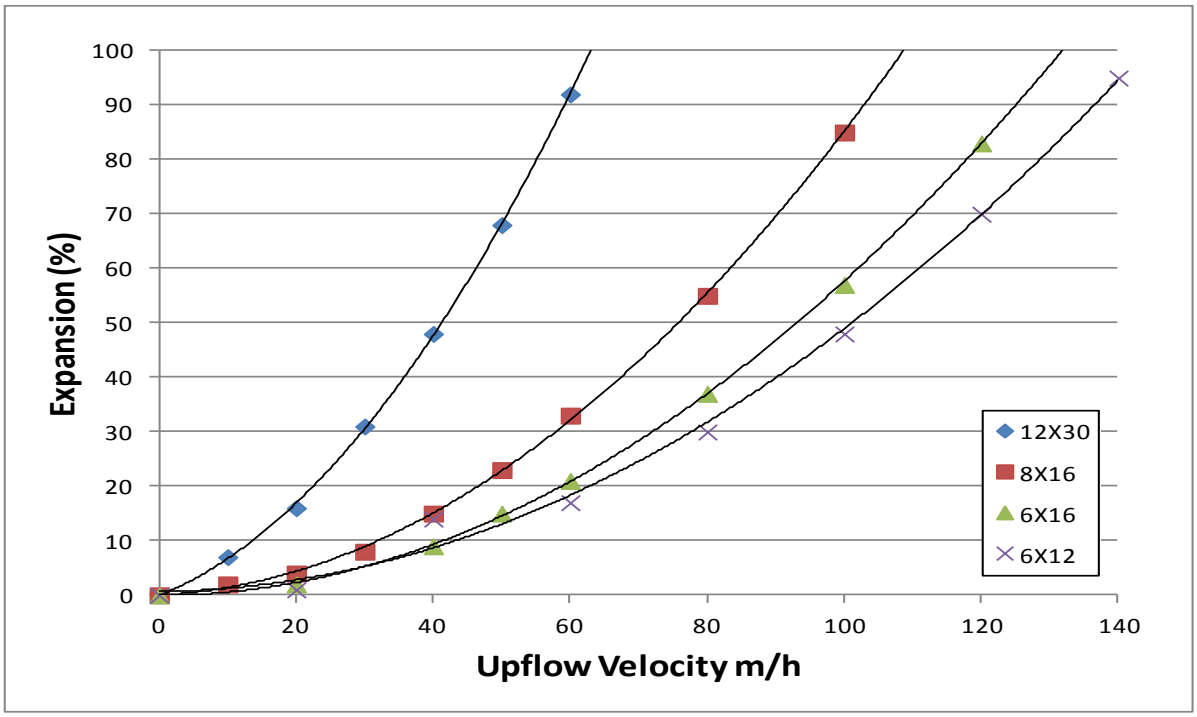

Figure 1: Carbon bed expansion with water at 25°C

Carbon loading

It is important to get a reasonable estimation of the carbon loading to calculate the carbon throughput through the circuit. Figure 2 indicates loadings for five operating plants. The solution grades entering tanks are plotted against the carbon in the tanks. The derived relationship is:

$$\text{Loading (g/t)} = 7{,}630 \times \text{incoming solution grade (mg/l)}^{0.8909}$$

This relationship is a reasonable approximation for loading, but it should be noted that the various operations used to derive this relationship have differences in the way that they operate their circuits. One of the primary differences is the retention time in the circuit. For similar carbon activity, the carbon that remains in the circuit for longer will have higher loadings, but as this occurs, the solution tails can increase. The relationship of these variables is discussed further below.

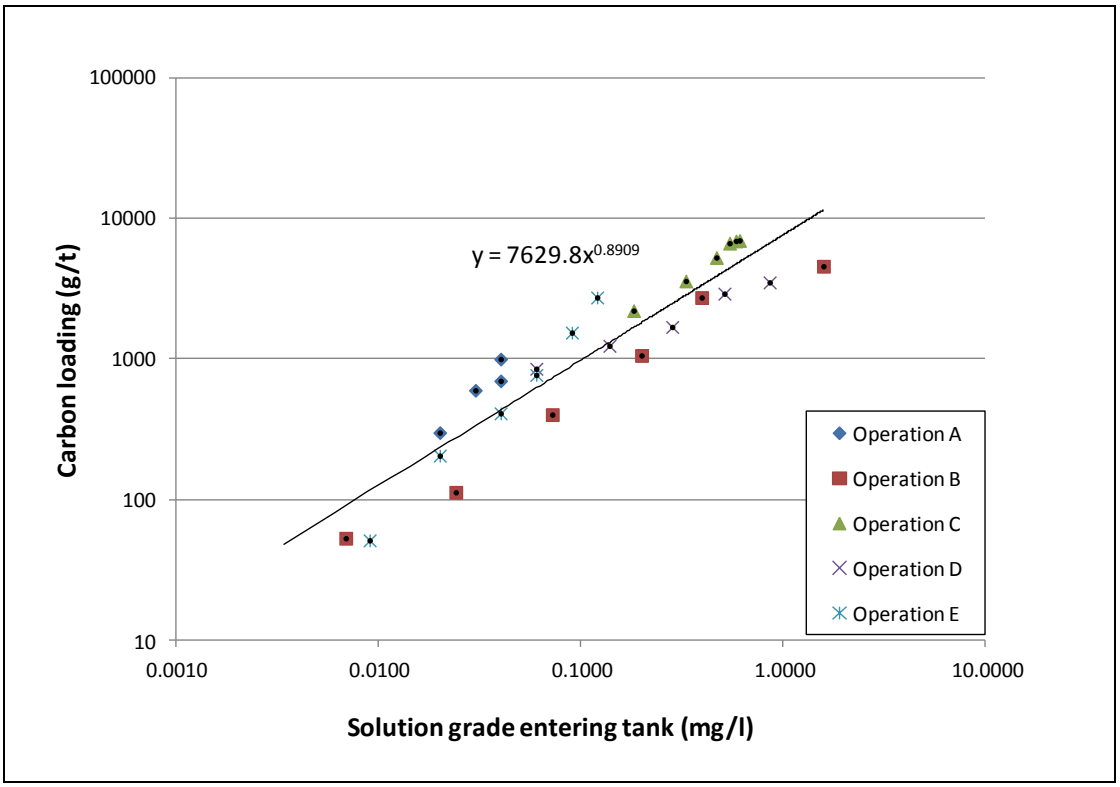

Figure 2: CIC carbon loadings

Number of stages

There are operations with as few as three stages. If the solution tail is not critical, fewer stages are more cost effective, but additional stages will allow for better overall solution recovery when combined with some of the additional carbon management practices discussed here. Typical circuits have around five stages.

Carbon circuit retention time

In Figure 2 it can be seen that some of the operations trend above the average, and some below. In general the loading will increase with higher retention time. Operations are not very consistent in their approach to retention. In Figure 2, the highest retention times are over 50 days, and the lowest around five days. In general a good average is normally around 14 days, and this is similar to CIP/CIL circuits.

Carbon transfer

Carbon is typically pumped from a takeoff point just above the distributor plate to the CIC tank above it in the train, using a recessed impeller pump. To create the best metal loading profile, backmixing should be minimized. The best way to do this is to move loaded carbon from the top tank to elution, then to move

from the second tank to the top tank, and continue down the train. Once the bottom tank has had the carbon pumped from it, the barren carbon returning from elution is added back to it.

Carbon batch size

The carbon batch size in the CIC tanks will typically be matched to the elution batch size, but does not have to be. Once the loading, number of stages and circuit retention have been fixed, the batch per stage is known. This batch size of a chosen carbon type, and selected fluidization (Figure 1) will dictate the column height. Typically the columns would be designed for at least 100% expansion with some additional freeboard.

Carbon attrition

In CIP/CIL and CIC tanks carbon attrition is unavoidable, and over time new carbon has to be added back. The majority of the attrition occurs during the acid wash/elution/regeneration cycle. A well designed elution circuit will have a closed water transfer circuit where any fine carbon can be removed from a central tank. This carbon can either be sold if there is sufficient contained gold, or discarded.

In the CIC circuit there is also potential for some attrition, especially with the interstage pumping. It is important to use pumps that are gentle on the carbon. At the back end of the CIC, there will also be a fine carbon screen that will recover smaller carbon particles so that it doesn't report with the solution to the barren pond. The finer carbon that gets through these screens is also a potential problem, and can adsorb gold in ponds or the heap downstream, and solution filters are often installed at the back end of the CIC to prevent this from happening.

Carbon elution and regeneration

In a well-designed elution circuit, the carbon that is returned to the circuit will have the following characteristics:

- Very low gold loading. The lower the gold loading (i.e. the better the elution efficiency) the better will be the solution tail of the CIC circuit.
- Good carbon activity. There are foulants in all circuits like oils, flocculants etc. If there is limited fouling, it is sometimes possible to regenerate only a percentage of the carbon rather than 100%. Where foulants are present, it is generally better to regenerate all the carbon after each elution.
- No fine carbon. If the returned carbon contains fines, the fines will exit the CIC circuit, and adsorb gold in a pond or the heap itself, and this gold will be lost.

CIC innovation

There have not been many innovations in CIC circuit design in recent history. There is potential to improve the performance of these circuits in the following ways:

- carousel operation; and
- conical tank design concept.

Carousel operation

There have been a number of CIP plants installed in a carousel arrangement. The advantages of this are that the carbon does not have to be moved between tanks. This results in less attrition, less backmixing with carbon in adjacent tanks, and better control of the carbon batch size in the particular vessel. Less backmixing allows for more consistent loading of the carbon since each stage is discreet. In most applications, the carbon batch size can be linked to the elution vessel capacity. A typical carousel circuit has flexibility for the feed to enter any of the tanks first. A schematic arrangement is shown in Figure 3 below.

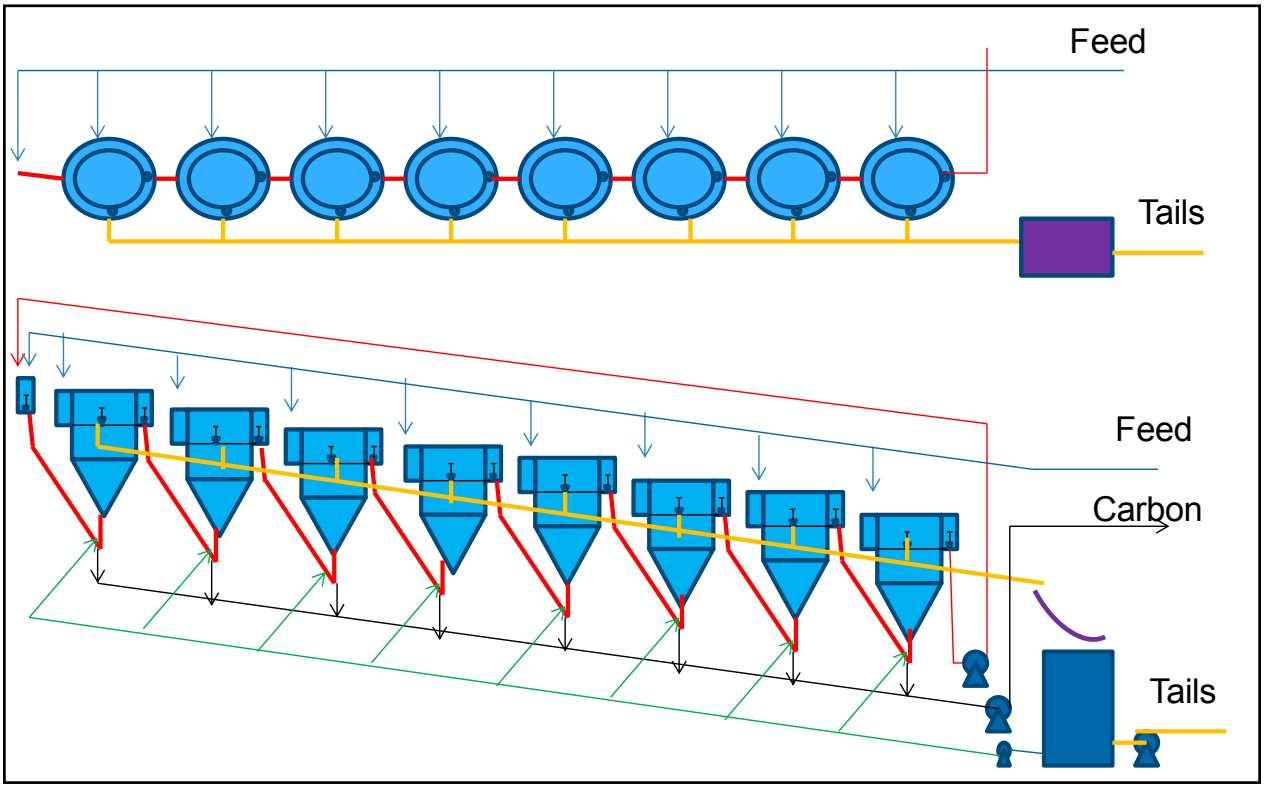

Figure 3: Carousel concept for CIC circuits

The carousel concept is characterized by the fact that any of the tanks can act as the loaded carbon tank. In order to achieve this it is necessary to have an additional solution pump at the tail end of the last CIC. Similarly, there needs to be flexibility for the solution tail to be taken from any one of the CIC tanks to the safety screen and tailings pumpbox. Each time a batch of carbon is ready to be eluted, the tank is taken offline and the tank below it becomes the feed tank. After elution, the carbon is added back to the tank it was taken from and this becomes the last tank in the train. With this configuration there is no necessity for interstage pumps, and this has been done successfully with a number of CIP/CIL operations. Interstage pumping is one of the places where carbon degradation occurs, although the majority of the carbon loss normally results from regeneration.

Figure 4 is a more detailed schematic configuration of a typical CIC circuit. In this case the solution flows by gravity from one vessel to the next. The design head differential has to consider the pressure drops due to piping and through the distributor orifices. In addition there is a "density component" that occurs due to the combination of wet carbon in water having a higher density than the incoming water.

The piping often enters centrally since this gives the most even distribution to a circular CIC vessel, although in some instances the piping drops down externally and enters the chamber at the bottom of the CIC from the side. The distributor plate has openings that keep the carbon from entering the lower chamber, and there are various options including "bubble caps" that are similar to those used in distillation columns.

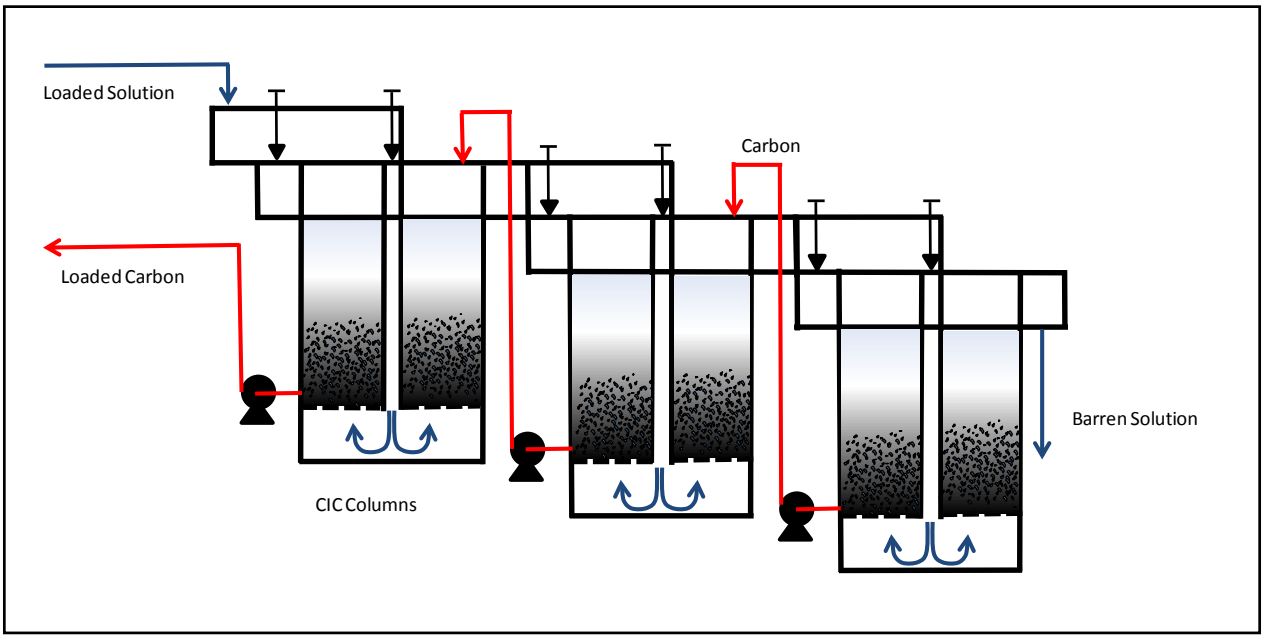

Figure 4: Detailed CIC tank arrangement

Conical tank design concept

Figure 5 shows a conceptual design of a CIC column for a carousel circuit. In theory the carousel CIC vessels could be the same as those shown in Figure 4; however the concept shown in Figure 5 offers a number of advantages. The loaded carbon can be more easily pumped out as a batch. It is important in a carousel arrangement to recover all of the carbon. In a conventional CIC circuit is not important as carbon is being pumped from the second tank, and the top tank is always the loaded carbon tank. In contrast, with a carousel CIC, the loaded carbon tank becomes the last tank in the train when it is brought back online. If there is any residual loaded carbon present in this last tank, the solution tails grade is likely to increase.

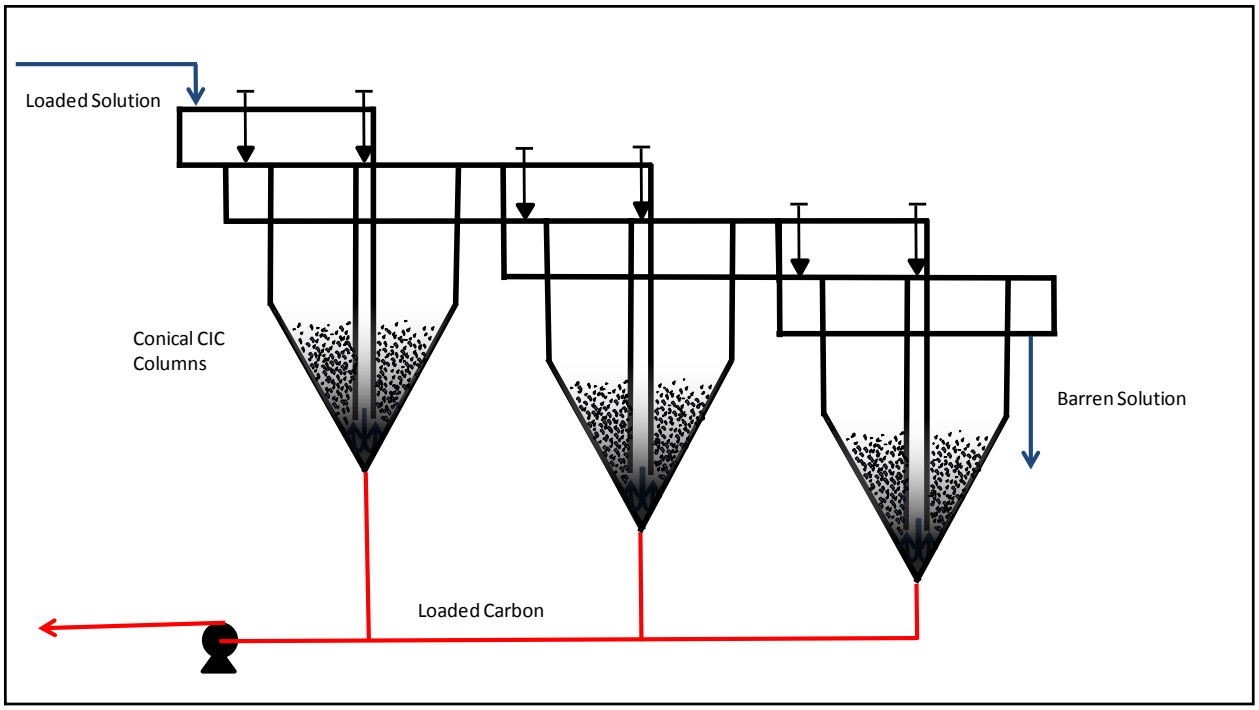

Figure 5: Carousel CIC tank arrangement

Some of the advantages of the conical arrangement shown are:

- There is a cost saving in that the distribution plate is not necessary and this is one of the more complex components of the CIC tank.
- Removing a complete batch of carbon for elution is easier, since there is no distribution plate. The carbon can be pumped from the lowest point and the column completely emptied so that there are no remaining loaded carbon particles that could affect the solution recovery when the CIC comes back online as the tailings tank.

- The conical arrangement will allow more flexibility in solution throughput rate. With a cylindrical vessel velocity has to be controlled within a fairly narrow range to achieve the fluidization calculated in Figure 1, whereas with a conical vessel the fluidization will self-regulate to an extent.

Conclusion

CIC circuits are well proven and have not undergone much technical innovation over the years. Since there are likely to be more heap leaches implemented in the future, it is likely that there will be more focus on optimizing the process. Some of the innovations discussed here have the potential to improve the operation of these circuits, with resulting improvements in the overall process economics.

The effect of elevated temperature on copper discharge performance during cold stripping of loaded carbon

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Abstract

This study describes the introduction of elevated temperature during cold stripping of loaded carbon in an adsorption, desorption and regeneration (ADR) plant. This is a practice applied at Eldorado Gold's Tuprag Kisladag Gold Mine in Turkey.

Cold stripping of copper at ambient temperature was slow, and reached only 37%. After elevating the temperature to 50 – 60°C, the copper discharge performance increased dramatically and reached around 70%. This was achieved without losing any gold.

The cold stripping and warm stripping processes will be explained, using real plant data from more than four years of operation. Results achieved before and after increasing the temperature will be presented in this paper.

Introduction

The Kisladag ADR plant has a cold stripping circuit ahead of hot stripping to remove as much copper as possible prior to sending the pregnant solution to electrowinning. A pressurized ZADRA hot stripping circuit follows the cold stripping process.

In regular practice, the cold stripping was performed at ambient conditions. Caustic soda (sodium hydroxide) was added to maintain an alkaline medium, while cyanide was added to facilitate the removal of copper. Process time was from two hours to four hours, with an average time of three hours, depending on the copper load of carbon.

Copper discharge performances were about 35 – 40%, as illustrated in Table 1. Although this is not a high recovery, the subsequent electrowinning process did not suffer, due to low copper levels in loaded carbon.

As the copper levels increased in the pregnant solution from heap leach and impacted on carbon loading (Table 2), copper removal began be increasingly important. To deal with the problem, the cold stripping conditions were adjusted. Stripping time was increased, cyanide concentration was changed, etc. There was no significant improvement: extending stripping time helped a little, but this caused a reduction of the available time for hot stripping, and therefore a reduction of the overall ADR capacity.

Table 1: Copper removal during cold stripping

Time intervals (three months)	Cu in carbon before cold stripping (ppm)	Cu in carbon after cold stripping (ppm)	Copper discharge recovery (%)
Sept. 2008 – Dec. 2008	1,184	772	35.8
Dec. 2008 – March 2009	1,082	723	33.5
March 2009 – June 2009	1,317	805	39.0
June 2009 – Sept. 2009	1,635	896	41.2

Table 2: Copper in pregnant solution (CIC feed)

Years	Copper (ppm)
2009	131
2010	182
2011	203
2012	243
2013	313

It was decided to elevate the temperature to 60°C to improve copper removal. There were concerns that the gold might also drop, but this did not happen. On the other hand, copper removal performance improved dramatically.

Copper removal at ADR, warm stripping of copper before hot stripping

Kisladag Gold Mine ADR has three parallel units of column trains; each has five tanks. Two of the trains are in production, leaving one train on standby. The main copper removal takes place here, since cyanide solution is added ahead of CIC (carbon in column) columns, to the feed box of trains. As illustrated in Table 3, an average of 99.6% of the copper is discharged with the effect of cyanide addition. The remaining 0.4% is enough to place a significant copper load on carbon. The average gold and copper loads on carbon from 2009 to 2013 are shown in Table 4. Further copper removal is necessary before the electrowinning operation. After several attempts to increase copper discharge during cold stripping as explained above, the temperature was elevated and a significant improvement in copper removal was achieved.

Table 3: Copper removal in CIC columns

Years	Cu in CIC feed (ppm)	Cu in CIC discharge (ppm)	Copper adsorbed on carbon (%)	Copper discharged at CIC tails (%)
2009	130.7	130.0	0.54	99.46
2010	181.7	181.2	0.31	99.69
2011	203.3	202.0	0.65	99.35
2012	243.1	242.1	0.42	99.58
2013	313.4	312.7	0.24	99.76

Cold stripping process

A cold stripping solution of 7 m^3 volume is prepared from barren solution before the cold stripping process starts. Caustic soda is added, 10 g/L, to maintain an alkaline environment. Cyanide is added, 2,500 ppm concentration, to strip the copper. Cyanide make-ups are done to maintain constant cyanide levels during the process. The process normally takes about three hours. The solution is circulated between the cold stripping tank and the stripping vessel.

Warm stripping process

The hot solution is obtained from barren electrolyte solution after electrowinning is completed. The hot stripping solution tank (45 m^3 volume) is located next to the cold stripping solution tank; a pipe connection with a valve has been made between the tanks; for each warm stripping, a 7 m^3 of barren solution at about 60°C is transferred from the hot stripping tank to the cold stripping tank (now, a warm stripping tank) – this has eliminated a heater for the cold stripping tank.

On the other hand, this is also like a bleeding of the barren electrolyte solution to the warm stripping process, and therefore, eliminated the need for the discharge of barren electrolyte solution at regular intervals. The warm stripping solution is circulated between the warm stripping tank and stripping columns for about three hours. Caustic and cyanide concentrations are the same as cold stripping and are 10 g/L for caustic and 2,500 ppm for cyanide, respectively. Cyanide make-ups are done when necessary. The hot and warm stripping tanks are shown in Figure 1.

Figure 1: Warm and hot stripping solution tanks

Table 4: Average copper on loaded carbon

Years	Copper (ppm)	Au (ppm)
2009	1,521	2,652
2010	2,180	3,018
2011	2,034	3,225
2012	2,069	3,139
2013	2,427	2,936

Table 5 illustrates the copper removal performance before and after the warm stripping process. The net gain is 32.4%.

Table 5: Comparison of copper removal before and after elevated temperature

Period	Cold stripping copper removal (%)	Warm stripping copper removal (%)
2008 – 2009	37.8	
2009 – 2012		70.2

Conclusion

The copper removal at ADR is a staged process; it begins by adding cyanide to CICs. The bulk of copper is removed at this step. The remaining removal is done during cold/warm stripping. The success of cold/warm stripping depends on succeeding in a vast amount of copper removal at CICs. Failure of this will directly affect warm stripping performance. However, the success of cold stripping is highly dependent on elevated temperatures at the Kisladag ADR plant.

Migrating from cold stripping to warm stripping at the Kisladag ADR plant has improved copper removal by 32.4%. This has the important effect of maintaining a high gold grade at Dore, and also provides advantageous electrowinning conditions. The implementation of warm stripping ahead of hot stripping to plants which have high copper loads at carbon is recommended.

Operational techniques to recover metal values from heap inventory and in situ chemical alteration prior to closure

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Abstract

The key to optimal metal recovery from a cell on a heap leach pad is to match the solution application rate to the permeability of the material for the duration of the leach cycle and to maximize the grade of the solution flowing downward from the cells to collection. Material properties and concepts of solution flow will be presented, demonstrating how operators may optimize solution management and metal recovery, thus reducing recoverable metals found in inventory.

A heap is mature when the ore is no longer being placed on the pad. This is usually due to achievement of the permitted height, or the mine discontinuing ore excavation. Generally closure permits require the resloping of the sides of the heap prior to capping. This side-slope disturbance allows operations to leach these new surfaces. Designs and operational techniques are shown to recover additional metal values from this phase of leaching.

Mature heap leach pads may contain about 6% of the recoverable metal values even after the final rinse operation. The heap may contain zones where the chemistry affects the quality of the drain-down solution as meteoric water passes through the pad. Hydro-Jex^{TM} is a 3-D leach technology invented to reduce heap rinse time, change the chemistry of targeted zones, and recover the stranded metal inventory from the pad. This technology is used widely in the United States and is available to aid operators worldwide during leaching and rinsing before heap leach closure. Examples of the use of this technology are presented.

Introduction

Nearly 10% of the world's gold production is from heap leaching operations − 236 metric tons gold/yr (Marsden, 2006). Thus, heap leaching is an important process for the mining industry.

After working many years as a metallurgical manager for several large heap leach operations and utilizing the experience to conduct a variety of field experiments and tests, the author has developed several techniques and methods that improved metal recovery at Carlin-type heap leach operations, thus

reducing the metal inventory and promoting rapid closure. These operational concepts and technology are presented.

Methodology

Solution application

Optimization of heap leach solution application involves managing unsaturated flow through unconsolidated and unsaturated particles with a wide range of particle sizes, from crushed to run-of-mine (ROM), and combinations thereof. The stacked ore has a variety of local void spaces or voidage. The rocks have internal particle cracks or micropores where diffusion of solution in and out via capillary action controls the internal metal dissolution. The optimal solution application rate provides just a thin fluid film on all the rock particles, with sufficient reagents to support reagent diffusion to the site of the target metal. With the proper concentration of reagents and fluid the target metal is leached. The dissolved metal then diffuses out to the thin fluid film on the rock surface as pregnant solution. Freshly applied solution then replaces the particle's fluid film and washes away, or sweeps the pregnant solution downward for collection. The applied solution contains fresh reagents, to begin the diffusion and leach cycle again. Adding too much solution will cause solution path bottlenecks to fill the voidage and force solution horizontally to the path of least resistance and cause channeling, while diluting the amount of the dissolved metal in the pregnant solution. Adding too little solution provides insufficient fluid film on each particle, resulting in a lack of sufficient reagents for optimal diffusion and dissolution of the target metal. This practice promotes slower leaching kinetics, or a longer leach cycle.

Figure 1: Schematic image of rock particles (VS), microporosity (ε), solution film (Vl), void (Vg) space (Bartlett, 1998)

The key to good solution management is to optimize leaching without dilution of the pregnant solution. Classifying the solution application rate to four distinct periods helps operators to understand solution management and optimize leaching. The periods are: ore wetting, leaching, rinsing, and drain down. Operational solution application experiments were done on a number of heap leach cells (91 × 91 × 9.1 m high) with crushed and ROM mixed Carlin-type ore. These operational experiments were modeled to optimize solution application rates. The models showed the best method was to regulate the initial application rate for wetting to 0.1 to 0.17 l/hr/m^2 (0.004 to 0.008 gpm/min/ft^2) for about seven days, which matched the average wetting hydraulic conductivity for the Carlin-type ores at this cell height. The ideal leaching cycle or second period had a solution application rate of about 50% of the wetting period. This was accomplished by turning off every other emitter line. Optimally, each emitter line was cycled on/off every few days to a week during the leach cycle. At the end of the leach cycle, the rinsing period was initiated by again turning on all the emitter lines for seven days to adequately rinse all the leached metal values from the particle fluid film, thus obtaining good sweep efficiency. This whole application rate scenario allows solution to be applied by timing the wetting, leaching, and rinsing period of the leach cycle to optimize metal recovery and obtain the highest grade of pregnant solution from the ore. As the ore size distribution, hydraulic conductivity and microporosity varies, so should the application rate for the 3 solution application periods, following guidelines modeled from large and small columns testwork, as shown in Figure 2.

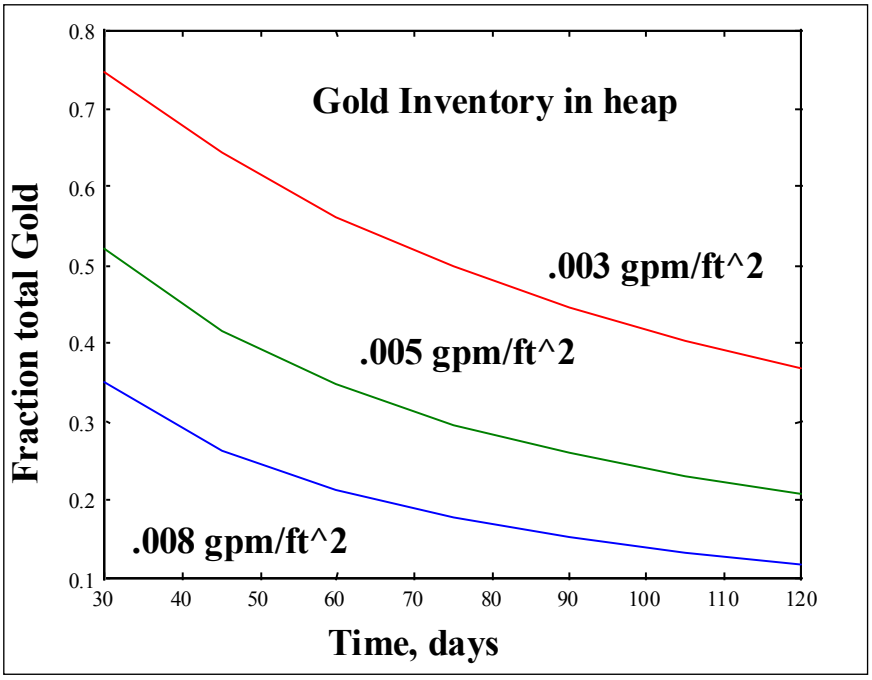

Figure 2: The fraction of total gold remaining in a heap after different application rates of 0.06, 0.11 and 0.17 liters solution per hour per square meter (Seal, 2004)

As the ore is stacked higher and a heap grows taller, the ore weight settles and the voidage at the lower lifts compresses with a resultant lower permeability. Figure 3 shows the change in permeability for a typical Carlin-type heap to a permitted height of 91 m.

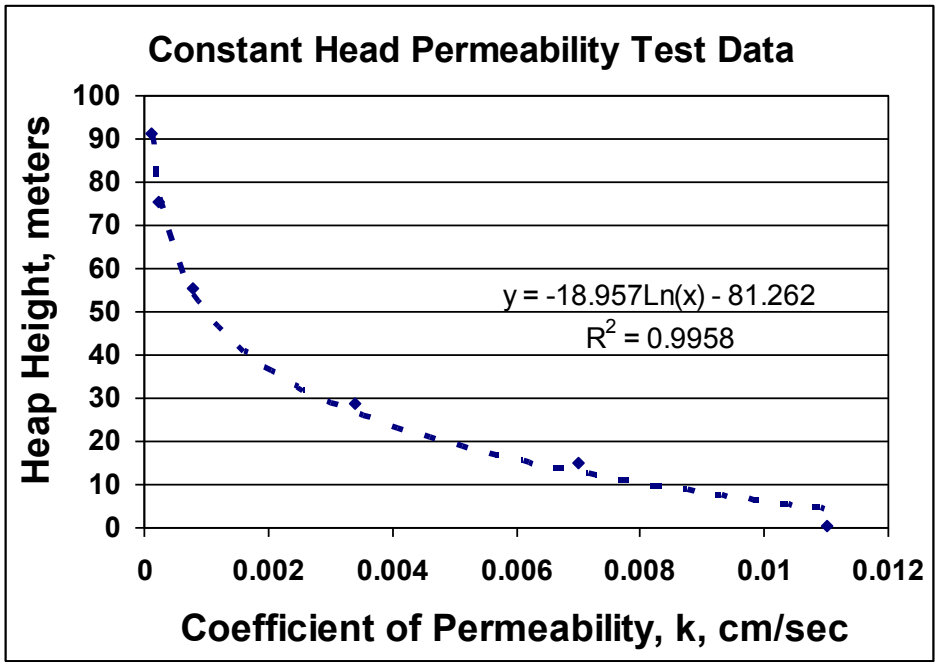

Figure 3: The coefficient of permeability in cm/sec as per depth in a heap, derived from laboratory testing of Carlin-type ore (Seal, 2004)

Reagent concentration in solution

Due to the variable chemistry of the material found in a given ore, it is difficult to determine the optimal set point for the concentration of cyanide in the barren solution that is added to the top of a cell on a heap leach pad. From lab column tests on a cell's ore type, reagent consumption data is collected and the total cyanide utilized is calculated during the test. The total cyanide utilized during the column test leach cycle, divided by the amount of solution applied, provides the heap leach operator with the initial solution cyanide concentration for the barren solution to be applied to the top of the heap. A method for optimizing cyanide addition was conducted by taking daily pregnant solution samples of the solution flowing from the bottom of the heap and tracking the quantity of free cyanide (CN^-). If sufficient free cyanide is flowing from the bottom of the heap, then sufficient cyanide is in the heap for leaching. Thus more cyanide is added to the barren solution, step by step until 1−3 ppm of free cyanide flows from the bottom of the heap in the pregnant solution. If more than 5 to 10 ppm of free cyanide is in the pregnant solution, then the cyanide solution added to the barren solution should be trimmed down. Too much cyanide in the pregnant solution will retard the CIC absorption efficiency (Marsden and House, 2006).

The operator must also consider the length of time it takes for the applied solution to come out of the heap, or break through, and only adjust the concentration of the cyanide periodically to account for the heap's conductivity or solution velocity through the heap. Experience on large Carlin-type heaps of 91 m height allowed monthly cyanide addition adjustment.

Side-slope leaching

The volume of ore found in the side slopes for a typical heap varies according to the heap design and height, which can be a substantial quantity. One heap on the Carlin Trend had about 30% of the ore in side slopes, which was not leached very well due to solution management and ore-stacking schedules. The closure plan for the heap required contouring to a 2.5:1 slope angle. Operationally, the side slopes were wetted with irrigation sprays to cover each lift and side slope, in order to reduce the metal inventory by re-leaching and evaporate solution accumulated in the spring months (Figure 4).

Figure 4: A picture of spray leaching and evaporation on a Carlin-type heap leach operation (Seal, 2003)

The side slopes were then allowed to dry, and during the summer months the slopes were re-contoured using a dozer, taking care not to overtop or puncture the liner under and adjacent to the bottom lift. This operation mixed, rechanneled, and exposed under-leached ore to be leached.

Figure 5: The dozer side-slope contouring of a Carlin-type heap leach operation (Seal, 2003)

Emitters were then strung vertically with special built point-source emitters and pressure regulators to keep an even flow of barren solution to the under-leached ore, as shown in Figure 6. Leach-pad crews would stage the tubing reels on top of the heap and walk down with several lines, placing the tubes to the bottom of the pad, and then take a ride to the top to keep the piping project moving quickly. Other operations used horizontally strung emitters. Daily recovery from the heap leach pad increased from an average 14 troy ounces (t oz) gold/day to an average of 80 t oz gold/day and a peak of 160 t oz gold/day when the side slopes were under leach (as shown in Figure 11).

Figure 6: Side-slope leaching with vertical emitters and pressure regulators on a Carlin-type heap leach operation (Seal, 2003)

Hydro-Jex

A heap is labeled "mature" when the ore is stacked to the permitted height and leached. Reagents continue to be added to the barren solution to remove inventory from the heap and side slopes. The metal recovery drops to a small consistent average value that pays for the reagents and pumping costs, but before rinsing and evaporation of solution prior to closure. Drilling heaps with hole sample analytical data analysis (such as mine bench kriging) can determine the location and quantity of metal in inventory as shown in Figure 7. Experience on heap leach pads located on the Carlin Trend showed that about 6-8% of the recoverable gold values remain in the interior of the heap. This inventory is due to stacking schedules and solution management, compaction under haul roads, incomplete ripping of the surface of the cells before leaching, chemical precipitation in the heap, clay zones or ore that is not adequately agglomerated, heap settlement, migration of fines, and compaction (Seal, 2008).

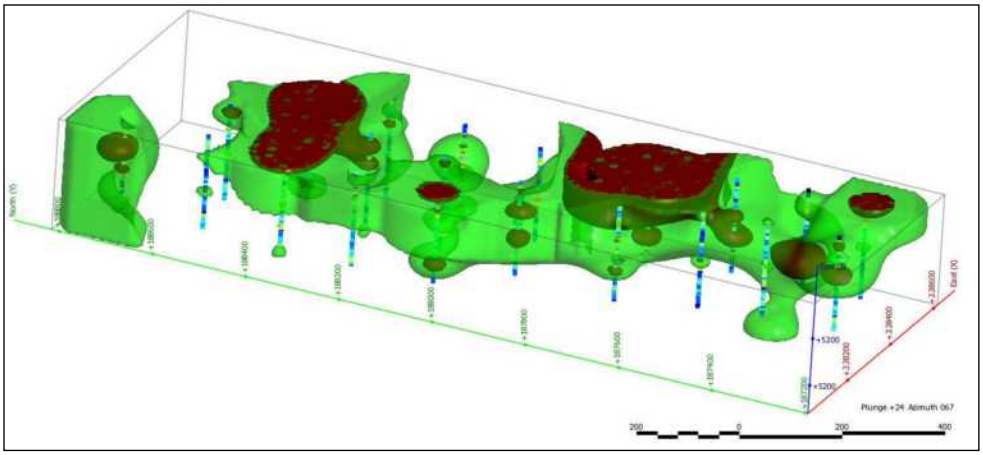

Figure 7: Kriged gold assay data from drill samples identifying under-leached zones in the heap interior for a Carlin-type heap leach operation, green 0.34 g/MT, red 0.48 g/MT (Seal, 2008)

The key to recovering this metal inventory is to transport the lixiviants to the unleached or under-leached zones to leach the metal values and rinse the pregnant film off the individual rock surfaces.

Geophysical resistivity measurements were conducted on the surface of the heap leach pad to identify the interior dry zones (noted in red) with under-leached inventory, as shown in Figure 8.

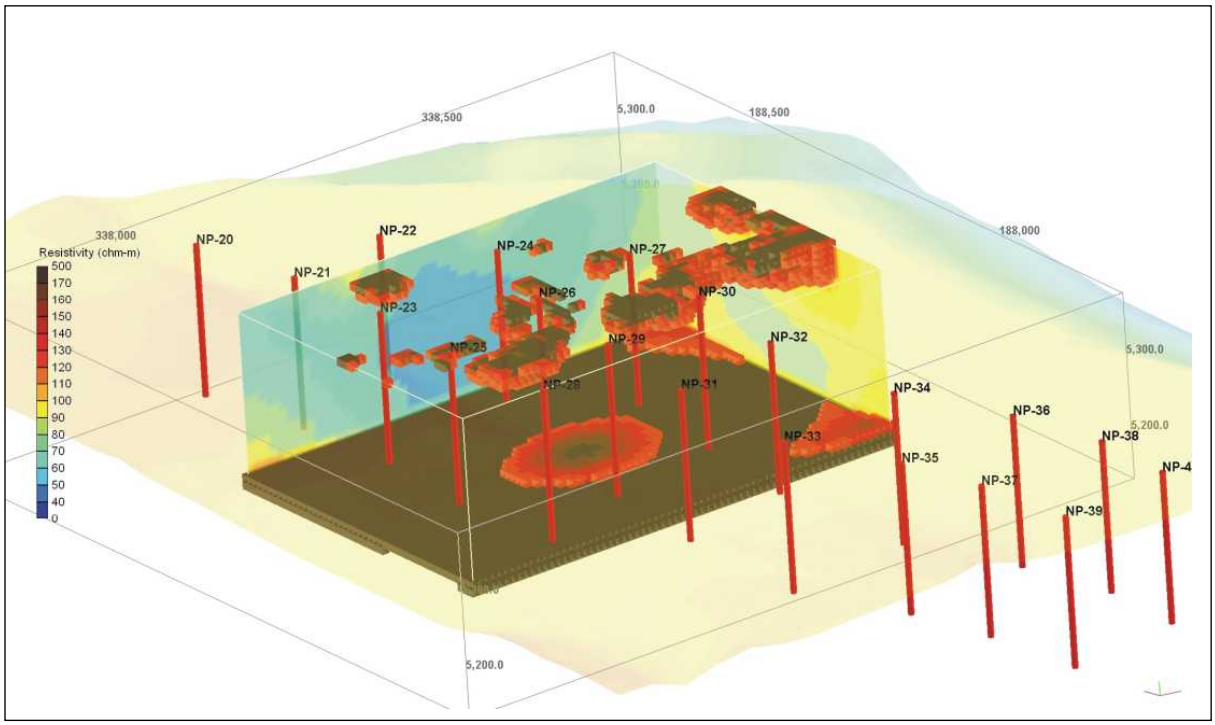

Figure 8: Plotted geophysical resistivity data from a surface array identifying under-leached zones in the heap interior for a Carlin-type heap leach operation (Seal, 2008)

The challenge of leaching these interior dry zones was solved by the use of Hydro-Jex, which was invented and patented as a doctoral research project (Seal, 2004). The technology basically involves drilling and sampling a heap leach pad and installing a well with zonal perforations. The zones are isolated using standard drill tools, and high-pressure solution is pumped in, to open solution pathways and channels, achieving 3-D leaching. In addition, any pumpable solution or slurry can be metered into a specific location in the interior of the heap.

Due to the encasement of all the solutions delivered into the heap's interior, strong reagents can be used without surface wildlife exposure. The reagents are mixed and diluted with other fluids that make up the pregnant solution flowing from the heap. The horizontal component of the solution profile ranges from a 20 to 30 m radius depending on the depth of the targeted zone and the size distribution of the rock in the heap. Geotechnical studies show a slope stability factor of greater than 1.5 during the pumping and the follow-up rinse process (Seal, 2008).

Figure 8: Hydro-Jex operations on a valley-fill heap leach operation (Seal et al., 2011)

The injection pump is initially horizontal with very little vertical component, until in-heap solution friction allows gravity to slope the pumped solution profile as shown in Figure 9, where sensors were placed adjacent to the Hydro-Jex well. With low internal pH in the heap, milk-of-lime slurries were added to the injection operation to change the pH in the zone and alter the pregnant solution pH, as shown in Figure 10. In a normal gold mining heap, the voidage has sufficient oxygen to dissolve into the fluid film encompassing the rock particles.

As heap leach pads reach higher elevations, the voidage volume decreases and if organic carbon and/or sulfides are present in the heap, then the oxygen concentration in the voidage decreases or is non-existent. Leaching gold and silver requires dissolved oxygen and if oxygen is depleted, then leaching kinetics are reduced. The Hydro-Jex technology mixes sparged air into the injection fluid to reach 11−13 ppm dissolved oxygen, as determined in bench scale tests (Seal, 2004).

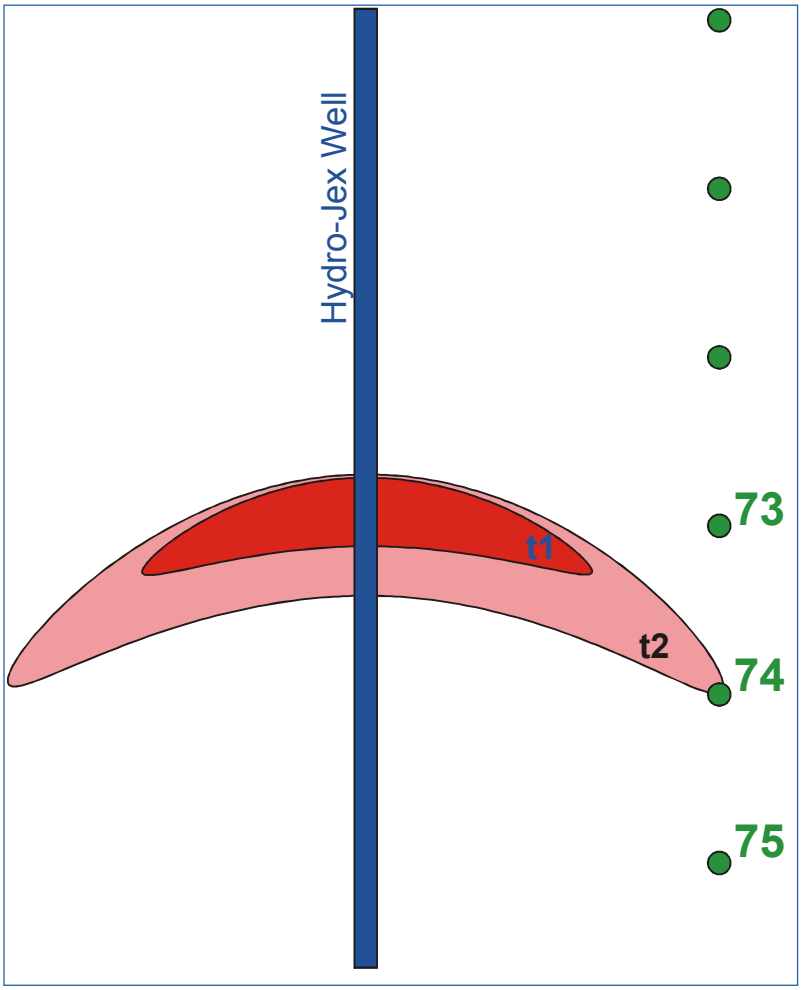

Figure 9: Hydro-Jex pumping plume in a valley-fill heap leach operation (Seal et al., 2011)

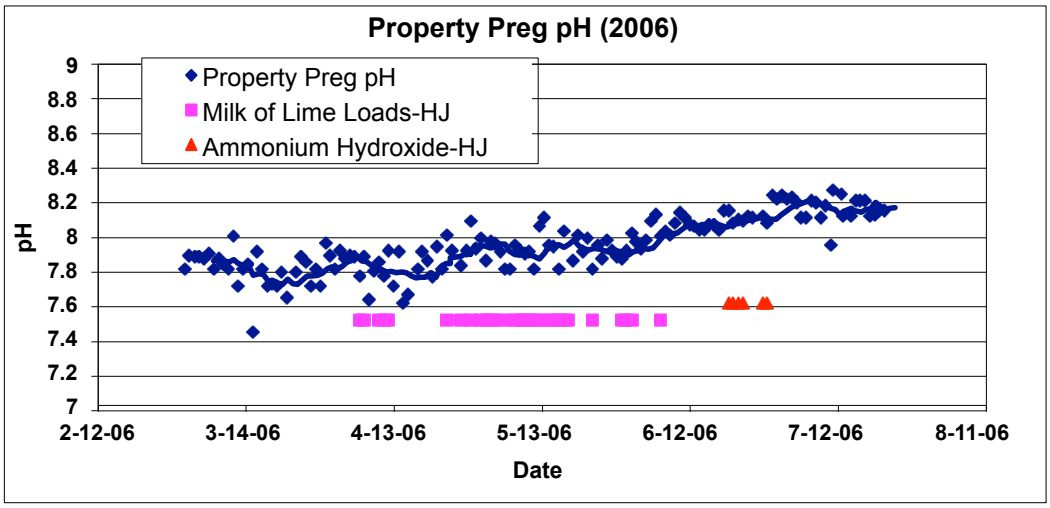

Figure 10: pH changes in the pregnant solution with to milk-of-lime and ammonium hydroxide injections into the heap interior via Hydro-Jex (Seal, 2008)

Post injection, the wells are revisited periodically with barren solution to each zone to rinse the dissolved metal values and re-establish fresh reagents using the channels established during the injection pumping. This system can operate concurrently with the closure plan and can shorten the rinse time, while recovering previously unrecoverable metal inventory. This allows operators to recover the reclamation bond earlier.

Conclusion

The main operational key to heap leaching efficiency is to recover the maximum amount of metals from the heap in the shortest time. The effect of side-slope recontouring and re-leaching is displayed in Figure 11. Gold production from Hydro-Jex wells is shown in Figure 12: at Lone Tree an average of 15.2 t oz gold was recovered per meter of a Hydro-Jex well during stimulation and rinsing. Prior to the stimulation and Hydro-Jex pumping, the weekly gold production was about 10 t oz per week (Seal, 2008).

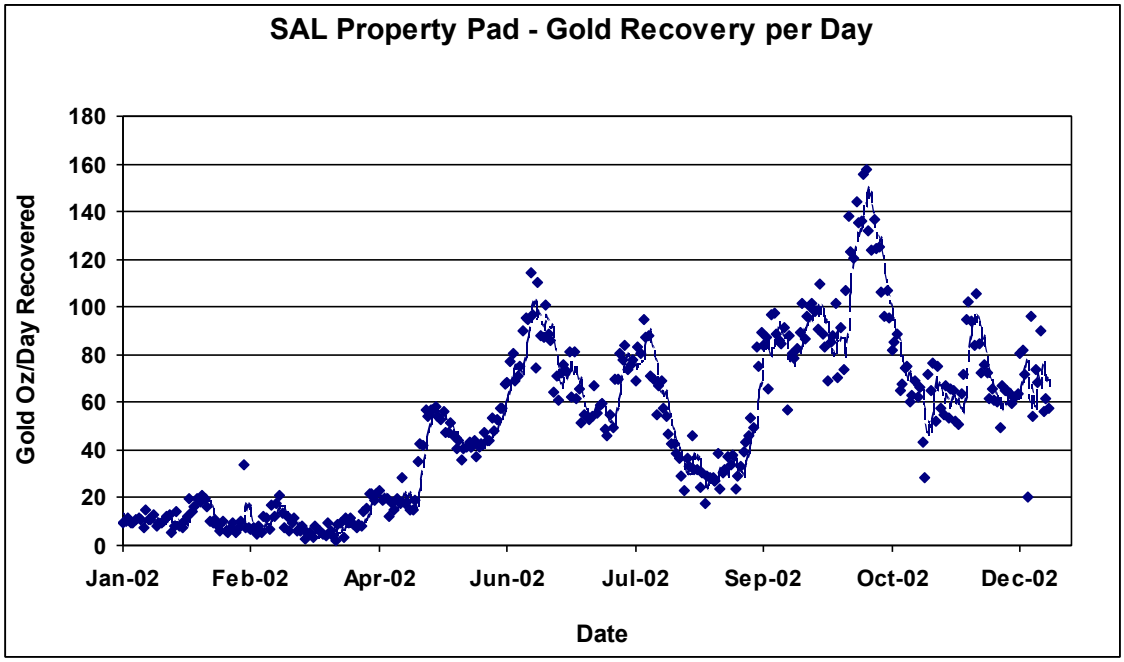

Figure 11: Increase in gold production due to side-slope recontour and re-leaching (Seal, 2003)

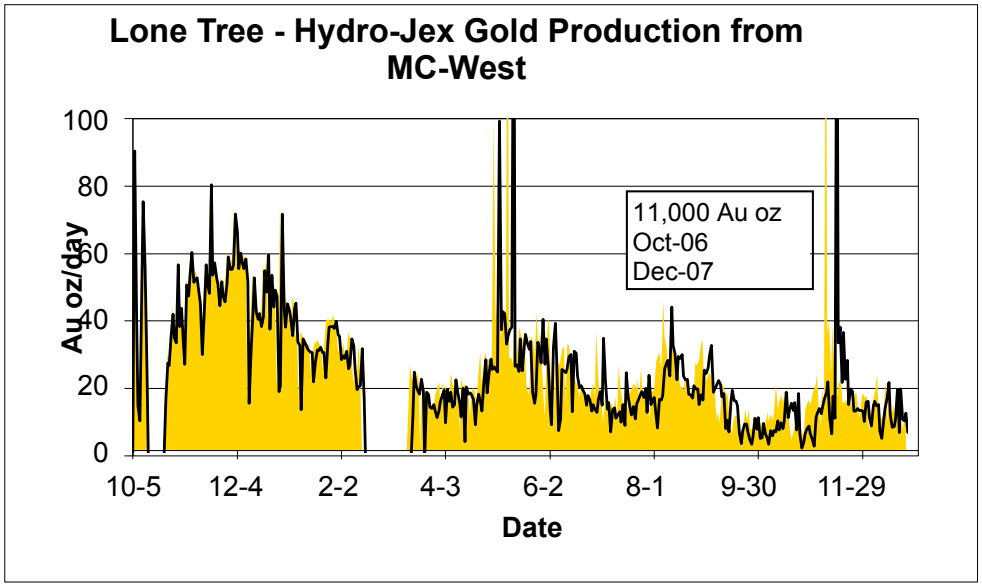

Figure 12: Increase in gold production from 10 t oz per week due to Hydro-Jex stimulation on the Lone Tree MC West pad (Seal, 2008)

Heap-leach operators have a wide variety of techniques to maximize gold production by optimizing solution management, recontouring and releaching side slopes, and by using the Hydro-Jex technology to target and stimulate dry zones of underleached inventory of recoverable metal values and to change the internal heap leach chemistry while shortening the rinse and closure cycle. In addition, Hydro-Jex offers a new solution management tool for operators of heaps.

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Development of accurate metal production forecasts for a heap leach project using METSIM®dynamic simulation and defensible column leach testing data

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Abstract

Development of precious metals or copper heap leach projects typically follows a sequence of steps, from feasibility study through engineering, detail design, construction, and operation. Critical to the success of these projects is correct estimation of leach kinetics and heap design. The column leach tests performed in the laboratory to simulate the heap leach process are conducted on representative samples of the ore to be processed. The metallurgical data developed include the leach kinetics, which are determined using optimized leaching parameters. However, the leach data developed in a metallurgical laboratory do not typically reflect the true leach cycle and contact times experienced in the actual heap leach process.

Dynamic simulation of the heap leach process allows operations personnel to fill the gap. METSIM metallurgical simulation software is able to dynamically model heap leach processes for copper and precious metal ores. Incorporating defensible metallurgical laboratory column leach test data into a METSIM model allows for development of accurate metal extraction projections. These metal extractions are useful to forecast production on a monthly and annual basis and they may be part of a comprehensive dynamic model of a life-of-mine (LOM) heap leach. Reliable laboratory column leach test data and METSIM heap leach modeling provide a predictive tool that has proven valuable in the development and operation of heap leach projects.

Introduction

One of the most challenging tasks at a heap leach operation is the development of a production forecast. This information is usually required by management to set benchmarks for production. The amount of precious metal produced is directly proportional to the volume and tenor of the pregnant solution processed in the metal recovery plant.

The metallurgical department develops the precious metal leach kinetics by means of column leach testing. In this type of study, column tests are used to evaluate a heap leach process using a batch type of testing methodology. It is impractical to conduct continuous column leach testing for evaluation of a heap leach process in the laboratory. Therefore, a combination of column data and computer software must be employed for simulation of a heap leach process. PROWARE® has developed metallurgical simulation software for precious metals heap leaching. This paper outlines a methodology for developing defensible metallurgical data that may be used to develop accurate production forecasts using METSIM software.

Leach kinetics estimation

Practical aspects of a column leach study

Important technical and economic variables associated with precious metals heap leaching are well known, since this technology has been fully commercialized. The testing procedure employed by commercial firms tends to parallel processes utilized by large mining companies, since the unknowns relative to leaching chemistry and economics are the same in both cases. A typical testing program consists of mineralogical characterization, sample preparation, and leach testing. The test procedure may be simple or complex, depending on the desired outcome of the testing.

Sample origin

A project under development usually employs a competent geologist to identify and define ore horizons. In many instances, a commercial testing firm must rely on the client geologist to provide a description of the ore mineralogy. In these cases, it is of utmost importance that the testing personnel specifically state the condition of samples received and fully specify all sample composite instructions; this ensures that samples prepared for testing are representative of ore horizons in the deposit.

Ore mineralogy

The specific mineralogical nature of the samples to be tested may be thoroughly defined, completely unknown, or somewhere in between. If no mineralogical information exists, it is prudent to examine the selected samples megascopically and microscopically. This information contributes specific knowledge to the subsequent precious metal extraction task by identifying the existence of minerals that could impact processing decisions. Host rock components should also be identified, along with the consequent potential for reagent consumption. It is important to characterize the mineralogy of sulfide ores, since sulfide minerals tend to be associated with gold and exhibit distinctive leaching characteristics.

Initial ore characterization

Samples may be received in a variety of forms, including unsorted bulk samples, large and small diameter diamond cores, and percussion diamond drill cuttings. Samples that are used to replicate heap leach response in test columns are contacted for specific time periods with a defined lixiviant concentration. Characterization of samples is therefore critical to testing, and different samples will require slightly different pretreatment for column leaching.

Every ore type identified in the deposit should be subjected to an ore characterization test prior to inclusion into the column leach program. The initial characterization test varies from organization to organization, but usually consists of a bottle roll low cyanide concentration leach test conducted for 24 to 72 hours and using a sample split that has been reduced to a size distribution between 4,750 and 106 microns. This test yields important data about the expected ultimate precious metal extraction from the sample and the potential maximum reagent consumption. The metallurgical data allows testers to select initial test parameters to minimize the number of column leach experiments required for the project.

Bottle roll studies provide information about the maximum reagent consumption characteristics for different ore types. Experience has shown that, for many typical materials, the amount of sodium cyanide consumed in an industrial-scale leach operation will be a fraction of the quantity of cyanide consumed in a minus106 micron bottle roll test.

A refinement of the characterization bottle roll leach study is conducted in small diameter columns. These mini-columns are typically 75 mm in diameter and 1.6 m tall. Each column is filled with a carefully prepared sample consisting of approximately 10 kg of minus 12.5 mm crushed material. Columns are arranged in a matrix fashion and allow researchers to study a number of variables, including the following:

- agglomeration or no cure agglomeration;
- quantity of cyanide or binders added to cure; and
- cure time – the rest period after cure treatment.

The mini-columns are operated for 14 to 28 days (2 to 4 weeks), during which researchers macroscopically study the physical condition of the ore under different leaching regimes. Usually, one or two standard tests that are not treated with strong lixiviant prior to leaching are included in the experimental matrix for each ore. Behavior of the effluent solution during the rather short mini-column leach cycle helps to define the criteria for larger test columns, thereby reducing the number of large diameter column tests to a minimum.

Sample preparation for a column leach study

Ideally, the crushed ore sample in an actual column study should replicate a hypothetical differential increment column in a heap. In order to approach this ideal situation as closely as possible, the test sample should be sized and deposited in the column in a manner that avoids segregation.

The procedure used to reduce the size of test samples is important. Sloppy sample preparation procedures can result in the generation of confusing column leach data. A recommended sample preparation procedure includes crushing the entire test sample through the coarsest top size and screening the entire sample on a stack with a minimum of five sieve sizes. Each fraction should be stored separately in a covered drum. Individual column test charges are reconstituted by splitting representative sample weights of each sieve fraction and mixing the resulting smaller fractions.

Since the head sample assay will be used to determine the precious metal recovery, it is recommended practice to obtain the assay sample from individual fractions at the full weight at the standard column charge. For example, if a full column charge is 90 kg (typical for one 20 cm diameter by 1.8 m tall leach column), and if the minus 50 mm by plus 25 mm fraction is 12 weight percent, then 11 kg of minus 50 by plus 25 mm material should constitute a representative split of the original assay sample. A typical procedure involves reducing the entire 11 kg sample to 1.7 mm, thoroughly blending the minus 1.7 mm sample by repeated riffle splitting and then pulverizing several one kilogram riffle splits through 150 micron for duplicate assay. This procedure tends to be laborious. However, it provides reproducible feed size fraction assay samples. As well as crushing, screening, and separating size fractions from the original sample, it is recommended that finer crush sizes be developed from the master samples. This can be performed by stage crushing the proper weight of oversized material and redistributing the crushed product screen fraction. This entire procedure tends to replicate the particle size distribution that is experienced in industrial practice.

One of the most important elements of precious metal heap leach studies is the ability to generate duplicate ore columns with exactly equivalent size distributions. Consequently, a sample preparation procedure that will yield equivalent column samples from a master ore composite must be employed. It can be difficult and time consuming to produce equivalent size distributions for laboratory testing. Usually, the common cone-and-quartering sample splitting system is not adequate. Columns of exactly equivalent size distribution can only be assured if the dry master test sample is screened into specific sieve sizes and the charge for each column is reconstituted by reference to the direct weight of these fractions.

The geological and mineralogical character of most economic precious metal ore bodies changes substantially at different ore horizons. These differences may radically influence leaching characteristics. For a large ore body, it is prudent to perform column tests at equivalent size distributions on all major ore

types. This means it is necessary to develop a preliminary mine plan to ascertain the schedule in which various ore horizons are to be extracted. It may also be useful to prepare composite samples which represent one-, two-or three-year increments of ore production. If development of the ore body is to be partially or fully financed, lending institutions are always interested in the sequence of cash production for the project.

Sample agglomeration

Agglomeration is a condition in which individual particles of a crushed mineral product loosely adhere to one another. Complete agglomeration is a condition in which even the finest particulate fractions present in a sample are incorporated into loosely bound aggregates.

A simple laboratory agglomeration procedure consists of placing the individual size fractions of the sample in a bucket or small cement mixer with a binder, such as Portland cement and lime, water (or a cure solution), lixiviant solution, and/or surfactant reagents, while slowly rolling the solid material. After a certain quantity of liquid has been added (usually between 4 and 10% by weight) the solid material will form loosely adhering particulate aggregates or agglomerates. This agglomerated material can be charged carefully into a column of any height avoiding segregation within the column.

It is recommended that initial tests should always be conducted using agglomerated ore samples, especially if it is anticipated that ore will be crushed prior to leaching. Operating permits usually contain regulations regarding levels of particulate emissions surrounding the size reduction and materials handling facilities. As a result, it could be argued that ore samples should be tested in both agglomerated and non-agglomerated conditions.

Particulate (dust) emissions are commonly abated by applying water or barren solution directly on to the material being crushed. These solutions are usually added as a mist or spray at truck dump pockets and material handling transfer points. The quantity of water used is typically judged by the disappearance of dust at the point of application. This water quantity is also the level at which crushed material becomes agglomerated. For this reason, it is likely that ore that is reduced through at least a primary crusher will reach a heap leach pad in partially or fully agglomerated form.

If the ore to be leached is crushed through a single primary stage, this provides an opportunity to add a cure solution. In actual practice, the cure solution may be applied at any cyanide concentration consistent with the maximum concentration desired in the pregnant solution. This solution may be added as a spray at conveyor belts, at transfer points, or in an agglomerating drum.

In the laboratory, a precisely measured volume of cyanide solution is carefully sprayed directly into the agglomeration device prior to charging the ore into the column. The optimum quantity of sodium cyanide added in the cure pretreatment can only be determined through column experimentation. The cure

dosage will depend upon crush size and ore mineralogy (sulfide, oxide, or mixed). Mini-column testing will reduce the number of large-scale tests required for clear definition and optimization of the important leach parameters.

Column leach test procedure

The column leach test procedure is relatively simple. Lixiviant solution (artificial or mature) is introduced at a constant flow to the top of the column and collected at the bottom. In larger diameter columns, the lixiviant may be distributed equally over the surface of the ore. The lixiviant solution must be delivered at a constant flow rate at all times for the duration of the test program.

Many different devices have been used to feed the lixiviant solution to the column, including intravenous drip apparatus, syringe pumps, constant head tanks (with or without timing devices), peristaltic pumps, and various types of positive displacement pumps. Although more expensive than most solution application devices, electronically actuated positive displacement diaphragm pumps are recommended for column irrigation.

The leach solution percolates the column and discharges as effluent solution. The effluent solution is gathered on a periodic basis, the volume is measured, and it is then assayed for elements and ionic species of interest. The test procedure is terminated either after a specific number of days have elapsed or when the column effluent no longer contains measurable quantities of soluble precious metal. The column is then drained, washed with water, and discharged. The leached residue is usually dried, weighed, screened into specific sieve fractions, and prepared for assay to determine insoluble losses.

Column testing can produce large amounts of data. The usual procedure is to use a computer spreadsheet and database management software to store and manipulate this information. Handling information in this manner is convenient for experimenters since interim progress reports are generated in a straightforward fashion.

At the termination of the leach test, the final column height is noted together with volumetric information associated with the wash and drain cycles. The leached residue is screened and assayed for elements of interest, particularly gold and silver. The column test information provides important test parameters, such as overall precious metal extraction and reagent consumption. Manipulation of this information also allows for the development of particle size degradation.

Test variables and parameters

Critical leach parameters commonly evaluated in the course of a column leach study include cure solution rates, application, and concentration. The cure solution may be added by a top-down methodology if it is anticipated that the ore will be leached at a run-of-mine (ROM) particle size distribution. If it is anticipated that the ore may be crushed, then the cure may be added during the agglomeration procedure.

The optimum cyanide quantity and concentration for the cure technique must be determined by experimentation. For many ores, the expenditure of time and money associated with this experimentation is worthwhile. A substantial reduction in the cyanide consumption relative to leaching without the cure technique may be realized as a result of the tests. The initial cyanide dosing used in the cure test is usually derived from the ore characterization test previously described.

The aging of ores upon extraction influences cyanide consumption and cure application. The impact of cure aging can only be investigated through test leach columns. On some silver ores, a considerable enhancement in silver disengagement rate and percolation characteristics have been observed when the ore columns are aged for periods from 5 to 15 days after a cure pretreatment. The application of binders improves porosity in the agglomerated material, assisting the cure process further.

Crush size

Commercially available crushing equipment tends to produce specific size distributions when operated in open circuit. These size distributions can be modified (usually at the expense of throughput) by operating the same crushing devices in closed circuit with classification screens.

Separation efficiencies on grizzlies and screening equipment are imposed by the ore size distribution, hardness, and moisture content. Recall that most modem crushing plants are required to operate within strict dust emission guidelines, and this restriction results in the production of a moist final product which is partially or fully agglomerated. The final product from each crushing stage depends upon the specific equipment assemblage, which is usually a function of design throughput. Typical top size distribution ranges have been developed through experience and are available in the literature provided by crushing equipment manufacturers.

Commercial crushing equipment tends to produce a rather narrow range of sizes, depending upon the circuit configuration selected. Ore sample top sizes are limited to100, 50, 25, and 12.5 mm, which covers the size ranges that are readily produced by conventional crushing facility equipment arrangements. If the quantity of available sample is limited, the test charge can be crushed to 12.5 mm top size and subjected to the previously described standard column leach procedure. If an acceptable extraction level results from this simplified test, coarser size distributions can be investigated when additional sample becomes available.

The data generated by size distribution studies may be presented in a number of formats. One option is a bar chart presenting the percentage of precious metal extraction by screen fraction. A depletion in weight for each of the coarser size fractions when leach feed is compared to leach residue is a common observation in column leach analysis. Presentation of data in this fashion will allow for development of economic parameters, which will indicate the optimum crush size.

Leach testing on different size distributions provides other valuable data as well. A plot of the percentage of precious metal extracted versus leach time for two separate ore size distributions may be prepared for comparison. Analysis of these data allows researchers to predict the precious metal disengagement rate and develop optimum heap placement and irrigation cycles.

Column diameter and height

The diameter of the test leach column used and the height to which the column is filled are influenced by several considerations. If only a small quantity of sample is available, small columns are used to conduct scoping studies. This situation is common when the only sample available from a specific ore horizon is derived from a small diameter diamond drill core. For test material top size limited at 12.5 mm, samples as small as 4 kg in total weight can be studied in 50 mm diameter by 1.2 m tall leach columns.

A rule of thumb relating maximum ore particle size to column diameter has been derived from sampling theory. In general, the diameter of the leach column should be three to four times larger than the maximum dimension of the largest particle in the test charge. Columns having the same size distribution but differing heights may exhibit different precious metal extractions with time and always have different pregnant liquor assays. When larger quantities of ore become available, a more or less standard leach test requiring 90 to 360 kg per column charge can be conducted in 150 to 300 mm diameter columns with an ore height varying between 1.5 and 3 m. Larger scale testing is always necessary and recommended to fully define all the leach parameters required to develop the design criteria for the project.

It is not unusual to conduct 6.0 m tall corroborative leach tests in columns up to 1 m in diameter. Test charges for this type of study will vary between 300 and 6,400 kg. These large-scale tests are designed to yield sufficient effluent (10 to 80 L per day) to operate an associated metal recovery system to fully quantify expected precious metal recovery and reagent consumption. Although column studies can be conducted on ore samples as small as 4 kg, the development of engineering design criteria requires the acquisition of larger ore samples. Leach columns ranging from 0.2 to 1 m diameter by 6 m tall charged with approximately 300 to 6,400 kg of ore typically replicate commercial results. Columns larger than this size are commonly used to corroborate previous studies or to produce a large quantity of pregnant solution for subsequent precious metal extraction testing.

Irrigation rate

There has been a concerted effort in the precious metal heap leaching industry to diminish the average heap irrigation rate. Many gold and silver properties irrigate the heap at a higher flow rate (for example, 12 L/hr/m^2) for an initial 14 to 21 day leaching period in order to quickly recover the most amenable portion of the precious metal content of the ore. After this time has passed, a lower irrigation rate (sometimes as low as 1.2 L/hr/m^2) is then utilized for the remainder of the leach cycle. This procedure

tends to maximize the precious metal assay and minimize the volume of the pregnant solution to be treated in the metal recovery plant.

Strategies required to minimize the pregnant solution volume produced from precious metal heap leach operations have included cyclic altering of the irrigation rate as mentioned above, intermittent (on-off) leaching schemes, and recirculation of solution generated near the end of the leach cycle through an intermediate solution catchment. All of these strategies can be replicated in the laboratory, provided that there is a sufficient quantity of representative ore available to constitute the required number of necessary leach columns.

Although the typical irrigation rate used for precious metal leaching is 12.2 L/hr/m^2, specific applications at volumetric flows greater or less than this rate are well known. It is necessary that each ore under study be subjected to scoping tests, which involve variation of the irrigation rate. It has been demonstrated through column experimentation that, for certain materials, there is an optimum solution application rate at which precious metal extraction is maximized.

In a column leach test operation, the flow rate is the single most difficult variable to continuously control. Utilization of small columns (those less than 100 mm in diameter) necessitates the employment of very low flow rates. If the proper irrigation rates are to be achieved, maintenance of these low flow rates over a period of weeks or months is also required for a successful column leach test. Frequent surveillance and expensive pumping equipment will usually maintain a flow rate within 5% of the desired level. If substantial variations in daily flow rate are experienced, only cumulative flow may be reported to mask the daily flow differential. Consequently, daily flow data must be included in the experimental report so that an independent judgment can be made relative to the precision of the irrigation rate.

Lixiviant concentration

Column leach tests are also useful in evaluating the cyanide concentration used in a heap leach process. Study of this parameter will result in significant savings in sodium cyanide and enhanced leach kinetics. Silver dissolution has been consistently shown to be sensitive to cyanide concentration. Silver mineralization requires a cyanide concentration of at least 500 mg/L free cyanide. Gold dissolution, however, may be achieved at lower cyanide concentrations.

Heap design criteria

Leach pad capacity

Before dynamically simulating a heap leach, a three-dimensional structure representing the ultimate heap design must be generated. METSIM has the tools to either build a block model directly, through manual input of the size and shape of each lift, or to import survey and/or AutoCAD drawings. Upon completion

of this "wire-frame" heap, METSIM then uses the material specific gravity and bulk density, along with the volume of the heap, to calculate the total solids capacity. In the case of dynamic or on/off pads, this capacity represents the total live capacity of the heap. Figure 1 shows the plan view of two heap configurations in METSIM. A permanent stacked pad is shown in the bottom right of Figure 1. The permanent pad was generated using AutoCAD contour lines.

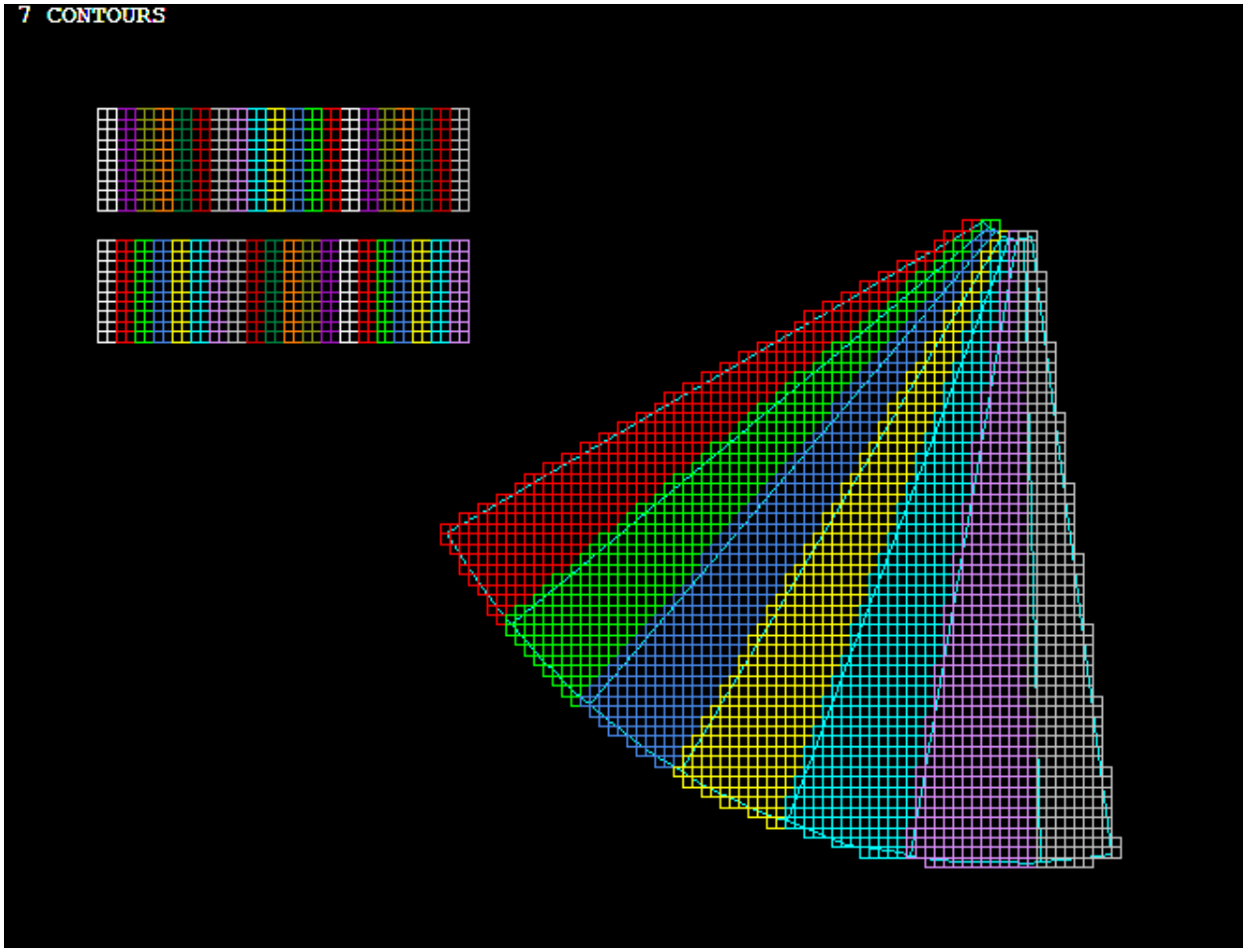

Figure 1: Example of a plan view of two heap configurations in METSIM. An on/off configuration built through manual input of cell size and number of cells is seen in the top left portion of the screen

Leach cycle

As Figure 1shows, the heap is broken into color-coded cells. A cell represents an area of the heap that is to be piped and leached as a single unit and may or may not drain to a specific location. Each cell in METSIM is assigned a leach cycle, which may include up to 15 different steps, and includes the length of time and application rate, as seen in Figure 2.

- Heap Leach Cycle Definition

CN 1 * Cycle Number

CO 3 Maximum flow in cubic meters per hour * Control Option

	Days	Stream	Rate	Limit	Label	Step
N,L	3	0	0	0	Piping	1 Col 1 = duration, days
N,L	35	702	0.05	1050	ILS	2 Step 1 controls the
N,L	10	0	0	0	Drain	3 start of the cycle.
N,L	60	703	0.07	1300	BLS	4 <999 Start immediately,
N,L	10	0	0	0	Drain	5 ≥999 Start when needed
N,L	15	704	0.03	350	Wash	6 to maintain limit.
N,L	10	0	0	0	Drain	7 + Newest blocks first
N,L	3	¯715	0	1500	Reclaim	8 - Oldest blocks first
N,L	0	0	0	0	0	9 Col 2 = stream number
N,L	0	0	0	0	0	10 Col 3 = spray rate
N,L	0	0	0	0	0	11 in lpm/m2 or gpm/ft2
N,L	0	0	0	0	0	12 Col 4 = Limit in blocks,
N,L	0	0	0	0	0	13 lpm, m3h, m3d, or gpm
N,L	0	0	0	0	0	14 See Control option.
N,L	0	0	0	0	0	15 Col 5 = Description

Figure 2: Example of a heap leach cycle in METSIM

As the heap is filled with ore during the simulation, the flows to and from each cell are controlled by the leach cycle. Flowrates are then recorded for all Intermediate Leach Solution (ILS), raffinate, drainage, and other streams for each day. Leach cycles may be created for each cell in the heap, or all cells may be assigned the same leach cycle, allowing the user complete control over all solution flows in the circuit.

Leach pad drain configuration

The control of each drainage stream may have a large impact on downstream plant designs and total metal recoveries, and may ultimately influence project success or failure. Using METSIM, numerous drainage configurations may be evaluated to optimize solution flows over the changing conditions of the project life. To best manage the drain design, METSIM allows for each cell to drain to a designated location or stream. For dump leach operations, this may be a single pond for all cells in the heap, whereas on/off operations may include a specific drainage pipe for each cell. Each of the drainage streams will witness varying flows and solution tenors for each day of operation and must be controlled accordingly. Figure 3 shows a heap leach process flow diagram in METSIM where streams 707,708, and751-770 represent heap drainage.

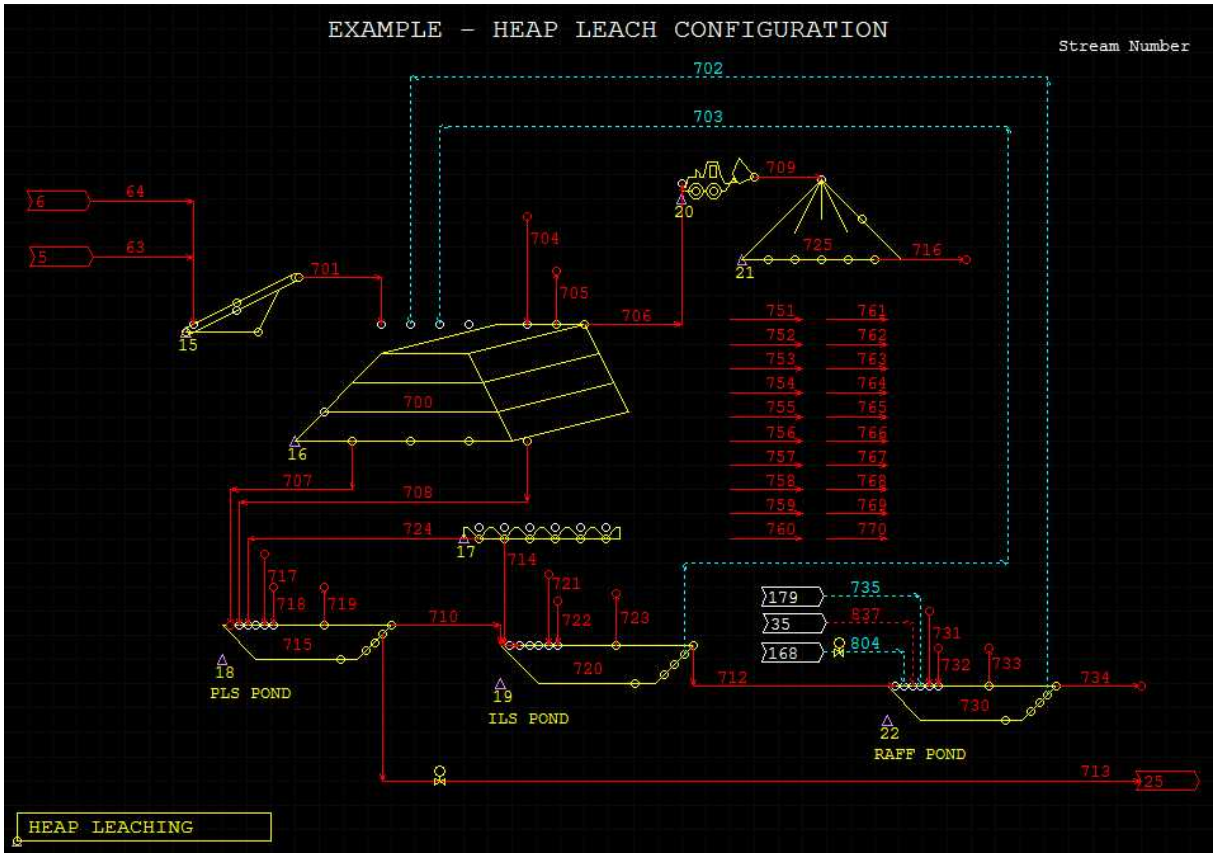

Figure 3: Process flow diagram for the heap leach drainage and pond system in METSIM. Streams 751-770 represent internal drainage streams assigned to specific cells

Using the heap leach drain unit operation (17 in Figure 3), drainage streams may be diverted based on cut-off grades, total flow capacities, or lixiviant stream number. Drainage rates and field moistures, obtained through testwork, are also included for each cell in the model. Dynamically simulating various piping designs and operating set points allows for thorough examination of all options and their results, therefore leading to better project performance.

Solution management and water balance

Once the design for the solution systems to and from the heap has been determined, the task of solution management and water balancing over the life of the mine is manageable. The lixiviant and drainage streams are recorded each day of the simulation as are all water make-up streams, daily precipitation to the heap and ponds, daily evaporation from the heap and ponds, and all recycle stream flows from downstream processes. Every unit and stream is balanced throughout the simulation. Appropriate plans can then be made to handle wet and dry seasons and major weather events.

Heap leach simulation

Leach kinetics extrapolation

Once the test-work has been completed and the heap system has been designed, these two critical pieces are merged to track solution tenors and metal recoveries. Initially, all column data is entered into METSIM. The best curve-fitting algorithm available is included in METSIM and used to determine the recovery rates based on the given data. For a single ore sample, various recovery rates are possible, and these have a major impact on the process. Figure 4 shows a curve fit performed in METSIM, where two recovery rates are applied to fit the data points.

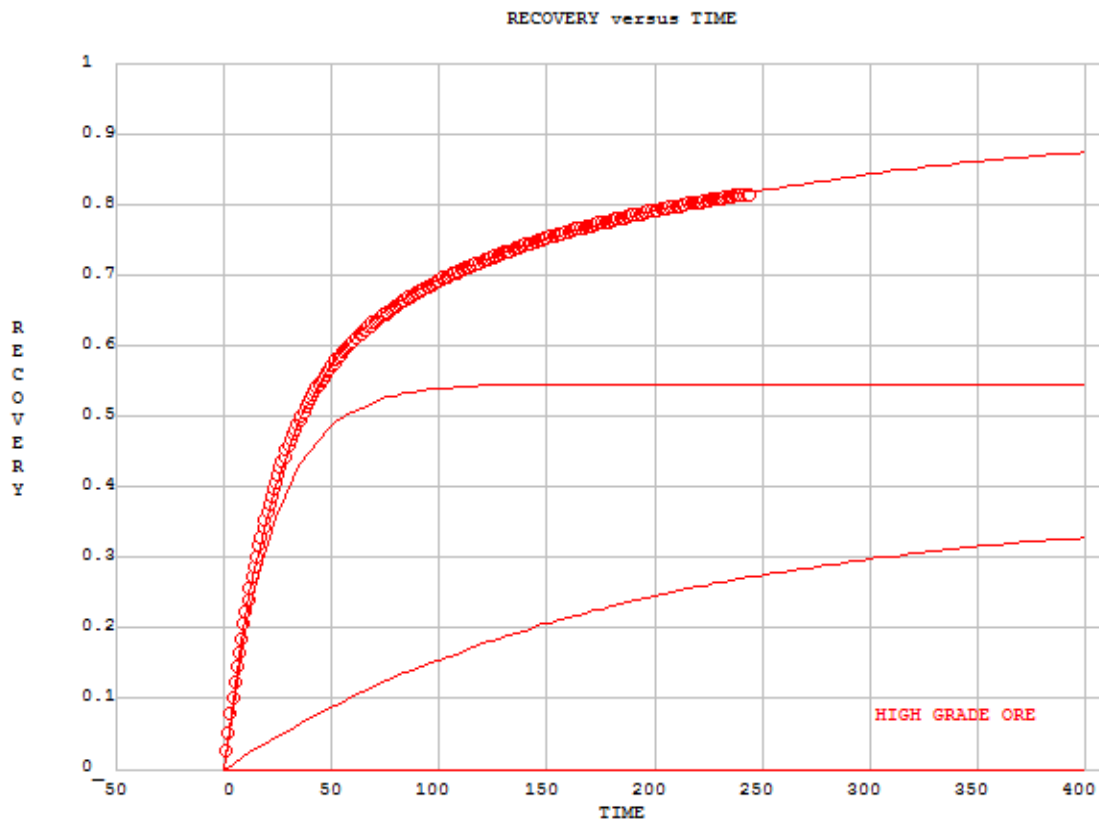

Figure 4: Curve fit in METSIM using two rates

Circles represent data points from column testwork, while the top curve-fit line is the overall recovery curve. The middle line in Figure 4 represents the fast rate recovery curve, and the bottom line represents the slow rate recovery curve.

Each recovery rate is then applied to a specific component in the model based on the ore type, particle size, or mineralogy. During the dynamic simulation, METSIM analyzes the solid and aqueous contents of each cell in the heap and applies the appropriate recovery rate. In the event that a reagent is not available (i.e., has been completely consumed), any reaction that requires that reagent will have no

recovery. This method of calculating and tracking reaction extents ensures that operating conditions are taken into consideration when determining ultimate metal recoveries.

When multiple tests on the same ore type yield slightly different results, it is necessary to further examine the cause of the differences. By plotting several tests against each other, as shown in Figure 5, it is possible either to determine the most representative data for calculating recovery rates, or to average the curves into a single curve that represents overall recovery.

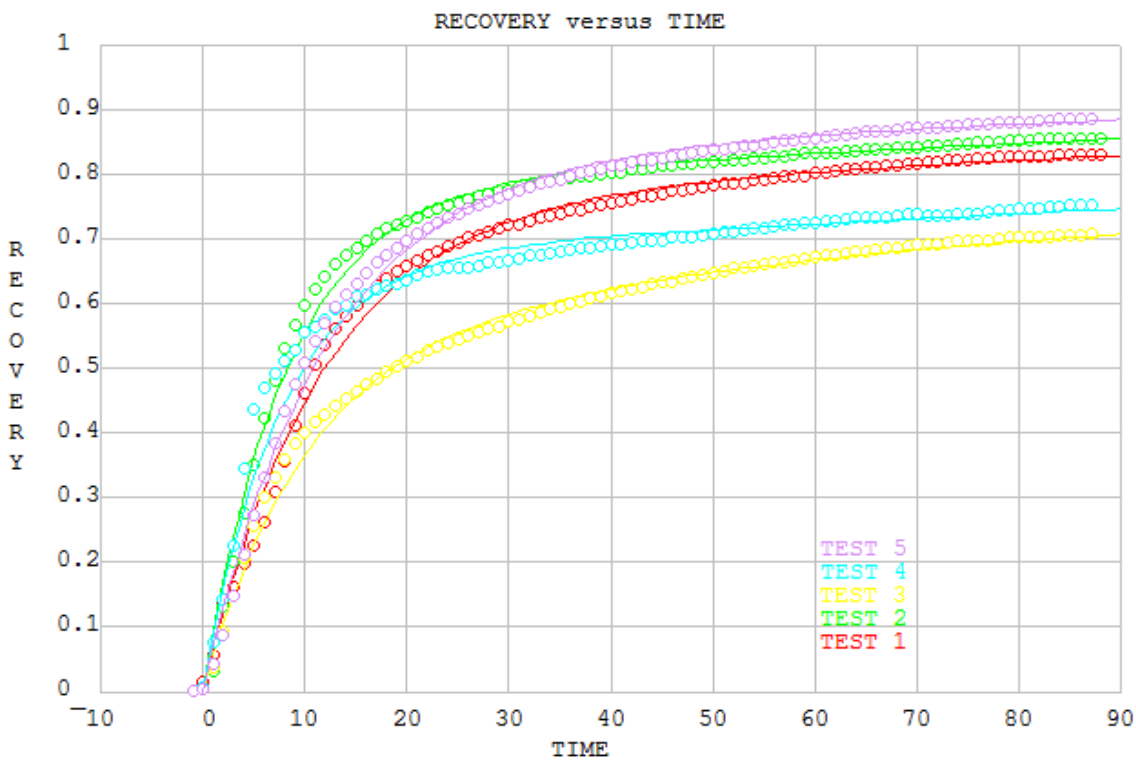

Figure 5: Recovery curves from five different tests plotted together in METSIM

Spreadsheet models become unmanageable with the large number of factors influencing the behaviors of such plants. Through curve fitting of test column data and dynamic simulation using METSIM, accurate process simulation and determination of the recoveries of heap leach operations is possible.

Conclusion

Development of accurate precious metals production forecast requires the following:

- Samples that are representative of the production period under evaluation (The production period could range from selected years of production to LOM for the project.)

- A set of metallurgical procedures that have been tried and proven to simulate actual practice as closely as possible in a metallurgical laboratory
- Evaluation of all the leach parameters to be included in the design criteria for the project

If a heap leach project is already in operation and the quality of the data is not suitable for the METSIM model, it is recommended to conduct a column leach study using full size columns. The samples to be used in the study should be representative of current production. The metallurgical data developed in conjunction with a dynamic model will provide the desired production forecast for the project. For a heap leach facility already in production, it is a good idea to develop a dynamic model to corroborate current or past production to ensure that the leach kinetics employed simulate the heap leach process.

Use of multi-depth and horizontal pan samplers to evaluate pregnant liquor solution chemistry in near real time

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Abstract

A multi-depth monitoring system (Westbay) was modified to sample pregnant liquor solution (PLS) from two depths (approximately 2.4 m and 6 m below ground surface) within the upper-most lift and underlying lifts in a test section of a large-scale copper heap leach operation under unsaturated flow conditions. In the same test section, specially designed horizontal samplers (stainless steel pans) were buried at the bottom of the upper-most lift (approximately 6 m below ground surface) during ore stacking operations. PLS was collected from the multi-depth wells and horizontal pans to evaluate parameter changes over time.

For the first 30 days of the leaching cycle, samples were collected daily from six Westbay zones (three wells at two depths per well) and three horizontal samplers. From day 30 to 90 of the leaching cycle, PLS samples were collected weekly.

A total of 303 samples of PLS collected from the Westbay and horizontal samplers in the test section of the heap were analyzed for key parameters of interest, which included Cu, Fe (II), Fe (III), Total Fe, H_2SO_4, pH, Oxidation Reduction Potential (ORP), Bacteria, Dissolved Oxygen (DO), and Electrical Conductivity (EC). The PLS samples were also analyzed for 52 major and minor elements.

The key parameters of interest from the analyzed samples of PLS were plotted as a function of concentration verses time during the leach cycle to evaluate time-based trends. Piper Charts were plotted to evaluate the dominant chemical characteristics of the PLS samples. Correlation and regression analysis were used to evaluate the potential to predict key PLS parameters from more basic parameter measurements.

PLS chemistry can be plotted over time during the leach cycle at multiple locations in the heap and used to improve the understanding of the metallurgy (changes in chemistry, consumption of acid and

completeness of leaching) in the in the upper-most lift in near real-time. This is compared with having PLS data from one or a very limited number of points, typical for most heaps, before it goes to the solvent extraction electrowinning (SXEW) plant.

Introduction

For the past three years, Schlumberger have been working with the mining industry to develop and conduct proof-of-concept testing of monitoring applications that can provide in situ measurements of the hydrometallurgical processes in a heap under leach. The objective of this work is to provide operators with information that can be used to increase efficiency of production, reduce costs and minimize risks. The key objectives established at the start of proof of concept testing included the following:

- Measure copper-in-solution production from the uppermost lift of a multi-lift heap or stockpile in near real-time.
- Track copper-in-solution inventory in near real time.
- Track PLS grade "front" as it moves through the heap/stockpile.
- Track copper extraction from a given block of ore in near real time.
- Measure solution and gas permeability and moisture content at multiple representative locations within the heap.

This article provides the results of in situ PLS sample collection and analysis from an engineered multi-lift heap with a combination multi-depth (Westbay) wells and specially designed horizontal pan PLS samplers. The owner and location of the heap are confidential.

Technique

A test section of the upper-most lift and underlying lifts of an engineered heap under leach was instrumented with Westbay wells and specially designed horizontal pan solution samplers that were used to collect PLS samples from multiple locations and levels within the upper-most lift. Although other instruments were installed and used to take measurements in the test section, their use and results are not within the scope of this article. The layout for the Westbay wells and horizontal pan samplers is presented in Figure 1.

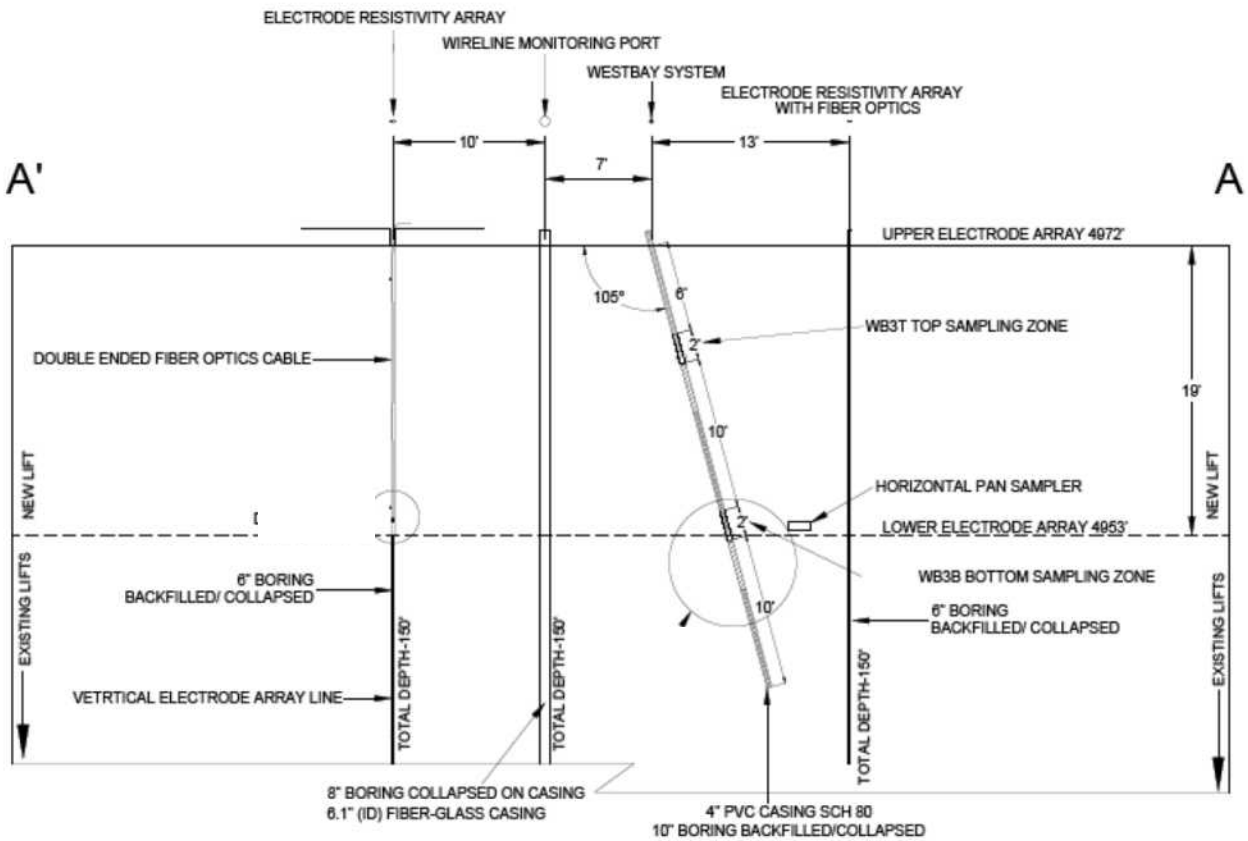

Figure 1: Test section design and layout of instrumentation

Three 25-cm diameter boreholes for the Westbay wells were sonic drilled at a 15 degree angle from vertical to a depth of 9 m in the upper-most lift. Three Westbay wells, specially modified to collect PLS samples under unsaturated flow conditions, were installed in the boreholes and completed with two sampling zones at approximately 2.4 m and 6 m total vertical depth (TVD). To mitigate compaction caused by the track-mounted drill rig, the pilot area was ripped before leaching. Three horizontal pan samplers were placed at the bottom of the uppermost lift in proximity to the Westbay wells, prior to placement of the fresh lift.

Raffinate was applied to the test section with drip emitters, and monitoring was conducted for the first 90 days of the leach cycle. For the first 30 days of the leaching cycle, samples were collected daily from six Westbay zones (two depths in three wells) and three horizontal samplers. From day 30 to 90 of the leaching cycle, PLS samples were collected weekly. Over 300 distinct PLS samples were collected and analyzed in the heap operator's laboratories. Samples were analyzed for Cu, Fe, Fe(II), Fe(III), H_2SO_4 (free acid equivalent), pH, ORP, DO, bacteria, and electrical conductivity (CE). The PLS samples were also analyzed for other major and trace elements.

Results and discussion

Figure 2 provides a plot of Cu, Fe(II), Fe(III), H_2SO_4, and CE from Westbay and horizontal pan PLS samples in the test section of the heap during 90 days of leaching.

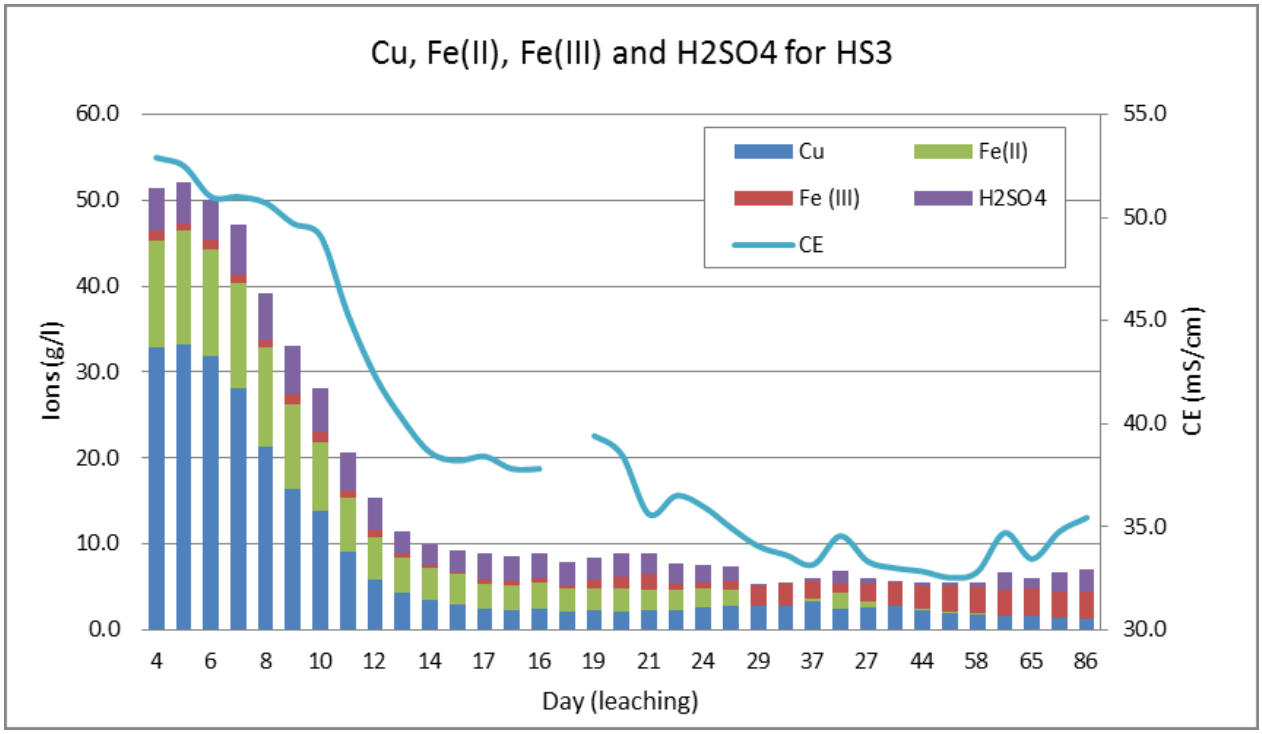

Figure 2: PLS grade, chemistry and CE through the leach cycle

As shown in Figure 2, most of the Cu was recovered during the initial 15 days of leaching, and then levels off. Concentrations of Fe(III) are relatively higher in the initial 30 days of leaching, after which Fe(II) becomes more prevalent. Free acid also decreases after 30 days of leaching. There is a strong correlation between CE and the concentrations of ions in the PLS.

As shown in the Piper chart in Figure 3, Mg-SO_4 was dominant in the leach solutions in samples collected over time with some mixing from Na-K components. As Cl was not analyzed, some proxy values were used. The solutions are dominated by Mg as the major gangue cation, which is usually the case as Ca is limited by gypsum solubility and Na is limited by sodium jarosite. Mg does not have solubility controls and builds up in solution to very high levels that are dictated by leach solution losses such as bleed streams, seepage etc.

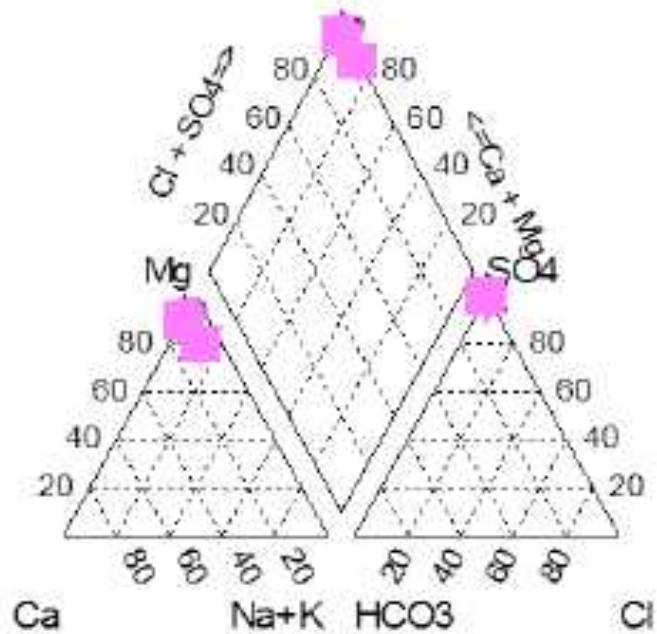

Figure 3: Piper chart for Westbay well samples

A correlation matrix was developed for analyzed PLS parameters from Westbay and horizontal pan samplers. Multivariable regression analysis was used to evaluate the ability to predict a parameter measurement (outcome variable) from combinations of other measurements (predictor variables). If it is possible to accurately and reliably predict certain parameters without taking actual measurements, it could potentially lower the cost of data acquisition while maintaining the value and utility of the data.

Regression analysis for copper predicted from CE, total Fe predicted from Cu, and CE predicted from Cu and free acid are shown in Figures 4, 5 and 6, respectively.

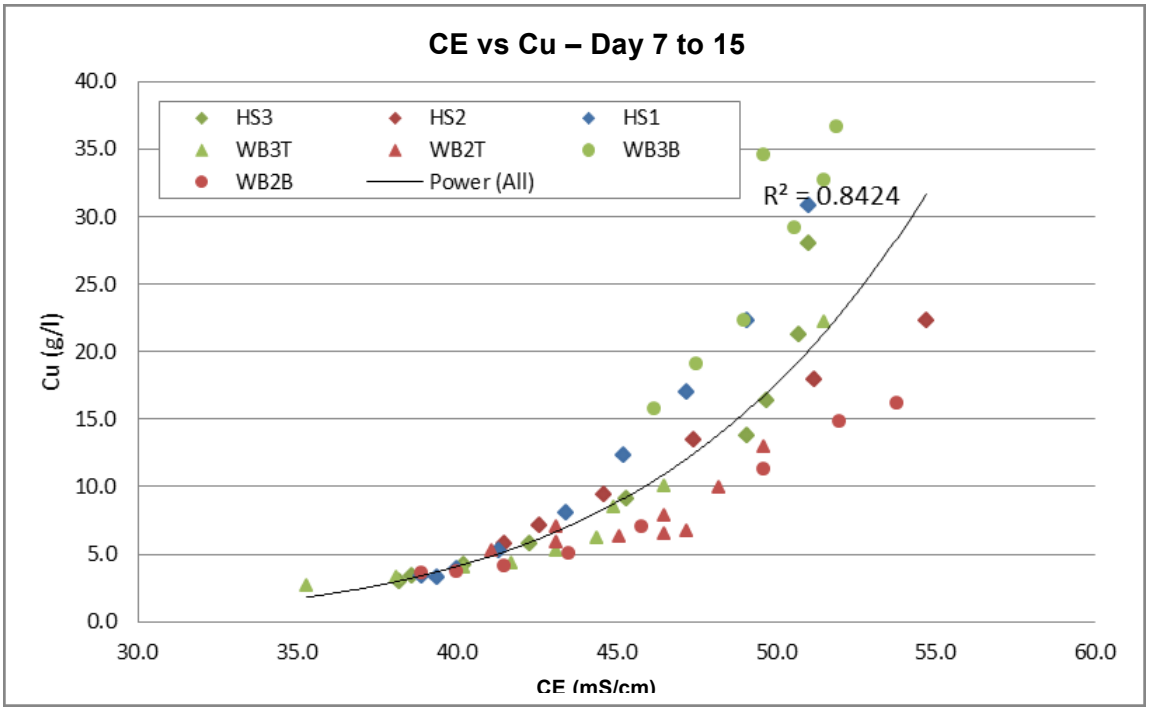

Figure 4: Cu predicted from CE

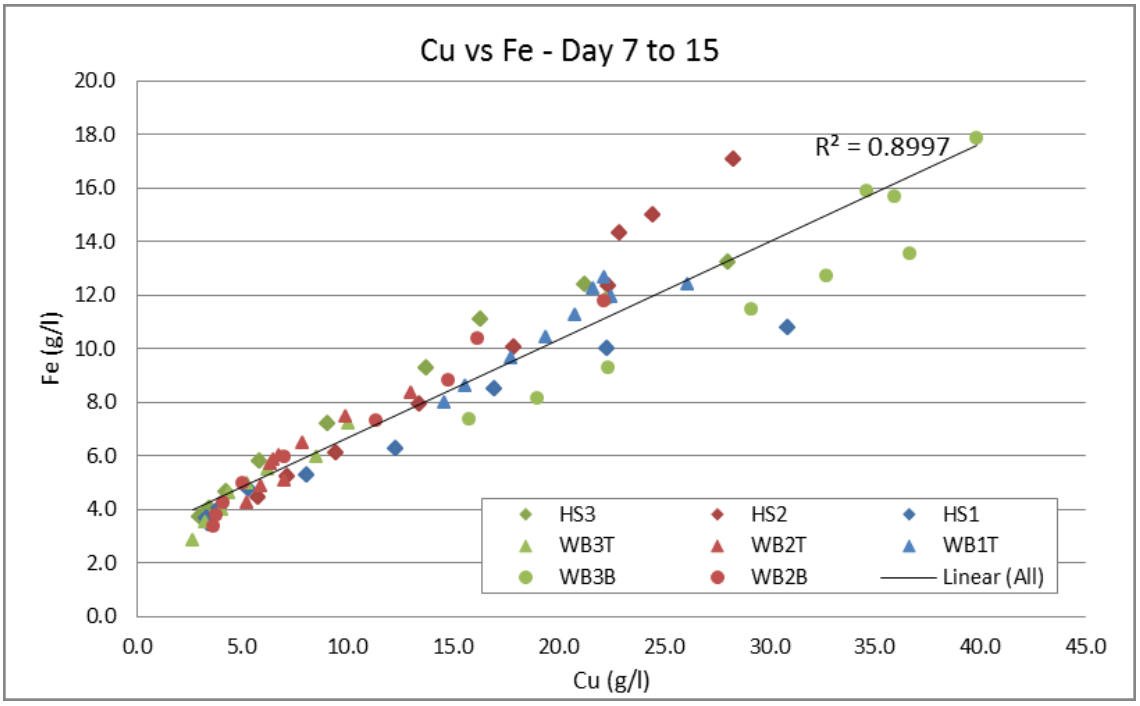

Figure 5: Fe predicted from Cu

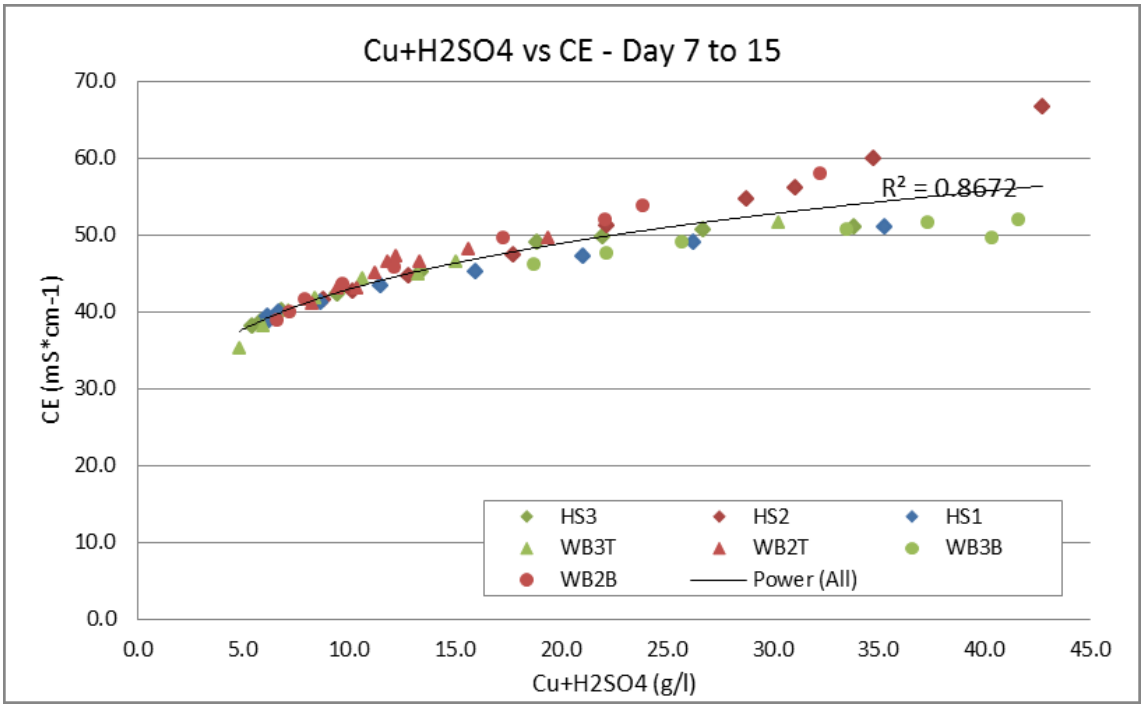

Figure 6: CE predicted from Cu and free acid

As shown in Figures 4, 5, and 6, relatively high R^2 values, a measure of the goodness of fit of the data, indicate there is good potential to analyze more easily-obtained parameters, such as CE for example, in order to develop a predictive model for Cu and Fe.

Conclusion

A test section of the upper-most lift and underlying lifts of an engineered heap under leach was instrumented with multi-depth (Westbay) wells and specially-designed horizontal pan solution samplers that were successfully used to collect PLS samples from multiple locations and levels within the upper-most lift.

PLS chemistry can be plotted over time during the leach cycle at multiple locations in the heap and used to improve the understanding of the metallurgy (changes in chemistry, consumption of acid and completeness of leaching) in the upper-most lift in near real time. This is compared with having PLS data from one or a very limited number of points, which is typical for most heaps, just before it goes to the SXEW plant.

The specially modified Westbay wells proved to provide a reliable and economic method to collect PLS samples in near real-time form the upper-most lift and can be used for Cu and precious metal leaching applications. They have a small footprint and are non-disruptive to the operation of the heap and can be installed in boreholes or buried in place, as needed to accommodate the design of the heap.

Horizontal pan sampler also proved to be effective but must be located near the edge of the heap in order to collect the PLS samples.

Results from this investigation indicate there is good potential to analyze more easily-obtained parameters, such as CE for example, in order to develop a predictive model for Cu and Fe. As a consequence, installation of monitoring wells in a leach pad monitored regularly for CE can enhance the evaluation of PLS and lead to improved understanding of leaching efficiency and completeness.

The use of fiber optics and advanced wireline geophysics to determine pregnant liquor solution front, depth and velocity

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Abstract

A test section of the upper-most lift and underlying lifts of an engineered heap under leach was specially instrumented with advanced wireline (WL) geophysics monitoring ports, fiber optics and resistivity electrode arrays. Resistivity measurements from an Array Induction Tool (AIT), electrical resistivity tomography (ERT) and distributed temperature sensing (DTS) fiber optics were used in part, to make in situ measurements and evaluate the hydrodynamics of the leaching process in near real time. Additional advanced geophysics WL logs were acquired using the Accelerator Porosity Sonde (APS), which measures volumetric water content and saturation, and the Triple Detector Litho-Density (TLD) tool, which measures bulk density, porosity, volumetric air content and saturation.

ERT inversion modeling provided high resolution 3D resistivity measurements, which were used to evaluate the distribution of moisture in the test section of the heap. DTS measured changes in temperature between fresh ore in the upper-most lift and the leaching solution and to a lesser extent, exothermic reactions. The AIT provided high-precision bulk resistivity measurements in a 90-inch radius around the monitored port.

Integrating the data from the three in situ measurements provides distinct advantages. For example, DTS data were used to correct ERT and AIT resistivity measurements, which made the assessment of moisture distribution and volumetric water content in the heap more accurate and reliable. The DTS data, in combination with bulk resistivity measurements provided the ability track the pregnant liquor solution (PLS) front in the uppermost lift and to estimate PLS front velocity.

Problems in the heap with pooling, channeling, longer lag and breakthrough times, increased solution in inventory, toe saturation, and slope stability can be detected in near real time with the in situ

measurements evaluated. Properly integrated data can provide the heap leach operator with information to increase production, reduce costs and reduce risks.

Introduction

For the past three years, Schlumberger have been working with the mining industry to develop and conduct proof of concept testing of monitoring applications that can provide in situ measurements of the hydrometallurgical processes in a heap under leach. The objective of this work is to provide operators with information that can be used to increase production, reduce costs and minimize risks. The key objectives established at the start of proof of concept testing included the following:

- Measure copper-in-solution production from the uppermost lift of a multi-lift heap or stockpile in near real-time.
- Track copper-in-solution inventory in near real-time.
- Track PLS grade "front" as it moves through the heap/stockpile.
- Track copper extraction from a given block of ore in near real time.
- Measure solution and gas permeability and moisture content at multiple representative locations within the heap.

This article provides the results of measurements made in an engineered multi-lift heap with a combination of conventional and advanced geophysics applications and focuses on the distribution of resistivity, moisture, saturation, and temperature. The owner and location of the heap are confidential.

Technique

A test section of the upper-most lift and underlying lifts of an engineered heap under leach was specially instrumented with advanced wireline (WL) geophysics monitoring ports, fiber optics and resistivity electrode arrays. Resistivity measurements from an Array Induction Tool (AIT), electrical resistivity tomography (ERT) and fiber optics distributed temperature sensing (DTS) were used in part, to make in situ measurements and evaluate the hydrodynamics of the leaching process in near real time. Additional advanced geophysics WL logs were acquired using the Accelerator Porosity Sonde (APS), which measures volumetric water content and saturation, and the Triple Detector Litho-Density (TLD) tool, which measures bulk density, porosity, volumetric air content, and saturation. A diagram that shows the design and instrumentation layout for the test section is in Figure 1.

ERT arrays were installed at the bottom (before ore stacking) and top (after ore stacking) of the uppermost lift. A total of nine sonic borings were drilled through the upper-most and underlying lifts to a total depth of 150 feet, representing a total of eight lifts. Three borings were cased with fiberglass and

used for WL logging. Fiber optics cables and ERT electrode arrays were installed together in three of the borings, and ERT electrode arrays only in three of the borings, all to total depths of approximately 43 m (140 feet). Although other instruments were installed and used to take measurements in the test section, their use and results are not within the scope of this article.

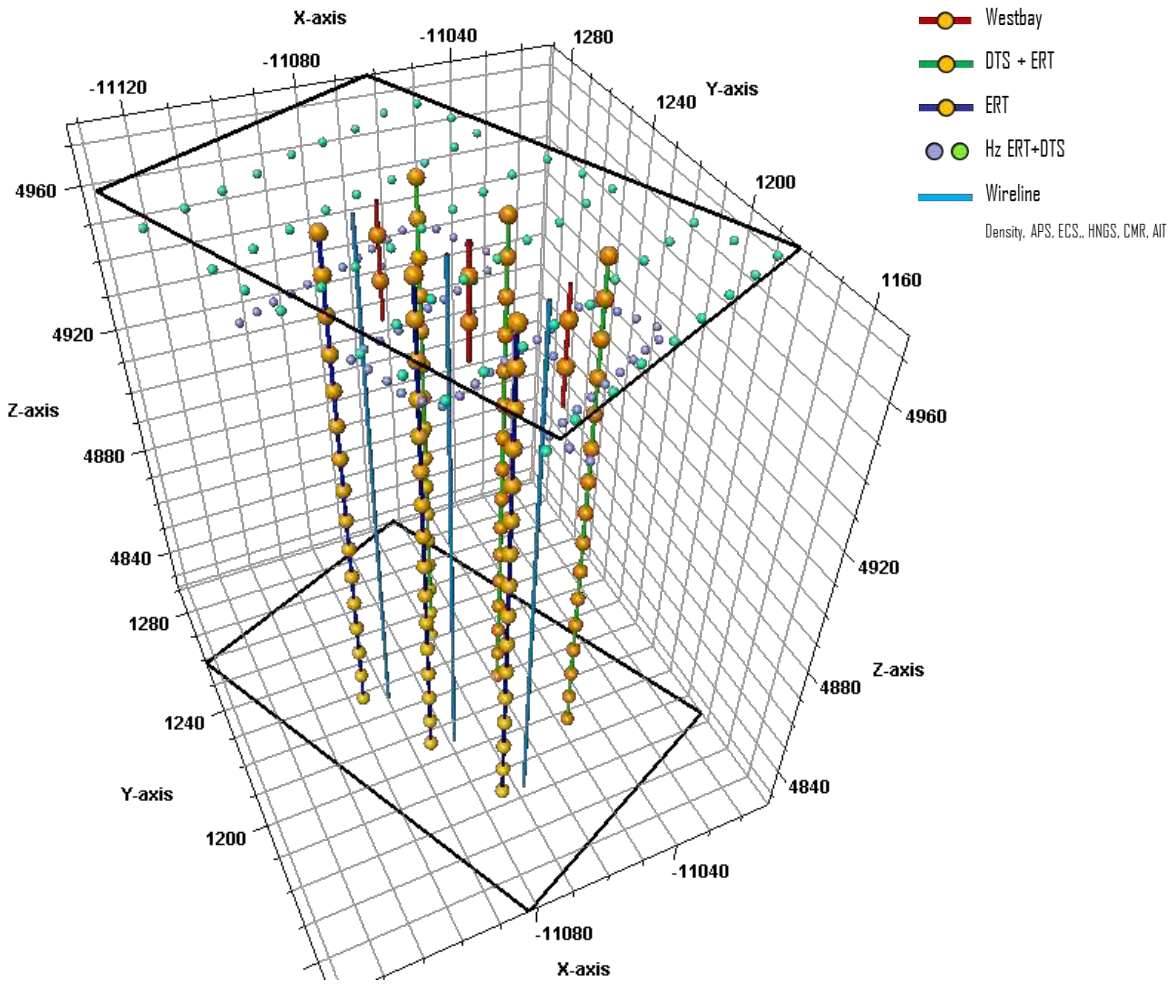

Figure 1: Test section design and layout of instrumentation

Raffinate was applied to the test section with drip emitters and monitoring was conducted for the first 90 days of the leach cycle. Frequency of measurements is summarized in Table 1.

Table 1: Monitoring frequency by method

Method	Month 1	Month 2 through 3
AIT	2 to 4/week	1/month
APS	2 to 4/week	1/month
TLD	2 to 4/week	1/month
ERT	2/day	1/week
DTS	140/day	1/week

The ERT instrumentation monitoring and data processing was performed by HydroGEOPHYSICS (HGI). Advanced WL logs from AIT, APS and TLD and DTS data were processed by Schlumberger with proprietary software and methods.

Results and discussion

Resistivity, water content and saturation measurements

The AIT logs provided highly precise, continuous bulk resistivity measurements for a radius of 2.3 m for the total depth of the investigation. The high precision and sensitivity of the AIT measurements detected small changes in resistivity with a high degree of precision and accuracy. AIT measurements were also effective to calibrate and ground-truth ERT measurements.

Elemental Analysis (ELAN) software was used to perform integrated log analysis and calculate volumetric moisture and saturation from APS and TLD logs. Figure 2 shows volumetric moisture and saturation over time for one of the WL logging ports prior to baseline reading and for two logging events during the leaching of the test section.

The logs in Figure 2 show variability in water content (left log) and saturation (right log) through the leaching cycle and indicate areas of higher water content and saturation above drier zones at certain depths, attributed primarily to compaction. In situ WL measurements of copper (Cu) (not reported in this article) also indicated there was historic Cu left behind in these zones.

The configuration of ERT arrays in the test section provided high resolution 3D resistivity distribution. 3D ERT results at different times from the beginning to the end of the leaching experiment are shown in Figure 3.

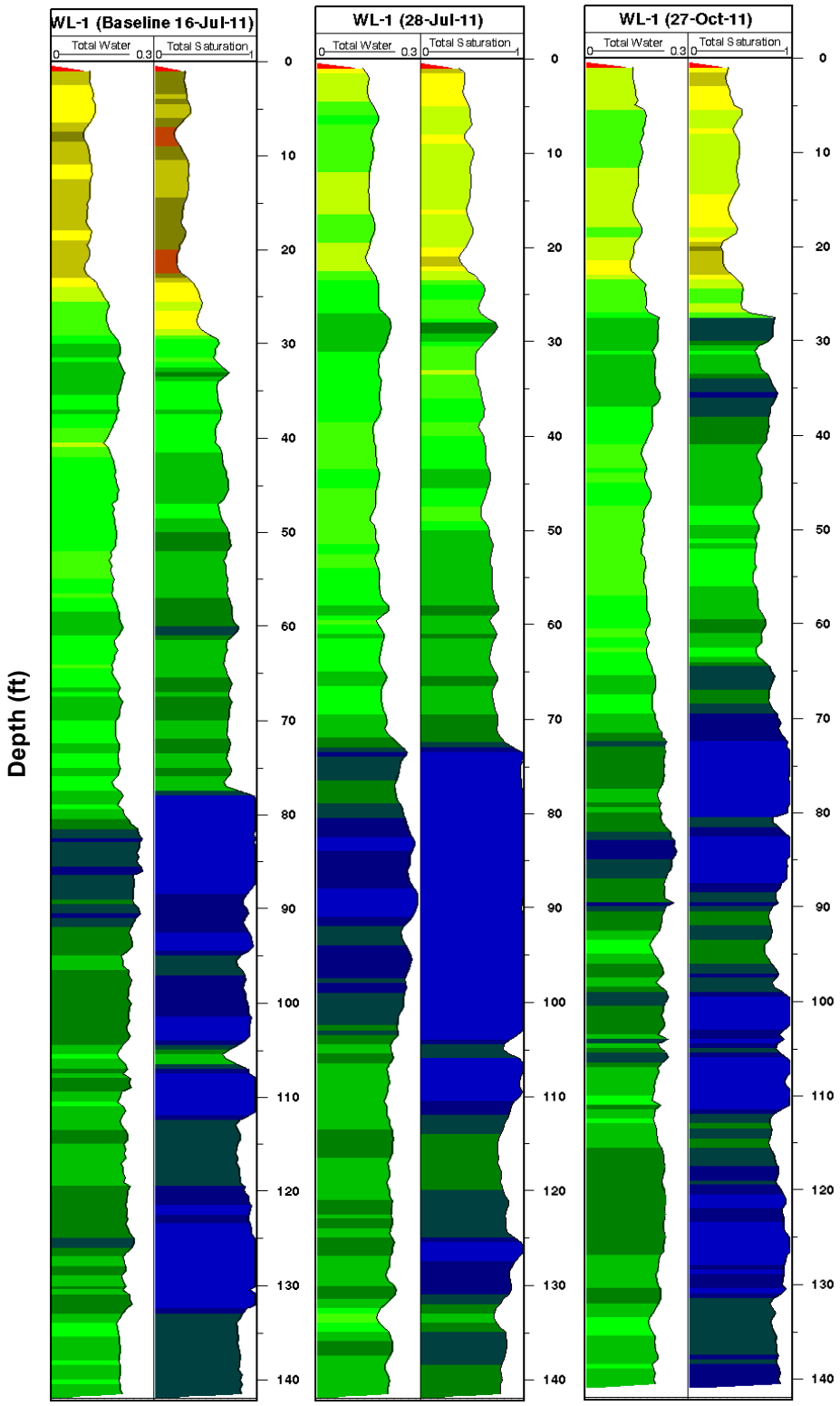

Figure 2: Volumetric water content and saturation from APS and TLD

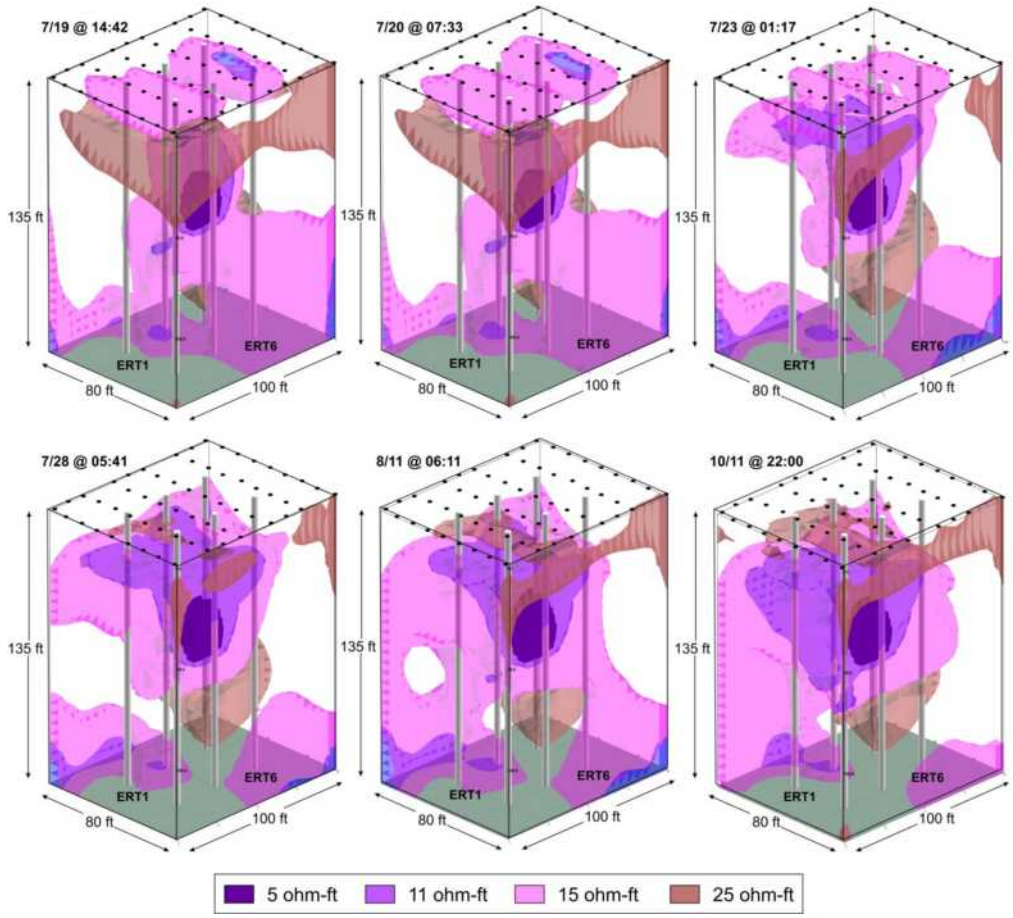

Figure 3: 3D electrical resistivity tomography models (from HGI)

Distributed temperature profiling

Monitoring the temperature profile inside the test section of the heap provided valuable information about the leaching process, which included but was not limited to the following:

- Tracking the leaching solution front.
- Estimating PLS front velocity and hydraulic conductivity.
- Identifying permeability/heterogeneity problems.
- Estimating exothermic/endothermic chemical reactions activity.

The temperature inside the heap was driven by the temperature of the raffinate. The contribution of exothermal chemical reactions was small in comparison. Figure 4 provides 3D temperature profiles with depth and time for the test section.

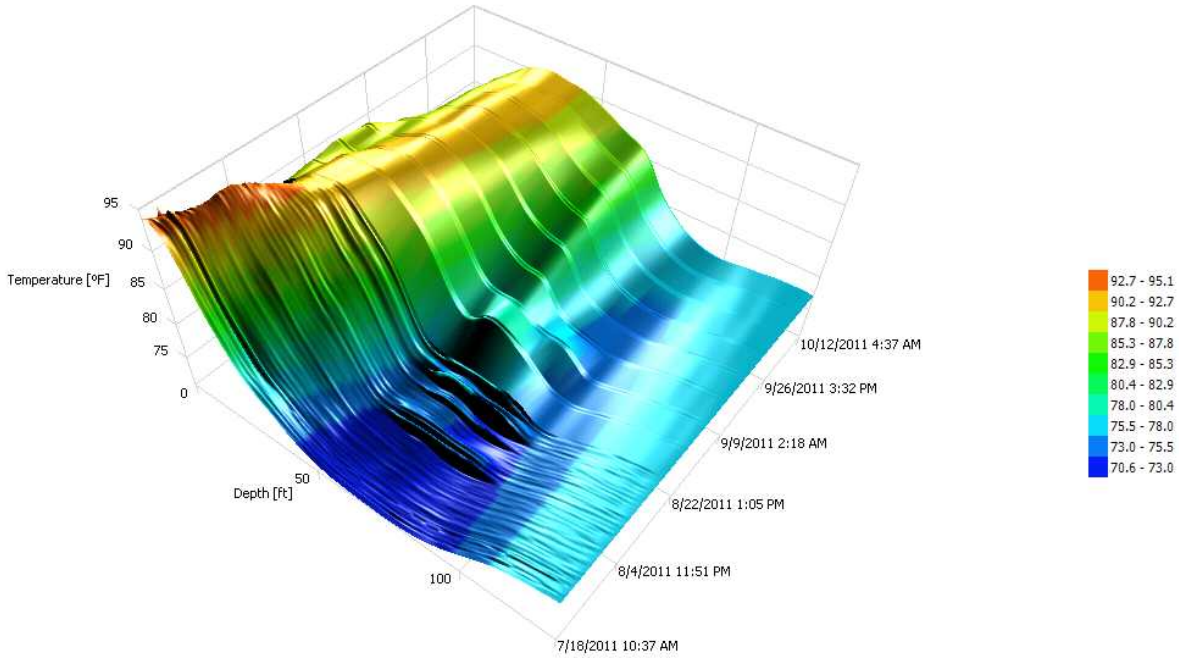

Figure 4: 3D temperature profiles with depth and time

The PLS front was tracked by monitoring changes in temperature, prior to and during the heap test section leach period, and fluid velocity was calculated. Temperature data were also used to correct ERT and AIT resistivity measurements.

Integration of data sets

Integration of multiple data sets provides distinct benefits and advantages over applications that provide single measurements, which include the following:

- Optimizes data collection and results and provides a more robust interpretation and understanding of in situ leaching processes.
- DTS measurements not only allow tracking the PLS front and velocity but can also be used to correct temperature sensitive ERT and AIT resistivity measurements.
- Multiple measurements of resistivity, moisture and temperature facilitate the quality assurance and quality control of monitoring data and interpretations though comparisons and cross checking of results by mass and energy balance accounting.

Figure 5 provides an overlay of temperature corrected ERT and AIT resistivity from the test section. As can be seen in Figure 5, agreement between the two measurements is quite good to a depth of

approximately 24 m/80 feet, but at greater depths, ERT resistivity is not as accurate or reliable as AIT bulk resistivity measurements.

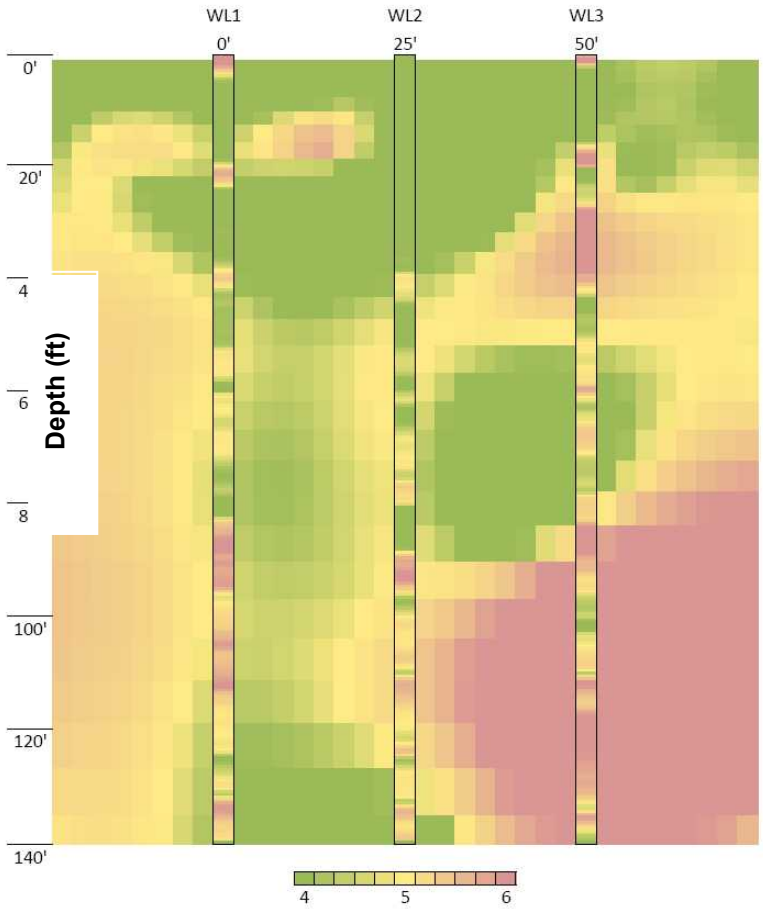

Figure 5: Overlay of ERT/AIT resistivity (ohm/m) for day 5

Resistivity is very sensitive to moisture changes in the heap. Compared to AIT resistivity measurements, ERT does not have the resolution or accuracy required to precisely track the PLS front. Figure 6 shows the average change in AIT resistivity (baseline verses current readings) resulting from the downward percolation of the PLS measured by DTS measurements.

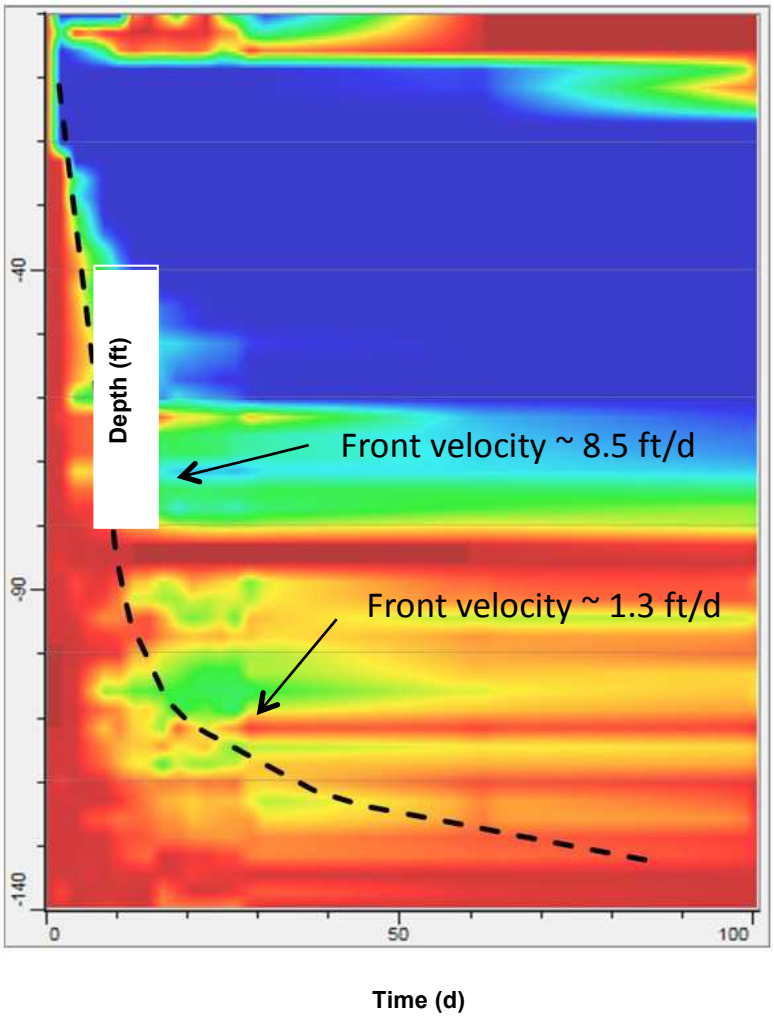

Figure 6: Average delta resistivity and PLS front velocity calculated from DTS

DTS effectively detects the PLS front, but there is a thermal lag associated with the time it takes for the temperature from the raffinate to warm the materials at depth in the heap. This thermal lag time could be corrected with a model that accounts for resistivity, which responds immediately to the increase in moisture from the PLS.

Conclusion

A test section of the upper-most lift and underlying lifts of an engineered heap under leach was instrumented and monitored with advanced wireline (WL) geophysics, fiber optics, and resistivity applications. These applications successfully and accurately measured temperature, induction and distributed resistivity, volumetric moisture, and saturation in the upper-most and underlying lifts in the heap. Measurements provided valuable information on the hydrodynamics of the leaching process in near real-time.

The AIT logs provided highly precise, continuous bulk resistivity measurements. The high precision and sensitivity of the AIT measurements detected small changes in resistivity with a high degree of accuracy. ERT arrays in the test section provided high resolution 3D resistivity distribution. AIT measurements were effective to ground truth ERT measurements, which improved reliability.

Temperature profiling with fiber optics combined with AIT resistivity made it possible to track the PLS front and calculate velocity and hydraulic conductivity. Temperature data were also used to correct ERT and AIT resistivity measurements, which further increased the accuracy and reliability of measurements.

Integration of multiple data sets provided distinct benefits and advantages over single measurements by providing a more robust interpretation and understanding of in situ leaching processes. Problems in the heap with pooling, channeling, longer lag and breakthrough times, increased solution in inventory, toe saturation and slope stability can be detected in near real-time with the in situ measurements evaluated. Properly integrated data sets can provide the heap leach operator with information in near real-time that can be used to increase production, reduce costs, and minimize risks.

Lessons learned in heap leaching technology implementation

Omar Caceres, Rio Tinto Copper Projects, USA

Abstract

Heap leaching technology is examined every time discussions about new project development or expansions are considered by mining companies, especially when economic downturns or high comparative analysis of capital expenditure (Capex) escalation factors appear on the horizon. In reality, very few heap leaching projects have turned into successful operations. This paper asks how much of this is caused by our inability to deliver studies within a reasonable timeframe and for an adequate cost/reward ratio.

With few exceptions, there are no solid work plans that allow for a rapid transition from test work to engineering studies without taking significant risks. In general, mining companies and engineering consultants repeat the same mistakes over and over: extensive test work that does not add value to engineering development; test work performed on the wrong scale, which leads to poorly defined process design criteria or to unrealistic assumptions for business case evaluations; incorrect interpretations of benchmark information from different minerals or, even worse, from different metals; failure to consider site-specific conditions during the test-work plan design, and so on. One of the main effects of these errors is that developing a project using heap leaching technology takes a long time, which leads to a significant reduction of shareholders' interest in investing in this technology.

The challenge for the heap leaching community is not only to work around the technical issues, but also to enhance the benefits associated with this technology and therefore improve its attractiveness. These benefits include increasing the mineral reserve inventory by turning waste into ore, improving mining development flexibility and pit optimization, lowering energy requirements, creating optionality for downstream process integration and final product specifications, and allowing modularization to reduce working capital during implementation. This paper sets aside the complex technical challenges of chemical reactions, bacteria performance, process control, and energy balances; instead, it discusses how to use good planning to expedite heap leaching implementation without introducing additional risks to projects.

Introduction

This paper shares the author's experience in developing heap leaching technology for application in two challenging arenas: nickel laterite and chalcopyrite dominant copper ores. It outlines what should be investigated at the concept study, pre-feasibility, and feasibility stages of project development, and proposes a series of absolute criteria that must be completed before moving from one stage to the next.

The paper does not include metallurgical data or theories about how to achieve higher recoveries, lower acid consumption, shorter leach cycles, or other key parameters that have been seen as critical success factors for future operations since, in the author's opinion, the results from one particular study can only be used within their original context and are not indicative of the results of other studies. This paper establishes a common language and defines a check list of minimum tests to be completed during each stage of project development. While it aims to set the basis for future studies, it is open to challenges and additions.

Keeping the focus

The following are a series of questions and answers that emphasize the importance of maintaining a focus on the primary objective of each stage of project development, from the metallurgical investigation perspective.

Concept study (CS)

The top priority here should be to understand as much as possible about the mineral resources so that you can confidently select the preferred metallurgical extractive process. This understanding is gained through in-house data (if available), expert consultants, and a well done risk assessment that covers the physical constraints for project development.

What is the level of understanding of the mineral resources available?

Make sure that you have finalized the exploration project. Ore body knowledge is the basis for any metallurgical development, regardless of the process technology to be applied. Understanding the overall mineralogy of the deposit and its sequence will simplify the decision about which metallurgical process to apply to the project. A clear understanding of the overall mineralogy of the deposit is necessary in order to establish the technical constraints that may eliminate an option. The feasibility of heap leaching is strongly dependent on mineral association, gangue reactivity, and hydrodynamic characteristics, among other factors. At the CS level, these factors are best assessed through ore body knowledge rather than through non-representative test work.

A significant amount of time and money can be saved by delaying metallurgical tests until sufficient understanding of the ore deposit is available. Early test work based on a few exploration samples leads to the creation of false expectations and assumptions regarding metallurgical performance. It also adds complexity to the process flow sheet and increases capital and time required to develop the project.

Does "representative of the sample" sound like a familiar challenge?

We all love shipping samples to external laboratories or ask for money to build our own testing facilities as soon as possible to run different kinds of tests to produce data. The results are almost always meaningless and can lead to incorrect conclusions to either abandon projects or fast-track them. Remember that, at the CS stage, it is better to invest that time and money on creating more understanding of the ore deposit. Set the required metallurgical assumptions based on the best information you can get at a lower cost: in house data, benchmark data, or a simple consultation with an expert.

Heap leaching is classified as a low recovery process, but it is also low Capex and low operating expenditure (Opex) when compared with the high recovery alternatives. Ore body knowledge is fundamental to a complete comparison between producing more metal while expending significantly more Capex and Opex and producing less with a lower investment. This distinction is significant for a green field project because a lack of information can lead to the wrong conclusion; it is not such a big problem for a brown field expansion where detailed ore body knowledge is available.

Are you biased towards a specific metallurgical process?

Depending on who is involved in the concept study, the project can start heading in the wrong direction at the CS stage because some options are either over- or under-estimated, influencing the decision where there is no strong evidence. Surprisingly, heap leaching is still considered by many to be a novel technology. Therefore, organizations with low risk tolerance will not consider it as a viable option, limiting their opportunities to maximize the value of a particular deposit.

A list of technically absolute criteria must be defined and used to select the preferred metallurgical process to be investigated in more detail during the pre-feasibility stage. This is fundamental to optimizing the time and money required to complete the pre-feasibility stage, since you cannot move your project from the CS stage without producing a narrow lists of options for the extractive technology to be used.

The points above regarding ore body knowledge help to establish the rationale and assumptions for estimating project economics. The important lesson here is to avoid being optimistic in these assumptions. For example, if you are doing a green field project, don't assume that it will be the best in the industry because that is unrealistic when you are at the CS level. At the risk of sounding repetitive, it is not necessary to do test work at this stage to produce the assumptions needed to evaluate the project. A

detailed understanding of the mineral resources is far more useful than a few tests done on non-representative samples.

Pre-feasibility study (PFS)

This stage of the project development must be divided into two stages: a first stage, to obtain the necessary samples to complete a comprehensive program that delivers well-supported process design criteria, and a second stage, where the engineering design and financial evaluation is completed.

What is sufficient test work?

Ideally, sufficient test work means that each component of the process design criteria is supported by some kind of test completed on representative samples. This is not just about leaching tests to define ultimate extraction and acid consumption. It is more complex than that.

The PFS should not officially begin until representative samples are available to complete the required test work. The bigger question is, "What is meant by 'representative'?" The ore body knowledge will give you the answer. The mineral association and ore types present in the deposit, the variability of the ore body, and its sequence along the expected life, among other factors, will tell you what samples you need and in what order.

Experts covering all areas of the preliminary process flow sheet should be involved in producing a test work program that covers all components of the process design criteria. Since what happens in one area of the process will affect everything downstream, tests should not be restricted to the leaching portion only. It is very likely that most of the tests will depend on the leaching test, since samples will be generated after these are completed (that is, solvent extraction (SX) circuit will need PLS, leached ore disposal, neutralization requirements, etc.).

As a preferred option, run the test work at the project site if your budget, time, and external constraints like permitting allow for this. You will learn more through the experience of performing the tests than through data received from a lab service provider. This is a challenge for a green field project, but the reward is worth the effort. In addition, if you have the right human resourcing strategy, you will be creating your future operations team.

If this is not an option and you have to go through a third party, make sure that you maintain frequent communication with the team performing the tests and that test documentation is high quality. Having a team member permanently located at the lab facility is preferable.

What is the right scale for leaching tests?

Table 1 shows an example of when the heap height and the solution application rate are used to determine whether or not a test shall be included.

Table 1: Leaching test scales

Operation parameter	Unit	Commercial scale	Pilot/large scale test	Medium scale test	Small scale test
Heap height	m	10	10	5	1
Solution application rate	$L/m^2/h$	5	5	5	5
Solution to ore ratio	L/h/t	0.28	0.28	0.56	2.78

A small-scale test (25 cm × 100 cm column) results in a factor of 10 in this example; this gives an idea of what happens chemically to a small fraction of the ore near the heap surface. Some people use the small columns in series to simulate the reactions over a full sized heap and maintain the reactions profile. Although this may accurately represent the chemistry involved, it will not accurately represent the hydrodynamic response of the ore, and both are interdependent. A medium-scale test (100 cm × 500 cm column) will give you a factor of 2 in this example; this may be efficient, depending on the extent of gangue reactivity and the degradation of the ore. The pilot or large-scale test (200 cm × 1,000 cm column) is the preferred alternative since it should give a factor of 1 and will allow you to collect all the necessary information for the design criteria.

The best practice is to use the small-scale tests only to characterize the different mineral components of the deposit and establish the range of expected maximum metal extraction that will be obtained. If blends are used, some blends should be tested to establish their impact against the single mineral components. Keep in mind, however, that the blends must be defined by the mine production plan.

The decision between medium and large scales will depend on the expected complexity of the metallurgical process combined with the hydrodynamic aspects. If you are expecting high gangue reactivity or significant changes in the hydrodynamic properties, go straight to the large-scale test. Performing a large-scale test does not necessarily mean that you have to build a pilot facility, but it does mean that you should at least build columns targeting the height you are planning for commercial application.

While there are obvious cost differences in the capital required to set up a testing facility at different scales, there is not a big difference in cost to run it. In fact, many laboratories charge almost the same for running the same tests at different scales. If you manage to set up a testing facility near the project location, it should be able to support the future operation.

Minimize the number of small-scale tests. They are only indicative of the metal extraction that you would expect without interference from gangue reactions or constraints caused by the hydrodynamic characteristics of the ore along the targeted leaching height profile.

The true constraint is the availability of a representative sample, but this should not be used as an excuse to move forward with your project without testing. If you do this, you could be headed for disaster. It will be much better to wait for that sample.

How many tests should you do?

Perform as many tests as your time and budget allow. The challenge is to ensure that they are the right tests, at the right scale, and using a representative sample.

When it comes to leaching, the quantity of tests is not as important as doing tests at the right scale. If enough samples are available, it should be a general rule to run multiple large scale (factor of 1) tests instead of many small columns. In the end, running a large column may be even cheaper, and the results will be more valuable to the project.

In the hydrodynamic area, it is recommended to test one sample from each meter of a leaching column test. The idea is to build a large database that allows statistical analysis. The purpose is to establish an operational window for the leaching unit and for the ultimate heap height target as inputs for the leach pad design.

It is important to note that samples from small- or medium-scale tests should be used only to define the range of operational parameters and metallurgical outcomes but not to define process design criteria. In the same way, drawing conclusions and defining design criteria using only fresh ore samples is one of the most common mistakes in the industry, and must be avoided. The hydrodynamic section of the process design criteria must be produced using only samples from fully leached material obtained from a test that used at least one variable with a scale factor of 1:1 (i.e., a solution application of L/h/t of ore).

Are you planning to hire the engineering company at the beginning of the PFS?

This is one of the most common mistakes in project development. Typically, at the very beginning of the PFS, you still don't have the samples to conduct a metallurgical program to produce the design criteria, which means that there is no basis for the engineers to work. Any process-related work done by the engineers without the final version of the design criteria will have to be reviewed and likely modified. This is by far the most common cause for significant changes in feasibility studies.

As a best practice, do not outsource the development of the process design criteria. Instead, use test work results, in-house data, benchmark data, and experts' opinions. Make sure that your process team takes ownership of the process design document, since it is fundamental for the project's future operation and success.

In deciding on the type of engineering company to use, the preferred option is to choose one with the capabilities to move the project forward to feasibility and then execution. Continuity is fundamental for project success. If the budget does not allow for this, make sure that you limit the scope of work to

meet the minimum requirements within your company's approval process. There are many examples of engineering, procurement, and construction management (EPCM) contractors that, during the feasibility study, reject the work done by others during PFS for many reasons, including liabilities, performance guarantees, value engineering, applicable standards, and so on.

What should be studied along with the metallurgical program?

In a heap leaching project, the most important study to be completed along with the metallurgical investigation is a geotechnical and hydrodynamic investigation of the project site. Understanding these characteristics as early as possible will enable you to produce a viable design based on the parameters defined in the process design criteria. As an example, understanding the options in terms of leach pad design (for example, on/off pad, multi-lift, valley fill, or hill side) may change the way that the metallurgical program is defined. This is very important because, depending on the location, the earthworks and civil foundations can represent between 30% and 50% of the direct capital cost and can increase or reduce the project risk profile.

Are the results from the test work and the process design criteria used correctly in the engineering design and financial analysis?

The process team is responsible for ensuring that the process design criteria are properly translated into the engineering design. Special care must be taken during the value engineering exercise, since there is a strong tendency to introduce changes without assessing their impact on the process performance.

The mid-case value for all critical variables must be well supported by the test work results and the metallurgical process investigation. You can use other sources to complete a range analysis, but only to understand the potential outcomes of the process when it deviates from your mid case, and not to try to optimize your net present value (NPV). Do not assume better results than those indicated by your test work and apply an optimistic scale-up factor if there is a significant difference between the scale of the tests and the proposed design.

Feasibility study (FS)

This part of project development is all about value engineering and project optimization. The goal of these processes is to produce the best cost estimation possible to complete the project evaluation and support the business in the investment decision process. The best practice is to only move the project to this stage if the metallurgical process has been selected, including all unitary processes, and the general layout has been defined. There is no room for major changes at this stage.

Are you still running test work or are you planning to start new tests?

If you are still running tests, this may easily mean that your project is not ready for FS. If the test work is critical, then keep your project in PFS, get the results, and complete the evaluation to ensure that your main PFS deliverables are not going to change during FS.

Unlike at the PFS stage, the focus during FS must be on the front-end engineering design and planning for future execution. The process team should review the engineering deliverables and ensure that there are no changes introduced to the design that may impact the expected metallurgical performance. If the process team has extra resources, it should be responsible for the operational readiness program, which must be completed during the execution phase.

Lessons learned

Below is a summary of lessons that should be taken into account during the development of any heap leaching project to avoid repeating familiar mistakes.

- **Finish the exploration program first.** Ensure that you have well characterized mineral resources for the first 10 years of operation, considering the range of production rates that you are targeting. Wait until the geology and mining options are well understood.
- **Heap leaching cannot be fast-tracked**. Waiting until sufficient ore body knowledge is available, completing the test work, and having land access to do the geotechnical and hydrodynamic characterization of the project site takes time. But redoing the engineering design every time new information is available will always take longer and will be significantly more expensive.
- **Avoid changing your mineral resources inventory or your mine plan during PFS.** Freeze the mineral inventory and mine production plan as early as possible in PFS to ensure that the test work program is targeting the right ore blends sequence. Your focus must be on mineral resources that will be processed during the project payback period. Testing ore blends for later years does not add any value to the project. (An exception would be if future expansions are considered.)
- **If you don't have the right samples, don't do the tests.** Results from non-representative samples always mislead the business case evaluation one way or another. Performing tests to gain time always results in rework, longer development time, and higher study costs.
- **Select the right scale for your tests.** Document the rationale behind the scale of tests used to support the process design criteria. Make sure that at least two of the critical operational

variables are tested at 1:1 scale; the ratio between the solution application and the ore measured as $L/m^2/t$ of ore must be one of these.

- **Think like an operator, not a researcher, when performing metallurgical tests.** If you are running the metallurgical tests under ideal conditions, you are testing the concept but not the feasibility of a future operation. Design your test work to consider all external factors that may impact the ability to replicate the test work results at the commercial scale.
- **Ramp-up and steady operations are different, and should be tested separately.** Test work programs usually run in a steady state mode because they are focused on understanding the leaching performance under normal conditions. Depending on what drives the leaching process (for example, temperature, pH profile, air/liquid permeability, etc.), the test work program must consider how to replicate those drivers to understand their role during the ramp-up period. Consider the impact of this stage on the business evaluation and NPV calculation.
- **Differences in ore preparation between the lab scale tests and the commercial scale design can introduce significant risk to the project.** Crushing, agglomerating, and loading of small, medium, and large columns for leaching tests must be consistent with current heap leach commercial-scale practices that are likely to be used during the engineering design. Changes in the particle size distribution will change the leaching response of the ore as well as its hydrodynamic response, both of which are fundamental for the engineering design and economic evaluation.
- **Do not underestimate the hydrodynamic side of the test work program.** Understanding the permeability characteristics of the ore is fundamental to designing a heap leach operation. Do not use results from tests done on fresh ore samples only. Complete as many experiments as possible using fully leached ore samples and complete a statistical analysis to establish the operating window. Do not establish the operating window based on discrete analysis.
- **Accept the test work results as they are.** The values included in the process design criteria must be within the range of results obtained in the test work program.
- **Never assume that you will do better at commercial scale.** Defining scale-up factors is a subjective exercise. Depending on who is performing the business case analysis (for example, a junior company, a risk averse organization, or a group with heap leach experience), the outcomes may be completely different. This is acceptable only if there is consistency with the test work results.

- **Numerical models cannot replace the metallurgical test work program.** Models are great tools to simulate process scenarios (for example, different ore blends, changes in operation parameters, changes in external conditions, etc.) but they cannot replace test work results in defining process design criteria and inputs for financial evaluation. The only exception is in the case of predictive models that have been validated through large-scale tests.
- **Separate the technology development program from the project implementation program.** A project must have a clearly defined scope to succeed. Regardless of continuous improvement exercises, the scope must be frozen as early as possible in each stage of the project development. If high-impact opportunities arise, a full assessment, including a review of the test work plan, already defined design criteria, and the project risk profile, must be completed to ensure that the proposed changes are adding value to the overall project.
- **Benchmarks are indicative only.** Every ore deposit is different; therefore, there is no direct correlation between one development and other. Benchmark information should be used to set the context. Very importantly, a green field project should set its main metallurgical drivers to be below industry averages, since the industry leaders have been working on process improvements for years after their first implementation.
- **Keep the level of technology innovation under control.** A critical success factor for any project is to have a well-defined scope of work. While it is nice to have innovative technology in your project, make sure that the risk/reward equation remains balanced. Complete sufficient test work before including any novel unitary process in the process flow sheet that will be used by the engineering designers. In many cases, it is better to kick off a new project to investigate an alternative innovative technology than to change the scope in the middle of the PFS. If the heap leaching project is already in FS, there should be no option to start testing a new technology.
- **Maintain a focus on what needs to be done in each stage of project development.** The purpose of the concept study is to establish whether or not you have a business case based on the available mineral resources. Pre-feasibility is for selecting the best processing route to extract the most value out of those mineral resources; it is not a technology development program. Feasibility allows you to complete project optimization and detail the way the project is going to be executed; it is not meant for performing additional tests that may change the basis of the design.

Conclusion

You may be thinking that this paper is full of platitudes, but the truth is that heap leach projects rarely achieve their expected performance because they fail in one or more of the areas listed above.

At the concept study level, your top priority should be to thoroughly understand the mineral resources that you will be processing and the options for the project location that may impact the metallurgical process selection. Use in-house data (if available), benchmarking, and expert consultants, in combination with a well done risk assessment that covers the physical constraints for the project development, to develop assumptions for the business evaluation. No test work should be necessary.

The pre-feasibility study requires sufficient planning to ensure that you do everything right and in the correct order. Any mismatch between the basis for the test work program definition (i.e., the mineral inventory and the mine production plan) and the information used to do the business evaluation can misdirect the investment decision, since the test results may not be valid. This is the longest phase in the project development.

To produce the process design criteria, define the ore to be covered and the tests programs required; then, get the samples, perform the tests, obtain the results, and complete the necessary analysis. If possible, run the geotechnical and hydrodynamic site investigation at the same time. Do not hire the engineering consultant until you have completed this stage.

To complete the engineering design, ensure that the process design criteria is properly translated to the basis of design for each engineering discipline. Be aware of potential impacts from changes proposed during the value engineering exercises.

Perform the business evaluation by ensuring that the assumptions used to establish the financial benefits and the risk profile of the project are consistent with the test work results and process design criteria.

During the feasibility study, the process team should be focused on project optimization, assessing all the proposed improvement opportunities to ensure that the process inputs are still valid. No metallurgical test work should be done during this stage if its results are likely to change critical areas of the process design criteria. Only test work necessary to support equipment and material selection is acceptable. The success of this stage is measured by the number of late changes to the design; therefore, the target should be zero.

Efficient heating of heap leaching solutions to minimize GHG emissions and cost of energy

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Abstract

Inproheat is a 56-year-old privately held Western Canadian based company that specializes in combustion technology and applied heat transfer. For the past 38 years, the company has applied its proprietary Heat Transfer Technology, namely SubCom™, to a diverse range of aggressive solutions used in the mining sector.

Submerged combustion offers the highest possible thermal efficiencies, 99% on a higher heating value (HHV) basis and resultant lowest greenhouse gases (GHGs) for heating aggressive liquids with a combination of high total dissolved solids (TDS).

This paper focuses on the application of heating an aggressive low pH 1.0 to 2.0 raffinate solution at a copper heap leach operation. It discusses the technical design challenges that had to be overcome to utilize submerged combustion as the heating method and the resultant benefits realized. These challenges included selection of the most appropriate materials of construction for the tank and internal combustion chamber alloys. The raffinate flow rate was 130 US gallons per minute (USGPM) and was heated from 15°C to 35°C in an effort to increase its leaching potential as this high altitude operation was subject to extremely cold ambient conditions. The location of the heap leach operation was remote and at an elevation of 2,600 MASL, and all required diesel fuel to run the operation had to be trucked to the site. As a result, the operators wanted to ensure they achieved the maximum efficiency from their diesel fuel energy supply.

Introduction

This paper describes the design process and outcomes of a raffinate heating system, using submerged combustion technology, at a copper heap leach mining facility in Chile. The purpose of heating the raffinate was to improve leaching performance during the cooler evenings and winter months.

Traditional heating methods include indirect boilers/heat exchangers which typically operate at thermal efficiencies of around 75% to 85% in optimal conditions and require regular maintenance to ensure safety, due to their high-pressure operation. The thermal efficiencies of boiler systems tend to decrease with time as the heat exchangers foul. Furthermore, boiler systems require condensate return systems to maintain a reasonable efficiency; adding more components that require maintenance. Steam and condensate piping must be insulated for heat conservation and personnel protection, adding to the cost.

Submerged combustion is a proven technology in several heavy industrial applications, and perhaps most notably in the liquid natural gas (LNG) industry where it has been used for over 50 years to heat seawater to vaporize LNG. Relatively unknown in the mining industry, submerged combustion has the advantage of being able to heat any type of non-combustible liquid to near boiling point at thermal efficiencies approaching 100%, and with low maintenance requirements. Submerged combustion is a process in which the products of combustion are directly in contact with the liquid to be heated. The energy transfer occurs directly between the liquid and the hot gases, resulting in high efficiencies with no fouling issues. Figure 1 is a graph of thermal efficiencies for heating water in single stage contact and with a heat recovery system.

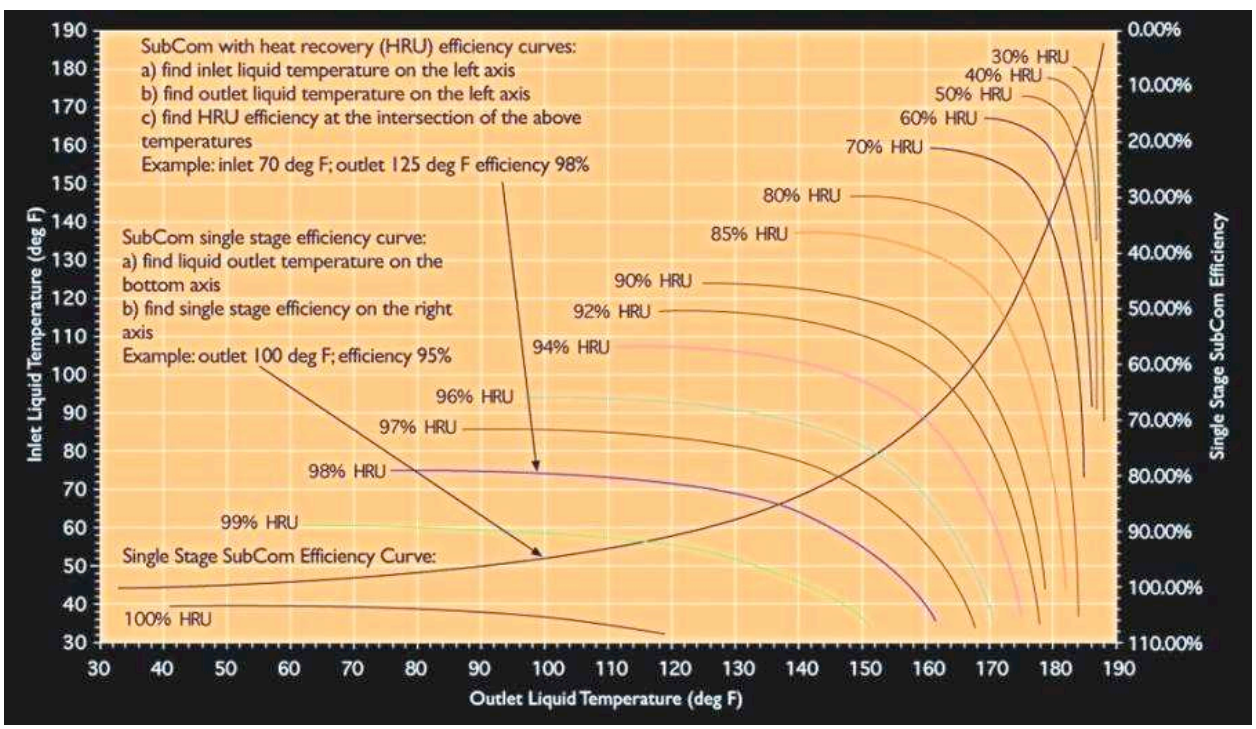

Figure 1: Thermal efficiency versus liquid discharge temperature

Submerged combustion technology

The raffinate heating system highlighted in this paper is trademarked SubCom™ and was developed by Inproheat Industries in the early 1970s in response to the global energy crisis. A SubCom™ liquid heating system consists of a fuel burner mounted at the top of a combustion chamber, which extends down from the top of a tank containing the liquid or slurry to be heated, as shown in Figure 2. The burner is connected to a blower that provides air for combustion as well as heat transfer. Pressure from the blower purges the liquid from the combustion chamber, allowing the flame to burn in a dry atmosphere without impingement, for complete combustion of the fuel. The products of combustion are vented through a series of carefully designed orifices around the lower circumference of the combustion chamber. The heat transfer takes place between the combustion gases and the liquid to be heated. An inherent property of the technology is that the temperatures of the exhaust gases that are released from the liquid are at the same, or close to the same, temperature as the heated liquid, which means high heat transfer efficiencies.

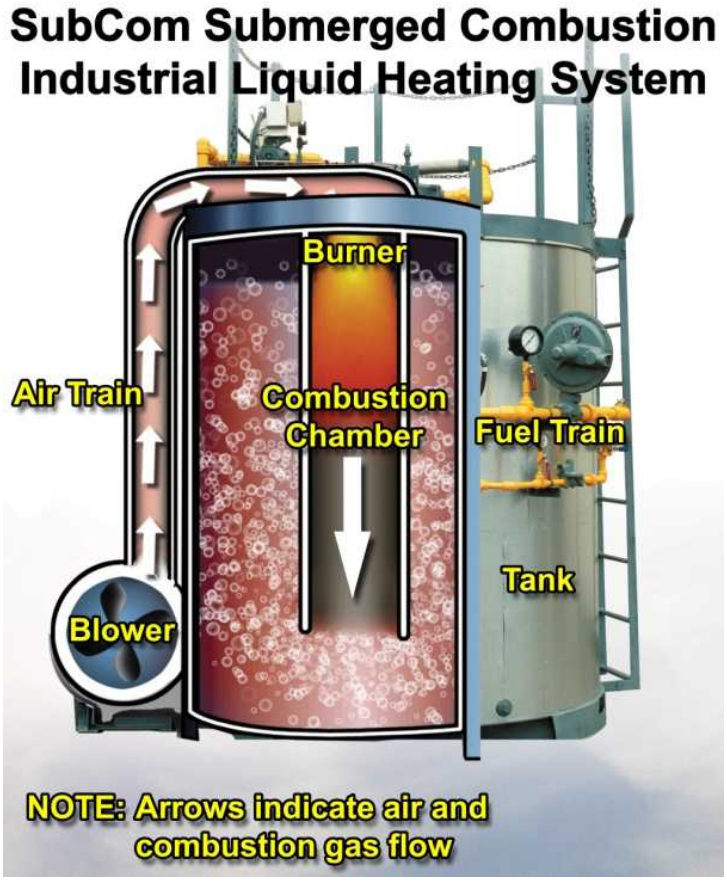

Figure 2: Typical SubCom™ system cutaway

GHG reduction with submerged combustion

Because SubCom™ is more efficient than alternative liquid heating technologies, such as steam boilers with indirect heat exchangers, there is a significant gain in thermal efficiency. This thermal efficiency advantage can be 20% or more. This translates into a directly proportionate reduction in fuel consumption and concomitant CO_2 emissions from the burning of fossil fuel.

SubCom™ applications in the mining industry

SubCom™ systems have been and are being used in a variety of applications in the mining industry.

A multi-burner system originally comprising ten 13 MMBTU/h (million British thermal units per hour) burners was installed at the PCS (Potash Corporation of Saskatchewan) Patience Lake potash operation in 1992. The system was modernized and upgraded with the addition of four more burners to bring the installed capacity to 182 MMBTU/h. The system heats brine to enable solution mining of potash from the former underground workings after the mine flooded in 1984. Without the heating of the brine, the operation would not be economically viable.

Recently a five burner SubCom™ system was installed at an integrated steel plant in Mexico to heat iron ore concentrate slurry with a 65% solids content. The burners were installed in an existing 15 m diameter × 15 m high agitated tank. The heating improves the slurry filtration performance in the pelletizing plant.

SubCom™ heaters have been used in mine water treatment to maintain the water temperature during winter months in cold climates.

A SubCom™ system has been in operation at an iodine recovery plant in Chile since 2008. A larger system is under construction for installation in 2013 to meet increased heating requirements at the operation.

Project overview

In 1995 Inproheat was approached to evaluate whether submerged combustion could be applied to heat raffinate solution. The primary objective was to minimize the high consumption of diesel fuel oil at the mine.

Run-of-mine ore is trucked to crushing circuits with primary, secondary and tertiary crushers. Crushed ore is screened and 13−19 mm material is sent to stackers that construct the leach pad. The ore is stacked to a height of 6−8 m. Fines are sent to agglomerating drums in which marble- to golf-ball-size ore nodules are formed with sulfuric acid to cure the pellets. The nodules are stacked for 15 or 30 days for oxide and sulfide ore respectively, to activate bacteria and consolidate pellet strength. The cured pellets

are then conveyed to the heap pads. Raffinate solution is constantly circulated to the heaps through an irrigation system. As the acidic raffinate percolates through the heap and contacts the ore it leaches the copper along with other impure metals. Pregnant leach solution (PLS) flows from the bottom of the heap on lined pads into ditches that take the PLS to collection basins and then to covered ponds. From there, the PLS flows by gravity to the SX/EW distribution point. Copper separation and recovery are done by conventional SX/EW to produce high grade copper cathode. Raffinate from the SX plants is heated from 15°C to 35°C before being pumped to the leach pads to increase its leaching potential. The primary source of energy for the mine is No. 2 diesel fuel.

The challenge

Raffinate is a mild solution of sulfuric acid with a pH of between 1.2 and 2.0. An investigation into suitable materials of construction for the system was required for the combustion chamber, tank, piping and valves. Typical of mining operations in Chile, the water contains chloride which can cause pitting and stress corrosion cracking in standard Austenitic stainless steels. In this case, there could be up to 2 grams per liter (gpl) of chloride in the raffinate. Inproheat's experience at the time with existing installations was limited to gaseous fuels. Using No. 2 diesel fuel had to be researched.

The elevation of the site, at 2,600 MASL, had to be taken into consideration when designing and selecting the combustion system components. The unit was to be mounted outside and capable of automatic operation with remote start/stop from the control room.

The requirement for high efficiency was dictated not only by economics but also by the logistics of trucking diesel fuel to the remote mine location.

The solution

Inproheat designed and manufactured a 10 MM BTU/h (2.5 MM kcal/h) submerged combustion system for raffinate heating. Metallurgical corrosion tests were undertaken to establish the best material for the combustion chamber. Carpenter 20Cb-3 alloy was selected for all combustion chamber components. The tank, vent stack and piping were made of fiberglass using an acid resistant Derakane 411-45 resin. To handle corrosive raffinate Inproheat selected Durco valves and a pump made of CD4MCu duplex stainless steel alloy. Propane was used as the burner pilot fuel and No. 2 diesel as the main fuel.

The unit was skid-mounted, pre-packaged and pre-wired, including the heating tank, submerged combustion burner, combustion air blower, propane pilot fuel train, diesel oil main fuel train, inlet water shutoff and control valve, discharge pump, and control panel. The fiberglass vent stack was shipped separately. The client provided raffinate inlet and outlet connections and fuel supply. Raffinate is fed by gravity from a large storage tank adjacent to the heating system. An inlet flow control valve maintains the

liquid level inside the heater tank via a level control loop. On the discharge side, a centrifugal pump removes heated raffinate from the heater.

Figure 3: Twin 10 MMBTU/h SubCom™ raffinate heaters

To address the technical uncertainties of the project Inproheat decided to simulate system operation with water in the Vancouver factory. The system was tested for approximately two weeks. A 50 Hz generator was used to match the frequency of power supply in Chile. A cold water supply of approximately 30 m^3/h was connected to the heater, and the high altitude operation of the combustion air blower was simulated by installing a restrictor plate on the inlet of the blower.

Initially, a problem of vortexing developed in the round-domed fiberglass tank. This problem was quickly resolved by the addition of baffles installed on the interior tank wall. Performance of the diesel oil burner exceeded expectations. Initial concerns of potential oil residue in the raffinator were alleviated by water discharge quality tests. All system functions and performance were tested before shipment to Chile.

Inproheat commissioned and started the system up in November 1995. A number of combustion tests were conducted. The raffinate discharge temperature was set at 35°C, with stack temperatures between 34°C and 36°C. The resulting overall system efficiency was calculated at 93% of the diesel oil

higher heating value, with USD$12/h (approx. USD$100,000/y) savings compared to the original conventional boiler/heat exchanger system which was rated at 64%. GHG emission was reduced by 31%.

Conclusion

The application of direct contact solution heating for aggressive mining solutions is ideally suited for optimizing energy utilization with minimal GHG generation. Small scale pilot projects could be utilized to explore potential benefits in situations where indirect heating systems have been the norm.

Pilot-scale drum agglomeration of an oxide ore with high fines and variable moisture

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Abstract

Heap leaching of gold, silver and copper ores is being conducted worldwide to extract these precious metals. A key component of successful heap leaching is the permeability of the stacked ore (heap). A South American mine is studying the feasibility of extracting gold from a stockpiled oxide ore by means of heap leaching. Due to the high percentage of fines, and specifically clays in the ore, the native permeability of the ore is low for heap leaching. Ore agglomeration is being proposed to improve the permeability of the ore. Bench-scale agglomeration tests showed that the agglomerate quality is sensitive to the moisture of the ore; agglomeration is poor or fails when the moisture content of the ore is outside a certain range. Field studies showed that the in situ ore moisture content of the stockpiled ore can be as high as 27%; thus, demonstrating the ability to effectively agglomerate wet ore up to the maximum moisture content was essential to establish the feasibility of the heap leach project.

The authors designed a three day pilot-scale agglomeration test to evaluate the hypothesis that the wet ore could be effectively agglomerated with proper adjustment of agglomeration drum parameters and with a blend of drier feed ore, if required. After an initial period of optimization of the drum parameters, the pilot-scale drum produced quality agglomerates for ore with initial moisture contents between 13% and 27%. We found that ore with initial moisture contents higher than optimal could be agglomerated by uniformly blending with a drier ore, and that this blending step could be conducted in continuous mode immediately upstream of the agglomeration drum. The agglomeration testing program demonstrated that by applying some adjustments during the agglomeration process, our wet, clayey ores could be successfully agglomerated.

Introduction

During the late 1920s and early 1930s, certain parts of the United States were identified as having large tailings piles with considerable amounts of residual copper. One plant in the south-west had approximately 15 million tonnes of tailings containing approximately 113 million kg of copper (Sullivan and Towne, 1930). While these resources were not economical to process through mill processing, leaching was identified as a suitable method for extracting the precious metal. Due to the high percentage of fines, the permeability of the tailings deposits was low, resulting in poor metal recoveries. In search of methods for improving the permeability of these tailings deposits with a high percentage of fines, the United States Bureau of Mines undertook an initiative to develop methods for extracting metals from fine-grained ores. This program demonstrated the advantages of agglomeration of fines and clays as a pretreatment process for materials that are difficult to treat by traditional heap leaching techniques. Since the 1970s, many mines have benefited from agglomeration technology (McClelland et al., 1983; Schweizer and Smith, 1987; Phifer, 1987; Butwell, 1990).

A South American gold and copper mine is studying the feasibility of heap leaching a run of mine oxide ore stockpiled from earlier mining operations. Due to the high percentage of fines and specifically clays in the ore, the agglomeration of the ore is being considered. Several small-scale agglomeration strength, stability, and permeability tests were performed, from which the agglomeration and heap lift height specifications were established. These tests showed that effective agglomeration of the ore requires that the moisture content of the ore is within a narrow range. Several previous studies by others such as McClelland et al. (1985), Butwell (1990); Fernández (2003); and Lawandowski and Kawatra (2009) emphasize the importance of achieving an optimal agglomeration moisture content for producing well-formed and permeable agglomerates. Typically, moisture is added in the agglomeration drum to bring the ore moisture of dry ore to the optimal agglomeration moisture content. The concern with the stockpiled oxide ore was that with in situ moisture of the ore as high as 27%, close to the surface of the stockpile, the initial ore moisture may be wet of optimal. Consequently, establishing feasibility of the project required demonstrating that the wet ore could be effectively agglomerated on a continuous (versus batch mode) operation.

The team hypothesized that, with proper adjustment of the agglomeration drum parameters and with pre-treatment of the feed ore, if required, the oxide ore could be effectively agglomerated in a continuous mode. To test this hypothesis, we conducted a pilot-scale agglomeration testing program. The pilot-scale agglomeration testing evaluated the ore agglomeration on pilot-scale equipment similar to that to be used in leaching operations and evaluated the performance of a range of feed ore conditions that could be encountered during leaching operations. A three day pilot-scale agglomeration testing program was

performed at FEECO International located in Green Bay, Wisconsin and included seven rounds of agglomeration testing.

Pilot-scale agglomeration equipment specifications

The pilot-scale agglomeration circuit consisted of the following (Figure 1):

- two belt feeders;
- cement silo with vibrating cement feeder;
- horizontal paddle mixer;
- drum agglomerator (approximately 1 m diameter by 3 m in length);
- four spray nozzles within the drum agglomerator.

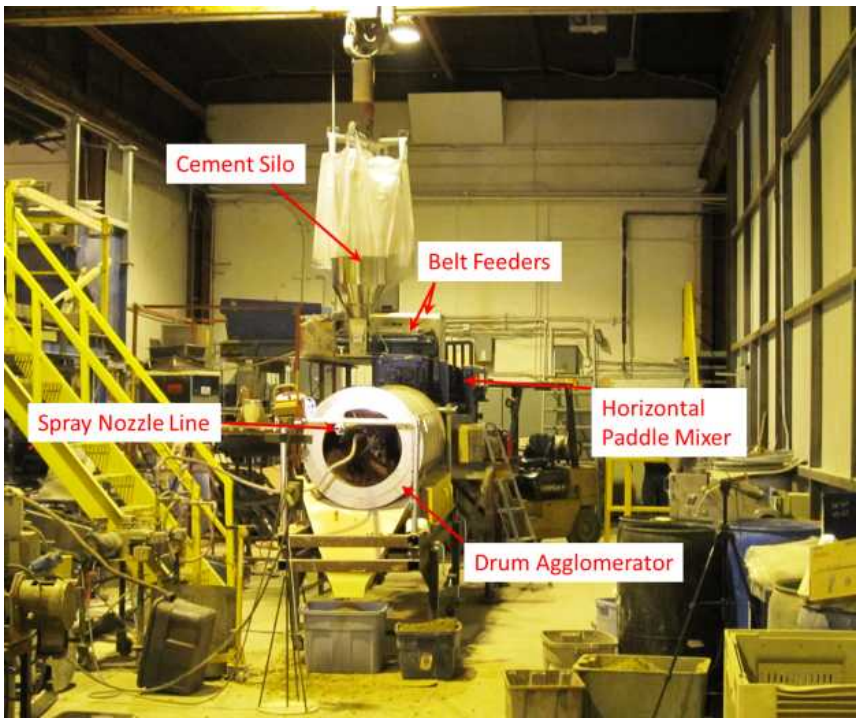

Figure 1: Pilot-scale agglomerator at FEECO laboratory, Green Bay, Wisconsin

The feed ores and cement were added to the paddle mixer and mixed prior to entering the agglomeration drum. Moisture was added, as required, by the spray nozzles located within the agglomeration drum. The drum was rotated at a slight angle (1.8°) so that ore traveled down the length of the agglomeration drum as it was pelletized and then discharged from the drum through a chute and into a plastic container. To increase the drum bed load, and the pellet residence time within the drum, plates were bolted onto the end of the drum (see Figure 2). The dimensions of the agglomerator used in the pilot-scale agglomeration testing are shown in Figure 3.

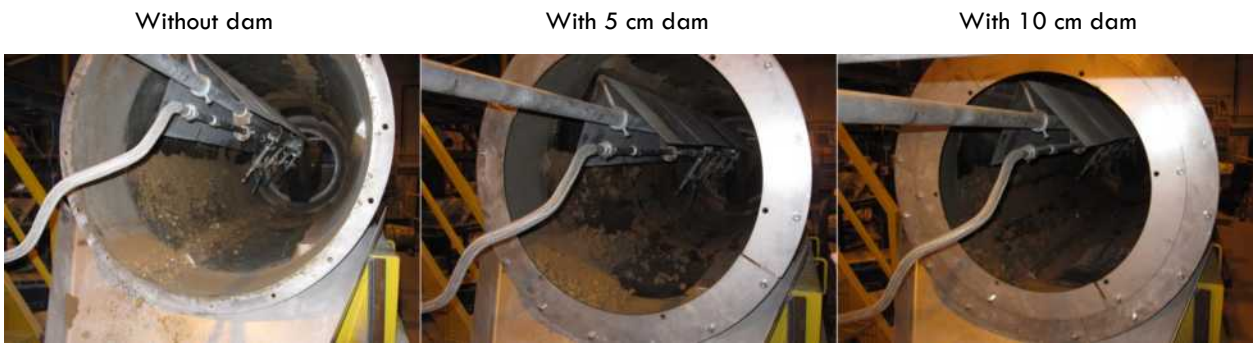

Figure 2: Pilot-scale agglomerator with different dam configurations

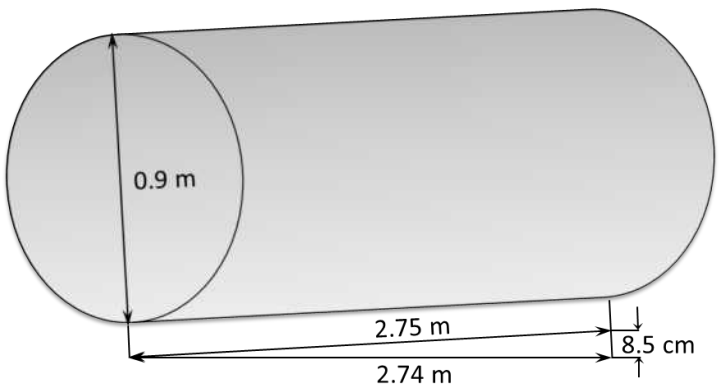

Figure 3: Pilot-scale agglomerator dimensions

Experimental procedure

The ore used for the agglomeration testing was classified as silty sand with clay and gravels. Prior to agglomeration, the ore was crushed to 2.54 cm and smaller size. The particle size characterization of the tested material is shown in Table 1 before and after crushing to 2.54 cm and smaller.

Seven rounds of agglomeration tests were conducted during three days of testing. The different rounds adjusted the drum rotation speed, initial ore moisture content, and drum dam height. The required cement addition had been established in previous screening tests and was not varied in these tests. Table 2 provides an overview of the key parameters used in each round.

Table 1: Particle size characterization of the ore used in pilot-scale agglomeration testing

Attribute	Ore samples (before crushing to 1 inch)	Ore samples (after crushing to 1 inch)
Sieve size	% Passing	% Passing
25 mm	90	100
4.75 mm (No. 4)	66	76
2.00 mm (No.10)	57	64
0.075 mm (No. 200)	28.5	32.1

Table 2: Summary of pilot-scale agglomeration testing rounds

Round	Ore initial moisture content	Additive	Drum dam height	Drum speed	Drum slope	Final agglomerate moisture content (%)
1	13%	15 kg/t cement	N/A	15 rpm	1.8°	15.5%
2	13%	15 kg/t cement	5 cm	15 rpm	1.8°	17.1%
3	13%	15 kg/t cement	5 cm	12 rpm	1.8°	17.6%
4	13%	15 kg/t cement	10 cm	12 rpm	1.8°	19.3%
5	19%	15 kg/t cement	10 cm	12 rpm	1.8°	19.5%
6	21%	15 kg/t cement	10 cm	12 rpm	1.8°	19.6%
7	27%	15 kg/t cement + dry ore at 9% moisture	10 cm	12 rpm	1.8°	16.7%

The first four rounds of agglomeration testing were conducted using the as-received ore, which had a moisture content of 13% (all moisture contents are reported on a dry basis). These rounds were to calibrate the system and establish baseline design parameters such as: the required drum rotation speed, dam height (see Figures 2 and 3), and configuration and application rate of the internal water sprinklers. The goal was to modify the agglomeration process by adjusting these parameters until stable and uniform (around 10 mm to 25 mm diameter) pellets were generated.

In general, during these rounds of testing increasing the added water generated larger pellets. After each round, the strengths of agglomerates were measured by repeatedly dropping the green (uncured) pellets from about a 1.5 m height until surface cracks were visible in the pellets. The target was to reach a drop number of 5 or larger. This was achieved in Round 4 by increasing the residence time compared to previous rounds.

Rounds 5, 6 and 7 focused on agglomeration of ore with initial moisture contents of 19%, 21%, and 27%, respectively. The ore was agglomerated successfully in these rounds; however, different approaches

and adjustments were required at each round. In Round 5, for example, water was added to the feeding ore while for Round 6 minimal to no water was required for successful agglomeration. In Round 7, the ore was mixed with drier ore (with moisture content of 9%) to reduce the moisture content of the ore to lower ranges and facilitate the agglomeration process. After adjusting the ratio of the wet ore (moisture content = 27%) to dry ore (moisture content = 9%) to 1.75 to 1.0, acceptable agglomeration was achieved in Round 7. However, the uniformity of the agglomerates was not as good, compared to the previous rounds (Rounds 4, 5, or 6).

Agglomerate quality testing

We conducted a series of tests on the agglomerates to develop some semi-quantitative metrics of agglomeration effectiveness and quality. The following parameters were measured in FEECO's on-site laboratory during the testing program to evaluate the strength and quality of the agglomerated ore:

- moisture content of the ore agglomerates as they exited the drum;
- drop number of the green agglomerates; and
- agglomeration breakdown.

The procedures for these measurements are described below.

Moisture content

Moisture contents were measured using a compact moisture analyzer MB35 manufactured by OHAUS. The pre- and post-agglomeration moisture content of the ore are shown in Figure 4.

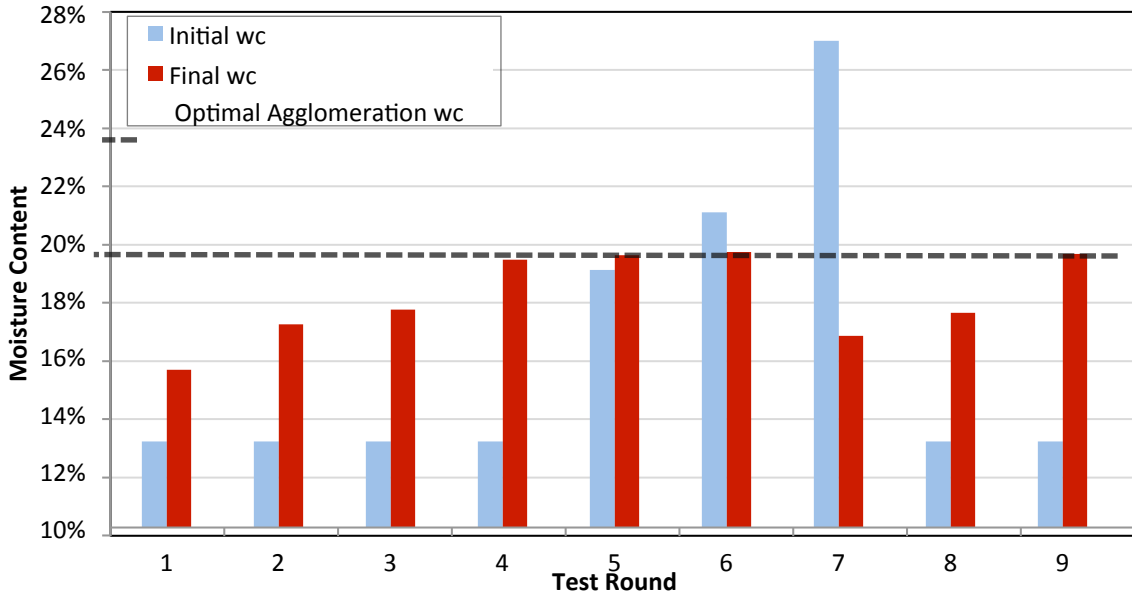

Figure 4: Moisture contents of different rounds of pilot-scale agglomeration testing

Drop number

As an indication of the strength of the agglomerates, during the proposed conveying and stacking process (just after agglomeration), the agglomerates were dropped from approximately 1.5 m onto a flat concrete surface multiple times until the agglomerate showed signs of cracking. The number of drops before visible cracking was recorded as the drop number.

Results

The observations and test results from the seven rounds of pilot-scale agglomeration testing are presented and discussed below.

Ore moisture

Visually competent agglomerates were formed with feed moisture contents ranging from 13% (as-received ore moisture content) to 27%. In the former case (Rounds 1 through 4), moisture was added in the agglomeration drum to increase the moisture content to that required for optimal agglomeration. In the case with an initial moisture content of 27% (Round 7), a drier feed ore (with moisture content of approximately 9%) was blended on a continual basis in the horizontal paddle mixture at a ratio of approximately 1 part drier material to 1.75 parts wetter material. In all cases, the final moisture content producing the best-formed agglomerates was approximately 19.5%. Figure 4 shows the pre- and post-agglomeration moisture content of the ore.

With adjustment of the volume of material in the agglomerator and the amount of moisture and/or blend material, the agglomerate size and density could be adjusted. We observed that the closer the feed ore moisture was to 19.5%, the faster the laboratory team was able to produce agglomerates and the better the agglomerate quality. We suspect that this is because the ore had a more uniform mixture of moisture that was near the optimal agglomeration moisture.

The moisture content measurements also indicate that the ore loses about 2% moisture during the agglomeration process if no moisture is added in the agglomeration drum (for the temperature and relative humidity conditions at FEECO laboratory). Thus the feed ore must have a pre-agglomeration moisture content of 21% to produce agglomerates with 19% moisture without any moisture addition in the drum.

Agglomerate size

The size of the agglomerates was sensitive to the test operating parameters, including moisture addition, location of moisture addition, and residence time in the drum. Once the feed ore, feed moisture, and ore residence time were properly adjusted, agglomerate sizes were predominately in the 8 mm to 15 mm range. An example of agglomerates from the Round 5 agglomeration test is shown in Figure 5.

Figure 5: Pilot-scale agglomerates produced from Round 5

Agglomerate strength and stability

Agglomerate strength, as measured by drop number and crush strength, depended on agglomerate quality and cure time. Agglomerates formed prior to optimization of the agglomeration and ore feed parameters had green drop numbers of between 1 and 2, while the agglomerates formed after the system was optimized had green drop numbers of 5 to 6 drops.

The agglomerates that were soaked overnight in tap water held their shape with little to no sign of disintegration the next morning.

Operating parameters

Several observations on the operating parameters were made during the testing. These observations include the following:

- An important factor in developing uniform and strong agglomerates is achieving a uniform mixture of the ore material with itself and with the cement prior to the mixture entering the agglomeration drum. This is particularly important when a drier feed ore is blended with a wetter ore.

- Placement and tuning of spray jets within the agglomerator greatly affects the agglomeration and can be as important as the amount of water that is added. We observed that proper placement of jets promotes nucleation and layering leading to dense, strong agglomerates. Improper location of spray jets can lead to particle coalescence, lower density, and weaker agglomerates.
- The residence time of the ore in the agglomeration drum is important. Without adequate residence time the agglomerates were not nearly as strong as when proper retention time was allowed.
- Bedload volume, or bed thickness, is also critical for proper compaction of agglomerates. The addition of dams to the pilot scale agglomeration drum increased bedload and provided compactive force to agglomerates prior to discharge from the drum.
- The effect of changing the drum agglomerator slope was not studied.

Conclusions

The pilot-scale testing demonstrated that agglomeration of ore over a wide range of initial moisture contents is possible. Specifically, the test results show that ore with initial moisture contents of up to 27% can be agglomerated. Wetter ore may require blending with a drier ore prior to agglomeration. Producing a drier feed ore may require some preconditioning in field such as disking and drying of a portion of the ore prior to agglomeration (Butwell, 1990). Although this testing program indicated that the ore with high moisture contents can be successfully agglomerated, wet ore could pose other operational challenges in processes, such as crushing and material handling. The outcome of the test is that moisture content in agglomeration is not the limiting factor.

The most successful rounds produced agglomerates with green strength of 5 or more drop numbers. Most particles were spherical in shape with diameters between 8 and 15 mm. No signs of free fines or oversize agglomerated pellets were observed during successful agglomeration. These results indicate stable agglomerates can be formed for our ore through judicious agglomeration techniques.

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A process flow sheet for an oxide gold heap leach facility to manage clayey ore within a limited land area

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Abstract

In order to maximize metals extraction from ore bodies, operators are increasingly looking for ways to treat ore with more complex mineralogy. Development of these resources may require innovative methods to process difficult ores at mines with challenging site conditions. A heap leach facility being planned at a South American mine to recover gold from the oxidized zone of a copper-gold porphyry deposit provides an example of an innovative process flow design. The key aspects influencing the flow sheet are related to the ore properties and the location of the project site. The ore is weathered and has high clay and moisture content. Hydraulic testing of the ore shows that agglomeration and a low lift height will be required to maintain permeability of the heaped ore. Metallurgical testing shows that a high dose of cyanide needs to be added during the agglomeration step to obtain the target gold recovery; the cyanide must be quickly removed from the process water and heap solids to meet environmental requirements. Further, the project site has limited suitable terrain for project development and high annual precipitation. The leach pad will be operated as an on-off type pad, and ore handling will be limited to the five-month drier season. The process described in the flow sheet involves cycling solution through a series of leaching cells, one pregnant leach solution pond, and two barren leach solution ponds. Solution from only two cells at a time is sent through the gold extraction plant, and the solution from the remainder of the cells is recycled to one of two barren leach solution ponds, depending on the stage of rinsing of a given cell. Solution passing through the plant is sent to a cyanide treatment circuit where the cyanide is destroyed so that it is not re-introduced to a rinsing cell, thus promoting faster cyanide removal in the heap. Loading and off-loading of the leach pad cells is done concurrently to cause only minor disruptions to the water balance. A downstream water treatment plant is provided to ensure that discharge from the site is in compliance with the environmental regulations.

Introduction

In order to maximize extraction from ore bodies, operators are increasingly looking for ways to treat ore with more complex mineralogy. Development of these resources requires innovative methods to process difficult ores at mines with challenging site conditions and strict environmental standards. These challenges require continued innovation to process difficult ores more efficiently and sustainably. For example, heap leaching of both clayey low-grade ores and mill tailings with a high amount of fines and moisture has been demonstrated, but only after including special provisions for ore pre-conditioning, crushing, and agglomeration and restricting heap lift height (Pyper and Pangbourne, 1988; Butwell, 1990).

An approach to meeting these challenges is being developed by a mine in South America. The mine is designing a heap leach operation as an alternative for recovery of gold from ore taken from the oxidized zone of a copper-gold porphyry deposit. The ore resource is attractive because it has already been mined and stockpiled as part of early mining activities; however, the stockpiled ore is weathered and has a high amount of clay, in situ moisture, and cyanide-soluble copper. Further, the mine site is land-constrained, with little suitable area for a heap leaching operation. Characterization of the ore properties and site conditions has led to the development of project-specific design criteria that dictate a unique process flow. The characterization methods and development of the process flow sheet are conceptual examples of an approach to heap leaching difficult ores in difficult areas.

Background

A copper and gold mine in South America is currently mining gold and copper sulfides from an open pit at a rate of approximately 19,000 metric tonnes per day (t/day). The sulfide ore is processed with a flotation plant. The sulfide ore body was initially overlain by an oxide ore cap. During early mining operations, the mine stripped the oxide cap, of approximately 6.6 million metric tonnes (Mt), and stored it onsite in two stockpiles. Mineral analyses performed by the mine show that the stockpiled ore has an average gold grade of 1.36 grams per tonne (g/t) and an average silver grade of 1.76 g/t.

In 2011, the mine began developing plans for heap leaching the oxide ore. Several challenges to processing the ore were known at the outset of the project. These challenges included the following:

- The ore has high cyanide-soluble copper.
- The ore is highly weathered with a high percentage of fines.
- The mine has limited available land area.
- The area has high average annual precipitation (on the order of 1,400 millimeters per year [mm/year]).

We recognized that the ore characteristics and site challenges would necessitate a unique leach pad configuration and process flow. We therefore conducted a detailed characterization program to better understand the challenges related to ore properties and to develop project-specific design criteria that addressed the ore properties, land constraints, and site weather. The design criteria then guided the development of the process flow sheet and leach pad configuration.

Ore characterization

A detailed characterization of the properties of the stockpiled ore was conducted. This characterization was developed in four phases and is discussed below.

Phase 1 characterization: Survey of ore properties and selection of ore types

The purpose of the Phase 1 characterization was to understand basic properties and variability of the ore in the stockpiles and then to select representative ore types for further testing. The Phase 1 characterization of the ore properties started with a general survey of the ore in the stockpiles, followed by collection and testing of specific ore types. The purpose of the general survey was to provide a high-level understanding of ore types and variability in the stockpiles and to identify locations in the stockpiles that have ore that is representative of the average across all stockpiles and of the poor (i.e., bottom quartile) ore permeability. The survey was conducted by a field engineer who visually inspected the surface of stockpiles and selected 30 test pit locations that included a wide range of ore physical properties. The field engineer collected samples from each test pit and had the samples tested for particle size distribution and Atterberg limits. The field engineer then selected two of the test pits as representative of average and poor ore types, as characterized by permeability. The field engineer collected larger samples from test pits with the average ore and poor ore (later call the montmorillonite, or MMT, poor ore) and sent these samples for more detailed testing (Phase 2). Later in the project, additional poor ore types (feasibility, or FS, poor ore and kaolinite, or kao, poor ore) were collected and tested (Phase 3).

The findings from the Phase 1 characterization included the following:

- The fines content (minus 0.075 millimeters [mm]) of the stockpiled ore has large variability, ranging from about 10% to about 60%. The ore selected as an average ore has a fines content of about 15%, and the ore selected as the MMT poor ore has a fines content of about 28%.
- In situ ore moisture contents are variable but can be high, particularly for the ore with high fines content. The in situ moisture of the average ore was as high as about 19% at a depth of 4 m, and the in situ moisture content of the MMT poor ore was about 26%.

Phase 2 characterization: Testing of ore hydraulic properties

The purpose of the Phase 2 characterization was to determine the need for and develop criteria for ore agglomeration and also to determine a maximum heap lift thickness. A series of agglomeration screening tests with the average and MMT poor ore types was performed at laboratories in the United States. These tests evaluated a large range of agglomeration treatments with binder additions of cement, lime, and polymer. Agglomerated ores from a subset of these treatments were tested for permeability under loads, simulating loads within a stacked heap. A few initial tests were run to test the sensitivity of permeability to crush size. After these initial tests, the average and poor ores were crushed to minus 25 mm for the permeability tests.

A soil hydraulics laboratory in the United States ran the load-permeability tests using a dual-wall cell with a flexible membrane inside a rigid cylinder (15 cm diameter by 30 cm tall). The lab loaded ore into the cell, and the ore sample was then consolidated by increasing the wall pressure to mimic the lateral earth pressures within the heap. Testing personnel then conducted a series of saturated hydraulic conductivity tests at different side-wall and overburden pressures that simulated different heap loads (heights).

Results of the load-permeability testing are shown in Figure 1. The criterion for an acceptable load-permeability test is that the measured permeability (Ksat) at an estimated heap height must be at least a factor of ten times (10×) greater than the design leach solution application rate (10 l/hr/m^2). The 10× factor accounts for uncertainties in the scale-up from column-scale to full-scale heap loads. The findings from the load-permeability testing included the following:

- The average ore (coarser-grained ore), agglomerated without addition of cement, can support heap heights of up to approximately 20 m, using the 10× factor criterion. No improvement in load-permeability performance was seen when the average ore was agglomerated with 6 kg/t cement; however, the tests show that the average ore can support heap heights greater than 30 m when agglomerated with 15 kg/t cement.
- The Ksat of the minus 25 mm MMT poor ore is much lower than the Ksat of the average ore at all loads tested. The Ksat versus load of the MMT poor ore improves with the addition of cement during agglomeration; however, even for the sample agglomerated with 25 kg/t cement, the Ksat of the MMT poor ore falls below the 10× factor at an estimated heap height of about 5.2 m. The Ksat versus heap height curve is steep for all treatments of the MMT poor ore, indicating that the effectiveness of leaching the MMT poor ore will be sensitive to heap lift height.

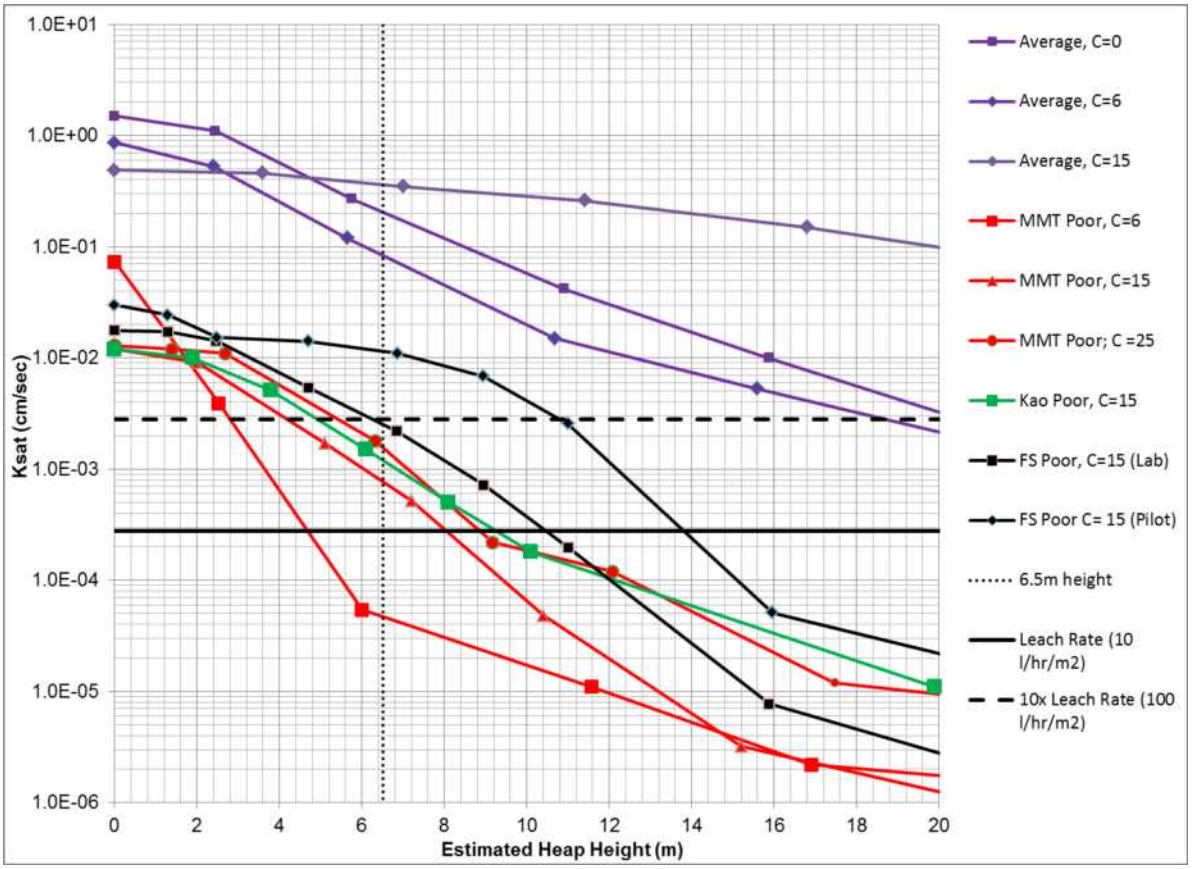

Figure 1: Saturated hydraulic conductivity (Ksat) versus estimated heap height for ores tested
(Notes: C is the amount of cement addition in kilograms per tonne; Lab is laboratory-scale agglomeration; and Pilot is pilot-scale agglomeration)

Phase 3 characterization: Verification sample collection and testing

The Phase 2 characterization indicated that designing the heap leach for the MMT poor ore type would require low lift heights and high agglomeration cement additions, which would limit design options and significantly increase capital and operating costs. Therefore, the purpose of the Phase 3 characterization was to re-evaluate the appropriateness of basing design criteria on the performance of the MMT poor ore by determining the mineralogy of the MMT poor ore type and the relative abundance of ores with similar mineralogy in the stockpiles.

The Phase 3 characterization involved a detailed assessment of clay types in the stockpiles. Boreholes were drilled on a 40 m × 40 m grid in both stockpiles (27 boreholes through Stockpile 1 and 69 boreholes through Stockpile 2). Borehole samples were composited in 2 m length intervals, and clay contents of the samples were determined using a spectral mapping instrument. The mapping indicated that the average kaolinite grades are 5% in Stockpile 1 and 7% in Stockpile 2; the average montmorillonite grades are 0.16% in Stockpile 1 and 0.38% in Stockpile 2. The mapping also found no correlation of clay with depth in the stockpiles.

Testing of the poor ore clay mineralogy through X-ray diffraction (XRD) found that the MMT poor ore sample contained an abnormally high amount of montmorillonite clay compared to the stockpile averages. We hypothesized that the montmorillonite may be a driver in the low permeability of the MMT poor ore. Accordingly, establishing design criteria based on the MMT poor ore would result in an overly conservative and expensive design.

To test our hypothesis, two additional ore samples were collected:

- The kaolinite poor ore (kao poor ore) was collected from an area with a high kaolinite to montmorillonite ratio. The particle size distribution (PSD) of the kao poor ore was manipulated in the laboratory to mimic the PSD of the MMT poor ore.
- The FS poor ore was collected after surveying areas of high kaolinite to find an ore with a PSD that was similar to the PSD of the MMT poor ore.

The load-permeability testing on the kao poor ore and FS poor ore was conducted with an ore top size of 25 mm and with the ore agglomerated with addition of 15 kg/t of cement. Multiple tests were conducted with the FS poor ore with different agglomeration methods and cure times. Shown in Figure 1 are two tests for the FS poor ore: one with the ore agglomerated in a continuous mode in a pilot-scale drum (pilot) and one with the ore agglomerated in a batch mode in a bucket at the laboratory (lab).
The findings from the verification load-permeability tests are outlined below.

The Ksat of the minus 25 mm kao poor ore is only approximately 50% higher than the minus 25 mm MMT poor ore agglomerated with 15 kg/t cement at estimated heap heights below about 7 m. This improvement is likely within the experimental accuracy limits. The improvement increases to about a factor of 10 between estimated heap heights between 7 m and 20 m. This may indicate that the kao poor ore is more competent under high loads than the MMT poor ore; however, the Ksat for both ores reduces rapidly under estimated heap heights above about 2 m.

The Ksat values for the minus 25 mm FS poor ore agglomerated in the drum show little decrease below estimated heap heights of about 8 m and are above the 10× factor to about 11 m. In comparison, the Ksat values for minus 25 mm FS poor ore agglomerated in the laboratory show an immediate decrease with load and drop below the 10× factor at an estimate heap height of 6.5 m. This result shows the importance of agglomeration quality on the heap leach hydraulic performance.

Phase 4 characterization: Testing of metallurgical properties

A metallurgical laboratory in Peru performed a series of column leach tests to understand the metallurgical properties of the ore. The ore used for the leach testing was a composite ore collected by the mine metallurgical team to target average gold and silver grades. Tests were conducted using column lengths of 3 m and 6 m and column diameters of 15 cm and 30 cm. The metallurgical lab varied several

design parameters during the testing, including ore top size, amount and method of cyanide addition, and amount of cement added during agglomeration.

Key findings from the metallurgical column leach tests include the following:

- Achieving consistently high gold recovery (between 85% and 90%), requires crushing the ore to minus 25 mm and adding 1 kg/t of NaCN during agglomeration. Gold recoveries showed little benefit from adding additional NaCN to the leach solution. The highest gold recoveries occurred when the ore was agglomerated with 15 kg/t cement.
- Gold recoveries for columns with minus 25 mm ore agglomerated using 15 kg/t cement and 1 kg/t NaCN were rapid, with greater than 85% recovery, typically occurring within 12 days or less, or a solution to ore ratio of around 1.
- Cyanide-soluble copper in the ore head assay was above 0.03% and peak copper concentrations in the leach solution consistently exceeded 1,000 mg/l and approached 2,000 mg/l. The high amount of cyanide-soluble copper is expected to be the driver behind the high cyanide consumption.

Site conditions

The mine site is compact, covering an area of approximately 540 hectares. Within this area are the open pit, a tailing storage facility (TSF), current and planned waste storage facilities (WSFs), topsoil stockpiles, sulfide plant processing facilities, a worker's camp, and offices and maintenance facilities. The little area that remains generally has steep terrain and is prone to epikarst and karst sink holes connecting to water supplies.

The site is also surrounded by sensitive environmental areas. Discharges into the environment are controlled through strict environmental regulations and demand the highest level of environmental stewardship and compliance for this project. Karst voids are known to sometimes provide a direct conduit to rivers and streams, so all potential leakage of process waters must be eliminated.

The site is located in an area of high rainfall. The average yearly rainfall at the site is over 1,400 mm, with over 2,100 mm recorded during the wettest years. On average, 80% of this rainfall occurs between October and April (wet season). During the wet season, rainfall occurs daily and the ground surface is generally always wet.

Design criteria

The physical and metallurgical properties of the ore and the site conditions dictated several design constraints. Relevant criteria based on these constraints are listed in Table 1.

Table 1: Relevant design criteria

Parameter	Criteria	Comment
Ore top size	25 mm	Required for gold recovery
Ore agglomeration		Required for permeability under load
Cement addition	15 kg/t	Required for gold recovery and for permeability under load
Cyanide addition (as NaCN)	1 kg/t	Required for gold recovery as a result of high cyanide-soluble copper
Water addition	Variable	Varies by ore type
Ore handling	Dry season only (May to September)	To minimize challenges associated with handling clayey ore
Stacking method	Conveyor to radial stacker	To minimize compaction of agglomerated ore
Maximum lift height	6.5 m	Based on load-permeability testing
Minimum leaching time	24 days	Two times greater than time required to reach 85% gold recovery in column testing
Leach solution application rate	10 l/hr/m^2	Standard leaching rate
Heap operational life	7 years	Required by mine plan

Leach pad configuration

The design criteria imposed several restrictions on the type and configuration of the leach pad, which in turn dictated several aspects of the process flow design. The available area at the site generally has steep terrain and is most amenable to a valley-fill or impoundment type leach pad. The requirement that the maximum lift height be 6.5 m precluded these options without the use of multiple stacked layers with interlift liners and drains; however, this option was rejected due to geotechnical concerns.

The alternative that was selected was to construct a single-lift leach pad on a WSF currently under construction. Construction of the WSF will be suspended during the construction and operation of the heap leach. At the time of suspension of the WSF construction, the available area for the leach pad will be 105,000 m^2, which is sufficient for a 6.5 m high heap operated for 7 years with 0.94 million tonnes being processed each year. Because the heap leach will be a single lift, the heap will be operated as a dynamic

(on-off) type heap. At the end of each leach cycle, the spent ore (ripios) from the heap will be off-loaded and sent to a WSF adjacent to the pad, where it will be stored with other mine waste.

The operation of an on-off type heap leach will impose additional conditions on the leaching process. Concurrent with crushing, agglomerating, stacking, and leaching of 1.1 million tonnes per year, the same amount of ripios will need to be off-loaded from the heap, transported, and placed in the ripios area. The ripios will need to be rinsed to reduce residual cyanide concentrations prior to off-loading to limit exposure of operators and equipment to residual levels of cyanide and to limit potential environmental releases of cyanide.

Process flow

The design criteria given in Table 1 and the additional requirements imposed by the on-off type leach pad necessitate a unique process flow, shown in Figure 2. Figure 2 shows two phases of the process flow. The first phase (top illustration) will occur during the drier season, and the second phase (bottom illustration) will occur during the wet season. Each illustration represents one point in the leach cycle.

The process described in the process flow diagram involves cycling solution through a series of leaching cells, one pregnant leach solution (PLS) pond, and two barren leach solution (BLS) ponds. Solution from only two cells at a time will be sent through the gold extraction plant, and the solution from the remainder of the cells will be recycled to one of two BLS ponds, depending on the stage of rinsing of a given cell. Solution passing through the plant is sent to a cyanide treatment circuit where the cyanide will be destroyed so that it is not re-introduced to a rinsing cell, thus promoting faster cyanide removal in the heap. Loading and off-loading of the leach pad cells will be done sequentially to cause only minor disruptions to the water balance. A downstream water treatment plant will treat BLS so that discharge from the site is in compliance with the relevant environmental regulations. Components of the process flow are further explained below.

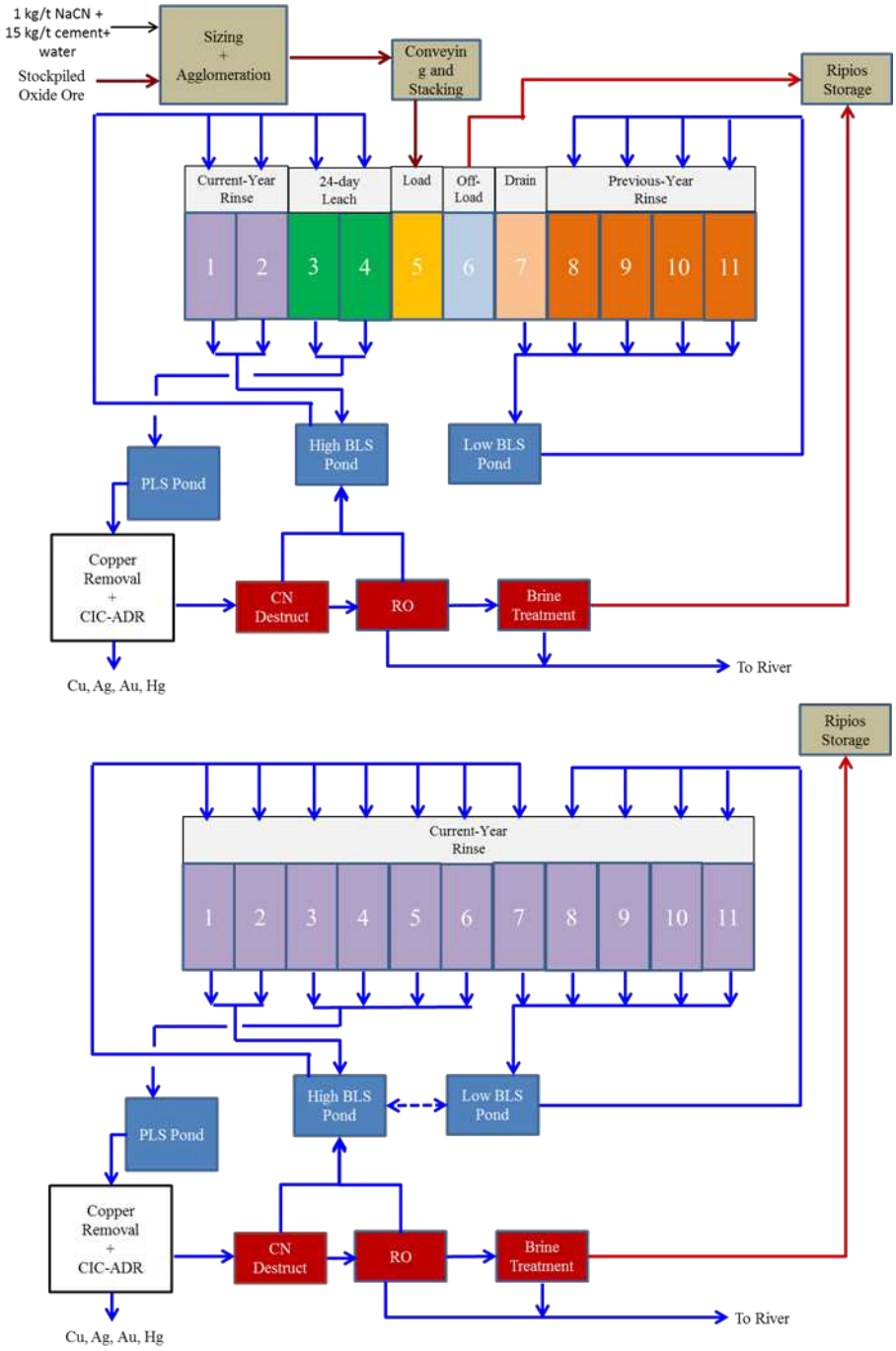

Figure 2: Process flow diagram for (top) dry season, May through September, and (bottom) wet season, October through April

Ore preparation

The stockpiled ore will be delivered to a crushing and agglomeration circuit where it will be crushed to a top size of 25 mm and then agglomerated using 15 kg/t cement. Sodium cyanide will be added to the ore

during agglomeration at a rate of 1 kg/t. Ore will be transported, crushed, and agglomerated only during the dry season (from about May through September).

Cell configuration and stacking

The leach pad will consist of 11 heap leach cells. Each cell will hold approximately the same volume of ore and has the same leaching area. The cells will not be physically separated but will be hydraulically separated by the collection piping. The leach pad will be plumbed so that solution draining from each of the 11 cells can be collected separately and directed to either the PLS pond or returned to one of the BLS ponds, depending on the point in the leaching cycle.

Cells will be stacked sequentially, starting with Cell 1. At the start of the dry season (approximately 1 May) in the first year of the heap leach operation, ore will be conveyed to Cell 1 and stacked to a lift height of 6.5 m. After Cell 1 is loaded, stackers will immediately move to Cell 2, and loading of Cell 2 will begin. This process will proceed until all 11 cells are loaded. Each cell will be loaded in approximately 12 days so that the pad is fully loaded in 132 days. No cell loading will occur the remainder of the year.

The start of the second year will begin 24 days earlier than the previous year (approximately 6 April). At this time, solution application to Cell 1 will terminate, and Cell 1 will be allowed to drain for 12 days. After the 12-day drain-down period, the ripios in Cell 1 will be unloaded and transported to the ripios storage facility. Simultaneously, solution application to Cell 2 will terminate, and Cell 2 will be allowed to drain. After 12 more days (approximately 1 May), Cell 3 will be allowed to drain, the ripios in Cell 2 will offloaded, and Cell 1 will be reloaded in 12 days. This pattern will continue until all cells have been drained, unloaded, and reloaded with fresh agglomerated ore. Figure 2 (top) shows Cell 5 being loaded during year 2 operation.

Loading and unloading in years 3 through 7 will proceed as in year 2.

Leach solution application

Each of the 11 cells will sequentially receive cyanide-free BLS. The start of solution application in the first year of the heap leach operation will begin at the beginning of the dry season (approximately 1 May). At this time, Cell 1 will be loaded with agglomerated ore, and the high BLS pond will supply BLS to Cell 1 at the rate of 10 l/hr/m^2. Twelve days from the start of leaching of Cell 1, Cell 2 will have been loaded with agglomerated ore, prepared for BLS application, and begin to receive BLS from the high BLS pond. Likewise, Cell 3 will be prepared and will start to receive solution from the high BLS pond 12 days after the start of leaching of Cell 2. This process will continue until all 11 cells are receiving solution from the high BLS pond at 121 days after Cell 1 began receiving BLS. The 11 cells will continue to receive

solution from the high BLS pond until solution application is terminated in Cell 1 in preparation for offloading ripios and reloading fresh ore for the year 2 cycle.

The sequence for BLS application to the cells in year 2 will be similar to the sequence for year 1 with the exception that when Cell 1 starts to receive solution from the high BLS pond for the second year, all other cells (Cells 2 through 11) – which will still be leaching the ore from the previous (year 1) cycle – will start receiving solution from the low BLS pond. When Cell 2 is prepared for leaching in year 2, it will start receiving solution from the high BLS pond, and Cells 3 through 11 will receive solution from the low BLS pond. This sequence will continue until all cells are leaching their second cycle of ore and are receiving solution from the high BLS pond. Figure 2 (top) shows Cells 1 through 4 receiving solution from the high BLS pond and Cells 8 through 11 receiving solution from the low BLS pond.

Solution application in years 3 through 7 will proceed as in year 2.

Leach solution collection

Each of the 11 cells will have a separate leach solution collection piping configuration. The collected solution will flow to a sloped collection pipe located at the down slope edge of the pad liner. During the first 132 days of a leach cycle, solution collected from cells that have been leaching for 24 days or less (for a given cycle) will be drained to the PLS pond. Solution from cells that are in the same leaching cycle as the cells with solution draining to the PLS pond will drain to the high BLS pond. Solution from cells that still contain ore from the previous leach cycle will drain to the low BLS pond. In this way, leachate from cells that contain ore from the current leach cycle will be kept separate from leachate from cells that contain ore from the previous leach cycle.

After the first 132 days of a given leach cycle, Cell 11 will have been leaching ore from the current cycle for 12 days, and Cell 10 will have been leaching ore from the current cycle for 24 days. Starting on day 133 of the cycle, solution from Cell 1 will again be sent to the PLS pond for the next 24 days. The drainage cycle then repeats, with two cells always draining to the PLS pond in a staggered pattern of 24 days each.

Process ponds

The PLS pond will receive PLS from two cells at a time. This pond will discharge PLS to the process plant at the same rate that it receives it (to maintain a constant water level). The operation of the PLS pond will not change throughout the year.

The high and low BLS ponds will be connected by a valved conduit, so that the ponds can function as a single pond when the valve is open and as separate ponds when the valve is closed. From about May through September, when cells are being loaded with fresh ore and are being brought into a new leach cycle, the valve will be closed and the two ponds will function independently. Near the end of October,

when all cells are leaching on the same yearly leach cycle, the valve will open, and the two BLS ponds will function as a single pond.

The reason for the separate high and low BLS ponds is to keep solution from cells in the current year's leach cycle separate from solution from cells in the previous year's leach cycle, so that cyanide is not introduced into cells where the ore has been under rinse for a long period of time and will soon be off-loaded as ripios. The separate high and low BLS ponds will prevent this residual cyanide from being applied to the cells that are completing their rinse from the previous leach cycle.

The sizing of the two BLS ponds accounts for the displaced solution in the heap during periods when cells will be drained, unloaded, and loaded. During this period, three cells will receive no BLS solution, and the solution that would be held in the three cells will drain to the BLS ponds.

Solution processing and water treatment

Solution entering the process plant from the PLS pond will move through a sulfidization, acidification, recycling, and thickening (SART) circuit to remove cyanide-solution copper. The solution will then pass through a carbon-in-column (CIC) adsorption desorption recovery (ADR) circuit to remove gold. All of the solution will then move through a cyanide destruction step. After the cyanide destruction step, the flow will be split. Approximately 60% of the flow will be recycled to the high BLS pond without further treatment. Approximately 40% of the flow will pass through a soda ash softening and reverse osmosis circuit. (The split in flow is determined through a technical and economic trade-off between costs for water treatment and the costs and challenges associated with recirculating and accumulating salts and other constituents in the heap leach circuit.) Treated water from the reverse osmosis (RO) unit will meet regulatory discharge requirements and can either be discharged to the river or recirculated to the high BLS pond, depending on the water balance of the heap leach circuit.

Unlike traditional gold heap leach operations, the process does not attempt to recirculate free cyanide in the leach cycle. Rather, a goal of the process is to remove cyanide as quickly as possible. Free cyanide is regenerated in the SART process, but the principal advantage of this regeneration is to have the cyanide in the form most easily removed by the cyanide destruction step. Rapid removal of cyanide promotes concurrent leaching and rinsing of the heap. As indicated in the metallurgical testing, high gold recovery occurs when the cyanide is supplied to the ore as a shock dose during ore agglomeration; adding more cyanide to the leach solution has little measureable benefit.

Conclusions

The process flow was designed to address the design constraints identified during the testing phases, the land constraints imposed by the mine site, and the environmental requirements. These process design features include the following:

- Shock addition of cyanide during agglomeration to promote early and rapid leaching of gold.
- No addition of cyanide to the BLS to promote concurrent rinsing with leaching of the heap.
- Separation of solutions containing higher and lower cyanide concentrations to avoid cross contamination of fresh ore with rinsed ore.
- Single 6.5 m lift leach pad to maintain sufficient permeability for leaching.
- On-off type leach pad operated in 11 cells that can be loaded, leached, drained, and unloaded simultaneously to allow continuous leaching in a relatively small area.
- Handling of ore during the dry season only to alleviate challenges associated with wet, clayey ore.

The process flow sheet demonstrates one project-specific method for handling difficult ores and/or challenging site conditions. As the project moves forward, the governing design criteria are being verified and optimized through further testing. We expect this process flow to be refined and to become even more efficient. For example, increasing the length of the period for ore handling will increase annual throughput, or conversely decrease the required pad size, subject to the required rinsing durations. Unique solutions, such as that presented here, will become increasingly necessary to maximize metals extraction.

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Characterization of heap leach ores using dye penetration and mineralogical analysis

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Abstract

Heap leaching is a cost-effective process method for a variety of ore types including gold and copper ores. Although climate and ore mineralogy may be conducive to heap leaching, mineralogical and ore availability constraints need to be evaluated to determine any potential limitations for maximum metal recovery. Column leaching in conjunction with dye penetration study and ore mineral availability are newly developed techniques, which integrate the metallurgical evaluation of the ore by column leaching with concurrent petrologic and mineralogical studies using the Tescan Integrated Mineral Analyzer (TIMA).

This paper will present two separate tests. The first is a mineralogical feed characterization of column tests for a porphyry-copper ore body using automated mineralogy techniques to determine Cu-deportment, Cu-mineral association, and total Cu availability based upon the association with surrounding gangue. The second study details the dye penetration of a telluride bearing gold ore, which incorporates TIMA investigation to determine mineral composition, texture and Au distribution with fluorescent dye tests in both a closed non-agitated system, and a column leach system where the gangue mineralogy controls the access of fluids into the rock.

Introduction

Ore availability studies quantify the amount of metal or mineral present in the ore with its accessibility to leach solutions as the availability of the ore mineral may be constrained by the gangue mineral, which surrounds it. As those fluids may ingress to the center of the rock fragment through fractures, veins, pore spacing and around grain boundaries, if the ore mineral is locked within resistant gangue mineral the ore value will not be captured. Using the newly developed TIMA the ore minerals and their associations with leachable gangue and non-leachable gangue can be quantified, providing a complete picture of the ore mineral relationship in the coarsely crushed rock.

Dye penetration studies utilize standard leach solutions doped with a fluorescent dye during column leaching tests, where samples are removed from the column at timed intervals and photographed under

ultraviolet light to determine the degree of fluid ingress into the coarse fragment. The distribution of the fluorescent dye highlights pathways as well as barriers to fluid penetration in test samples. Quantification of the degree of penetration per unit leach time and per recovered metal unit is then calculated once photographs have been processed using image analysis software.

Characterization using automated mineralogy

The mineralogical characterization of ores has often been underutilized in many industries over the years due to the limited statistical nature and significant expense of the analysis (Marsden, 2009). Since the wider acceptance of QEMSCAN in the late 1990s the use of automated scanning electron microscopy using equipment such as the quantitative evaluation of minerals by scanning electron microscopy (QEMSCAN), Mineral Liberation Analyzer MLA (Lotter et al., 2002; Winckers, 2002) and now the TIMA are often a standard part of material characterizations. The use of these instruments allows for a large and statistically relevant amount of data to be captured to effectively detail factors that control leaching behavior of an ore.

The TIMA system is a recent addition to the automated mineralogy market and incorporates the simultaneous acquisition of both back scatter electron (BSE) and energy dispersive X-ray (EDX) spectra to detect and analyze the mineral composition of a prepared sample. The acquisition time at each pixel is adjusted based upon the BSE and EDX signal strength, significantly shortening data acquisition times. The use of an yttrium aluminum garnet (YAG) scintillator BSE detector offers superior imaging and mineral detection allow for accurate mineral detection of ultra fine-grained particles. The mineral content and grain associations are determined in this manner to quantify the ore mineral associations.

Characterization of a low-grade Porphyry-Cu orebody

The characterization of three feed samples from a low-grade porphyry-copper ore body was carried out to detail the ore and gangue mineral compositions in conjunction with a column leach program. The column work employed the combination of acid and bacterial leaching to recover Cu from primary sulfide minerals. The samples were coarsely crushed to minus 20 mm and sized at 8 mm, 4 mm, 1 mm and 150 µm. Each fraction was prepared into replicate 30 mm polished sections with the number of replicates proportionate with each size fraction. Each polished section was analyzed to measure the total mineral abundance, copper deportment and copper-bearing mineral association. Based upon the association of the Cu-bearing minerals with the surrounding gangue minerals a distribution of Cu can be determined based upon the grouping of the gangue minerals resistant to acid solubility and those that are soluble in leach solutions, thus capable of providing access to the Cu-bearing minerals for dissolution.

As the characterization is carried out on coarse sized material some aspects of the overall petrography can be discerned, but the information is mainly focused on the Cu-bearing minerals. The mineral abundance in Table 1 details the composition of the samples, indicating a significant amount of phyllic alteration (potassium bearing mineral alteration such as orthoclase to sericite) in each sample. Figure 1 presents the deportment of Cu by mineral in each sample, which in this case is mostly chalcopyrite with lesser amounts of chalcocite (Col 1), bornite (Col 3) and other Cu-minerals, which include malachite, chrysocolla and brocanthite. The associations of the Cu-bearing minerals are presented in Figure 2 and indicate a strong relationship with orthoclase and muscovite, with moderate amounts associated with quartz and as free/liberated grains. The distribution of Cu in relation to these associations is then calculated as Cu-availability in the basic groupings of "Free", "Leachable" and "Resistant" as illustrated in Figure 3. This provides some indication of the total availability of Cu over time, but this also requires some correlation with penetration rates over time to provide a full understanding of leach rates of Cu.

Table 1: Mineral abundance per sample

Mineral Abundance Wt.,%	Col 1	Col 2	Col 3
Chalcopyrite	0.24	0.46	0.50
Bornite	0.00	0.00	0.01
Covellite	0.00	0.00	0.00
Chalcocite	0.04	0.00	0.01
Enargite	0.02	0.00	0.04
Other-Cu Minerals	0.00	0.00	0.01
Pyrite	2.07	2.81	4.64
Other Sulphides	2.94	2.27	6.48
Quartz	40.0	38.6	31.5
Plagioclase	0.03	0.10	4.68
Orthoclase	12.7	12.8	13.2
Muscovite/Sericite	27.6	28.5	28.2
Biot/Phlog	4.81	5.88	3.20
Chlorite	1.66	3.31	1.85
Sillimanite	6.91	3.71	3.36
Amphibole	0.00	0.00	0.61
Andradite	0.00	0.00	0.01
Epidote	0.02	0.01	0.39
Calcite	0.00	0.00	0.19
Fe-oxides	0.02	0.07	0.44
Other	1.02	1.48	0.93
Total	**100**	**100**	**100**

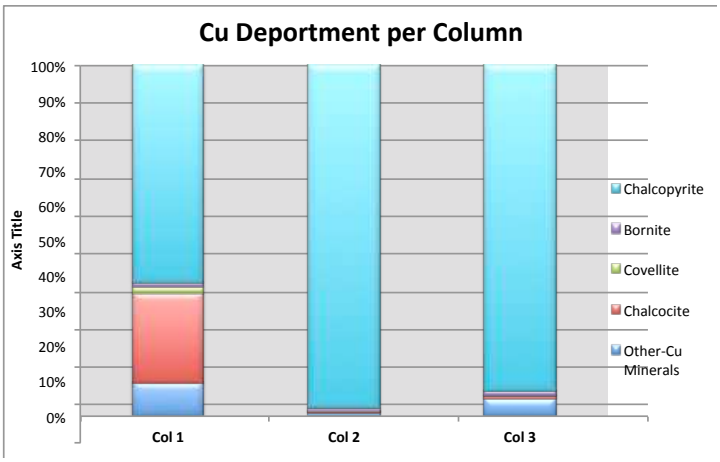

Figure 1: Cu deportment by mineral per sample

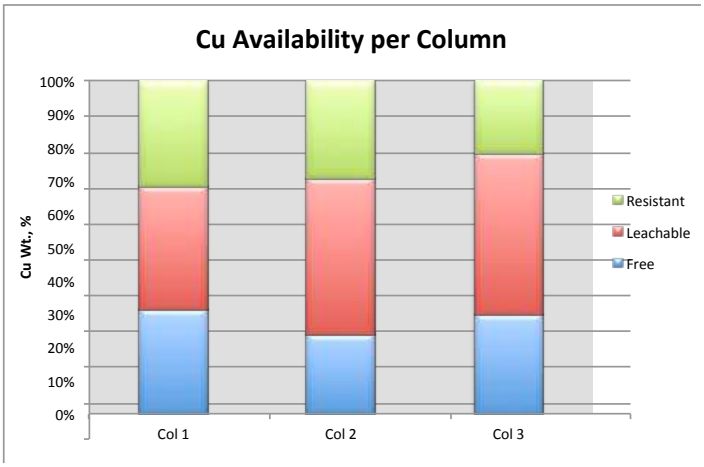

Figure 2: Cu availability by mineral per sample

Characterization of a gold ore using dye penetration and TIMA

Geology and mineralogy

The rate of penetration in the rock will be governed by the overall texture, mineral compositions, alteration facies and degree of metamorphism, which will restrict or accentuate the accessibility of fluid into ore bearing areas. Soluble mineral veins, fractures and highly porous zones may also accentuate areas in the rock to increase solution flow. In the case of gold, leachable gangue has little effect on the access of fluid and the absorption of leach solution is mainly dependent upon porosity, fracturing, veining and grain boundary permeation. Work completed on the Cresson gold deposit in the Cripple Creek district of

Colorado (Lane et. al., 2006) illustrates how these factors and the gangue mineralogy play an important role in metal recovery in a heap leach environment.

The Cresson deposit is composed of volcanic intrusive rocks and volcanogenic sediments where gold mineralization is associated with veins and porous cavities within the ore. The geology of the area is composed of an irregular shaped basin with lithologies that include syenite, tephriphonolite, phonolite and lamprophyre (Lane et. al., 2006). Initial mineralogical investigation identified two differing alteration types within the ore that include a phyllic alteration (sericitic or pottasic) zone dominated by fine muscovite (i.e. sericite) and a propylitic alteration zone dominated with lime minerals such as carbonate, chlorite and epidote which in-filled pore spaces. Measurements of porosity of both these alteration types indicated that the phyllic alteration rocks contained up to 5% macro and micro pore spacing and the propylitic alteration phases contained as little as 0.3% porosity (Lane et. al., 2006). Both these alteration types contain Au-bearing minerals that include calaverite, krennerite, montbrayite, petzite, sylvanite, hessite, steutzite, electrum and native gold (Lane et. al., 2006). Further TIMA analysis of breccia material from this region reveals the textural features that accentuate fluid transportation.

Figure 3 presents a TIMA image of a single breccia fragment and illustrates a large matrix of muscovite/sericite containing irregular and truncated veins of chlorite, carbonate as well as fractured pyrite and phenocrysts of arsenopyrite. The modal mineralogy as determined by TIMA is presented in Table 2 and indicates that more than 47% is comprised of quartz, ~27% of the mass is muscovite/sericite, with ~6% present as chlorite. TIMA also identified 0.24% native gold in this sample. The platy nature of both the muscovite/sericite and chlorite is conducive to fluid flow, albeit it is slower due to the increased surface area of the grain boundaries.

TIMA images of regions within the particle that contain native gold are presented in Figures 4 and 5, and illustrate fine grains of gold along contact boundaries of muscovite/sericite with quartz and adjacent to sulfide grains of arsenopyrite and fractured pyrite. Figures 6 and 7 present the corresponding SEM regions of these TIMA images and further illustrate the fine distribution of gold along these contact veins with chlorite and carbonate in the muscovite/sericite matrix. A total of 814 gold grains were observed by TIMA and the frequency size distribution is presented in Figure 8 and indicates the majority of the grains range in size between 4 μm and 8 μm. Figure 9 illustrates the size distribution of Au by weight where 71% of the mass are grains >64 μm.

Table 2: Mineral abundance Cripple Creek Breccia (CCB)

Mineral	Mass [%]
Quartz	47.5
Muscovite	26.8
Chlorite	6.34
Arsenopyrite	4.64
Plagioclase	3.50
Pyrite	2.85
Carbonates and Phosphates	2.32
Fe-Oxides	1.98
Biotite	1.73
Other minerals	0.92
Orthoclase	0.56
Other Silicates	0.26
Native Gold	0.24
Amphiboles-Pyroxene	0.22
Clays	0.08
Cu-sulphides	0.01
Total	100

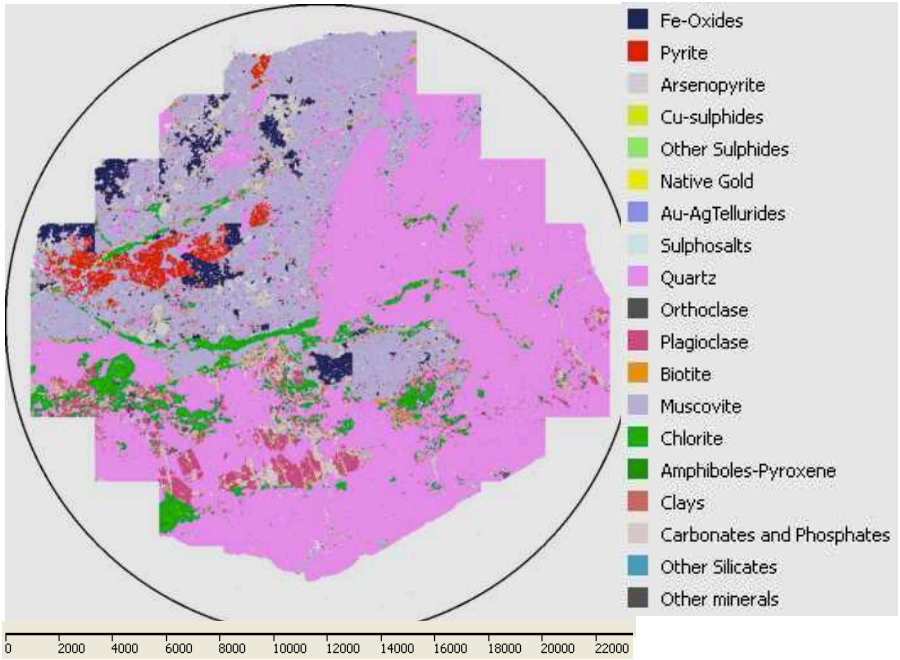

Figure 3: TIMA image of breccia with large muscovite/sericite fragment with minor chlorite and carbonate veining within a quartz matrix

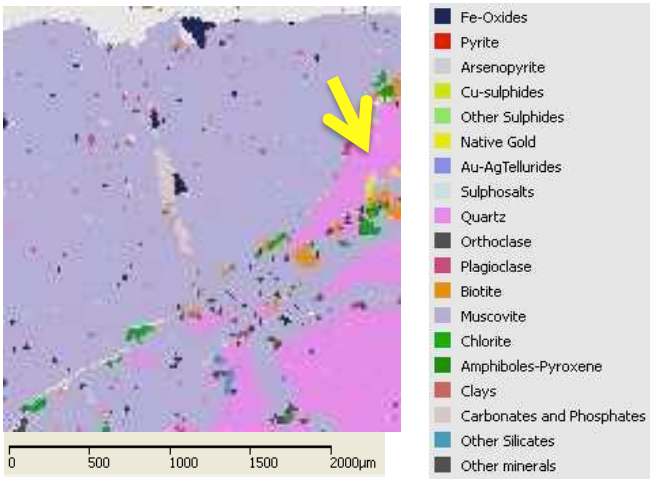

Figure 4: TIMA image of portion of CCB with fine gold along contact of quartz and muscovite/sericite and chlorite

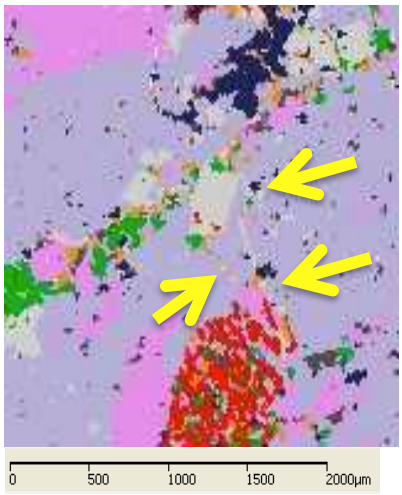

Figure 5: TIMA image of portion of CCB with fine gold in arsenopyrite and pyrite along muscovite/sericite contact

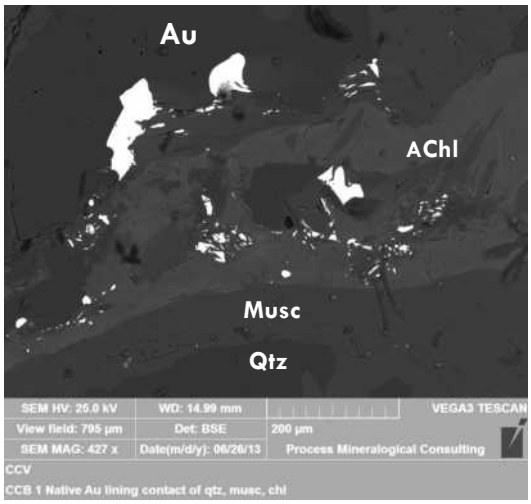

Figure 6: Back scatter electron image (BSI) of Figure 4 (and beyond)

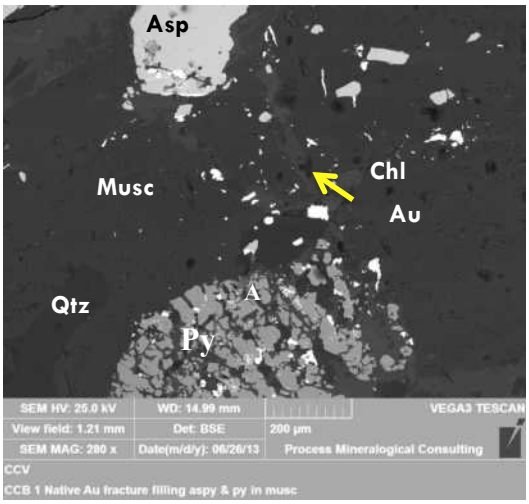

Figure 7: BSI of Figure 5 at higher magnification

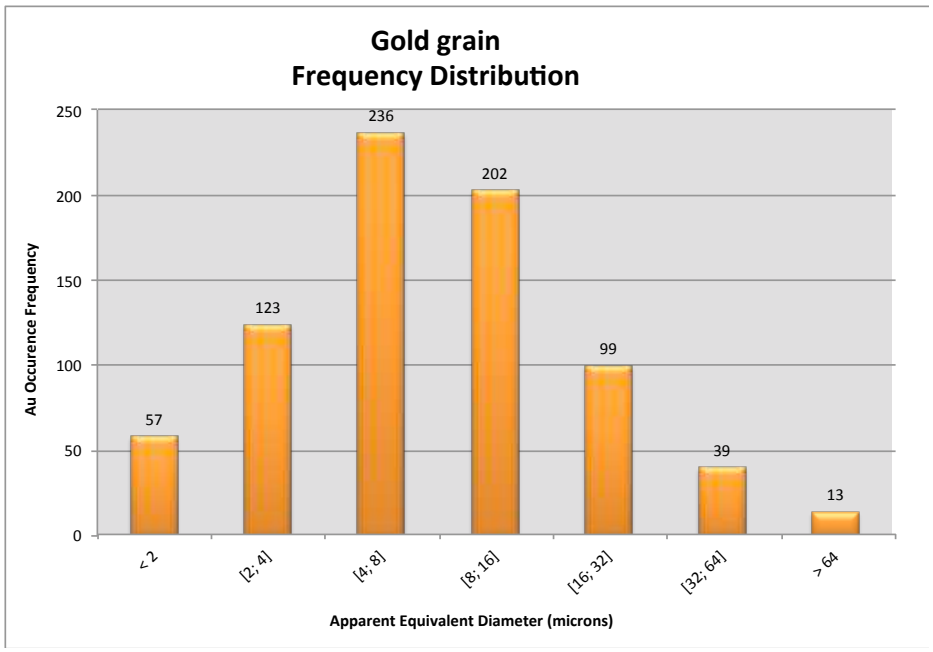

Figure 8: Frequency size distribution for gold grains observed by TIMA

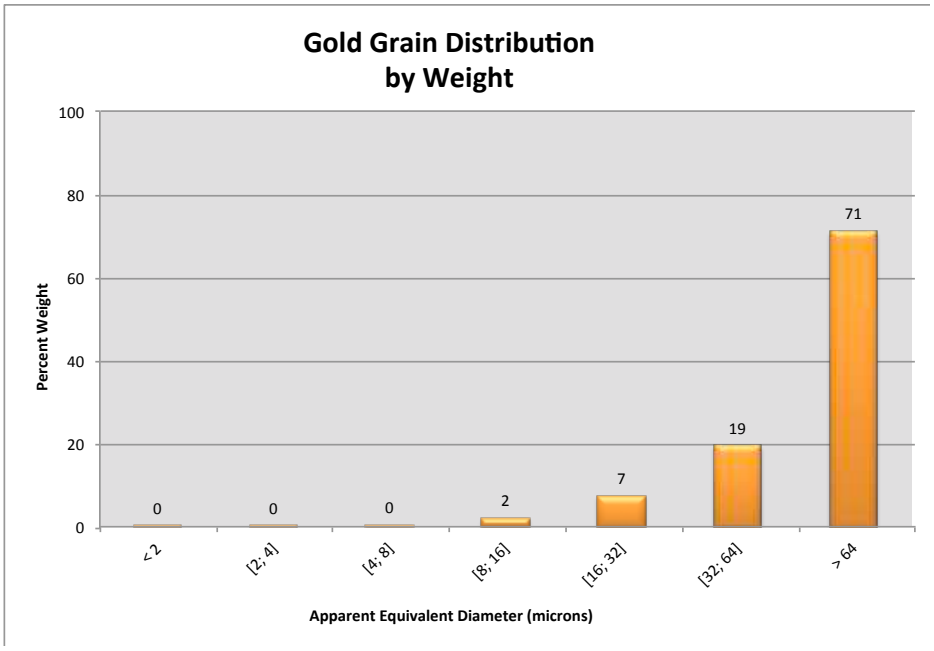

Figure 9: Size distribution of gold by weight as observed by TIMA

Dye penetration test work

A study was undertaken to elucidate pathways of penetration within the rock using a solution of sodium cyanide and ultra-violet sensitive dye (Lane et. al., 2006). The dye used was Pyranine Green 10G and is consistent with that used in the metal and ceramics failure analysis fields; it is manufactured to have an

affinity with highly basic solutions. Similar dyes are also available that have an affinity with acidic solution for Cu-leaching. Having a dye that is compatible with the actual solution leaves no doubt as to the penetration patterns that are present in the ore that control leaching. In order to ensure that the dye was compatible a number of tests were carried out to determine the effect of lime on the dye and the effect of the dye on carbon gold adsorption. In both cases no diverse effects were noted. The addition of the dye with the actual reagent allows for a direct correlation between penetration and extraction. In dye studies carried out by others the dye used was either an oil based product (Witte and Witte, 1984) or a food dye (Summersby et al., 2013) which limited the observation to penetration only. The test procedures employed 1grams / tonne g/t dye in a solution of 2 g/t NaCN solution for the submergence test and 1 gams / litre (g/L) dye to a leach solution of 0.5 g/L NaCN solution at a flow rate of 12 ml/min (Lane et. al., 2006).

Two methods of testing were employed to evaluate the Cripple Creek breccia. The first comprised the stagnant soaking of 50 mm^2 blocks cut from one larger rock within a sodium cyanide solution. The second procedure involved the column leaching of 100 kg of breccia having been crushed to minus 13 mm. Samples were taken from both procedures at timed intervals. These were then sectioned using a diamond saw to reveal the center of the fragment and the fresh surface was photographed under ultraviolet light using an SLR camera and macro lens. The digital images were then analyzed using the Clemex Vision Pro image analysis software package to determine the area penetrated in each particle. The area penetration was then correlated with both optical and SEM-EDS analyses and the recovery of gold in solution. During the submergence test the samples were removed at timed intervals of 8, 16, 32, 64, 128 and 256 hours. In the column test samples were collected after 2, 4, and 7 days, and once a week thereafter, until the maximum gold recovery was achieved.

The submergence test indicated that penetration of the rock can be high when in direct contact with solutions (>90% particle area), but as there was little to no flow of the solution, extraction of the gold out of the rock was limited (Lane et. al, 2006). Limitation of the penetration was consistent with lapilli fragments, oxide and sulfide veins as well as iron-stained material. Figures 9 and 11 illustrate the penetration patterns along veins and platy mineral grain boundaries. The maximum penetration of ~90% particle area is achieved after 64 hours with this method.

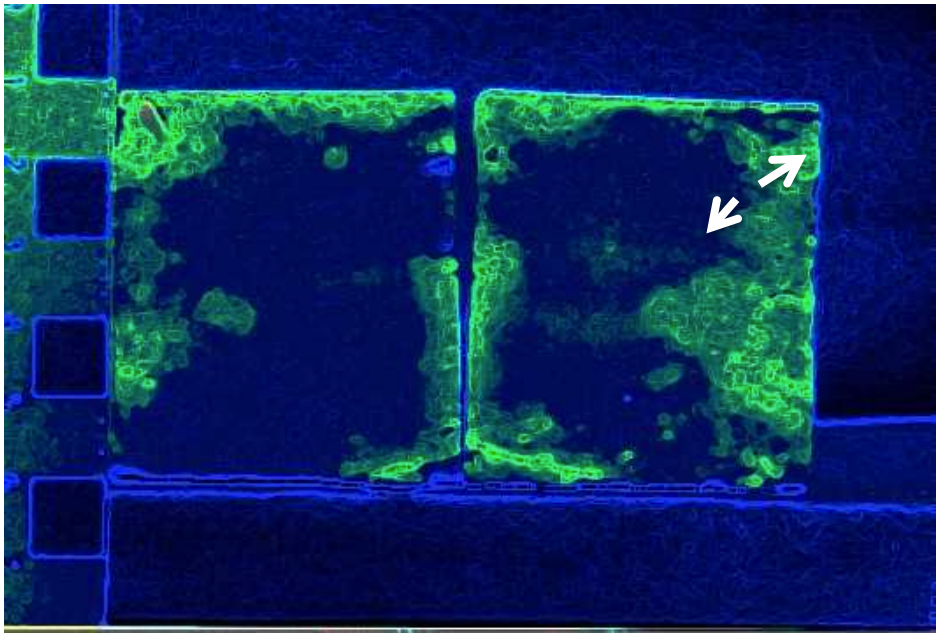

Figure 5: Submergence test 8 hours

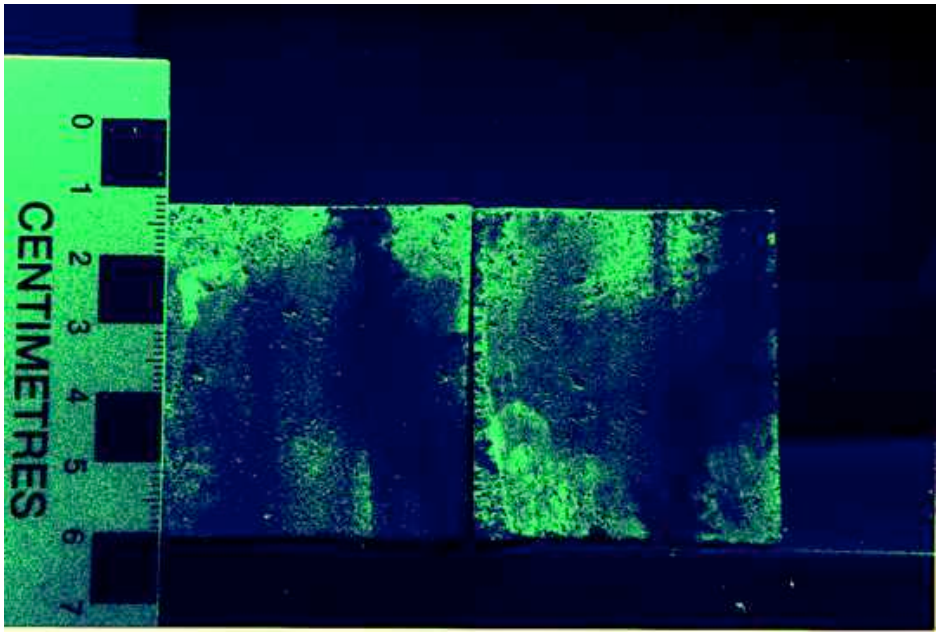

Figure 6: Submergence test 16 hours

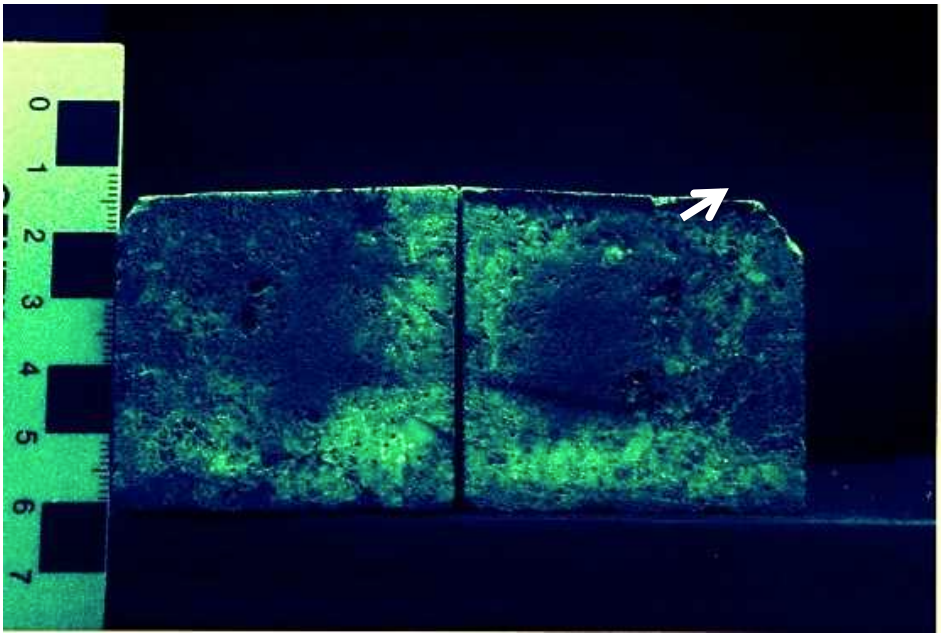

Figure 7: Submergence test 64 hours

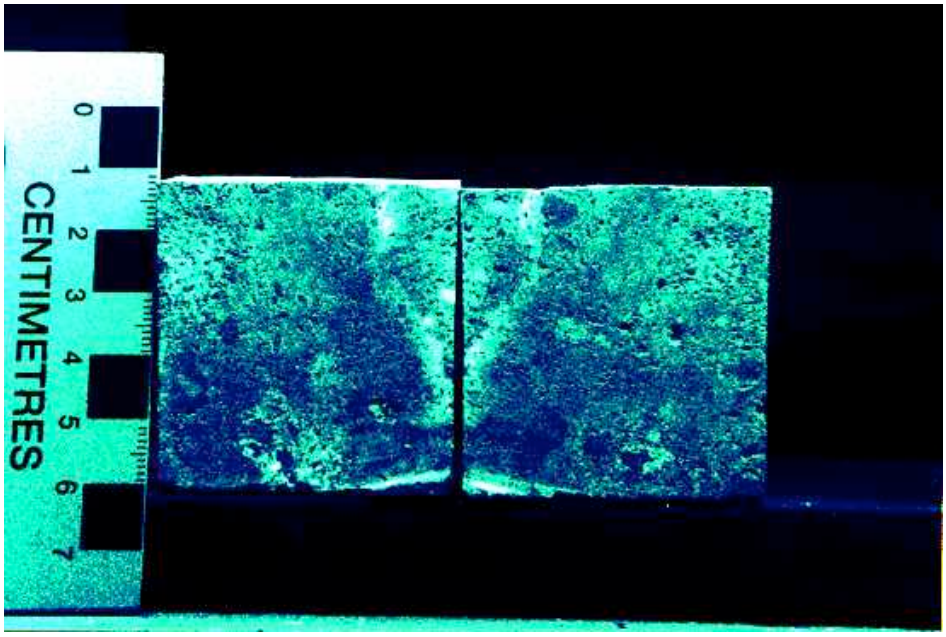

Figure 8: Submergence test 256 hours

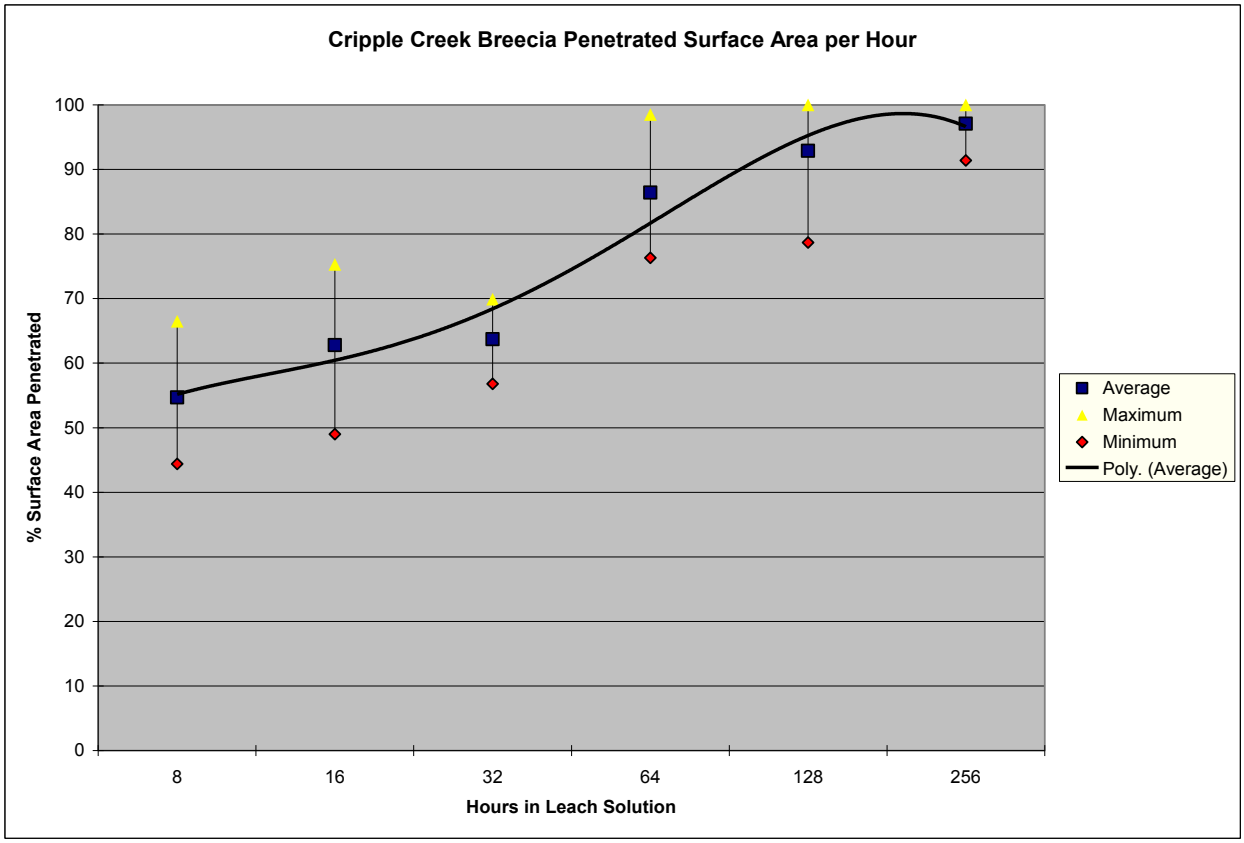

Figure 9: Submergence test illustrating solution penetration versus time in hours

The column test allowed for the correlation of area penetration with extraction over time. Figure 11 illustrates the surface area penetration of 50% of particle after 840 hours, or 35 days, which yields nearly 90% of the extractable gold. The relationship between penetration and gold extraction can thus be plotted and a formula extrapolated (Figure 12). The resulting formula $y = 0.0003\times^3 - 0.0372\times^2 + 1.3788\times$ predicts the possible percent extractable gold based on the area penetrated within the rock. This will be specific to the individual ore and may vary within the deposit depending on changes within the geologic environment.

The variation in ore geology in this work is evident in the sorting of the two alteration types observed, showing that the more porous rocks (phyllic alteration) are more quickly penetrated than the less porous propylitic rocks, which exhibit increased penetration times, as illustrated in Figure 13. The information gained by the TIMA investigation and optical mineralogy indicates that the gold is more associated with the propylitic alteration type, having more contact with the chlorite, carbonate and sulfide veining and contact areas which typically filled areas of porosity in the rock. This is also consistent with the penetration versus extractable gold chart; Figure 12 indicates a slow progression of fluids to yield

greater quantities of extractable gold. Figure 10 B presents assorted images of dyed fragments that illustrate the distribution of penetration.

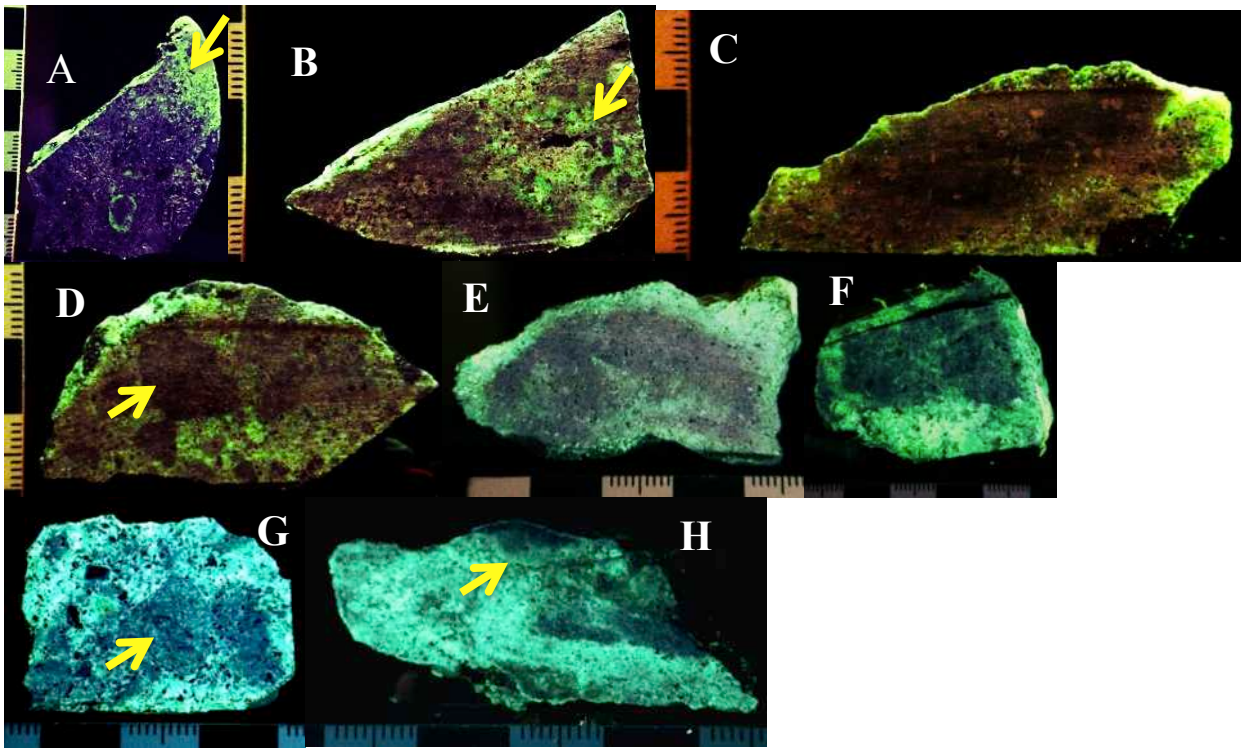

Figure 10: Particle images from the column test illustrating progressive penetration over time as follows: A – 2 days; B – 4 days; C – 7 days; D – 14 days; E – 21 days; F – 35 days; G – 63 days; H – 92 days. Image A clearly shows the influence of flow direction on particle penetration. Image B illustrates a large pore space that has advanced penetration to the center of the particle. Image D and G illustrate large more resistant lapilli fragments within the more sericitic matrix. Image H indicates the minor amount of penetration into the lapilli fragments after 92 days of leaching

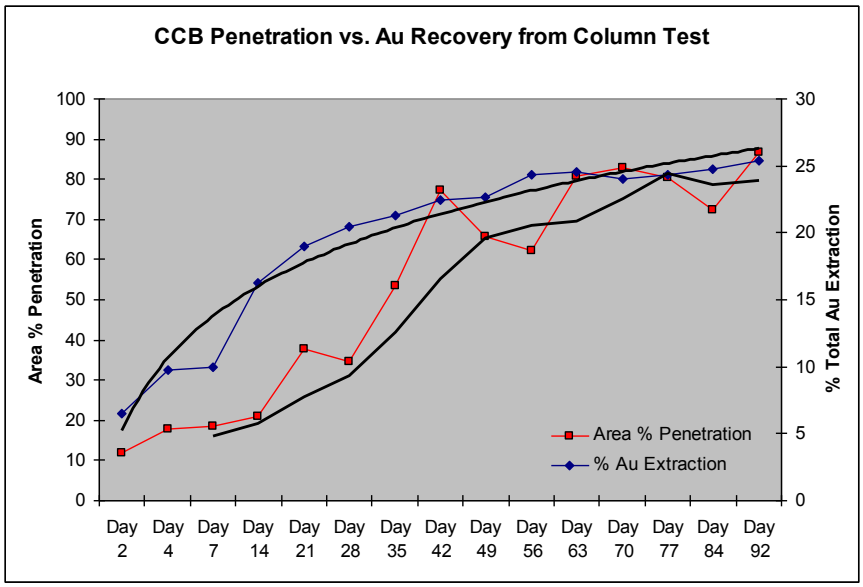

Figure 11: Area penetration and Au recovery plotted over time

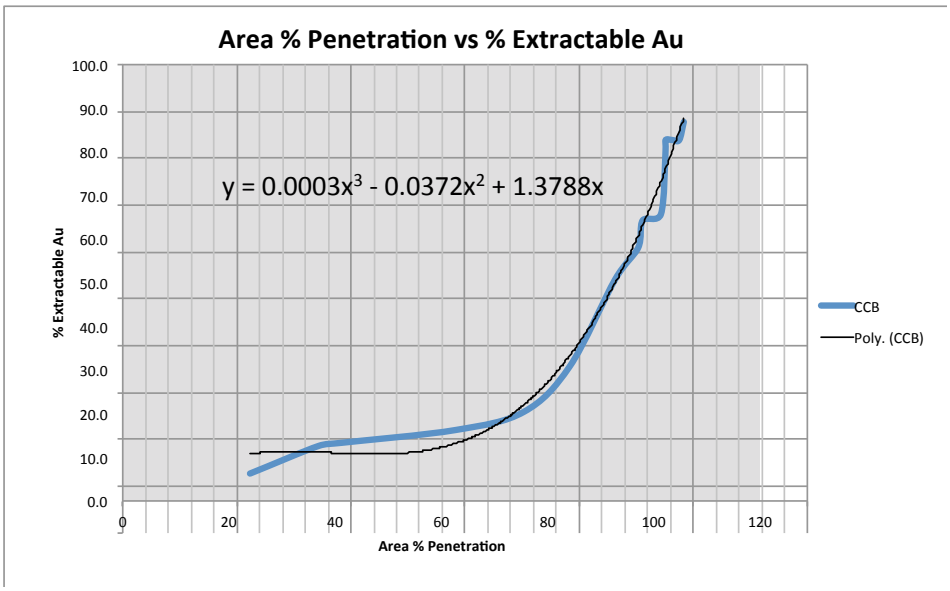

Figure 12: Correlation of area penetration % versus % extractable Au, illustrating that the Au association is more strongly associated with the less porous rock types

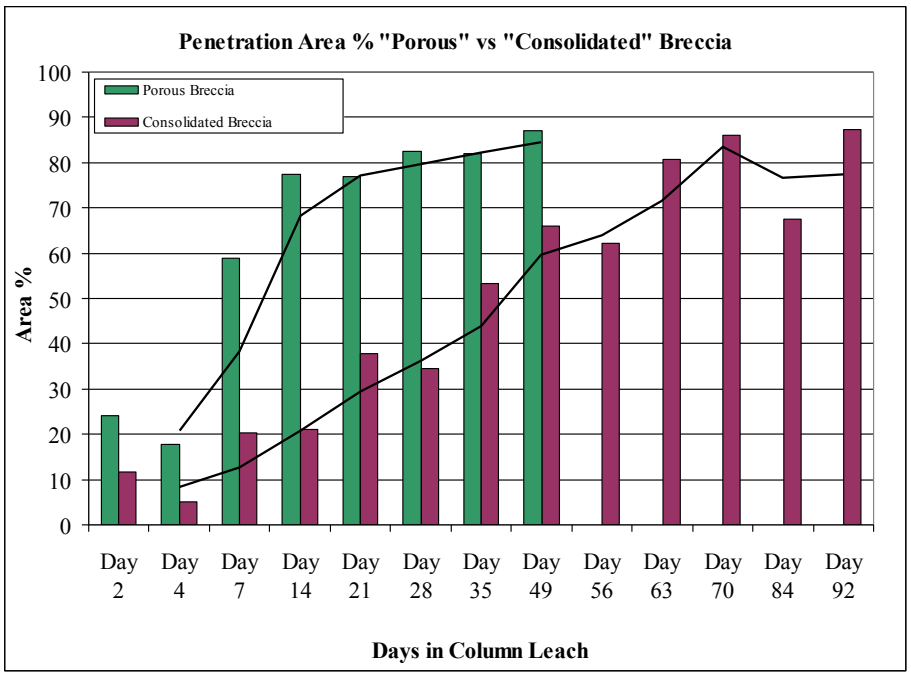

Figure 13: Alteration type penetration over time shows the more porous phyllic rock reaches maximum penetration much earlier than the more consolidated propylitic rock types

Conclusion

The use of automated mineralogical techniques such as TIMA can be successfully applied to heap leach ore characterization through determination of metal availability based upon association or by integration with column and dye leaching to determine controlling factors of leaching. The coarse mineralogy on

particles coupled with the penetration work can add value in assessing areas of the deposit which may respond in specific ways to leach solution, allowing for optimization of heap construction.

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Incorporating gangue and precipitation reactions within a comprehensive CFD model for copper heap leaching

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Mark Cross, Swansea University, UK

Abstract

The heap leach processing of new ore deposits offers considerable challenges for future projects as the mineralogy becomes more complex and the effects of gangue chemistry exert a greater influence on leach behavior. Employing leading edge computational fluid dynamics (CFD) techniques, a comprehensive model has been developed in order to capture the complex multi-phase interactions of a heap leach process. The CFD technology has been successfully employed to describe the critical heap process, which comprises a solid porous media with complex fluid flow, gas and liquid phases, competing solid-liquid reactions, dissolution of minerals and transport through the heap. The model accounts for many of the physical and chemical features of the heap, including transport and reaction kinetics for the important copper and iron sulfide minerals, ferrous oxidation, bacterial effects, thermal energy, acid and sulfate balances. More challenging is coupling the flow models with the reactive dissolution and the sometimes competing aqueous chemical and solid-liquid reactions.

The CFD model has been extended to incorporate dissolution of various species from gangue reactions and precipitation of hydronium and potassium jarosite, ferric hydroxide, alunite, and gypsum. In this contribution, the model structure for considering the leach reactions is described, including dissolution of various species from suites of very complex reaction sequences, also involving gangue reactions and precipitation, all of which are invariably closely coupled with thermal conditions. The model is applied to a typical copper sulfide ore and calibrated to mineral dissolution reaction rates and material hydrological properties. The model is used to simulate behavior for a sulfidized epithermal ore type, containing chalcopyrite, bornite, covellite, enargite, sphalerite, and other sulfide minerals. Simulations to examine the effect of temperature and aeration were performed as a two column in-series leach, with the pregnant solution recovered to a reservoir, uniformly mixed and recycled.

In the example provided in this study, aeration and thermal conditions of the heap were shown to have a significant effect not only on overall copper recovery but also on the concentrations of soluble and precipitated species.

Introduction

Heap leach operations typically employ large stockpiles (with a width and depth of hundreds of meters) of low grade ore reacting over timescales that may be months or years long. In the case of heap closure or tailings abatement of sulfide-containing materials, acid drainage may be an issue for tens and even hundreds of years. These hydrometallurgical problems offer considerable challenges in the development of effective computer models due to the wide range of physical and chemical phenomena present. These phenomena usually involve a series of interacting physico-chemical processes, which can include transport of liquid, i.e., raffinate, and air through the porous stockpile. In the leaching of copper ores, particularly those containing copper sulfides, a sequence of highly interdependent gas-liquid-solid reactions occur, some of which are impacted by the bacterial population acting as a catalyst on the reactions, and the generation and transport of thermal energy. These complex interactions make it difficult to evaluate the impact of process design and control systems on the operation of the heap in a reliable fashion. The trend is toward development of a variety of tools to support engineers in the optimal design and operation of stockpile leach operations. Ekenes and Caro (2012) have described the use of an engineered heap approach, employing the simulation tools utilized here, to improve profitability and optimize metal recovery from a low-grade copper heap leach.

Bartlett (1998) has provided an overview of the various physical and chemical aspects of the heap leach process, including a synopsis of the early development of numerical simulation models. In the past few years, several heap leach models based on computational fluid dynamics (CFD) platforms have been reported in the literature (Leahy, Davidson and Schwarz, 2005; Leahy, Davidson and Schwarz, 2006; Leahy, Davidson and Schwarz, 2007; Pantelis, Ritchie and Stepanyants, 2002). These models provide an excellent foundation for the handling of the transport, mass, and energy balances, but typically require additional methods and procedures to follow the dissolution of various species from the ore and any solution or precipitation reactions that might occur in the system.

As ore deposits and mineralogy become more complex, gangue chemistry will exert a greater influence on leach behavior (Jansen and Taylor, 2003; Baum, 1999). In this paper, a brief description of the basic structure and computational model formulation of a comprehensive CFD heap leach model is provided with a focus on the gangue dissolution and some important solution precipitation reactions that have been incorporated in a copper sulfide heap leach model.

Structure of the CFD model

The general heap leach model is implemented within a continuum physics software framework, thus allowing the model to take advantage of the CFD capability available within this structure. Using the PHYSICA platform (Croft et al., 1995) provides an open structure for incorporating the reaction chemistry through related modules. For CFD models, the geometry of the process is first defined in order to create a domain, that is, a section of the heap or a leach column is divided by a geometrical mesh to form a series of representative element volumes (REV). The physics and chemistry of the heap process are incorporated within the CFD framework, and conservation equations are applied to each REV and solved simultaneously for the entire domain over the simulation time period.

Fluid flow through porous media

Flow through variably saturated porous media is typically characterized by the Richards' equation. Heaps normally operate in the unsaturated flow regime, which has been well described by existing conventional numerical models, but heaps can also have areas of saturation, especially where permeability issues exist. Modeling systems containing both saturated and unsaturated flow regions is a challenge for most CFD codes. A computational procedure for handling both saturated and unsaturated conditions in the same domain environment was developed by McBride et al. (2006). The procedure is based on a transformation method and is fully integrated into the unstructured context within the PHYSICA code. The details of the CFD flow module used to calculate the fluxes of chemical species, gas and liquid flows, and heat balance are described by Bennett et al. (2003a, 2003b, and 2006) and McBride et al. (2005). Typically, van Genuchten equations are employed to solve for the flow.

Reaction chemistry

Each REV contains a solid fraction of ore defined by a size distribution of representative particle sizes, each with their own mineral properties, replicating measured data for the properties of specific ore types. Each characteristic particle size is modeled using a shrinking core equation for each of the reactive minerals specified. The overall chemistry balance for each REV is determined by summing the reaction products for each particle size fraction.

In the case of copper sulfide mineral leaching, the chemical reactions occur between raffinate species (typically ferric ions and/or acid) and reactive minerals in the solid phase. In addition, other liquid-liquid or liquid-solid reactions take place, such as the oxidation of ferrous ions to ferric and the precipitation of ferric iron salts. In the case of copper sulfide heap leaching, ferrous ion oxidation is modeled using an algorithm based on a population of ferrous-oxidizing bacteria. The bacteria are split between a fixed population that remains attached to minerals on particle surfaces and a mobile population

that can be advected through the heap. Separate models for growth rate and oxidation rate are formulated for multiple bacteria species. The PHYSICA macro algorithm for a comprehensive model of stockpile leaching is illustrated in Figure 1.

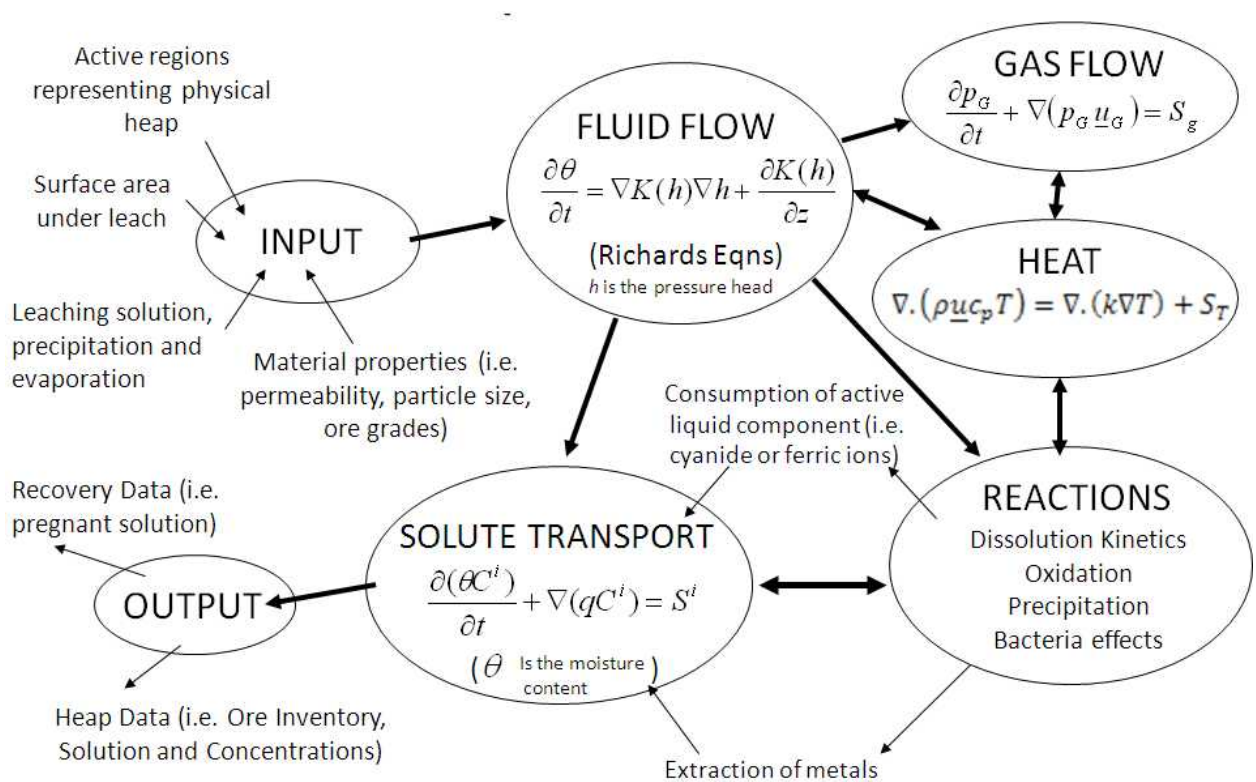

Figure 1: Schematic of macro-procedure for a comprehensive heap leach model

In addition to the standard dissolution reactions for copper sulfide minerals as described previously (Bennett et al., 2006), tracking of the following species considered to derive mainly from gangue minerals was added to the model: aluminum, magnesium, potassium, and calcium. The dissolution of these species was implemented using simple first order reaction within the shrinking core model for individual particle reactions. Once in the solution phase, the solubility and precipitation of several species can be affected by the local chemical environment. The following rules for species precipitation were developed for the copper sulfide mineral heap system.

Jarosite precipitation

Potassium jarosite

The precipitation of end-member potassium jarosite can be given as (Dutrizac, 2008):

$$3Fe_2(SO_4)_3 + K_2SO_4 + 12H_2O \rightarrow 2KFe_3(SO_4)_2(OH)_6 + 6H_2SO_4 \quad (1)$$

This is similar to the method outlined by Leahy and Schwarz (2009) where precipitation is related to the acid and ferric concentrations. An assumption is made here that when potassium is dissolved from sources in the ore, it immediately precipitates as potassium jarosite, regardless of temperature and acid concentration, and the reaction is not reversible. The only way that potassium can remain in solution is when there is not enough ferric present to stoichiometrically precipitate with potassium. This may occur at pH values greater than 4 where the ferric concentration is very low due to the solubility of ferric hydroxide.

Considering the stoichiometry for potassium jarosite, one mole of potassium will remove 3 moles of ferric and 2 moles of sulfate and will add back 6 moles of H^+ (assuming that the sulfuric acid product dissociates to H^+ and HSO_4^-). After potassium jarosite has been precipitated, the model looks at the further precipitation of ferric hydroxide and hydronium jarosite according to the following rules.

H^{3+} (Hydronium) jarosite

After precipitation of potassium jarosite, ferric precipitation is controlled by pH. At pH < 2, the precipitation of insoluble hydronium jarosite is allowed. At pH > 2, soluble ferric hydroxide is precipitated. The model assumes there is virtually no ferric in solution at pH > 4 (Stumm and Morgan, 1970). The hydronium jarosite precipitation reaction can be written as follows:

$$3Fe_2(SO_4)_3 + 14H_2O \rightarrow 2(H_3O)Fe_3(SO_4)_2(OH)_6 + 5H_2SO_4 \quad (2)$$

Considering the stoichiometry for H_3O^+-jarosite, one mole of hydronium jarosite formed will remove 3 moles of ferric and 2 moles of sulfate and will add back 5 moles of H^+ (assuming that the sulfuric acid product dissociates to H^+ and HSO_4^-). The relationship controlling hydronium jarosite precipitation can be written from the solubility product for the solid species on the right hand side of the equation above.

$$[Fe^{3+}] = \left(K_{sp}/(1E-70)\right)^{1/3} [H^+]^{5/3} [SO_4^{2-}]^{-2/3} \quad (3)$$

The pH level at which H_3O^+-jarosite is precipitated is < 2. The K_{Jar} equilibrium value is calculated to give the same solubility as the hydroxide model at the pH at which jarosite starts to form.

Ferric hydroxide

The activity of ferric ion is described to be pH dependent (Garrels and Christ, 1965):

$$Fe^{3+} + 3H_2O \leftrightarrow Fe(OH)_3 + 3H^+ \quad (4)$$

with a heat of reaction ΔH = −19.83 Kcal/mol.

Ferric hydroxide is generated through relating the maximum ferric to pH using:

$$[Fe^{3+}] = \left(K_{sp}/(1E-42)\right) \quad [H^+]^3 \tag{5}$$

Alunite precipitation

Alunite precipitation is assumed to be governed by (Hemley et al., 1969):

$$KAl_3(SO_4)_2(OH)_6 \leftrightarrow K^+ + 3Al^{3+} + 2SO_4^{2-} + 6OH^- \tag{6}$$

where the solubility product is given by:

$$K_{SP} = \frac{a_{K^+} a_{Al^{3+}}{}^3 a_{SO_4^{2-}}^2 a_{OH^-}^6}{a_{Alunite_3}} \tag{7}$$

The solubility of alunite is governed by the solubility product, and the equilibrium constant for the reaction such as:

$$[Al^{3+}] = \left(K_{sp}/(1E-84)\right)^{1/3} [H^+]^2 \, [K^+]^{-1/3} [SO_4^{2-}]^{-2/3} \tag{8}$$

Gypsum precipitation

Gypsum (dihydrate) is moderately water-soluble (~2.0 – 2.5 g/L at 25 °C) and, in contrast to most other salts, it exhibits a retrograde solubility, becoming less soluble at higher temperatures. The solubility of gypsum is governed by the solubility product, and the equilibrium constant for the reaction such as:

$$CaSO_4 \cdot 2H_2O \leftrightarrow Ca^{2+} + SO_4^{2-} + 2H_2O \tag{9}$$

Following Stumm and Morgan (1970), the solubility product is given by:

$$K_{SP} = \frac{a_{Ca^{2+}} a_{SO_4{}^{2-}}}{a_{CaSO_4}} \tag{10}$$

For the solid phase, a_{CaSO_4} = 1.

Then the solubility of gypsum in pure water at room temperature can be estimated by:

$$K_{SP} \approx [Ca^{2+}][SO_4^{2-}] = 10^{-4.6} \quad (11)$$

Leach simulation of a copper sulfide ore

The material and model parameters

Clusters of highly sulfidized epithermal deposits can be found in South American volcanic deposits, particularly in the Yanacocha district of northern Peru (Pilco, 2011). These "acid-sulfate" epithermal deposits show some similarity to porphyry deposits and may be amenable to heap leach processing. For this study, average metal and mineral concentrations were assumed for a deposit of this type. Table 1 provides the hypothetical component quantities used in the leach model set up for a deposit of this type (Pilco, 2011; Davies and Williams, 2005), which can contain chalcopyrite, bornite, covellite, enargite, sphalerite, and other sulfide minerals within the altered massive silicate host rock. The current version of the heap leach model uses a single rate expression for each gangue element, calibrated for a specific ore type, and does not consider origin of the elements from specific gangue mineralogy.

Table 1: Mineral and chemical composition (weight percent) of the ore

Size fraction (mm)	> 8	4–8	1–4	0.15–1
Malachite	0.061	0.092	0.122	0.26
Chalcopyrite	0.084	0.126	0.168	0.357
Chalcocite	0.048	0.073	0.097	0.206
Enargite	0.007	0.01	0.013	0.028
Pyrite	9.54	10.5	11.7	14.2
Sphalerite	0.2	0.2	0.25	0.3
Aluminum	6.5	6.5	6.5	6.5
Magnesium	0.9	0.9	0.9	0.9
Calcium	4.0	4.0	4.0	4.0
Potassium	3.0	3.0	3.0	3.0

Column leach simulations were performed with the conditions as outlined in Table 2. These conditions are similar to the authors' experiences with other column test work (Bennett et al., 2008). A schematic diagram of the column configuration for the leach simulations is shown in Figure 2. The simulations were performed assuming one dimensional flow. Columns are constructed of unit hexahedral elements 0.1 m high. A fixed flux of raffinate wash was applied to the top surface of column 1 for 10 days, a free drainage boundary condition was applied to the base of column 1 and this solution was recycled to the top surface of column 2. A free drainage boundary condition was applied to the base of column 2 and the recovered pregnant solution was fed into a reservoir were the solution was assumed fully mixed. The solution was recycled from the reservoir to the top surface of column 1 from day 10. The

500-day leach period simulation employed a typical time step of 900 seconds and took 30 minutes on a 2.2 GHz processor.

Table 2: Summary of input data for simulation of sulfide leach

Property	Column simulation
Height	1.75 m
Column diameter	0.25 m
Ore particle size distribution	12% <1mm, 36.4% <4mm, 63% <8mm, 100% <12mm
Ore density	2950 kg/m^3
Solid fraction	0.55
Particle porosity	5%
Leach cycle	500 days
Raffinate wash application rate	2.7778E-6 m/s for 10 days
Recycle application rate from day 10	2.7778E-6 m/s for 490 days
Raffinate wash ferric (g/L)	5.0
Raffinate wash pH	1.2
Air inlet velocity (m/s)	3.6E-5
Reservoir initial volume	1 liter
Reservoir Initial pH	1.5

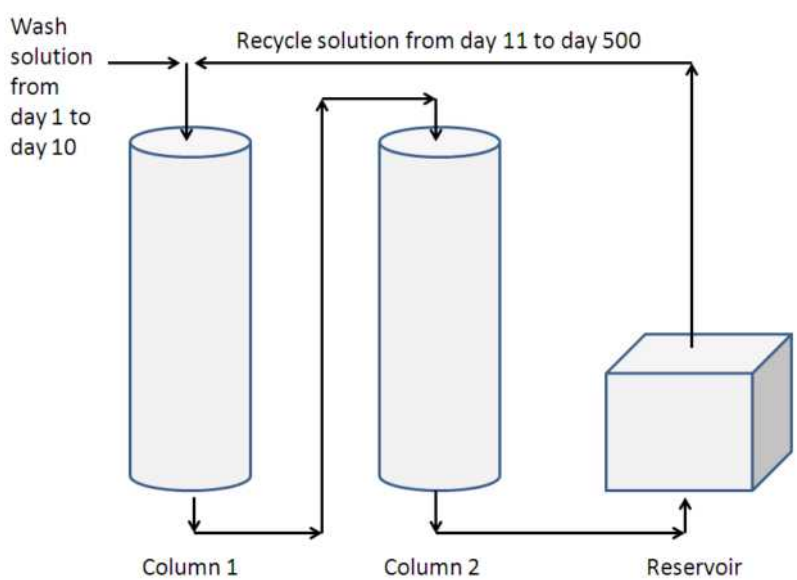

Figure 2: Schematic of the column configuration for leach simulations

Model simulations

Model simulations were performed for an initial base case with the column and solution reservoir maintained at a constant temperature of 50°C throughout the leach period. The initial case was performed

with aeration, and then simulations were conducted using the same leach conditions but without aeration. The fraction of mineral extraction of the copper, iron bearing and other minerals is shown in Figure 3 at 20°C and 50°C. The main difference between the cases with and without aeration is that extraction of the copper and iron sulfide minerals is substantially diminished without aeration, while extraction of the copper oxide mineral is hardly affected. The main difference between the cases at 20°C and 50°C is that metal extraction from the copper and iron sulfide minerals is reduced at lower temperature.

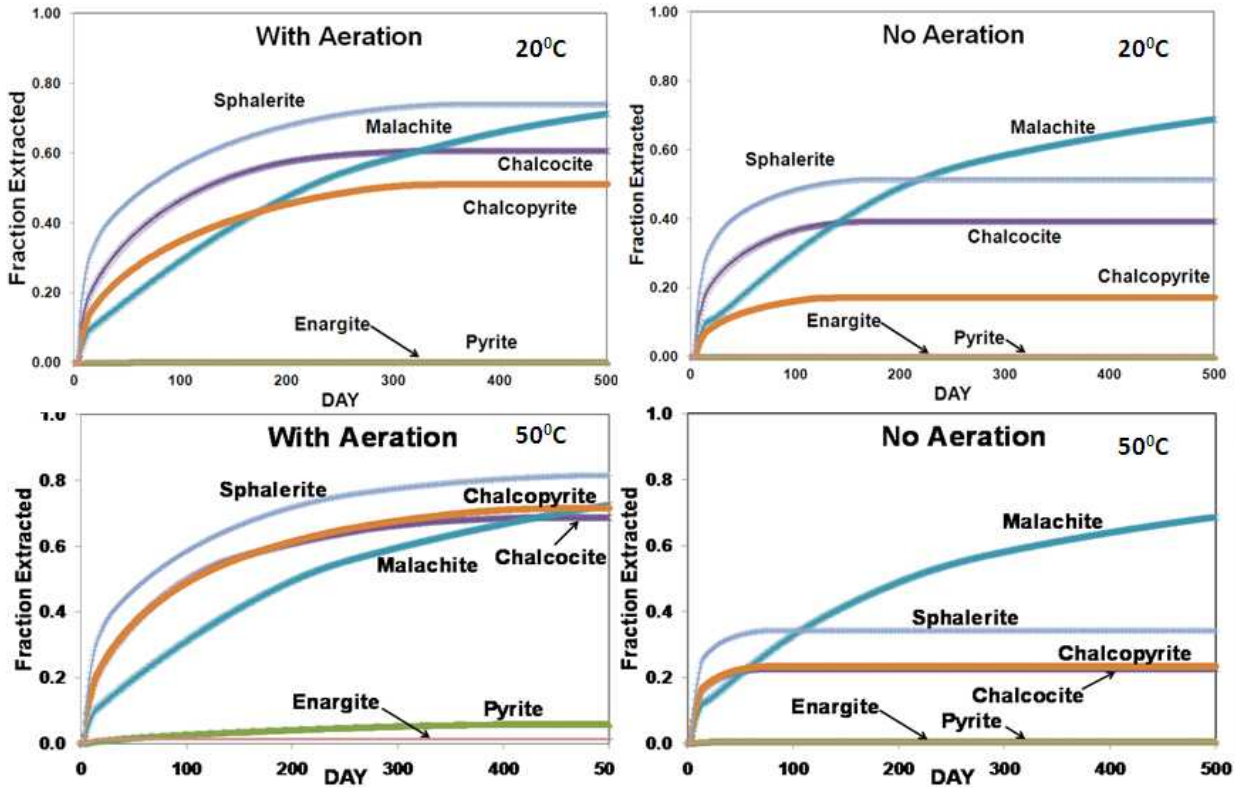

Figure 3: Simulation results showing fraction of mineral extraction with and without aeration

Acid consumption is a significant process concern in heap leach operations with these ore types. Acid was added to the reservoir every four days, initially 114 grams reducing to 85 grams on each occasion over the leach period. A total of 5.7 kg of acid was added, and additions were kept the same for all simulations. The pH of the reservoirs and the pH of the PLS discharge from the first column are initially high, in the 2−3 range, while the strongly acid-consuming species react. Subsequently, pH decreases to a range of 1.0 to 1.5, where good extraction and proper control of the ferric species can be expected.

Overall copper recovery and iron concentrations are shown in Figure 4 for a constant temperature of 50°C. Overall copper recovery is about 65% with air and decreases to about 30% with no air. Without

aeration in the system, the ferric concentration is depleted quite rapidly, and copper recovery is limited by the available ferric. Total iron in solution is about 5 g/L for the non-aerated case, while total iron in solution decreases over the leach period due to the precipitation events that occur.

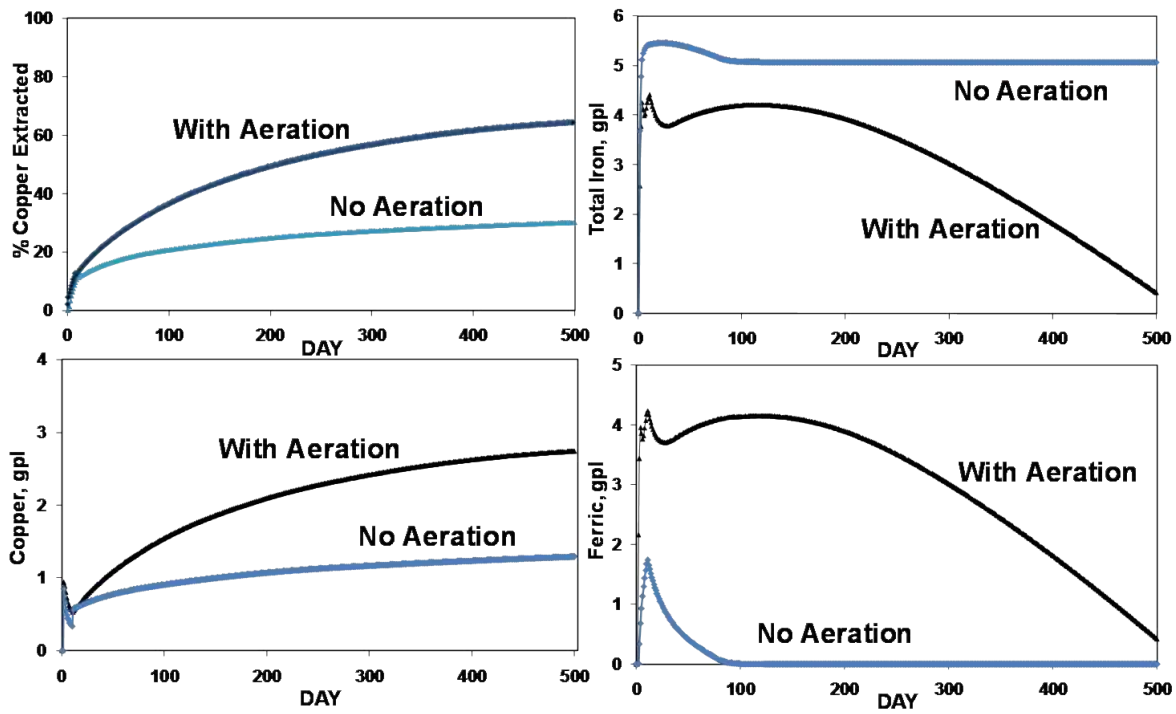

Figure 4: Simulation results showing copper recovery and copper, total iron and ferric concentrations in the reservoir PLS

The extraction of gangue elements, such as aluminum, magnesium, calcium, and arsenic, is shown in Figure 5, and concentrations in solution range between 0 and 6 g/L. Only a slight decrease in gangue element extraction is observed for the case without aeration. This is due to slightly higher pH conditions for the non-aerated simulation.

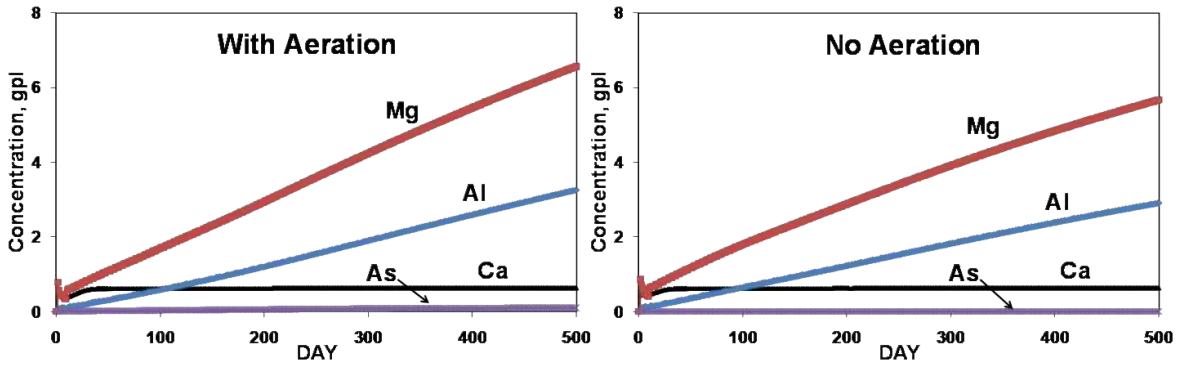

Figure 5: Simulation results showing gangue elements concentrations over the simulated leach period maintained at a constant temperature of 50°C

The precipitation of ferric is in the form of jarosite species, 0.165% without aeration and 1.66% precipitation with aerated conditions because of the pH and higher available ferric concentrations. The formation of jarosites in operating heaps is of concern since the fine nature of the precipitated particles can affect permeability of the system. However, this effect on solution flow was not considered in this study.

The precipitation of gypsum also occurs, about 0.88% for the aerated case, and with slightly less acid available in the non-aerated system there is slightly lower gypsum formation, 0.77%, in the latter stage of the simulated period. The sulfate concentrations were very similar for both cases, starting out initially at about 20 g/L and gradually increasing to about 120 g/L by the end of the simulation period. With less available ferric for reaction with the sulfide minerals, there is significantly lower sulfur, 0.0116%, expected in the non-aerated system, compared to 0.135% with aeration.

Column simulations with aeration at varying temperatures were performed with the solution reservoir maintained at a constant temperature of 5, 20, 35 and 50°C throughout the leach period. The results, shown in Figure 6, suggest that ferrous-ferric oxidation is limited at the lower temperatures. The ferric concentration is depleted earlier in the leach cycle resulting in lower copper recovery, which is limited by the available ferric. The copper recovery reduces from about 65% at 50°C to 35% at a temperature of 5°C.

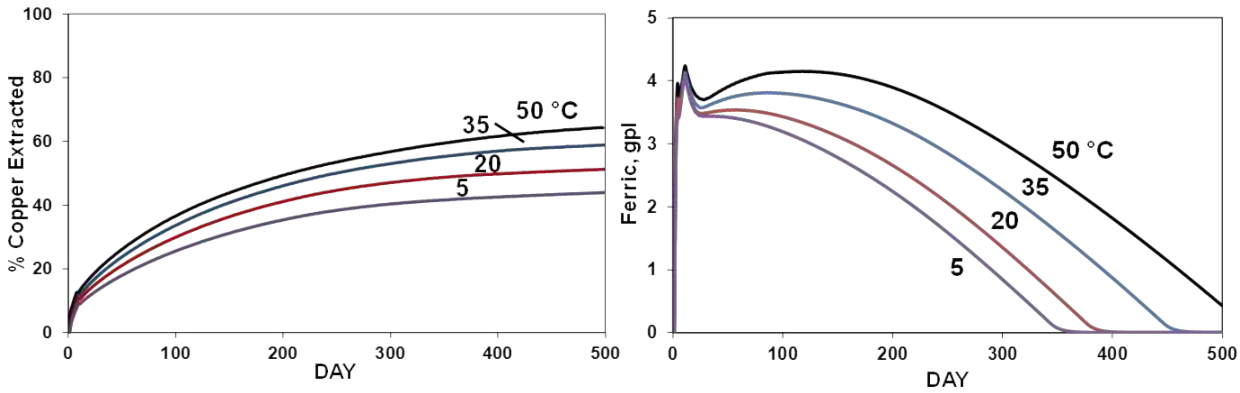

Figure 6: Copper recovery and ferric concentrations in the reservoir PLS

Conclusion

Modeling the leaching of copper sulfide ores involves taking into account many complex and interacting physical and chemical phenomena. A comprehensive model of the heap leach process using a CFD

structure coupled with appropriate rules to describe the chemistry of reactions within the solid and liquid phases was described. In the example provided in this study, aeration and thermal conditions of the heap were shown to have a significant effect not only on overall copper recovery but also on the concentrations of soluble and precipitated species. No aeration in the system reduces copper recovery by about 50%. Lower temperatures also reduce copper recovery in aerated systems by up to 50%. Heap leach models based on this methodology, and which are properly calibrated, offer a means to evaluate many of the influential heap parameters through computer simulation. These types of leach models can be employed by the process and/or design engineer to optimize design parameters, evaluate operating strategies, and effectively maximize the economic return using an engineered heap approach.

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PART 2

HEAP LEACH FACILITIES DESIGN AND OPERATIONS

On the use of geotextile filters in heap leach pads

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Abstract

Drainage over a barrier, similar to that in heap leach pads, is accomplished in many applications with drainage geocomposites. A drainage geocomposite consists of a geotextile filter bonded to a geonet. The geotextile is typically nonwoven needle-punched, while several different types of geonet cores are available from geosynthetics manufacturers. The current practice in heap leach pads involves no filter, whether geotextile or mineral, as the ore is stacked directly on the drainage layer. The introduction of a geotextile, associated invariably with a drainage geocomposite, raises a number of design and performance questions related to permeability, clogging, and piping. Of these, the clogging from chemical precipitates, slime, and scale is of greatest concern. Moreover, in the geosynthetics and mining literature related to heap leach pads, there are few references, if any, and little performance test data related to the use of geotextile as a filter.

Design criteria for geotextiles in many civil engineering applications were developed under saturated conditions with high gradient and are meant to prevent piping. Heap leach ores are generally accepted to be self-filtering with little or no loss of fines with the percolating solution under unsaturated conditions. Given the nature of chemical and biological processes in an ore, mining and metallurgical engineers are mainly concerned with clogging of the ore and subsequently of the geotextile, assuming one is used. Ores are generally tested in columns, cribs or silos, and pilot-scale projects prior to full commercialization of a project. Such tests on drainage geocomposites are recommended to evaluate any changes in hydraulic properties of a heap leach pad due to the introduction of a geotextile filter.

Introduction

Generally, gravel or a select ore is the drainage medium in heap leach pads. The solution percolates down through the ore body and then flows laterally within a pipe network embedded within the drainage layer. This type of "drainage-over-barrier" function is achieved most optimally by drainage geocomposites in several applications including landfills, waste lagoons, green roofs, and retaining walls. Drainage geocomposites are being proposed by manufacturers and engineers as an alternative to granular drainage layers in heap leach pads (Smith and Zhao, 2004; Smith and Li, 2012). Figure 1 presents the current

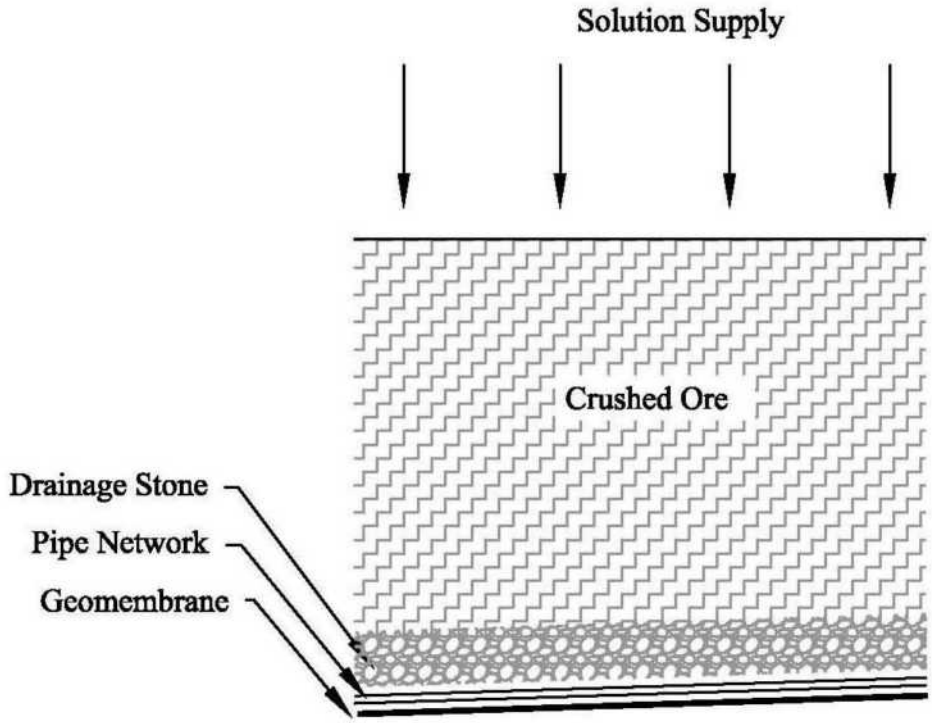
practice of using a granular drainage layer as well as the alternative, which uses a drainage geocomposite. A very obvious difference between the two systems is the use of a geotextile filter with a drainage geocomposite compared with a complete absence of such a filter in granular drainage layers. The geotextile filter, shown in Figure 1(b) is the topic of this paper.

A drainage geocomposite consists of a polymeric core bonded on one or both sides to a filter geotextile. The latter is typically of a nonwoven needle-punched type, although a few products are available with a woven geotextile. The opening size of the geotextile filters on commercially available drainage geocomposites being proposed for use in heap leach pads ranges from 0.001 mm to 0.2 mm (Bhatia, et al., 1996; Aydilek and Edil, 2003). Contrast this with the optimum particle size for heap leach pad ores, which is in the 10 to 38 mm range (Dhawan et al., 2012) and is significantly larger than the opening size range in the nonwoven needle-punched geotextiles that are the standard on drainage geocomposites. Given this difference, mining engineers, metallurgists, and hydrogeologists are often concerned with any negative (i.e., restrictive) effect of introducing a filter in an application where none has been used historically. Yet geotextiles have been used successfully by geotechnical and geosynthetic engineers since the 1950s in numerous applications, including dams, where consequences of a failure are significantly more severe than in heap leach pads. Unfortunately, no test data exists to resolve the concerns related to the use of geotextile filters in heap leach pads. A search for technical literature on performance testing of geotextile filters in heap leach pads does not return any references. Many publications on the ore leaching tests do not show a need for a filter, as ores are typically self-filtering under partially saturated conditions. However, it is necessary to use a geotextile filter if a drainage geocomposite is to be used; therefore, the use of a geotextile filter must be addressed.

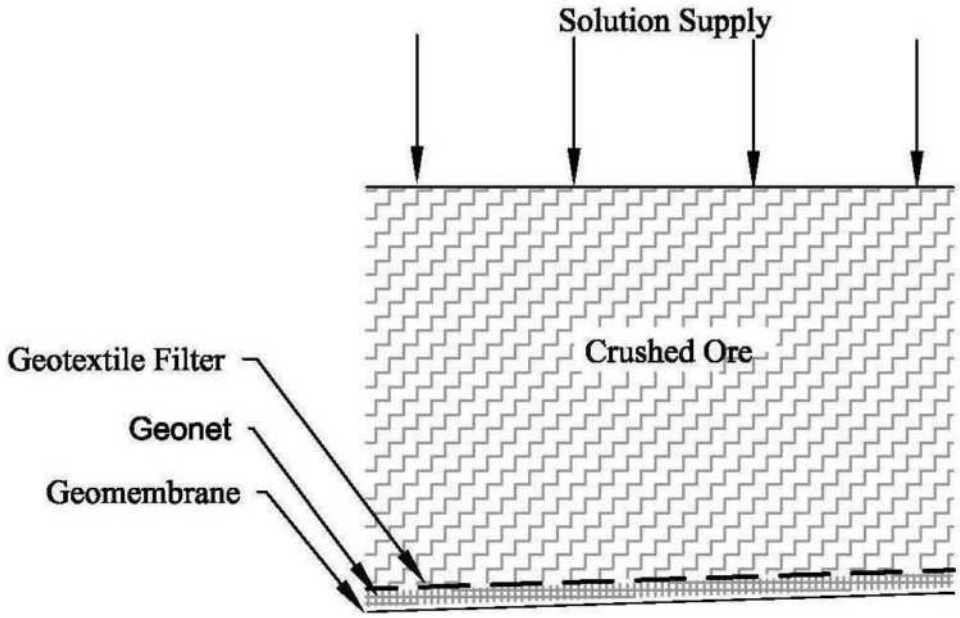
Heap leach mineralogy and hydrogeology relevant to geotextile filters

The performance and design of geotextile filters is generally expressed in terms of piping, clogging, permeability, and chemical compatibility. These four concepts can be applied to the specific conditions of heap leach pads when considering the use of a geotextile filter. The requirements related to the use of geotextile filters can then be discussed under following four subheadings:

- ore gradation versus need for retention;
- ore dissolution, scaling, slime, and precipitates versus geotextile clogging;
- solution application rates versus geotextile permeability; and
- heap chemical environment versus geotextile durability.



(a) Conventional drainage



(b) Geocomposite drainage alternative

Figure 1: Cross-sections of the current state of practice and the drainage geocomposite as an alternative

Since there has been minimal or no performance evaluation of geotextile filters to address specific conditions of heap leach pads, the above subheadings are discussed below based on what is known from testing of ores without filters.

Ore gradation versus need for retention

Heap ores are typically broadly graded with a gradation curve that has a concave upwards shape, as shown in two of the three curves in Figure 2. Generally, less than 10% fines are dispersed within a highly coarse matrix of gravel; in some cases, there are larger size particles. In geotechnical and geosynthetic literature (Lafleur, et al., 1989; Kovacs, 1981; Sherard, 1979; Kenny and Lau, 1985; and others), such soils are classified as internally unstable with a potential for movement of internal particles. Such an internal loss of fines is referred to as piping and must be avoided if a drainage geocomposite is to function properly. The opening size of the geotextile must be made small enough to retain the fines and prevent a clogging of the drainage geocomposite core. However, concerns with the particle migration through broadly graded soils are based on testing of natural or manufactured (glass bead) soils under saturated conditions and high gradients. Heap leach column, bin, and field tests rarely ever show piping, and it is generally accepted that piping of fines would not occur. A theoretical analysis by Kunkel (2008) based on

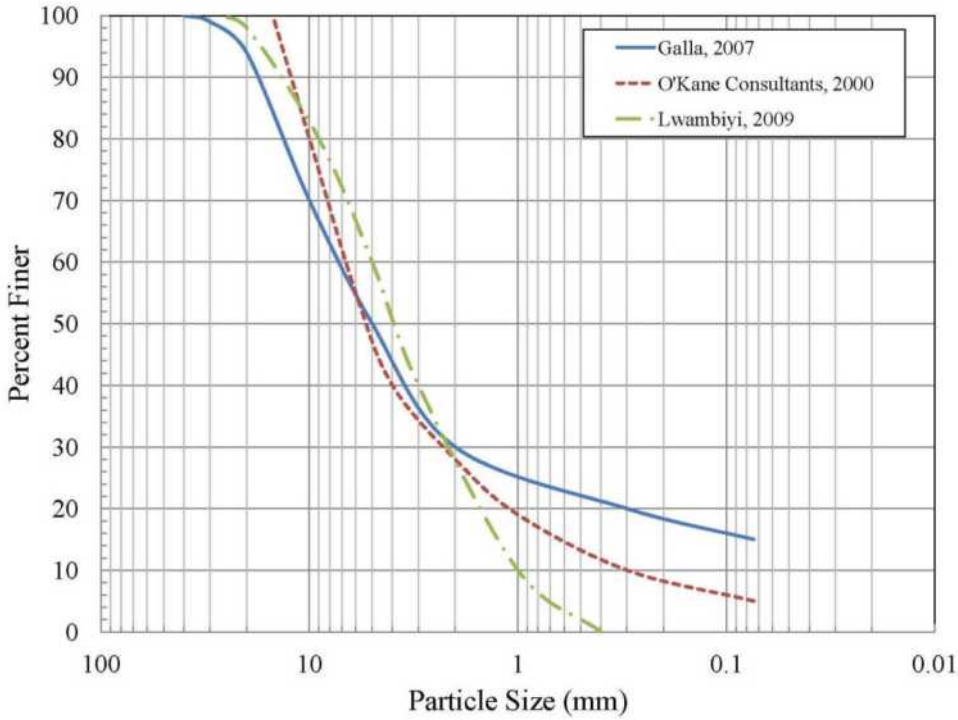


Figure 2: Gradations from literature as an example of filter requirements

the work of Or (2008) show the piping to be unlikely in heap leach pads. It is rare to come across fine particle loss from heap leach columns, as can be seen from papers by O'Kane Consultants (2000), Bouffard and West-Sells (2009), Lwambiyi et al. (2009), and Galla (2007), to name just a few. Column tests currently ongoing at laboratories with and without geotextiles do not show loss of fines even when no filter is used.

Lacking any evidence of piping, one can determine the opening size of a geotextile filter based on the coarser part of the gradation curve. The design procedure presented by Luettich et al. (1992) is a good candidate for this purpose as it includes the effect of the shape of the curve and density of the base material. For relatively low densities relevant to heap leach pads, the method can be presented as

$$O_{95} < (9/C'_u) * d'_{50}$$

Where

O_{95} = apparent opening size (in mm) of the geotextile based on ASTM D4751

C'_u = linear coefficient of uniformity

d'_{50} = linear soil particle size distribution of which 50% is finer (in mm)

C'_u and d'_{50} are obtained from a straight line drawn tangent to the particle size distribution curve at 50% passing value. After drawing tangents to the curves in Figure 2 and applying the above equation, one obtains the following values:

- $O_{95} < 9.6$ mm for Galla (2007)
- $O_{95} < 15.4$ mm for O'Kane Consultants (2000)
- $O_{95} < 7.2$ mm for Lwambiyi et al. (2009)

Currently, maximum opening size with geotextiles is around 0.2 mm in the nonwoven needle-punched type and 0.5 mm for woven geotextile bonded to geonets. The values above point to a need for geotextiles with possibly a larger opening size than that for most commercially-available products. Since retention is not a critical requirement, one would use as open a geotextile as dictated by more critical requirements discussed in the following sections. But geotechnical engineers are used to designing for retention of a finer fraction of broadly graded soils and would feel uneasy about using an open geotextile unless extensive testing is performed.

Ore dissolution, scaling, slime, and precipitates versus geotextile clogging

The leaching of gold and silver is typically done with a weak sodium cyanide (NaCN) solution while copper, uranium, and nickel are leached primarily with a weak sulfuric acid (H_2SO_4) solution. Additional chemicals are added to the ore body (such as gypsum, cement, and lime to gold and silver ore, and ferric sulfate to copper sulfide ores) depending on the type of the ore, requirements of the chemical reactions,

and need for agglomeration. Metal-bearing minerals occur within gangue which can be non-reactive (like quartz) to highly reactive (like calcite). As a leach solution flows down a heap, it reacts with ore and gangue minerals in a great many hydrolysis and redox reactions. The permeability of ores can be negatively affected by excessive clogging caused by precipitates of ferric hydroxide, and fine particles of gypsum, cement, and clay. Under optimum conditions, the effect of these can be minor, with little or no negative influence on the leaching process. However, difficult ore types or poor process control can generate scaling and slime to the point of making an ore completely clogged (Cousins, 2011; Dreier, 1999; Dhawan et al., 2012; Ghorbani et al., 2011; Milligan and Muhtadi, 1988). The use of a geotextile as a filter must take greater account of clogging than of piping, since the former is indeed a concern while the latter appears to be a non-issue. This is the opposite of most other applications of geotextiles, where filters are sized primarily based on the retention criteria. The geotextile opening size, therefore, needs to be based on clogging rather than retention requirements.

Precipitates and particles would likely adhere to the densest points in the fiber matrix of a geotextile and then accumulate with time and grow outward to cover more and more porous area. Since little data exists on the actual clogging of geotextiles in heap leach pads, the author presents a hypothetical scenario of clogging around the fiber matrix in Figure 3. The rate of clogging, if clogging does occur at all, will vary depending on many factors, such as the type of ore, pH, and the nature of gangue. In Figure 4, different clogging rates are presented based on the nature of the ore and other operating conditions. The actual shape of these curves must be determined based on laboratory and field testing. The geotextile filter needs to perform the filter function for a definite time period, which ranges from 70 to 600 days (Dhawan et al., 2012). If clogging of a geotextile does indeed take place, it must not restrict flow within the active leach time. Fortunately, the leach times are several orders of magnitude less than the design life of geotextiles in many other applications such as dams, roadways, landfills, and earth retention structures.

Solution application rate versus hydraulic conductivity

Solution application rates in heap leach pads range from 4 to 15 liters per hour per square meter (Galea et al., 2010). This translates into a percolation rate of 1×10^{-4} cm/sec to 4×10^{-4} cm/sec. Flow through heap leach pads is considered to be unsaturated (Kunkel, 2008; Bouffard and West-Sells, 2009; O'Kane Consultants, 2000); therefore, one must consider it to be unsaturated through a geotextile filter as well. Unsaturated hydraulic conductivity is a function of the volumetric water content and is typically much lower than saturated hydraulic conductivity. Therefore, the effect of scale, slime, and precipitates on the geotextile must be considered under unsaturated conditions prevalent in heap leach pads. Tests currently being conducted on geotextiles using column leaching tests do not show any reduction in flow through columns due to the use of geotextiles.

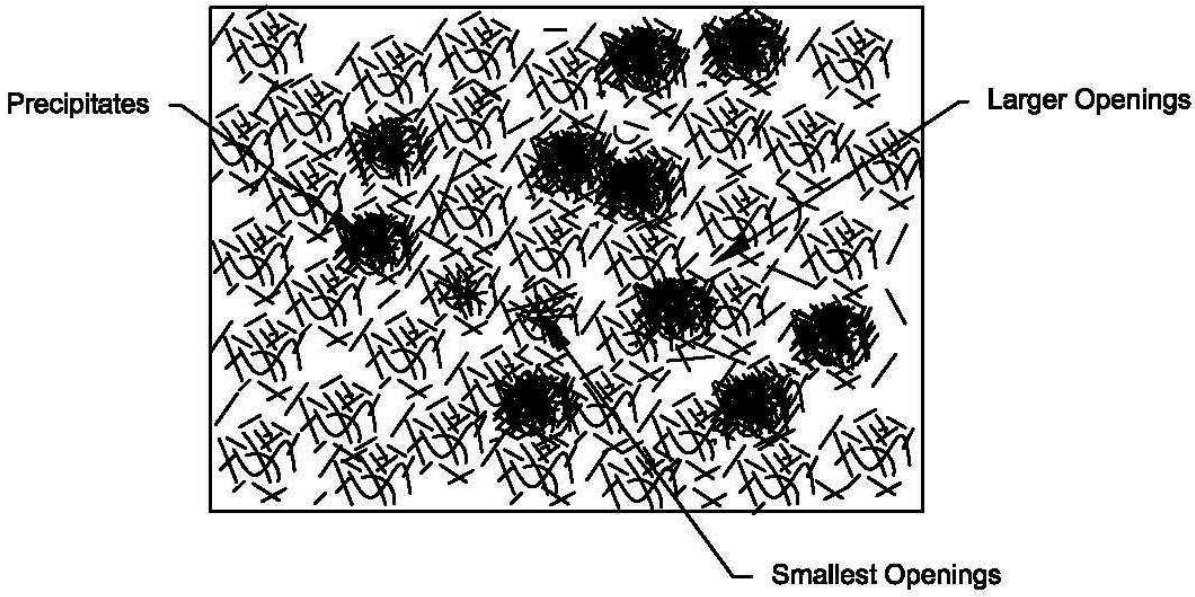


Figure 3: A hypothetical scenario of clogging of a nonwoven needle-punched geotextile showing the range of openings and precipitates deposits, starting with the smallest openings

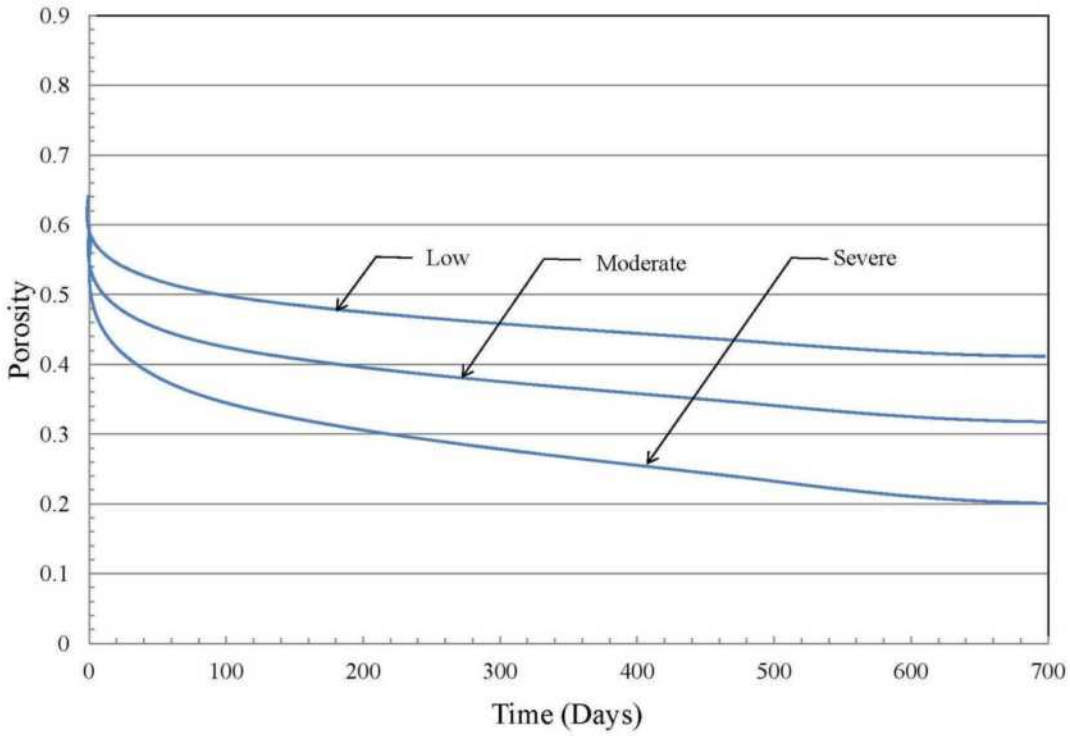


Figure 4: Hypothetical curves representing change in geotextile porosity with time as a result of chemical and particulate clogging

The current design procedures require that the geotextile hydraulic conductivity be 10 to 100 times the hydraulic conductivity of the base soil being filtered. Since this method is based on saturated conditions, its applicability to heap leach pads can be questionable. Using this requirement for heap leach pads, the geotextile's saturated hydraulic conductivity must be 10 to 100 times the saturated hydraulic conductivity of the ore. Saturated hydraulic conductivity of ores falls within a broad range from 1×10^{-4} to 10 cm/sec (Lupo and Dolzel, 2013; O'Kane Consultants, 2000). Although fines reduce the hydraulic conductivity of ores, clean gravels (approximately similar in size to ores) have a value on the order of 0.1 to 40 cm/sec (Cedergren, 1989). The hydraulic conductivity of ores can be much higher than that of soils with which geotextiles are typically used as filters. On the other hand, percolation rates through ores are much lower than their permeability values. Therefore, a filter geotextile can be selected based on the percolation rates, although a more conservative design may be one based on a multiplier to the ore hydraulic conductivity.

Heap chemical environment versus geotextile durability

Dilute sulfuric acid and sodium cyanide are the primary leaching agents used in heap leaching. Sodium cyanide has no negative effect on geosynthetics (Hornsey et al., 2010). High molecular weight polymers that are used to manufacture geosynthetics, including geotextiles, are generally resistant to dilute acids, including sulfuric acid. Chemical resistance tests performed on geotextiles at low concentrations of leaching agents show no negative effect on mechanical and physical properties (Gulec et al., 2005). However, tests performed by Abdelaal et al. (2011) show a depletion of antioxidants in HDPE geomembranes over time. Nonwoven needle-punched geotextiles have a high surface area that, when exposed to acids at a higher temperature in some leach pads, can lead to some loss of mechanical properties over long lifetimes (i.e., 30+ years). Leaching time of 60 to 600 days or even five years of design life is too short an exposure time to be a concern for mechanical properties of geotextiles. It is necessary, however, to generate and publish actual data to illustrate this to mining and metallurgical engineers who are not familiar with the polymer properties of geotextiles.

Testing protocol to evaluate geotextile filters

The literature on mining or geosynthetics includes few publications on the use of geotextile filters in heap leach pads. The above sections show that the physical, chemical, and hydrological conditions prevalent in heap leach pads are very different from those in other applications of geotextile filters such as dams, landfills, and roadways. Although geotextile filters are one of the most researched of the geosynthetic categories, little data exists on the evaluation of these materials in heap leach pads. For geotextile filters to be accepted and used in heap leach pads, filter performance, and specifically the chemical clogging

concern, must be investigated. For this purpose, test methods and procedures that are already used in heap leach pads must be adapted to the evaluation of geotextile filters. The hydrogeological and metallurgical response of heap leach pads is generally based on the following three tests of increasing complexity:

- column leach tests;
- crib or silo tests; and
- field tests.

Column leach tests

Column leach tests are used to evaluate the metallurgical response of an ore, including acid consumption, metal recovery with time, pregnant leach solution (PLS) composition, and water balance. Columns are typically from 10 to 30 cm in diameter and 2.5 to 6 m high. Irrigation is performed at the top and the solution is collected at the bottom. The solution is analyzed and various parameters are plotted with time. When using the test to evaluate a filter, the objective is different from that of a standard column leach test. With a filter, the goal of the test is to measure geotextile clogging, piping, and flow rate. As such, the test procedure and equipment must be modified when evaluating a filter geotextile.

Figure 5 shows a column construction with a geotextile, although it is possible to set up the test in many different ways. The most important details related to the geotextile are to ensure that there is a geonet below the geotextile, that the edges are sealed so that there is no leakage, and that there is a rigid plate below the geonet. It is preferable not to have the geotextile bonded to the geonet, even if a geocomposite (geotextile bonded to a geonet) is what may be used in the field. The test is run exactly like any other heap leach column test, but the solution is analyzed for suspended solids in addition to the characteristics analyzed in other routine tests.

Any loss of fines through the filter will show in suspended solids in a solution that is collected and analyzed periodically. The geotextile chemical clogging is evaluated based on a microscopic examination of the geotextile test specimens retrieved at the end of a test. A comparison of irrigation and effluent rates, although useful, will not provide a measurement of clogging of geotextile as these values are much lower than the available flow capacity of geotextiles.

Crib or silo tests

Large sized boxes made of steel and wood, or silos made of steel, are used to build a prototype of a heap leach pad with the aim of obtaining a better understanding of the hydrodynamics of flow than that afforded by a column test. Cribs or silos can vary significantly in size, depending on the objectives of the test and the actual heap leach it is meant to model. Bouffard and West-Sells (2009) report using a box of 2 × 2 × 7 m, while Brown and Root (1996) utilized an octagonal box with an inside diameter of 5 m and a

height of 7.6 m. The ore being evaluated is placed in layers, just like it would be in the field. Commercial level irrigation systems (for example, drip tubes spaced suitably apart) can be used for the purpose of irrigation. This test would generally better represent the hydrological conditions of a heap due to the scale and construction effects. Figure 6 shows a proposed crib test with a geotextile and a geonet. As with a column test, a geotextile and a geonet are used, although the two products may be factory-bonded in an actual project. An inclined slope is built within the container to simulate field conditions. A steel plate with a geomembrane over it represents a barrier layer in the field. The geonet is installed over the geomembrane to represent a synthetic drainage layer. The geotextile is installed over the geonet, followed by the placement of the ore. The fact that the geomembrane-geonet-geotextile friction angle is only a few degrees should be considered when building the box.

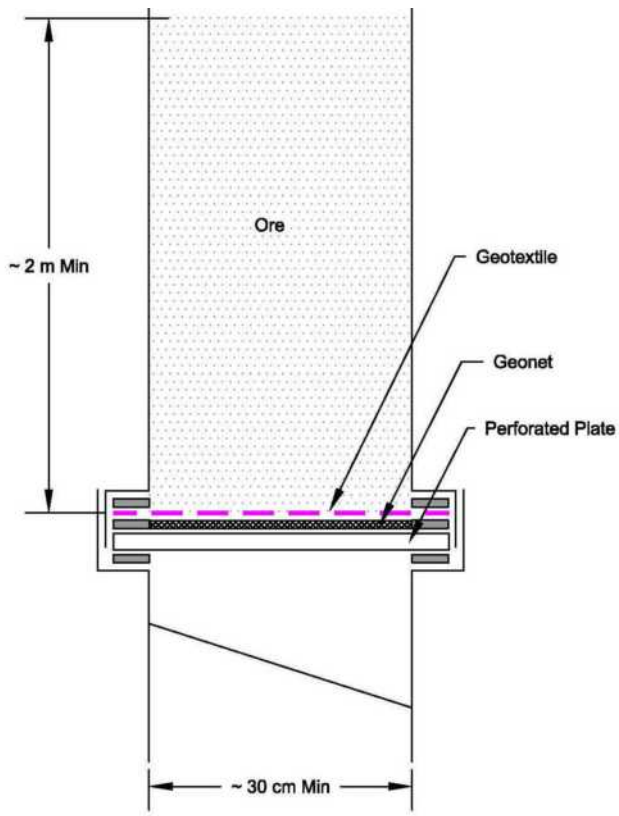


Figure 5: Column tests with a geotextile to evaluate geotextile filter performance

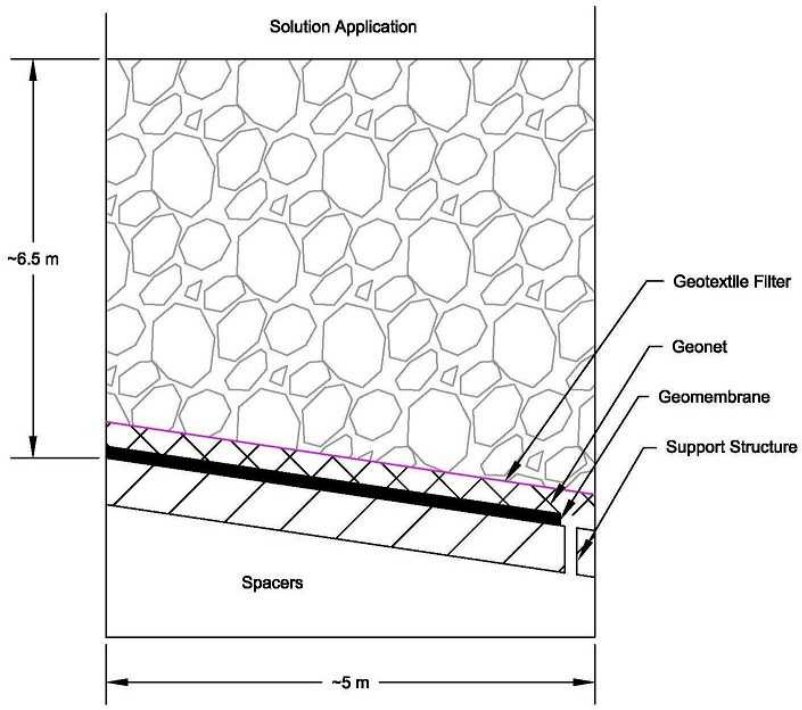


Figure 6: The cross-section of a 5 × 5 × 6.5 m crib for testing a geotextile filter

A crib test with a geotextile and a geonet involves both a downward flow through the ore and the geotextile and a lateral flow through the geonet, as would occur in the field. The ore can be instrumented with moisture sensors to measure the degree of saturation with depth in order to evaluate the effect of the geotextile. A comparison of inflow and outflow as well as a measurement of total suspended solids in the effluent would give an indication of the overall response of the geotextile. The geotextile and the geonet can be retrieved at the end of the test for a physical examination of the material. A microscopic examination of the geotextile retrieved at the end of the test will give any indication of the chemical clogging.

Field tests

For many important projects, the metallurgical response of an ore obtained from column and crib tests is confirmed with a demonstration heap leach pad prior to full-scale construction. Field tests are especially important for understanding the hydrodynamic response of an ore, including bulk density, porosity, degree of saturation with depth, and distribution of void space and flow paths. Field tests are especially important for drainage geocomposites, as there is no prior experience with these materials in heap leach pads. Considering that drainage geocomposites are less than 10 mm in thickness, it is likely that the material will become completely saturated at the toe while remaining only partially saturated for most of the slope. The effect of the length of the slope on the clogging of the geotextile, or even the core, can be

fully understood from field tests. Bouffard and West-Sells (2009) used a 55 × 55 × 3.4 m heap leach pad in their study and concluded that the hydrodynamic response of the ore from field tests could not be repeated in column or even crib tests. Webb et al. (2008) installed 24 free-draining lysimeters atop an operational heap leach pad; five measured approximately 6 × 15 × 0.6 m in size and nineteen were 1.8 m in diameter and 0.6 m deep. They concluded that fluid flow in their study was highly heterogeneous and was dominated by capillary flow.

Figure 7 shows a cross-section of a proposed field experiment with a geotextile and geonet. The ore body above the geotextile can be instrumented with moisture and temperature sensors to obtain an indication of pressure buildup, especially at the toe. The drainage swale at the toe is used for recording flow out of the system and comparing it to the irrigation rate. The PLS solution can be tested periodically to obtain an indication of migration of fines through the system. At the end of the test, both geotextile and geonet material samples are carefully retrieved to evaluate any occurrence of clogging. The geotextile must be observed under a microscope to determine the nature of precipitate and slime, if any.

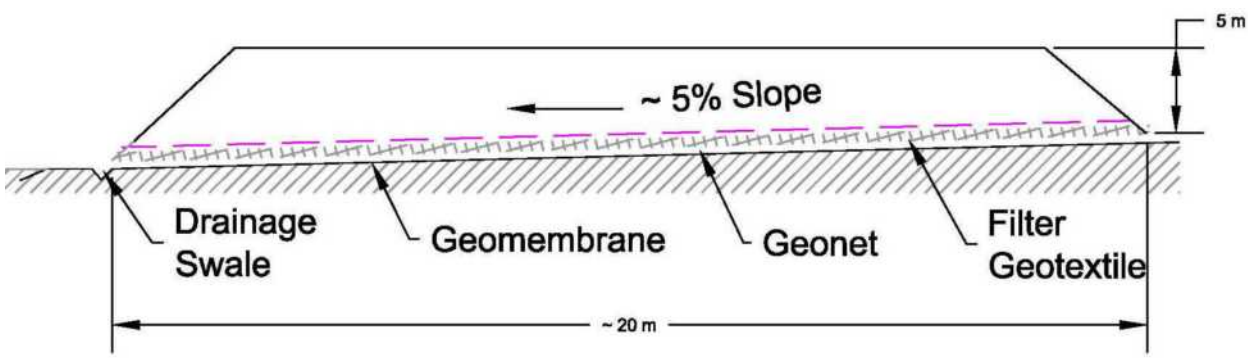


Figure 7: Proposed field test to evaluate filter geotextile

Conclusion

Geotextiles are used as filters in many applications, including dams, landfills, walls, roads, and buildings. The use of geotextiles in these applications has been supported by extensive research by experts since the 1960s. Most of the design methods developed for filter geotextiles were based on laboratory tests on natural and manufactured soils under high gradients and, in some cases, using slurries. As such, these test methods generally favor the retention of particles by the geotextiles. Research and testing on heap leach ores have shown that most ores are self-filtering with little or no concern for particle movement and loss. Chemical clogging, as well as clogging from scale and slime, is seen to be the primary concern in the use of the geotextiles. While geotextiles with very small openings are preferable in many applications, heap ores appear to favor the use of more open geotextiles that can handle clogging, if clogging does indeed occur.

Practically no data exists within the geosynthetic or mining literature on the use of geotextiles as filters in heap leach pads. This situation can be rectified by testing geotextile filters as ores are currently studied in heap leach projects, that is, in three stages of tests: column, crib, and field. Column tests are simple and relatively inexpensive. A microscopic examination of geotextiles from column tests is necessary to understand the nature of the clogging. Just the flow and mineralogical data from columns will yield little information about the condition of the geotextile. Larger scale crib and field tests are required in order to evaluate the hydrological behavior of a geotextile within heap leach pads. While some clogging of the geotextile filter will inevitably occur, with the ore itself, the geotextile must not restrict the flow over the leach time within a project. Fortunately, these leach times are short compared to numerous projects where the design life for geotextiles is in the range of 50 or more years. Geotextile evaluation under simulated conditions will show the type of geotextile that performs best for the unique conditions of heap leach pads.

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A case study of the Ocampo Phase 1 heap leach expansion

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Russell Tremayne, Aurico Gold, Inc., Portugal

Abstract

The Ocampo mine completed an expansion of the Phase 1 gold/silver leach pad in 2010. The expansion effectively changed the Phase 1 flat pad with external pregnant leach solution (PLS) pond to a valley fill leach pad with an internal sump to remove the PLS. The expansion required a berm raise at the toe of the new pad, affecting a portion of the lower valley, and an additional installation of approximately 49,000 m^2 of geomembrane liner. The conversion provided an additional ore storage capacity of approximately 10 million tonnes and extended the leach pad life for over three years. Amazingly, the expansion was completed without interrupting leach pad operations, in record time, and was performed during the winter wet season.

Site description

The Ocampo mine is located in the Sierra Madre Occidental Mountains of north-west Mexico, between the cities of Chihuahua and Hermosillo. The total property area covers over 12,000 hectares. The project elevations range from 1,600 to 2,200 meters above mean sea level (masl). Geologically the site is located close to the western margin of the central volcanic belt. The volcanic belt is comprised of a thick sequence of volcanic extrusive rocks such as rhyolite and andesite tuffs and flows and intrusive dacite porhyrys. The Sierra Madre Occidental is part of the Mexican Highlands section of the Basin and Range tectonic province of western North America, as defined by Dohrenwend (1987) and the US Geological Survey (USGS, 1969).

The Ocampo mine has been mining gold and silver for over 150 years. Historically all mining was performed using underground methods to extract high grade ores. Currently the mine facilities comprise

both heap leaching and mill technologies to process ore extracted from open pit and underground mining operations.

The open pit operations are expected to produce an average of 10 to 12,000 tonnes per day of ore to be stacked and processed on the leach pad facility at the site. Ocampo utilizes conveyor systems to transport the crushed fresh ore to the heap leach facility. The conveyor system includes overland conveyors, portable conveyors (grasshoppers) and a radial stacker.

The ore is leached with a cyanide solution at a flow rate that ranges from 20,000 to 27,000 m^3 per day. Under the first leach pad configuration, pregnant solution was collected in the external PLS pond, located near the toe of the leach pad, and then pumped to the Merrill-Crowe plant located directly adjacent to the south-east corner of the leach pad.

Original leach pad configuration

The PLS pond was separated from the leach pad by a 5 meter high berm and a lined open area which was several meters wide. PLS would collect at the toe of the heap and then flow through an open channel into the external pond. The leach pad was lined with a 2 mm thick linear low density polyethylene (LLDPE) geomembrane overlying 30 cm of compacted low permeability soil liner material. The soil liner material was replaced by a geosynthetic clay liner (GCL) on areas with steeper slopes located near the upper back half of the pad. The PLS pond was double lined with geomembrane and a geonet between the layers to act as a leak collection and detection layer. The pond bottom was graded with a sump where any leakage could be detected, collected and removed from between the two geomembranes.

Prior to the initial construction of the Phase 1 leach pad the area was a basin with several minor drainages which flowed approximately north to south to a larger drainage. Extensive grading was performed during initial construction of the Phase 1 leach pad to provide continuous, even, flat slopes within the bottom 300 meters of the leach pad basin area. A network of collection pipes was installed on top of the liner to help collect and remove PLS from the heap. A photo of the existing phase leach pad showing the PLS pond and toe of the heap is presented in Figure 1.

Figure 1: Ocampo Phase 1 leach pad before the expansion

Leach pad expansion features

The Phase 1 leach pad was already located in a basin, so that the changes from a flat pad to a valley fill did not require large quantities of earth to be moved. The valley side slopes were simply not being used to contain the heap. This is shown in the picture of the west slope, above.

The expansion included clearing and grubbing the side slopes, smoothing out the grades on sides and grading these areas to drain into the pond from the sides, constructing a toe berm along the new base of the heap, raising the berm on the downstream side of the pond, adding a new primary liner in the pond, and installing additional PLS collection pipes which were plumbed into vertical riser pipes to remove the solution.

A grading plan of the Phase 1 leach pad at the time of the expansion is presented in Figure 2 and the grading plan for the expansion is presented in Figure 3. The area outlined in red in Figure 3 was the area that required re-grading as part of the expansion.

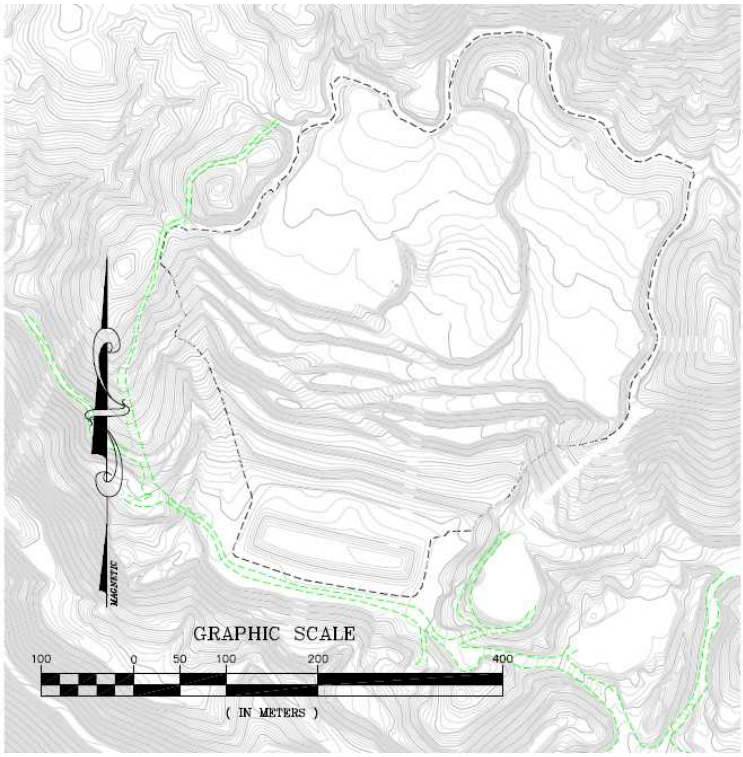

Figure 2: Pond and heap grade prior to expansion

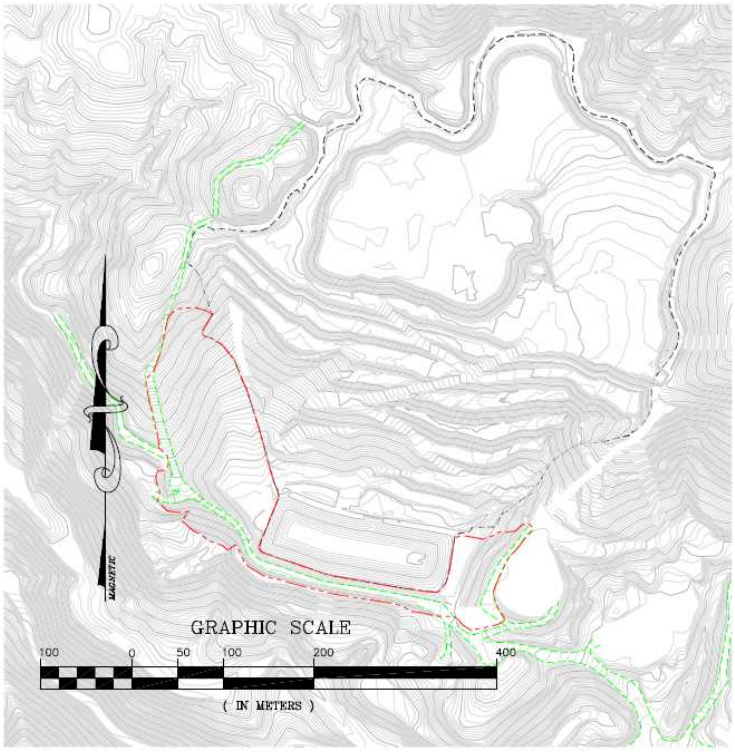

Figure 3: Phase 1 expansion graded area (February 2010)

To improve stability, the downstream embankment of the pond was raised to an elevation of 1,749 masl. This raise, referred to as the stability berm, has an internal slope of 1.5H:1V and an external slope of 2H:1V. The 4 meter wide berm crest provided enough room to accommodate a narrow service road and the geosynthetics anchorage system. Additionally, the lower portions of the leach pad and toe buttress were lined with double sided textured (DST) LLDPE overlying 30 centimeters of compacted low permeability soil, as shown in Figure 3. The upper, steeper areas of the expansion were lined with doubled sided textured LLDPE over a GCL to form a composite liner system.

Grading included 5,240 m^3 of cut and 38,900 m^3 of fill. Approximately 3,000 m^3 of low permeability soil liner material was placed and compacted. In addition, 700 m^3 of fresh ore was added to the sump to serve as a leak detection collection layer. Geosynthetics included 34,500 m^2 of GCL material, 10,500 m^2 of geonet, and 45,500 m^2 of LLDPE.

The solution pond had a double liner system (upper primary geomembrane and lower secondary geomembrane) with a geonet between. In order to avoid any potential release of PLS into the environment, Gammon decided that the existing pond liner system should remain in place. In order to avoid any chance of solution loss a new composite liner system was installed on top of the existing system. The liner system for the PLS pond consists of the following, from top to bottom:

- ore;
- 2.0 mm thick, DST, LLDPE geomembrane;
- GCL;
- ore leak detection layer (for pond floor) or geonet (for pond sideslope);
- existing primary geomembrane;
- existing geonet;
- existing secondary geomembrane; and
- existing subgrade.

Schematics of the revised liner systems for the PLS pond floor and sideslopes are shown in Figures 4 and 5, respectively.

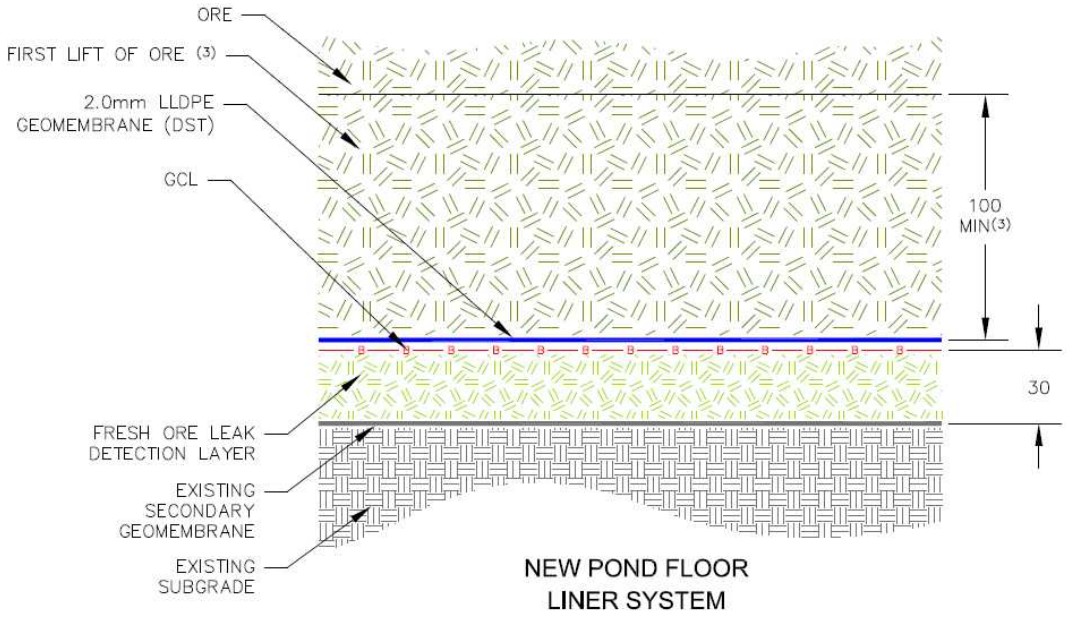

Figure 4: Revised pond floor liner system

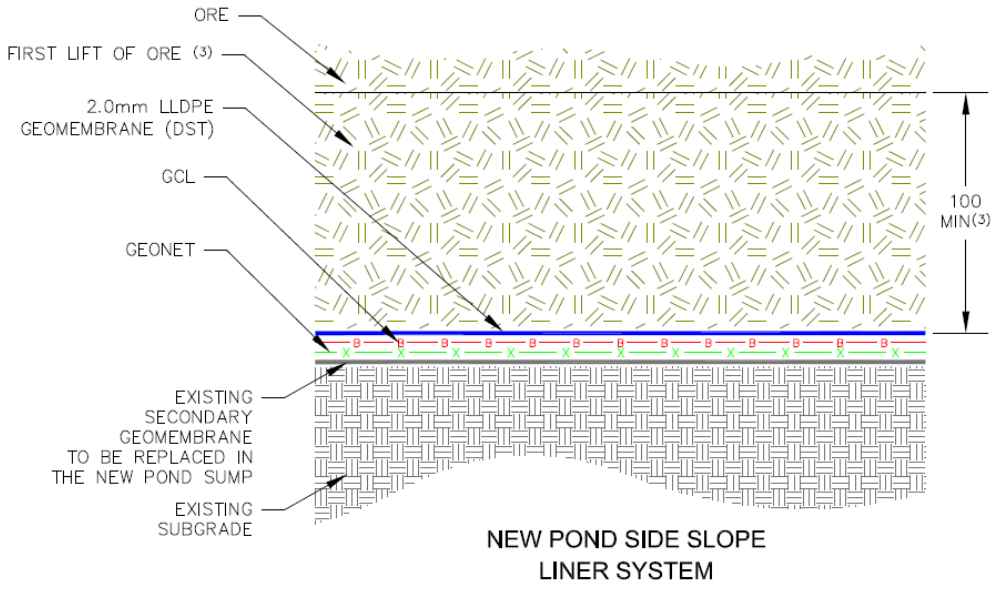

Figure 5: Revised pond sideslope liner system

The schedule of the expansion construction was critical in order to avoid disruption of the leach pad operations. Therefore the first parts of the expansion were re-grading of the west and east sides and southern berm rise (stability berm). These areas were then lined and the new liners were welded to the

existing liner system. Then the inlet channel to the pond was blocked with sand bags and a temporary solution pumping system was installed on the upstream side of the embankment. This temporary PLS collection and removal system is shown in Figure 6. The operation of this temporary PLS pumping system provided sufficient work area to install the new liner system and the solution recovery structures.

Figure 6: Temporary PLS removal sump

Stability of the expanded heap

Based on available laboratory data and published literature the critical interface for stability for this liner system is expected to be between the geonet and geomembrane. Based on test data reported by Koerner and Narejo (2005) for geomembrane/geonet interface strengths, the effective friction angle and cohesion were assumed to be 11 degrees and 0 psf, respectively. This interface strength was found to be consistent with testing performed by Vector Engineering, Inc.

Previous studies have been conducted to determine the seismic characteristics of this area. A study released by Knight Piésold and Co. (2007) resulted in a recommended design peak ground acceleration ranging from 0.04 to 0.08 g, corresponding to an earthquake with a 500-years return period and an assumed magnitude of M7.5. For the purposes of the stability analysis model, a horizontal ground acceleration of 0.08 g was utilized, this being the maximum recommended peak ground acceleration.

For design purposes, the phreatic surface within the internal pond was modeled at the crest and 1 meter over the surface of the geomembrane liner in all other areas.

Five cross-sections of the expanded heap were analyzed for stability. Section A which passes through the highest portion of the proposed heap and through the middle of the pond was found to be the most critical section and is shown on the proposed stacking plan presented in Figure 7. Stability analyses

were performed using the SLIDE 5.0 computer program which was developed by Rocscience (2003). SLIDE 5.0 enables the user to conduct limit equilibrium slope stability calculations using a variety of methods. Several methods may be used to determine the failure surface with the lowest factor of safety (i.e. the critical slip surface). The cross-sections were analyzed for circular and noncircular failure surfaces using the Spencer method (1967, 1973, 1981).

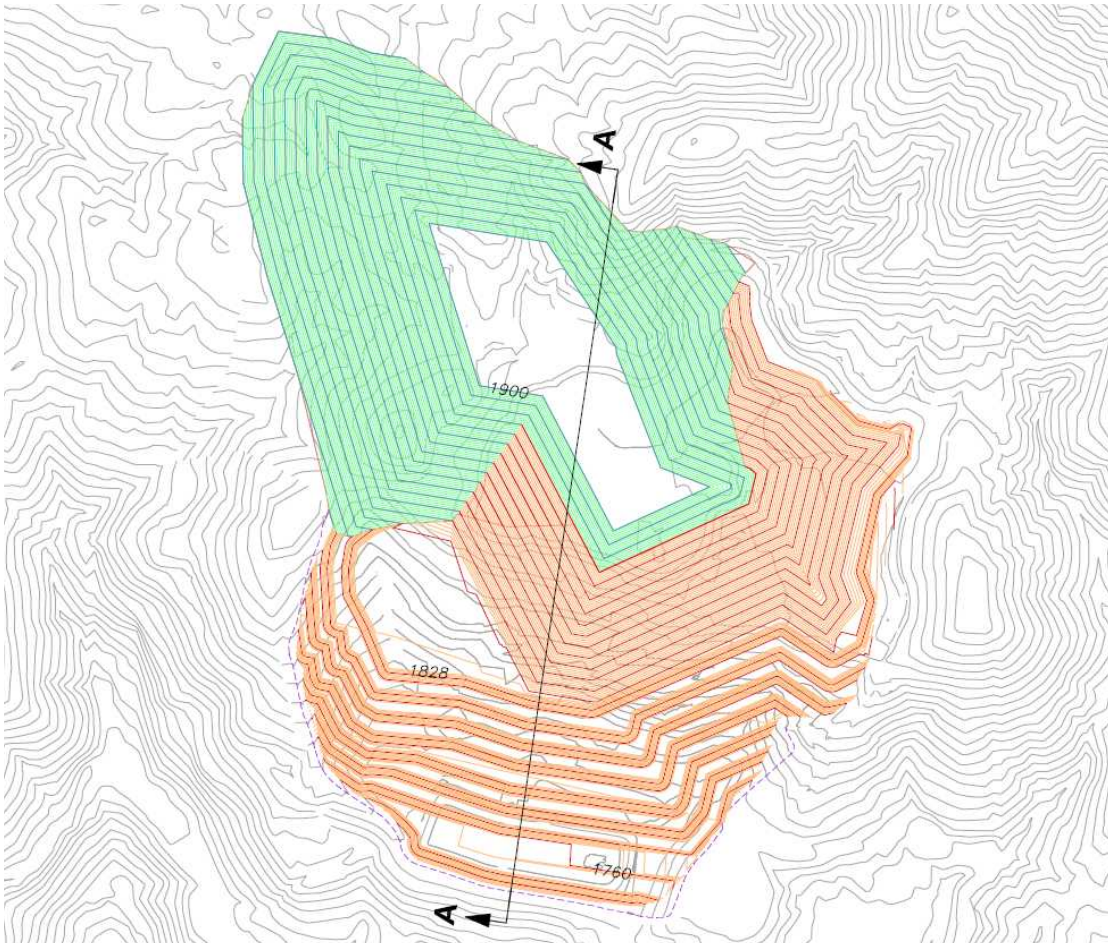

Figure 7: Proposed stacking plan and location of the critical cross section

The Spencer method satisfies both force and moment equilibrium, thereby yielding a more rigorous solution relative to other commonly used methods. The critical mode of failure for the proposed leach pad configuration was shear along the liner system. The results of the stability analyses for Section A are summarized in Table 1.

Table 1: Summary of stability analysis results

Description of section A	Static factor of safety			Pseudo-static factor of safety (block)
	Block along liner	Circular through ore	Through toe berm	
Berm to el. 1,749m	1.4	1.7	1.5	1.1

Static and pseudo-static factors of safety (FS) of greater than or equal to 1.4 and 1.1, respectively, were considered acceptable for the expansion design.

Construction considerations

The earthwork for the leach pad expansion was performed by mine personnel, and Soluciones Ambientales Integrales S.A. De C.V. was responsible for the GCL and geomembrane liner installations. Vector Engineering, Inc. provided the design as well as construction quality control and quality assurance services for the project.

Several items were critical in order to construct the expansion during the winter wet season and while the leach pad was in full operation. On several occasions construction was delayed by rain and snow storms. Liner was not deployed during times of high wind or precipitation, which resulted in delaying construction by about 30 days from the original schedule.

The temporary sump, up-gradient of the pond berm, was critical for the expansion to occur without interrupting the process. Once that system was in place the bottom of the pond could be cleaned out, the new liner could be added, and the vertical standpipes could be installed.

The final change of going from using the temporary sump to the vertical standpipes was the last hurdle and involved first filling the new sump (old pond) with ore and raising the outer berm while the temporary system was still being used. The final step was filling the gap between these areas with ore and then continuing to stack the heap to the final configuration.

Figure 8 shows the reconfigured heap in operation in 2012.

Figure 8: Re-configured heap in operation, 2012

Conclusions

The Ocampo Phase 1 heap leach pad was transformed from a flat pad with an external PLS pond to a valley fill pad with an internal PLS removal system without disrupting operations. This transition allowed for an additional leach pad capacity of approximately 10 million tonnes and provided an additional three years of capacity, while only requiring approximately 46,000 m^2 of new liner.

In mountainous terrain a valley fill leach pad with an external or internal PLS pond is usually the most efficient means of constructing a leach pad. Stability can be maintained even under seismic loading conditions by constructing benches in the subgrade and toe buttresses, although stacking can be a challenge. Textured geomembrane liners are also normally needed for stability purposes. Geosynthetic clay liners also allow designers to use composite liner systems in steep terrain where conventional low permeability soil liners cannot be properly constructed.

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Structured geomembranes in mine applications

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Abstract

Structured or embossed high density polyethylene (HDPE), linear low density polyethylene (LLDPE) geomembranes have been available to mine owners and designers for over 10 years. Their use in new leach pads and expansions and final closure designs has been steadily increasing, especially over the past 10 years, as owners and designers discover and demand the consistent high-quality characteristics of this type of geomembrane – the results of its unique manufacturing processes. This paper will discuss the structured or embossed geomembrane concept and manufacturing process and present comparative testing that illustrates the major advantages of implementing this type of product in these applications. Both technical and economic advantages will be illustrated with examples of recent cost-effective case history solutions.

Materials available for single, double-lined, and final cover systems are changing the way ponds, pads, and final closures are constructed using these types of liners. Cost savings come in various forms, such as lower material costs, lower installation costs, faster third-party inspection times, and better performance. We are all looking for ways to decrease the bottom line and increase our profits, and liners are a leader in this movement. The mining industry will be interested in this presentation as the increased quality means fewer problems; savings from new systems means mining companies will have capital to spend in other areas to increase the inflow of cash. These structured systems will be presented in this paper.

Polyethylene flexible membrane

A geomembrane is a planar, relatively impermeable (1×10^{-13} cm/sec), synthetic sheet manufactured from low permeability materials and used as a barrier to control fluid migration in a project. A geomembrane can be used inside an earth mass or liner, and as an interface or surface impoundment.

Geomembranes, predominantly polyethylenes, are used for a variety of reasons. They have become the standard material used in the mining industry for fluid control. They are produced in a factory under rigorous quality control conditions, install more rapidly than alternative products, and cost less than their soil counterparts. They have undergone quantifiable testing in both manufacturing and installation. What were thought to be impossible designs from a practical and/or economic standpoint are possible using flexible membranes, which have a long life expectancy. Structured membranes have played an important role in improving the ease of constructability of projects with high interface friction angles. This is among the many advantages of using synthetic materials in mining applications.

Structured (embossed) geomembrane texture

During the flat die manufacturing process for geomembranes, hot extruded polymer sheets are run between two counter-rotating hot embossing rollers that contain uniform structural die shapes to form a molded or "embossed" structured or textured surface that is an integral part of the sheet but does not affect its core thickness. (Table 1 provides a list of the different types of embossed/structured liners.) This method has been used for over 10 years and was designed to overcome problems that are commonly found with other types of liner production: non-uniformity, variable area coverage, variable peaks and valleys, variable thickness, and reduction in mechanical properties. Figure 1 illustrates the production method and Figures 2, 3, and 4 show examples of the surface texture generated by the flat die molded surface manufacturing process. A major advantage of structuring is its ability to create very different surface textures on the upper and lower geomembrane sheet surfaces, thus customizing the surfaces for specific applications (for example, drainage on top and aggressive friction surface on the bottom).

Table 1: Types of structured membranes

Texturing	
Single	Regular texturing on one side, either top or bottom
Double	Regular texturing on both sides
Grip	Aggressive on one side
Micro grip	Aggressive on one side, regular texturing on opposite side
Double grip	Aggressive texturing on top and bottom sides
Drainage/separation	
Drain liner	Drainage medium on upper or lower as required
Micro drain	Drainage medium with texturing on upper or lower as required
Drain/aggressive	Drainage medium with lower aggressive surface

Figure 1: Flat die calendaring manufacture (smooth sheet production)

Figure 2: Flat die calendaring manufacture (textured)

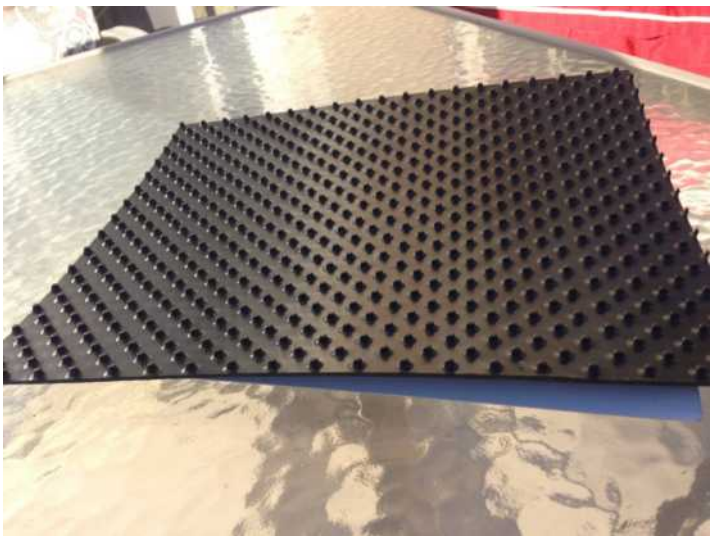


Figure 3: Drainage medium combines separation and impermeability in one product

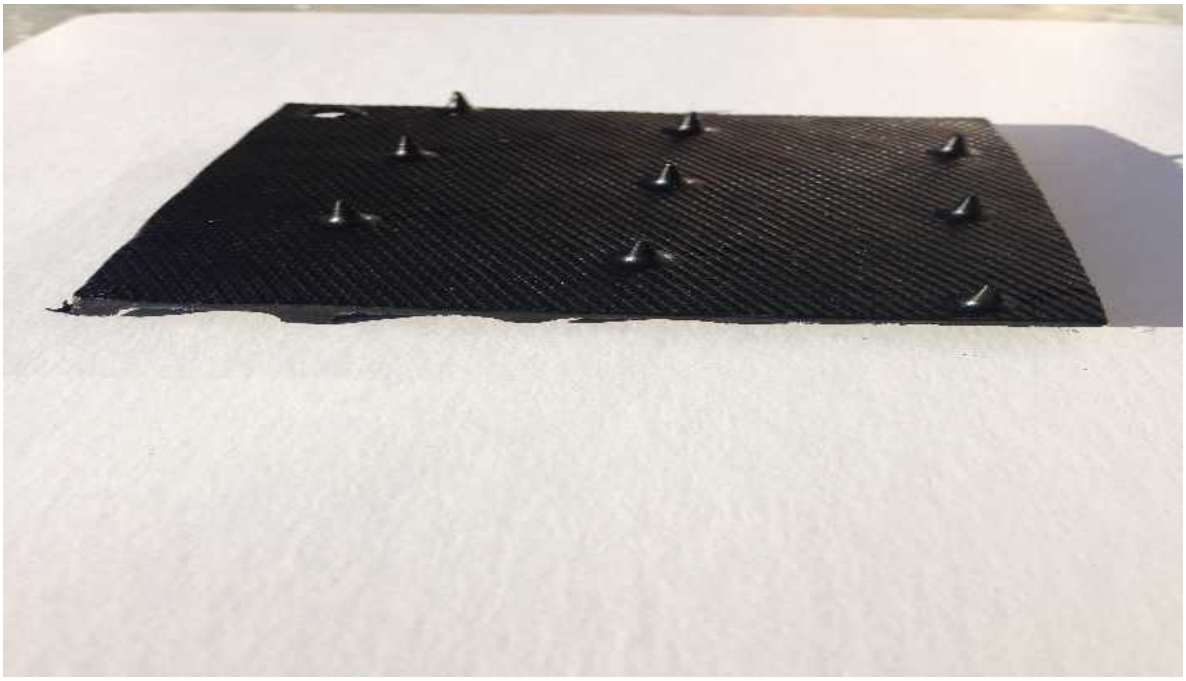
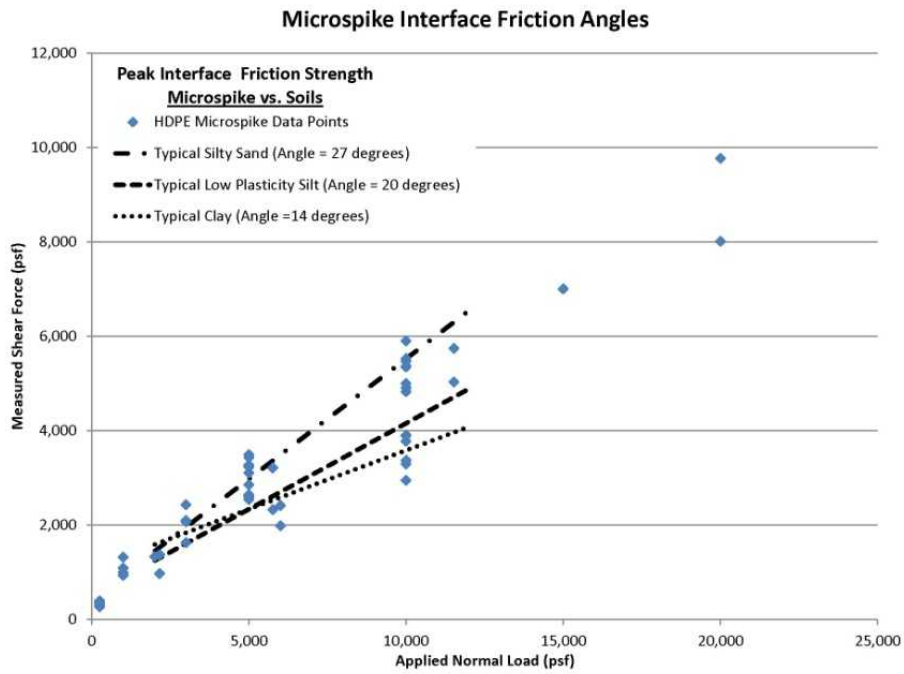


Figure 4: Aggressive texturing

This method of manufacturing is well known for its consistent interface and thickness and other properties that are paramount in design of any kind of slopes. Constant mechanical properties such as tensile stress, tear, and elongation at yield are over the industry standard, with elongation at break more than double the standard. This is a valuable characteristic on heap leach pads where loading methods can be questionable at times.

Interface characteristics

Numerous tests have shown that the interface between liner and soils and between liner and fabrics is very good. The fact that it is consistent texturing is positive (see Figure 5). Embossed surface textures exhibit high interface shear strength and lower post peak strength loss at lower normal stresses than those found in other designs.



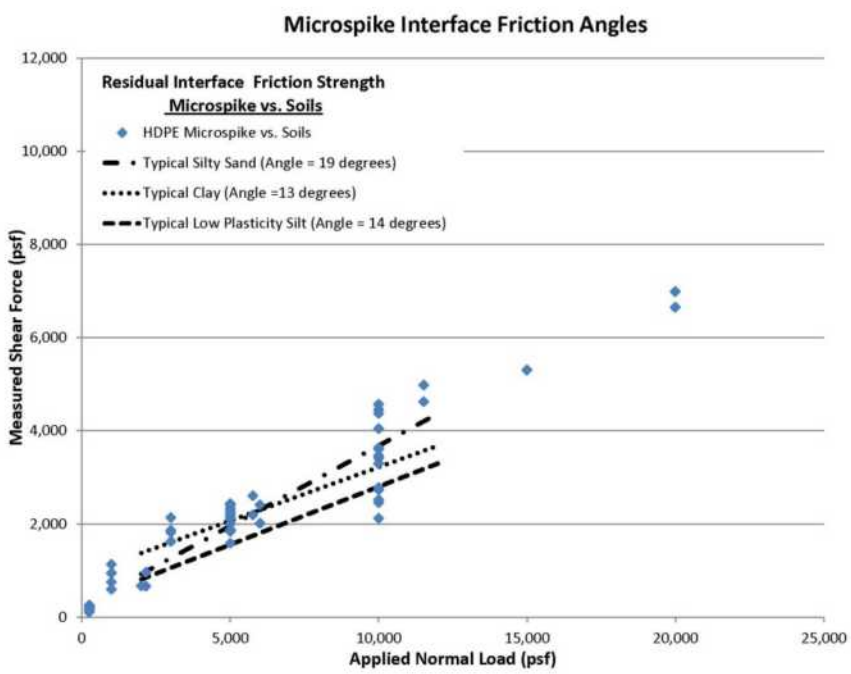


Figure 5: Interface friction angles

The advantage of calendared or embossed manufacturing of liners is that the texturing of the liners is a direct impression of the rollers used. The rollers are very consistent but can also be developed and tested to possibly improve design.

Figure 6: Consistent texturing

Mechanical properties reduction

Reduced mechanical properties of manufacturing of a required sheet thickness due to a texturing process such as other methods of manufacturing must be considered, especially for the long term where increasing stresses due to subsidence or localized settlements will occur and affect the out-of-plane (multiaxial) response as well as seam strengths under stress as can be found on heap leach pads. Using the flat die extrusion process, the geomembrane's mechanical, tensile, elongation, and other properties are closer to the values of a smooth sheet and do not change from roll to roll, as imperfections or thickness variations are not introduced during manufacture. See Table 2 for a comparison of flat sheet extruded liners and the industry standard, published by Geosynthetics Research Institute.

Table 2: Comparison of calendared and industry standard liners

Comparison with GRI GM 13			
60 mil HDPE Smooth			
Tensile properties	ASTM D6693 Type IV	GRI GM 13	**Structured liner**
Strength @ Yield	lb/in	126	**132**
Break strength	lb/in	228	**240**
Yield elongation	%	12	**13**
Break elongation	%	700	700
Tear resistance	lb	42	**45**
Puncture resistance	lb	108	**120**

60 mil HDPE Textured			
Tensile Properties	ASTM D6693 Type IV	GRI GM 13	**Structured liner**
Asperity height	mils	10	**16**
Strength @ Yield	lb/in	126	**132**
Break strength	lb/in	228	**240**
Yield elongation	%	12	**13**
Break elongation	%	100	**350**
Tear resistance	lb	42	**45**
Puncture resistance	lb	108	**120**

Interaction at the shear surface

Depending on the project design requirements (e.g., steep slopes, seismic response, construction and service loading, etc.), the peak and large displacement (post peak) interface strengths must be taken into consideration. For example, according to Stark and Richardson (2000) and Richardson and Thiel (2001), coextruded textured geomembranes exhibit large post-peak strength loss against geotextiles due to geotextile fiber tearing, pullout, and shear orientation. In addition to geotextile fiber/texture interaction, the texture itself may comb (lay over), causing greatly reduced post-peak shear strength (Stark and Richardson, 2000). Embossed surface textures, on the other hand, exhibit higher interface shear strength and lower post-peak strength loss at lower normal stresses commonly found in landfill closure designs.

Figure 7: Slide failure due to inconsistent texturing

Constructability with geotextile surfaces

Some designs require a textured geomembrane to be placed directly on a geosynthetic clay liner (GCL), or a geonet composite or geotextile to be placed directly over the textured geomembrane surface. This requires interfacing a nonwoven geotextile with the textured surface. The "Velcro" effect or "hook and loop" adhesion found in other textured surfaces is often problematic during field placement and requires very careful positioning or the use of a slip sheet. Embossed geomembrane surfaces, on the other hand, allow positioning of geotextiles and geocomposites without major difficulty. Quantifying the "hook and loop" phenomenon, and in particular the effects on interface shear and the textured surface during shear, has been the subject of extensive testing (Hebeler et al., 2005; Giroud, 2004; Frost et al., 2002). It has been put forward that the adhesion of certain textured liners against GCLs may cause a pulling effect when the surface is subjected to large temperature variations. This reduces or eliminates the overlap and therefore proper coverage. To the author's knowledge, this does not happen with smooth materials.

Geomembranes manufactured with surfaces textured by embossing provide consistently uniform quality texture that will supply the requisite interface shear strength. Additionally, as regards CQA field testing and laboratory conformance testing, structured or embossed textured geomembranes will provide a consistent value from roll to roll and across the roll width, thus providing requisite design reliability. This is generally not the case with other textured geomembranes, where "… the consistency of the texturing both across the roll and roll to roll should be a concern to the engineering community … What good is direct shear testing if the material provided is not consistent with respect to texturing?" (Sieracke, 2005) (See Figures 6 and 7.)

Table 3 is a summary of several design considerations that should be addressed when selecting a textured geomembrane to enhance slope stability factors of safety.

Table 3: Summary of properties for design considerations

Design consideration	Embossed
Consistent thickness (cross roll)	Yes
Consistent texture (cross roll)	Yes
Consistent asperity heights	Yes
Asperity heights > 15 mil	Yes
Consistent shear testing (cross roll)	Yes
Affect on multiaxial stress-strain (settlement/subsidence)	No
Texture combing during shear	No
Post peak reduction in shear strength	Yes
Easily placed with geotextile surfaces	Yes
Increased QC and CQA costs	No

Textured geomembranes perform well in combination with other structured membranes. In one heap project, because of leakage concerns, a triple layer system was developed using SuperGripNet for the tertiary base layer, MicroDrain as the secondary layer, and MicroSpike for the primary layer. The sandwich was tested for puncture and direct shear interface performance. The hydrostatic puncture test (ASTM D5514) was set up with 3 in of clayey sand as sub-grade against the spikes of the SGN and 3 in of gravelly sand nominally compacted as the upper drainage layer. After 101 hours at 12,000 psf, there were no punctures or holes. The interface direct shear (ASTM D5321) was tested under the same set-up at 3,000, 6,000, and 12,000 psf at 0.04 shear rates. This test reported a peak angle of 23° and a residual of 19°.

Mine site test pad for leach pad closure

A test pad to define suitability for use of structured liners under cover material was initiated by a large gold mine's mine closure division to confirm suitability of drain/liner on future mine closures (see Figure 3). The design engineer in Elko, Nevada, decided that the material to be used at one of the mines to be closed should be tested. Four 13.16 m^2 pieces of DrainLiner material, two of 2 mm and two of 1.5 mm, were delivered to the Cortez Hills Mine owned by Barrick. One piece each of 1.5 mm and 2 mm DrainLiner was placed with 10 oz of nonwoven heat-burnished fabric (HB down) on top. The cover soil comprising 0.304-0.46 m minus sub-grade with limited fines, much coarser than intended for the mine, was placed over the top of the liner using a Caterpillar D9 Bulldozer. When the cover soil had been placed to 0.941 m and had been tracked over many times by the bulldozer, a fully loaded mine water truck/scraper was brought in to make ten passes over the cover; this was also much more than was to be expected on the mine project.

After the material was tested, the liner was exposed for damage. As the material was completely undamaged, testers decided to reduce the cover soil to 0.6 m, to see if this depth would be sufficient. When the 0.6 m cover had been placed and the water truck completed its ten passes, a track hoe was brought in to carefully remove the overburden. The fabric, when exposed, was peeled back with the help of the track hoe to expose the liner. There appeared to be very little superficial damage to the liner and no penetrations of the liner. This was confirmed when the material was taken back to the engineer's lab and tested with a vacuum box.

The mine had been proposing to use a 2 mm liner with a geocomposite covering the liner for a drainage medium. This testing provided enough information to allow the engineer to use 1.5 mm drain/liner and 10 oz/sy fabric, saving the client considerable costs in material and installation.

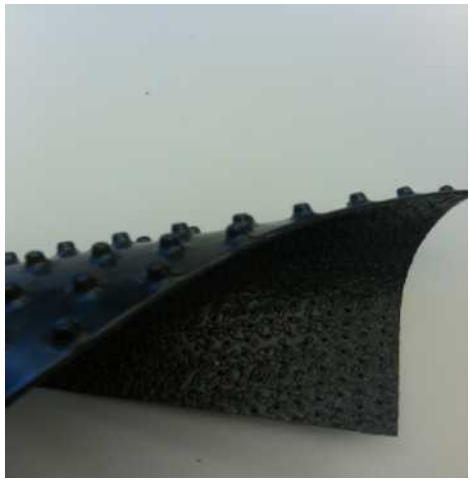


Figure 8: Flat die calendaring manufacture (textured/drain)

Separation for double-lined ponds

Before the drain/liner became an integral part of the liner, it was conventional to install three layers of material − liner, net, and liner −as a leak detection system and a means of reducing head pressure, which could potentially force dangerous liquids used in mining into the ground water. The new drain/liner product (see Figure 3) integrates the net into the liner, creating a system where there are only two materials to deal with in design and construction. This reduces freight, handling, and deployment costs, and results in a substantial cost saving in applications. This product can be used in ponds or in double-lined leach pads with high compressive strength, low creep, and sustainable transmissivity under loads.

Net can be extremely dangerous to walk on and virtually impossible to maneuver on slopes to deploy, making it very hard to perform quality assurance. With the drain/liner structured surface attached to the liner, walking up and down liners is very simple and safe. This is not only a safety factor; it also increases production by making the deployment of the primary liner much easier. A primary liner with a

textured side up provides for much safer walking around ponds, especially if wet, and drain/liner strategically attached around the ponds provides safer egress from ponds.

Summary

Structured flat die extrusion methods provide a variety of materials (see Table 1 and Figure 8) that give design engineers a wider range of options to more closely fit the requirements of the project they are designing. This creates a greater factor of safety, which is a benefit to all.

The advantages of the embossed sheet are listed below:

- The core or base thickness of the material is not affected by the manufacturing process; it is virtually a smooth sheet with spikes. This is seen in the performance of the embossed material, which has specifications closer to those of a smooth sheet than other types of texturing.
- Consistency between materials is constant because of the simple nature of manufacture.
- Tensile and strain properties are not affected.
- The material is completely homogeneous.
- The integral textured profile is embedded in the sheet.
- The integral drain profile is embedded in the sheet.
- It is cost efficient.
- It incorporates a drainage liner with a filter fabric for use in mine closures.

Use of a more aggressive textured material makes it possible to work with on-site materials instead of importing soils. For example, with a mine in Nevada, aggressive texturing was used in a design instead of importing of soils to replace the fine-grained natural soils, and what would have been a major expense was diverted.

There are applications for structured liners in all stages of a mine's life: ponds, tails dams, leach pads, and closure. They are easy to use and cost efficient, and they provide materials that have not been available before, making the impossible possible.

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Design considerations for geosynthetic clay liners in heap leach pad liner systems

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Abstract

Traditionally, leach pad lining systems have consisted of a single geomembrane liner placed directly over a prepared subgrade of locally available soil or occasionally compacted clay. Heap fills are constructed by placing a layer of highly-permeable drainage stone (overliner) over the geomembrane. Crushed ore is then placed on the leach pad in 3 m to 10 m thick lifts, often reaching final heights of 100 to 160 m. The crushed ore is irrigated with a chemical solution which dissolves the target metals from the ore. The nature of the leaching solution depends on the metal being targeted. Low pH sulfuric acid solutions are generally used to leach copper, uranium and nickel; high pH sodium cyanide solutions are used to leach gold and silver. The metal-laden pregnant leach solution (PLS) passes down through the ore pile and is captured in a drainage system. Metals are extracted from the leach solution and the solution is then recycled back onto the leach pile.

When under load, geomembranes are vulnerable to damage from large stones both in the soil subgrade and in the overlying drainage layer. Although intact geomembranes are virtually impermeable, installed geomembranes will have a small number of holes due to imperfect seams or damage during construction, overliner placement, and subsequent filling operations. These holes serve as open pathways for leakage into the soil below. Smith and Welkner (1995) estimated liner leakage rates ranging from 5 to 10,000 L/ha/day, depending on the type of liner and level of construction quality assurance (CQA). Thiel and Smith (2003) reported liner leakage rates up to 2,000 L/ha/day for one valley fill facility with heads ranging from 15 to 35 m, and total leakage greater than 500,000 L/day have been measured at another facility.

Other factors that are often overlooked include over-stressing that can occur next to pipes (up to 130% of the nominal load) and potential for increased geomembrane deformation due to high temperatures in bio-heap leach facilities (60° C) (Smith, 2012).

Composite liner systems

To reduce leakage through defects, a low-permeability layer can be used beneath the geomembrane to form a "composite" liner system. The low-permeability material beneath the geomembrane is typically either a compacted soil (clay or silt) liner or a bentonite clay-based geosynthetic clay liner (GCL). These systems have been used for base liners for leach pads as well as covers and base liners at municipal solid waste landfills in the USA for more than 20 years. Compacted soil liners are typically constructed at wet of optimum moisture content to achieve a maximum hydraulic conductivity of either 10^{-6} or 10^{-7} cm/sec, depending on local regulatory requirements. While preparing soil liners wet of optimum moisture content is beneficial for hydraulic performance, it can be detrimental for slope stability, as most famously demonstrated at the Kettleman Hills landfill (Mitchell et al., 1990).

Based on liner leakage measurements collected by the US Environmental Protection Agency (USEPA) at 287 landfill cells, spanning 91 sites (Bonaparte et al., 2002), GCL-based composite liner systems have been shown to allow less leakage than clay-based composite liner systems. Due to improved hydraulic performance, limited availability of fine-grained soils at many mine sites, and speed of installation, GCLs are seeing increased use in heap leach pad liner systems.

Since conditions under an ore heap can be extreme – harsh chemical environments, high temperatures, enormous normal stresses aggravated by shearing or horizontal loads from sloping ground and stacking equipment, and coarse-grained angular soils – the use of geosynthetics and GCLs in these applications requires special design considerations. This paper will discuss designing with GCLs in leach pad liners, with respect to puncture protection for the geomembrane, chemical compatibility, and shear strength for slope stability. The information presented will include a review of the existing literature, the authors' project experience, and the latest research intended to improve GCL performance in these aggressive conditions.

GCLs as puncture protection

The use of well-graded, angular stone as the overliner layer directly above the geomembrane presents obvious puncture challenges. The landfill industry has addressed this but adding non-woven geotextiles for puncture protection, but this is often proscribed on leach pads for stability reasons. Additionally, it is common to encounter subgrade soils at mine sites containing appreciable quantities of gravel, leaving geomembranes vulnerable to puncture damage from below. In these settings, geomembrane damage could occur either with increasing normal stress (as ore is placed), or during shear displacement (due to dynamic construction loads, seismic forces, slope movement, or settlement).

Narejo et al. (2007) demonstrated that GCLs can effectively cushion geomembranes from subgrade puncture challenges, providing protection that was comparable to a 540 g/m^2 nonwoven geotextile. Compacted soil liners, unless devoid of stones, cannot be expected to offer the same protection; in fact, under the high normal loads seen at many leach pads, any stones in the soil subgrade present increased puncture risks. (Puncture risks from such stones are increased when the soil is compacted wet of optimum, leading to greater consolidation around the stones). Also, since the rate of leakage through defects in a composite liner decreases with decreasing hydraulic conductivity of the underlying soil layer, GCL-based composite liner systems are expected to allow substantially less leakage than soil-based composite liner systems should a puncture occur (Narejo et al., 2002).

Athanassopoulos et al. (2009) presented example calculations which demonstrated that the improved PLS recovery rate afforded by adding a GCL below the geomembrane could potentially translate to hundreds of thousands of dollars per year of added revenue, which over the life of the project, would exceed the cost of the initial investment in the GCL.

Athanassopoulos et al. (2009) performed a series of short-term static puncture tests on geomembrane-only and geomembrane/GCL composite liner systems at normal stresses as high as 5,172 kPa, or a very high equivalent ore depth of 270 m − 180 m with a factor of safety (FS) = 1.5. The results of the high-load puncture testing showed that geomembranes alone are generally expected to experience more puncture damage (puncturing or strain deformation past yield) from the overliner than a geomembrane with an underlying GCL. Geomembrane samples subjected to stresses greater than 2,586 kPa (135 m of ore) experienced over 300 permanent deformations per m^2. A geomembrane sample tested alone at the highest normal stress, 5,172 kPa, also had two punctures in a 0.09 m^2 area, each measuring 2 mm in diameter. The GCL's benefit, in terms of reducing biaxial strains in the geomembrane, appeared to increase with increasing normal stress.

Shear-induced puncture damage

Fox et al. (2013) performed a series of direct shear tests on geomembranes in contact with a compacted clay subgrade containing 20% gravel, to evaluate potential for shear-induced puncture damage. In a leach pad setting, especially those on steeper ground, shear forces could occur as a result of heavy loading or unloading equipment, seismic forces, or settlement causing downdrag. Shear forces will also be present at the leading face of any heap for the first lift of ore. Tests were performed both with and without a GCL placed between the geomembrane and the compacted clay liner at normal stresses ranging from 348 to 4,145 kPa (18 to 220 m of ore) to evaluate the level of protection provided by the GCL. Results indicate that geomembranes are vulnerable to severe puncture damage due to shear displacements under high

normal stress, and that including a GCL between the geomembrane and the compacted clay liner can essentially eliminate such damage.

Figure 1: Geomembrane puncture damage from 38 mm overliner, without underlying GCL (left) and with underlying GCL (right). Testing at 5,172 kPa for 24 hours

Chemical compatibility

GCLs are factory-manufactured liners containing sodium bentonite clay with a hydraulic conductivity of 5×10^{-9} cm/sec with deionized water. The amount of bentonite swelling, and therefore, the GCL hydraulic conductivity, can be influenced by the presence of divalent cations (e.g., calcium and magnesium), high ionic strength, and/or extreme pH solutions. GCL chemical compatibility is therefore an important consideration in heap leach pad applications because high-pH cyanide solutions are used to leach gold and silver, and low-pH sulfuric acid solutions are generally used to leach copper, uranium, and nickel laterites.

Relative to high-pH dilute cyanide leaching solutions, CETCO (2000) presented hydraulic conductivity results which showed that GCLs are compatible with gold PLS, with a long-term hydraulic conductivity, on the order of 10^{-9} cm/sec.

Relative to sulfuric acid leaching solutions, Jo et al. (2001) found that sodium bentonite exhibited approximately a 50% decrease in swell at pH values less than 3. As part of the same study, GCL permeability values on the order of 10^{-6} to 10^{-5} cm/sec were measured at pH values less than 2. Ruhl and Daniel (1997) found that when exposed to strong acid, a GCL's buffering capacity was not exhausted until after 15 pore volumes of flow. At the low water flow rates expected in a liner, it may take years for

the first 15 pore volumes to flow through the liner. By this time, the liner will likely be covered and compressed by several hundred feet of ore, so that the GCL's permeability will be greatly reduced.

Additionally, Shackelford et. al. (2000) and Jo et al. (2004) have shown that prehydration of a GCL with clean water prior to exposure to leachates can significantly improve hydraulic conductivity. As shown in USEPA (1996), GCLs placed on subgrade soils prepared at optimum, or wet of optimum, moisture content can hydrate within several weeks of placement. Accordingly, it is likely that a GCL placed on a moist subgrade will be at least partially hydrated with subgrade moisture before it is exposed to any aggressive acidic PLS. This benefit in chemical compatibility could be offset by decreased shear strength, as discussed below.

Athanassopoulos et al. (2009) performed permeability tests on both an intact GCL sample and a GCL sample that had been pierced by a large stone during an earlier high-stress puncture test. The permeant solution for these tests was a copper PLS collected from an active copper leach pad in the southwestern USA. Chemical analysis of the copper PLS is presented in Table 1. Permeability testing was performed at effective stresses ranging from 34.5 to 3,166 kPa (2 to 170 m of ore), to span the range of operational stages of a typical copper heap leach facility. Testing continued until specific termination criteria (steady-state flow and chemical equilibrium) were established between the effluent and influent. The results of these permeability tests showed that at low effective stress, the GCL permeability was greater than 10^{-6} cm/sec. As effective stress was increased to simulate increasingly higher ore heights on the liner, the permeability decreased significantly, reaching values less than 10^{-10} cm/sec at 1,440 kPa (75 m of ore). Interestingly, similar behavior was exhibited by the punctured specimen, as shown in Figure 2. The improvement in GCL hydraulic conductivity with increasing effective stress, even in harsh chemical solutions, is similar to the findings of Daniel (2000) and Thiel and Criley (2005). This is important because post-construction defects will most likely occur at greater ore depths.

Research is underway to evaluate the use of polymer-amended bentonites and bentonite-polymer nanocomposites for improved GCL hydraulic performance in contact with low-pH mining solutions (Scalia et al., 2011). Encouraging early results with low-pH uranium mill tailings and nickel laterite PLS are expected to be published later in 2013.

Table 1: Copper PLS chemistry

pH	1.8
Electrical conductivity	37,000 μmhos/cm
Aluminum	5,044 ppm
Calcium	262 ppm
Copper	802 ppm
Iron	1,788 ppm
Magnesium	498 ppm
Zinc	198 ppm

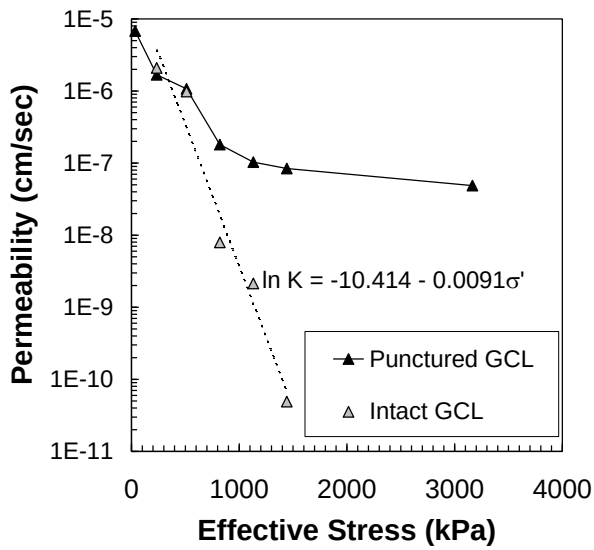

Figure 2: GCL Permeability with copper PLS

Shear strength and slope stability

Since many leach pads involve a combination of steep slopes and high normal stress, shear strength and heap stability is a critical design consideration. Historically, laboratory direct shear devices were limited to loads less than 690 kPa, representing approximately 36 m of ore, which falls far short of the maximum loads expected in most leach pads. In recent years, universities and commercial laboratories have gained the ability to perform shear testing at higher normal stresses. The following is a summary of some of these tests involving GCLs.

Athanassopoulos et al. (2009) tested different styles of textured geomembranes against needle-punch reinforced GCLs, at normal stresses up to 2,758 kPa (146 m of ore). Results, which are summarized in Table 3, showed peak secant angles ranging from 30° at 517 kPa (27 m of ore) to 20° at 2,758 kPa (146 m of ore), and corresponding large displacement secant angles decreased from 14° to 7°.

Thielmann et al. (2013) evaluated geomembrane/GCL interface strengths for ultra-high normal stresses up to 4,144 kPa (220 m of ore). Tests were conducted with the geomembrane/GCL liner materials placed between a lower layer of sand and an upper layer of coarse gravel, to replicate common leach pad field conditions. The peak friction angles (Table 2) ranged from 22° at 348 kPa (18 m of ore) to 15° at 4,144 kPa (220 m of ore), and corresponding large displacement secant angles decreased from 13° to 5°.

Table 2: Summary of past interface shear results on textured geomembrane/ GCL liner systems under high normal stress

Reference	Normal stress (kPa)	Equivalent ore height (m)	Failure mode	Peak secant angle	LD secant angle
Thielmann et al. (2013)	354	19	Interface	22°	13°
Blown film texturing	699	37	Interface	21°	9°
	1043	55	Partial internal	20°	8°
	2078	110	Partial internal	19°	5°
	3112	165	Internal	17°	5°
	4146	220	Internal	15°	5°
Athanassopoulos et al. (2009)	517	27	Interface	30°	14°
Structured texturing	1034	55	Interface	24°	12°
	1724	91	Interface	20°	9°
	2758	146	Interface	20°	7°
Athanassopoulos et al. (2009)	1034	55	Interface	24°	12°
Blown film texturing	1724	91	Interface	23°	10°
	2758	146	Partial internal	20°	7°

Note: Large displacement strengths in Thielmann et al. (2013) were measured at 200 mm of shear displacement. Large displacement strengths in Athanassopoulos et al. (2009) were measured at 75 mm of shear displacement

Table 2 is useful in showing how interface friction angles change with increasing normal stress. For slope stability analyses, the most important area of the heap is toward the toe (inside the "stability zone"). If a slope stability analysis is performed with a non-linear failure envelope, the sections of the liner under the deepest part of the ore would have the lowest peak and large displacement friction angles; however, in the critical "stability zone", towards the toe, the normal loads are lower and therefore the friction angles, and thus resistance to sliding, will be higher.

Breitenbach and Swan (1999) and Parra et al. (2010) observed that liner components placed in contact with coarse soils are expected to see an increase in strength, due to local out-of-plane deformation, or "dimpling", of the liner components under the gravel particles. These results suggest that the common practice of performing direct shear tests using rigid backing plates is conservative (perhaps overly conservative) with respect to shear strength of composite liners that are overlain by coarse soils.

At extremely high normal stress, the interface strength between a textured geomembrane and a needle-punch reinforced GCL can exceed the strength of the needle-punched reinforcement and the critical interface could occur internally within the GCL. The critical normal stress associated with this failure mode transition depends on the specific materials (e.g., GCL peel strength, geomembrane texturing and asperity height) and testing conditions (e.g., hydration/consolidation, displacement rate) and, as such, can vary over a wide range. Recent laboratory studies noted that this failure mode transition occurred at normal stresses ranging from 692 kPa (Fox and Ross, 2011), to 2,072 kPa (Thielmann et al., 2013), to 2,758 kPa (Athanassopoulos et al., 2009). Triplett and Fox (2001) observed no GCL internal failures for geomembrane/GCL interface tests conducted at normal stresses as high as 486 kPa. McCartney et al. (2009) observed no GCL internal failures in a database of 534 geomembrane/GCL interface tests performed at normal stresses as high as 965 kPa. The variability of normal stress at failure mode transition further highlights the need for project-specific shear tests using representative site materials and project-specific conditions.

Observations of internal shear failure of needle-punch reinforced GCLs have thus far been limited to the laboratory, as there are no known cases of internal shear failure of needle-punch reinforced GCLs in the field (Fox and Ross 2011; Koerner 2012). Nonetheless, the potential for both interface and internal failure should be considered for designs that subject hydrated GCLs to high normal stress levels. (Note that, unless the GCL is installed on a subgrade prepared at optimum, or wet of optimum moisture content, the assumption of a fully hydrated GCL across the entire failure plane is a conservative one.)

Peak versus residual shear strengths for design

When reviewing laboratory direct shear results, it is important to understand the differences between peak and residual (or large displacement) strengths. Despite years of debate, there is no consensus in the industry on the issue of whether to use peak or residual shear strengths for design. Various authors have suggested different approaches, with some summarized in Table 3 (from Smith, 2008).

Table 3: Summary of recommendations for use of peak vs. residual shear strength parameters

Author	Date	Recommendation
Stark and Poeppel	1994	Performed both peak and residual analyses
Liu et al.	1997	Use large displacement (post-peak) shear strengths
Gilbert	2001	Use residual or post-peak parameters for all cases
Thiel	2001	a) residual for all components, or b) residual for interface with lowest peak strength, or c) peak and with high FS value, or d) residual on side slopes and peak on base areas
7th IGS Workshop, Nice, France	2002	74% of participants recommended using values between peak and residual, 12% recommended peak and 4% recommended residual
Jones et al.	2000	Use post-peak for large equipment or steep slope field conditions (i.e. where large displacements are likely); otherwise use peak shear strength
Koerner	2003	Use peak shear strength when FS is greater than 1.5 except for seismic loading conditions or for unusual construction practices, where residual may be appropriate
Christie	2008	1) residual for all components and moderate FS value, or 2) use interface with lowest peak strength and high FS value, and 3) check lowest residual strength and FS >1.1

Improving stability of leach pads with low-strength interfaces

Breitenbach and Athanassopoulos (2013) presented a series of slope stability calculations for a hypothetical ore heap study section containing a leach pad liner with a very low strength bentonite clay to geomembrane liner interface (friction angle = 7°). The calculations evaluated the benefit of incorporating non-planar features, such as stability berms, shear keys, or backwall benches, along critical portions of the liner system. For one example study section (Figure 3), the use of stability berms (or "speed bumps") along the toe of the structure improved the FS against sliding from a clearly undesirable FS = 0.89 to a favorable FS = 1.4. These findings have important design implications for leach pad liner systems with low interface strengths. The incorporation of non-planar features in liner systems containing GCLs can improve overall stability and maintain an adequate FS, even in worst-case loss of GCL internal strength due to either rupture, pullout, or degradation of the needle-punched reinforcing fibers, or reduction in geomembrane/GCL interface strength due to extrusion of hydrated bentonite through the woven geotextile component of the GCL. It is important to note that for many leach pads the inclusion of an undulating subgrade can also reduce grading costs.

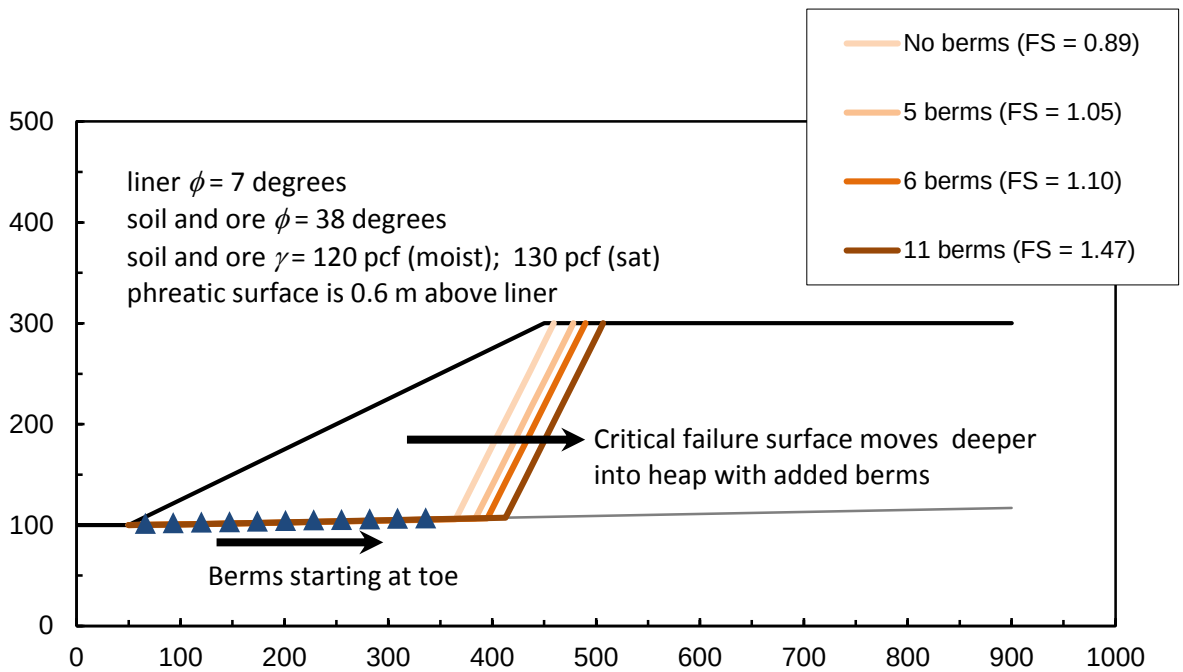

Figure 3: Improvement in slope stability with addition of stability berms

Conclusions

GCLs are being increasingly used in leach pads due to improved containment, the limited availability of fine-grained soils at many mine sites, speed of installation and reduced risk of cost overruns and construction delays. Since conditions under an ore heap can be extreme – harsh chemical environments, high temperatures, enormous normal stresses, and coarse-grained angular soils – the use of geosynthetics and GCLs in these applications requires special design considerations, including:

- **Puncture protection.** Stones in both the subgrade and overliner present obvious puncture challenges to the geomembrane, especially under high normal stress. GCLs have been shown to effectively cushion geomembranes from subgrade puncture challenges, providing protection that was comparable to a 540 g/m^2 nonwoven geotextile.
- **Chemical compatibility.** GCL hydraulic conductivity can be negatively impacted by high ionic strength and/or extreme pH solutions, which can be common in mining, especially with sulfuric acid leaching processes. Under low normal stresses, GCLs have been shown to have a hydraulic conductivity greater than 10^{-6} cm/s with copper PLS. However, as the effective stress was increased, to simulate increasingly higher ore heights on the liner system, the

permeability decreased significantly, reaching values less than 10^{-10} cm/sec at 1,440 kPa effective stress.

- **Shear strength.** High-load direct shear testing of geomembrane/GCL liner components showed peak secant angles ranging from 30° at 517 kPa (27 m of ore) to 15° at 4,144 kPa (220 m of ore), and corresponding large displacement secant angles of 14° to 5°. Although there have been no known cases of internal shear failure of needle-punch reinforced GCLs in the field, the potential for both interface and internal failure should be considered for designs that subject hydrated GCLs to high normal stress levels.
- **Slope stability improvement methods.** Breitenbach and Athanassopoulos (2013) presented a series of slope stability calculations which showed that the addition of a series of non-planar features along a low-strength liner system can drastically improve stability, even in worst-case conditions (friction angle = 7°). Depending on local terrain, incorporating natural undulations of the site subgrade into grading plan can reduce grading costs.

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Design, construction, and performance of closure cover systems for spent heap leach piles – a state-of-the-art review

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Abstract

Considerable research has been carried out in Canada and throughout the world on developing and progressing cover system design for reclamation or closure of various mine waste materials. Technical guidance documents have been written to clearly outline the preferred techniques for design, construction, and performance monitoring of closure cover systems for the mining industry. These techniques, which also apply to cover systems for spent heap leach piles, are reviewed in this paper.

Defining design objectives is the first step in developing an appropriate cover system design for closure of a spent heap leach pile. These vary from site to site but generally include objectives related to physical and chemical stabilization, as well as meeting land-use objectives and other societal values. Two key processes are fundamental to both defining and meeting these objectives; namely, the surface water balance and surface energy balance. Controlling oxygen ingress into spent heap leach piles is certainly more challenging than controlling water ingress, based on the authors' experience.

Cover system design alternatives most applicable to closure of spent heap leach piles include store-and-release or evapotranspiration (ET) cover systems, enhanced store-and-release cover systems, barrier-type cover systems, and cover systems with engineered layers. Selection of an appropriate cover system design requires completion of a receiving environment impacts analysis. The goal is to select a closure scenario that will attenuate peak concentrations of contaminants of concern in the receiving environment to levels that can be assimilated without adverse affects over the long term.

Three additional elements reviewed in this paper include landform design and surface water management, cover system construction, and performance monitoring. Developing an appropriate closure landform design and surface water management system are critical to ensure that the performance of the cover system will be sustainable over the long term. It is imperative that cover systems be constructed with a high level of quality control to ensure the as-constructed cover systems are representative of the

designs. Finally, direct measurement of field performance is the preferred methodology for measuring performance of a closure cover system for spent heap leach piles.

Introduction

Earthen or "dry" cover systems are typically required for closure of spent heap leach piles. The primary purpose of a cover system is restoration of the surface to a stable, natural condition while minimizing degradation of the surrounding environment following closure. Cover systems can have numerous design functions, including but not limited to isolation of waste, limiting the influx of atmospheric water and/or oxygen, controlling the upward movement of process-water constituents / oxidation products, and providing a medium for establishing sustainable vegetation (MEND, 2004; 2012; INAP, 2011).

Cover systems can be simple or complex, ranging from a single layer of earthen material to several layers of different material types, including native soils, suitable overburden, non-reactive waste materials, geosynthetic materials, and oxygen-consuming materials (MEND, 2004). The complexity of any given cover system design depends on several factors, including the climate regime at the site, reactivity and texture of the heap leach material, hydrogeologic setting of the facility, and both physical and chemical properties of locally available cover materials.

Design objectives of cover system designs applicable to closure of spent heap leach piles are reviewed first, followed by key characteristics of generic cover system designs, and finally, key aspects related to cover system design, landform design and surface water management, construction, and performance monitoring.

Background

A key design function of most cover systems placed over spent heap leach material is to protect the downstream receiving environment following closure of the facility (O'Kane and Wels, 2003). This is achieved by reducing "net percolation of meteoric water" (defined below) into the heap pile, which reduces effluent seepage volumes. This reduction in seepage volumes ideally limits peak concentrations of contaminants in receiving waters to levels that can be assimilated without adverse effects on the aquatic ecosystem.

Net percolation is the net result of meteoric water infiltrating into the cover profile (see Figure 1). Meteoric water will either be intercepted by vegetation, runoff the cover, or will infiltrate into the surface. Water that infiltrates will be stored in the "active zone" and a large majority will then subsequently exfiltrate back to the surface and evaporate, or be removed by transpiration. The infiltration can also move laterally downslope within and below the active zone. In addition, a percentage of the infiltrating

meteoric water will migrate beyond the active zone due to gravity overcoming the influence of evapotranspiration and moisture retention, and result in net percolation to the underlying waste.

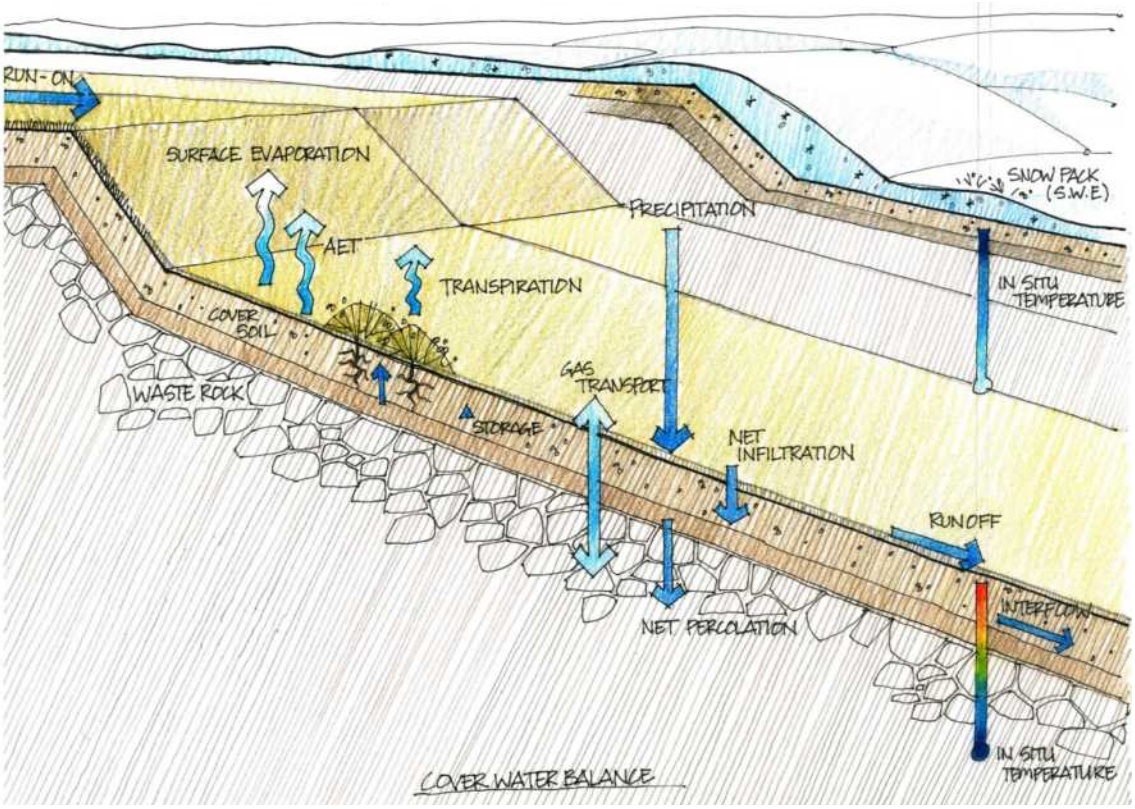

Figure 1: Schematic of hydrologic processes that influence performance of sloping mine waste cover systems (MEND, 2012)

Developing cover system design objectives

A cover system is often an integral component of a closure plan for any given heap leach facility. The objectives of a cover system may vary from site to site but generally include (from MEND, 2012):

1. Physical stabilization:
    - Provide dust and erosion control, particularly wind and water erosion of heap leach materials; and
    - Act as a barrier to prevent direct contact of the heap leach material by flora and fauna;
2. Chemical stabilization:
    - Chemical stabilization of mine waste through control of oxygen or water ingress; and
    - Contaminant release control through control of infiltration;
3. Meeting land-use objectives and other societal values:

- Provide a growth medium for establishment of sustainable vegetation; and
- Reclaim the area for desired post-closure land uses.

Two key processes are fundamental to both defining and meeting these objectives; namely, the surface water balance and surface energy balance (MEND, 2012). The surface water balance is a function of many factors such as climate, soil type, and hydrogeologic setting. Of these factors, the hydrogeologic setting of a heap leach facility exerts a predominant control on the cover system requirements. For example, the location of the final water table has a large influence on predicted amount of net infiltration, leaching processes, and oxygen ingress into a heap leach facility. In a setting where the water table interacts with stored waste, leaching (by water and any remnant acidity) will take place regardless of the ability of the cover system to control water infiltration. Therefore, the design of a cover system is highly dependent on the conceptual and detailed understanding of hydrogeologic conditions at the site, leading to an understanding of the surface water balance of the cover system.

Controlling oxygen ingress into spent heap leach piles is certainly more challenging compared to controlling water ingress based on the authors' experience. For tailings impoundments, one can arguably achieve oxygen ingress control by maintaining high saturation, or ideally, flooded conditions thereby reducing the issue of oxygen ingress to one of diffusion, as opposed to advection. However, for heap leach piles, in the absence of flooded conditions it is challenging to maintain sufficient saturation levels within the pile such that diffusion will be the dominant mechanism influencing gas transport. In this case, one must rely on high saturation conditions in a cover layer over the heap leach pile. Furthermore, given the elevation differences, there is the potential for temperature differences within the heap leach material compared to ambient conditions. Temperature differences may induce convective gas transport deep within the heap leach pile. Other convective transport phenomena, such as barometric pumping, wind action, and pressure differentials can also induce convective flow (Wels *et al.*, 2003). Therefore, it is clear that limiting oxygen availability to heap leach material is challenging. Finally, a cover system designed to limit oxygen ingress would need to adhere to a high level of construction quality control, seek to minimize the potential for differential settlement and cracking following construction, and would require a commitment to maintenance of these conditions.

Cover system design alternatives

MEND (2004; 2012) and INAP (2011) provide good overviews of cover system design alternatives applicable to closure of mine waste stockpiles, including spent heap leach piles. Table 1 summarizes four types of cover systems that are most applicable to closure of spent heap leach piles.

Table 1: Cover system design alternatives for closure of heap leach piles

Category	Best suited for:	Key attributes	Comments
Store-and-release cover system (also referred to as ET cover system)	• Arid and semi-arid climates • Higher net percolation rates (10–40% of MAP[1]) • Control of gas ingress/ egress not required	• ~1–2 m of well-graded soil or inert run-of-mine (ROM) waste material	• Sustainable vegetation cover is critical • Landform grading may be required to shed runoff waters
Enhanced store-and-release cover system	• Arid and semi-arid climates • Moderate net percolation rates (10–15% of MAP) • Control of gas ingress/ egress not required • Sites where higher soil water needed for revegetation efforts	• ~1–2 m of well-graded soil or inert ROM waste material overlying a reduced permeability layer (RPL) • RPL can be compacted weathered heap leach material at surface or capillary break layer	• Longevity of RPL must be addressed during design stage • More robust landform design required to prevent erosion
Barrier-type cover system	• Semi-humid and humid climates • Low net percolation rates (5–10% of MAP) • Control of gas ingress/ egress desired	• ~1 m growth medium/ protective layer overlying a low permeability layer (LPL) • LPL can be compacted clay, sand-bentonite, or permanently frozen layer	• Longevity of LPL must be addressed during design stage • More robust landform design required to prevent erosion
Cover system with engineered layers	• Semi-humid and humid climates • Very low net percolation rates (<5% of MAP) • Control of gas ingress / egress desired	• ~1 m growth medium / protective layer overlying a geo-membrane liner (e.g. LLDPE, GCL, BGM[2]) • Bedding and drainage layers usually required	• Costly • Longevity of geomembrane liner must be considered • Require robust landform design

1. MAP: mean annual precipitation

2. LLDPE: linear low-density polyethylene; GCL: geosynthetic clay liner; BGM: bituminous geomembrane liner

The mining industry has generally labeled cover systems for mine waste stockpiles as per the cover system's primary function (O'Kane and Ayres, 2012). Examples include:

1. "store-and-release" type cover systems;
2. "water-shedding" type cover systems; or
3. "capillary break" type cover systems.

However, this approach has led to a significant misunderstanding in regards to cover system performance expectations. For example, a "water-shedding" cover system will typically include a barrier or low permeability layer within the cover system and then an overlying growth medium layer. In reality, the growth medium layer is simply another label for a store-and-release cover layer because the functionality of the two is the same (i.e., store surface infiltration within the material, then

evapotranspirate moisture to release it back into the atmosphere). The underlying barrier layer is required to promote "water-shedding" for conditions when storage is overwhelmed in the growth medium layer (e.g., periods of high precipitation).

Cover system design methodology

A key component in developing a defensible cover system design for a spent heap leach pile is a receiving environment impacts analysis. The specific environmental impacts to be evaluated depend on the objective(s) of the proposed cover system design in conjunction with the site closure plan as well as local, state, and federal government commitments. Environmental impacts most commonly evaluated during a cover system design include:

- impacts on surface water quality;
- impacts on groundwater quality;
- impacts on air quality;
- impacts on vegetation; and
- impacts on wildlife.

The goal is to select a closure scenario that will attenuate peak concentrations for contaminants of concern in the receiving environment to levels that can be assimilated without adverse affects over the long term. Once the required criteria have been determined for closure of a heap leach facility, feasible cover system design alternatives can be developed and carried forward into a soil-plant-atmosphere (S-P-A) numerical modeling program. In addition, closure criteria, developed on a site-specific basis, provide the basis for measuring field performance of a cover system and ultimately, determination of whether the cover system is working (O'Kane and Ayres, 2012).

The design of a closure cover system and in particular, determination of predicted rates of net percolation over the long term, should involve S-P-A numeric simulations using a long-term climatic database. This database should be comprised of at least 50 to 100 years of daily records from local and regional meteorological stations. Each year of the long-term climate database should be run continuously for each cover design alternative, thereby taking into account antecedent moisture conditions. This allows curves, as shown in Figure 2, to be developed for each cover alternative, providing mining companies with a means of understanding "risk" or the "probability of exceeding" a certain net percolation rate for a given heap leach facility.

A key factor to consider during the design process is the anticipated climax vegetation species that will develop on the cover system. A vegetation cover is important not only for minimizing soil erosion,

aesthetics and creating wildlife habitat, but lower net percolation rates can be realized with a mature vegetation canopy due to higher rates of interception and evapotranspiration.

The authors suggest that for general context, cover systems that achieve:

4. "very low" net percolation rates are those that have a high probability for the net percolation rate for any given year to be between 1% to 5% of precipitation;
5. "low" net percolation rates are those that have a high probability for the net percolation rate for any given year to be between 5% to 15% of precipitation; and
6. "moderate" net percolation rates are those that have a high probability for the net percolation rate for any given year to be between 10% to 40% of precipitation.

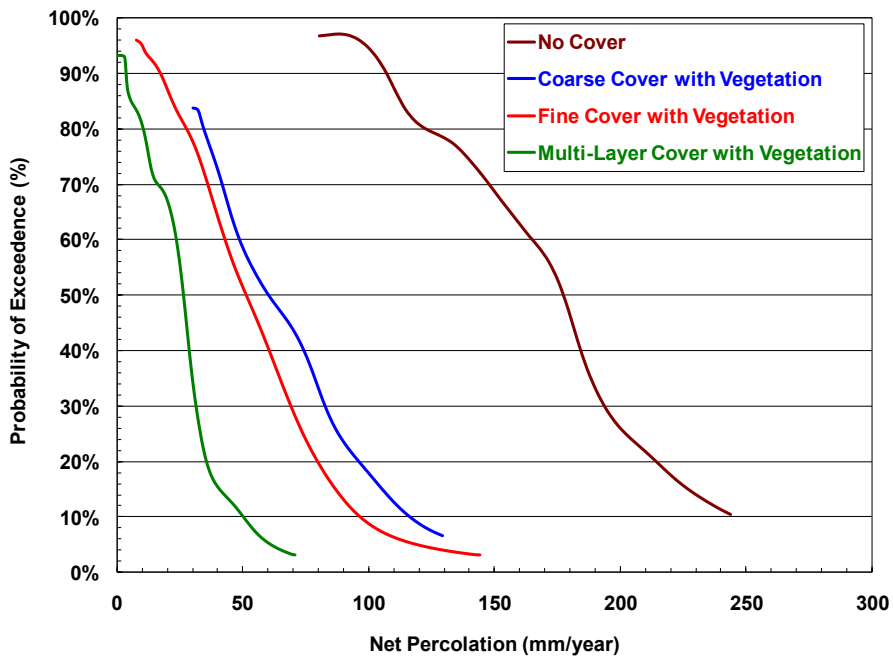

Figure 2: Example illustration of net percolation probability of exceedence curves generated from the results of continuous 100-year S-P-A simulations for a "no cover" scenario, and three different cover system design alternatives

Landform design and surface water management

Historically, rehabilitated mine landforms possess uniform slopes conforming to neat lines and grades. This lends itself to uniformity of design and construction, but does not necessarily achieve the mine closure objectives of minimum erosion and long-term sustainability. Uniform landforms represent immature topography, and are poised to evolve to lower energy states by shallow slope failures or accelerated erosion. In contrast, the development of a sustainable landscape for mine closure involves the

development of landforms that replicate natural landscapes. The replication of mature and relatively stable natural systems reduces the rate and risk of accelerated erosion.

Following a tour of 57 abandoned and partially reclaimed operating mines, McKenna and Dawson (1997) created an inventory of mine closure practices, physical performance of reclaimed areas, and environmental impacts of reclaimed and abandoned mines. The inventory shows that the greatest physical risk to the landscapes is associated with gully erosion and re-established surface water drainage courses. Poor surface water management and landform instability are common factors leading to failure of mine waste cover systems around the world (MEND, 2004). These factors are much more prevalent at sites situated in cold regions where processes such as frost heave and snowmelt can substantially diminish the integrity and performance of a reclaimed mine landform (MEND, 2012).

The incorporation of natural slope features into the final landform design for stockpiles not only improves aesthetics, but also emulates slopes that are in equilibrium with local conditions of rainfall, soil type, and vegetation cover (Ayres et al., 2006). The relatively small increase in costs for engineering and construction for creating natural landforms are more than offset by improved visual appeal, decreased slope maintenance costs, and improved long-term stability. In addition, constructing mine landforms that visually blend with the surrounding landscape has considerable public relations value for operators.

Landscape design is dependent on numerous factors, including climate, geology, soils, local hydrogeologic patterns, topography and final land use (MEND, 2007). A major challenge concerning landform design is the objective of long-term sustainability; design timeframes may be in the order of hundreds of years. The changes that will occur during this period are difficult to predict and quantify, yet will affect the system.

Processes that affect the evolution of a system can be grouped into physical, chemical, and biological categories, and each will affect the landform differently over time (see Figure 3). The appropriate approach is to account for site-specific processes that are anticipated to act on the closure landform, rather than have these processes result in an adverse effect on long-term performance. In general, the simpler the closure design, the less one has to be concerned about the potential effect of processes such as wet-dry cycling, frost action, and vegetation developments on long-term performance.

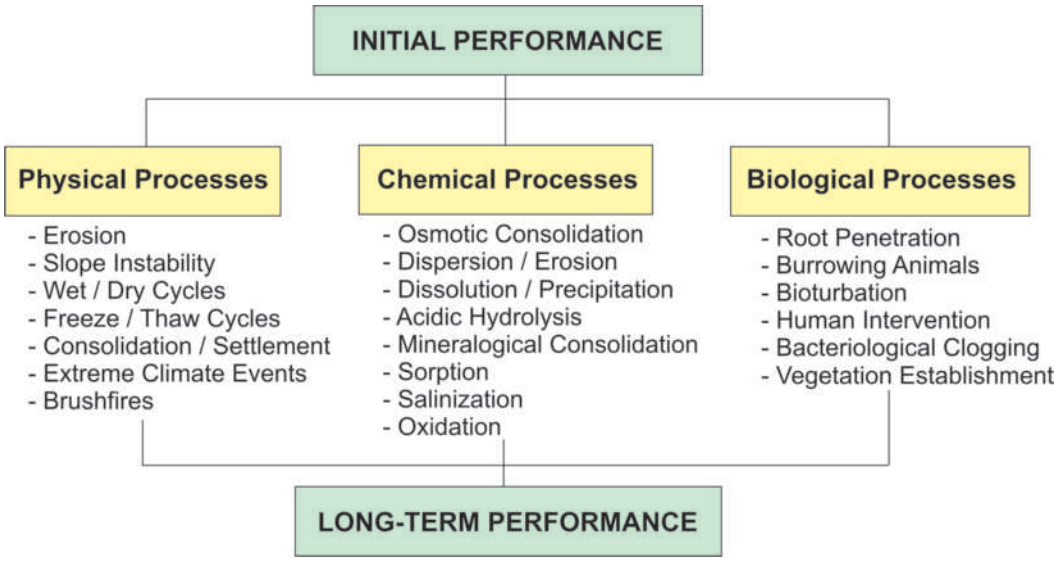

Figure 3: Conceptual illustrations of processes affecting long-term performance of closure landforms/cover systems (from INAP, 2003)

The authors strongly recommend that a failure modes and effects analysis (FMEA) be completed on the proposed final cover system and landform for closure of a heap leach pile. A FMEA is a top-down/ expert-system approach to risk identification and quantification, and mitigation-measure identification and prioritization. Its value and effectiveness depends on having experts with the appropriate knowledge and experience participate in the evaluation during which failure modes are identified, risks estimated, and appropriate mitigation measures proposed. The goal is to provide a useful analysis technique that can be used to assess the potential for, or likelihood of, failure of the proposed design and effects of such failures on human health and the surrounding ecosystem. Robertson and Shaw (2006) describe the FMEA approach in greater detail.

Key aspects of cover system construction

Several key issues associated with cover system construction are fundamental to long-term cover performance, many of which are atypical when compared to other types of construction (MEND, 2004). It is fundamental that a cover system be constructed with good quality control to ensure that the completed cover system is representative of the actual design. The effect of improper construction of a cover system, or poor QA/QC during construction, can have a dominant influence on cover system longevity. Based on the authors' experience, the following key issues are often overlooked or not given appropriate consideration during cover system construction:

- *Experienced, informed personnel for construction supervision:* in addition to having experience in earthworks construction, personnel or firms retained to supervise cover system construction should be informed and knowledgeable of the intended functions of all the design elements.
- *Continuous characterization of cover materials from borrow sources:* an appropriate geotechnical testing program including, at a minimum, particle size distribution analyses (Atterberg limits and Proctor compaction tests for cohesive materials) should commence as soon as a cover material borrow source is identified, and continue throughout cover system construction. This is critically important for a compacted cover layer comprised of local silt or clays, but is equally important for the growth medium layer to ensure an adequate fines content for moisture retention and plant growth.
- *Field compaction trials prior to full-scale construction:* if the cover system design includes a layer of compacted weathered heap leach material or local silty-clay soil, then field compaction trials should be carried out to ensure the two important elements − density and moisture content − are obtainable throughout construction. The trials are also useful to optimize compaction and test equipment selected for construction and identify unforeseen construction issues.
- *Construction of thicker moisture store-and-release layers using ROM waste materials:* these materials tend to be well-graded to gap-graded, which can lead to segregation during placement and the formation of coarser textured zones (i.e. preferential flow-paths for higher infiltration of atmospheric water and oxygen); INAP (2003) describes the adverse affect of ROM material segregation within a moisture store-and-release cover layer at BHP Billiton Iron Ore's (BHPBIO's) Mt. Whaleback operation; thicker profiles of material prone to segregation should be constructed in multiple lifts.

Performance monitoring of cover systems

Historically, one of the most common technologies used for evaluation of closure cover system performance has been water quality analyses of seepage discharged from a heap leach facility. Water samples are typically collected from collection ditches around the perimeter of the facility or from monitoring wells installed below or downgradient of the site. While this approach empirically describes a facility through monitoring of its cumulative effect at the base, it may take several decades before seepage water quality improves. This is due to the storage capacity in the heap leach material and the length of time it takes for the phreatic surface to equilibrate with local hydrogeologic conditions. Direct measurement of field performance is the preferred methodology for measuring performance of a cover system for reclaimed heap leach piles (MEND, 2004; 2007; 2012). This is the best method for demonstrating to all stakeholders that the cover system will perform as designed.

A "watershed" approach to monitoring is preferred in order to gain a better understanding of cover system performance under site-specific conditions (O'Kane, 2011). The rationale for utilizing a watershed approach is such that it allows for the complexity and challenges of cover system performance monitoring, which are apparent given the scale increase of a cover system from a point-scale (e.g. a test plot) to a macro-scale (e.g. a watershed). Although most monitoring techniques used in point-scale cover system monitoring can be applied for macro-scale cover system monitoring, the extent of performance monitoring for a macro-scale cover system is much broader than that for a point-scale cover system. The performance monitoring and evaluation of a macro-scale cover system considers the temporal and spatial variability of the field measured datasets. The monitoring frequency (scale) for obtaining sufficient data, which is associated with spatial instrumentation and temporal data acquisition, must be understood in order to deploy a cost-effective monitoring system.

Several factors need to be considered when designing a cover system performance monitoring program. Cover system performance will be different in upslope versus downslope areas due to differences in runoff and infiltration across a sloping surface. Heterogeneity in the particle size distribution of cover system material will also result in slight differences in cover system performance. Cover system performance monitoring systems should be automated as much as possible to avoid missing collection of field response data during key times of the year (e.g. during and following storm events). In addition, the use of automated systems (see Figure 4) for data collection greatly reduces the need for human intervention and in particular, demands placed on mine site personnel.

Figure 4: Automated meteorological and soil monitoring stations installed on a closure cover system for a spent heap leach pile

Conclusions

The recommended design philosophy for a heap leach facility cover system is one that integrates the heap leach material within its environmental context. This is in contrast to isolating the material from the environment to completely prevent the production of contaminated seepage. A cover system must be designed as an unsaturated system exposed to the atmosphere, the performance of which will be significantly influenced by seasonal, annual, and long-term site climate conditions.

The behavior of a cover system will change with time as a result of various physical, chemical, and biological processes. The key is to account for site-specific processes that are anticipated to act on the closure landform, rather than have these processes result in an adverse effect on long-term performance.

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Recent developments in mine storage pond LCRS design, construction, operation and monitoring

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Abstract

Leakage collection and recovery systems for Nevada process water storage ponds typically incorporate a double-synthetic liner with a drainage media sandwiched between the liners to form a means of draining and dissipating any leakage through the upper or "primary" liner. This eliminates the potential for transfer of hydrostatic head from the primary to the lower or "secondary" liner, thereby minimizing the potential for leakage through the secondary liner to the environment.

The drainage media typically convey flows to a gravel/rock-filled sump (underlain by the secondary liner), from which collected fluids are evacuated by pumping from a port installed into the sump. The port typically includes a screened section lying horizontally within the sump, and solid pipe outside the sump on the pond slope. The pipe is installed between the liners and accessed from the crest of the pond thus allowing pump installation and operation. Pump operations require a minimum operating hydrostatic head, below which the pump will trip.

In order to measure the rates of leakage through the primary liner, the operator must be able to measure the rate of filling of the sump, either by evacuation pumping at a similar rate to inflows being experienced, or by emptying the sump, measuring the time taken for the sump to fill to an elevation that is lower than the top of the sump, and using predetermined sump stage-storage characteristics to calculate the volume of water required for partial filling. Both of the above methods require pre-installation and monitoring of a pressure transducer(s) to measure hydrostatic head acting on the base of the sump (secondary liner elevation). Measurement of hydrostatic head that exceeds the elevation of the sump crest (primary liner elevation) means that a transfer of hydrostatic head is occurring from the primary to the secondary liner.

Both operators and regulators support the implementation of the above design because it eliminates the necessity for liner penetration using pipe "boots", which "always leak". However, maintaining a pump-operating head on the base of the single-lined LCRS (leachate collection and recovery system) sump may well result in leakage through the sump secondary liner, with similar unintended results.

Therefore, the authors have considered options for LCRS design, construction, operation and monitoring that include both measures employed to provide better containment of LCRS sump leakage that may result due to a requirement to maintain hydrostatic head on the secondary liner, and benefits of advances in liner fabrication and installation using "boot" systems, thereby allowing gravitation of flow from the LCRS sump that effectively maintains zero head on the secondary liner.

Introduction

For process water storage pond internal base and sideslopes, the practice of installing two liners with a dissipation drainage layer sandwiched in-between has been commonly used in USA, Mexican and Canadian mine water management solutions for the last two decades, and is also widely employed in international mine water management solutions. This leakage collection and recovery system, or LCRS, provides an efficient means of draining and dissipating any leakage through the upper or "primary" liner. This dissipation eliminates the potential for transfer of hydrostatic head from the primary to the lower or "secondary" liner, thereby minimizing the potential for leakage through the secondary liner to the environment, also minimizing loss of potential revenue to the operator, as is the case with a gold leach "pregnant solution pond".

Water pollution control regulations governing design, operation and closure of mining facilities under the Nevada Administrative Code (NAC) 445A.435 *Minimum design criteria: Ponds*, state:

> *1. All ponds which are intended to contain process fluids must have a primary synthetic liner and a secondary liner. Between the liners there must be a material which has the ability to rapidly transport any fluids entering it to a collection point which:*
>
> *(a) Is accessible; and*
>
> *(b) Has a system for recovering those fluids.*
>
> *2. When the material between the liners is unable to collect, transport and remove all liquids at a rate that will prevent hydraulic head transference from the primary liner to the secondary liner, the pond must be shut down.*

These regulations have caused industry to evolve to the current typical LCRS solution (see Figure 1), that includes installation of:

- A primary synthetic liner, typically 80 mil high density polyethylene (HDPE).

- A synthetic permeable layer to provide a preferential flow path towards a sump for collection and dissipation of water leaking through the primary liner (typically HDPE "geonet", or incorporated into secondary fabrication).
- A gravel filled sump located in the low point of the pond, which is enclosed by the primary liner on the top and the secondary liner at the base.
- A sump evacuation port consisting of a solid HDPE pipe that is installed between the primary and secondary liners on the embankment inside slope and is linked to a horizontal slotted HDPE pipe on the sump base.
- A submersible pump to evacuate fluids collected in the sump and maintain the water level in the sump to beneath the elevation of the primary liner at the sump.
- A secondary synthetic liner to provide resistance to downward migration of dynamic fluid within the double lined system into the environment. This is placed on the prepared pond subgrade which is compatible with liner installation standards.

This paper describes the authors' experience related to the above-summarized installation; in regard to both operational monitoring of LCRS systems installed in Nevada mine process water ponds, and improved containment of potential LCRS sump leakage exacerbated by hydrostatic head on the secondary liner necessitated by evacuation-pumping operations.

In addition, the authors have designed alternatives for LCRS construction, operation and monitoring that take advantage of advances in liner fabrication and installation using "boot" systems, which potentially eliminate the requirement for internal sump operation and allow operation that effectively maintains zero head on the secondary liner during operations.

Experience related to current LCRS installation and monitoring

This section defines the typical range of current methods of monitoring LCRS flows for the typical pond and sump installation described in the introduction, monitoring results and discussion thereof, potential risks associated with the methods, and potential methods of both improving monitoring accuracy, and sump containment potential, with the aim of decreasing risk of leakage from the LCRS sump.

Monitoring methodologies

Water Pollution Control (WPC) permits issued by the State of Nevada typically require that the LCRS flows not exceed 150 gallons per day (gpd) average over a calendar quarter, and 50 gpd averaged over a year. Additionally, NAC445A.435.2 requires that "*When the material between the liners is unable to collect, transport and remove all liquids at a rate that will prevent hydraulic head transference from the primary liner to the secondary liner, the pond must be shut down.*"

Monitoring methodologies for the LCRS system are broadly defined by a range incorporating the following minimum (Method 1) and optimum (Method 2) requirements:

- Method 1: Weekly sump evacuation until pump trips, indicating an almost-empty sump. Evacuated water (i.e., sump *outflow*) is pumped into a measuring container and the measured volume divided by time (7 days) to obtain a daily average flow rate.
- Method 2: Weekly sump evacuation until pump trips, and measurement via a pressure transducer of the hydrostatic head variation in the sump to determine the head increase (from almost-empty) and change in sump stored-water volume for a known time duration, to determine sump *inflow*. These are also collected weekly and averaged over the preceding quarter, or year, and the results compared to the permit limitations. Utilization of this method allows frequent or real-time measurement via a pressure transducer to confirm that the water level in the sump is less than the elevation of the primary liner at the sump, thus demonstrating that there is no hydrostatic head transfer from the primary to the secondary liner. Monitoring protocols for Method 2 include:
    (a) Initial measurement of the pressure transducer. If no daily or weekly increase in hydrostatic head is observed *and* the hydrostatic head is less than the elevation of the primary liner at the sump, there is no water getting into the sump and the sump does not need to be evacuated.
    (b) If there is an increase in sump hydrostatic head, perform the following sump evacuation protocols to determine the real-time inflow into the sump.
        1. Measure the hydrostatic head in the sump using the pressure transducer. (Note: Set the pressure transducer to record a depth reading at 15-minute intervals.)
        2. Evacuate the sump using a *low flow pump* until the sump runs "dry" (i.e., minimum pump operating depth).
        3. Measure the hydrostatic head in the sump immediately when sump runs "dry" and turn off the pump.
        4. Allow water levels in the sump to rise until the sump water levels reach a sufficiently high level to enable the inflow volume to be calculated (limited to the sump primary liner elevation to maintain compliance with NAC445A.435.2).
        5. Calculate the volume of water (V) that has flowed into the sump during the recovery period (t) using sump height-volume relationships that include the effects of sump drainage media porosity.
        6. Calculate the average inflow into the sump in gallons per minute and gallons per day (V/t). If the calculated average inflow rate exceeds LCRS limitations imposed

by NDEP, i.e., 150 gpd averaged over a quarter, and 50 gpd averaged over a year, a pump capable of maintaining an equivalent or higher flow rate (than the calculated average inflow rate to the sump), will be immediately installed in order to eliminate the potential for transfer of head from the primary to the secondary liner.

Monitoring data and discussion

Sump inflows of less than 150 gpd translate to an average inflow of approximately 0.1 gallon per minute (gpm), and for context, about 100 times less than a 10 gpm capacity pump. In addition, typical sump construction includes drainage media that by virtue of their permeability, can limit the potential for such a relatively high flow rate pump to effectively dissipate the full volume of water stored in the sump, particularly if the dimensions and capacity of the sump do not allow optimum recovery (i.e., are slower draining than the pump capacity).

Using Method 1, this can result in outflow measurements that underestimate the weekly inflows, and calculation of lower than actual values for LCRS flow recovery for comparison with permit limitations. The higher the pump flow capacity and the lower the drainage media permeability, the greater the risk of underestimation. In addition, if operated without a pressure transducer, this method provides no means of demonstrating that there is *no* hydrostatic head transfer from the primary to the secondary liner.

Method 2 protocols provide a more accurate means of calculating representative inflows to the sump provided that the system is not subject to the potential sump permeability and pumping limitations described for Method 1.

In order for either method to provide reasonably accurate data, the following design and construction elements must be addressed.

- The sump must be equipped with a pressure transducer to enable hydrostatic head in the sump to be continuously measured.
- The sump drainage media must have very high permeability to enable use of a high flow capacity pump.
- If the existing sump drainage media have low permeability, the sump pump must have low flow capacity, preferably of the same order of magnitude of the permit limitations, or less than 1 gpm.
- Porosity values of the sump drainage media are required to enable development of a reasonably accurate sump stage-capacity curve for use in determination of inflow volumes under Method 2. Alternatively, the stage-capacity curve can be developed by performing phased filling of the sump with a known volume of water, and correlating water depths in

the sump to the increased phase volume using the pressure transducer. This would be performed prior to completing installation of the sump primary liner, and could also serve as a tightness test by continuing to measure the hydrostatic head for a few days to see if any significant variations occur. If they do, it will be possible to use the transducer to pinpoint the elevation that a leak occurs.

- Construction of a level, uniform concrete sump foundation that facilitates sump liner measurement and factory fabrication to minimize potential for field weld failure.

Design alternatives for LCRS construction, operation and monitoring

The alternative design approach adopted addresses both industry and regulatory misconception that pipe-penetrations through geomembrane liners cannot be successfully engineered (i.e., the notion that "pipe-boots always leak"). More often than not, pipe boots leak because there is insufficient attention paid to both design and construction quality, particularly as far as field-fitting and field-welding are concerned. Technical advances in the ability to factory-fabricate custom HDPE design elements are not considered because the typical design and minimum monitoring methodologies discussed above are readily accepted by both industry and regulators as being adequate to ensure no leakage is occurring.

As previously discussed, a reasonable potential exists for miscalculation of leakage rates, non-detection of hydrostatic head transfer from the primary to the secondary liner, and leakage through the sump secondary liner (under operating head conditions). Potential miscalculation of leakage rates and non-detection of hydrostatic head transfer from the primary to the secondary liner can be addressed by performing the monitoring protocols provided in the preceding "Monitoring methodologies" section and considering design and construction recommendations provided in the previous "Monitoring data and discussion" subsection.

Options for addressing potential leakage through the sump secondary liner under operating head conditions are provided below, including:

- construction of a low permeability barrier beneath the sump secondary liner that allows for containment of potential leakage through the sump secondary liner for the anticipated operating life of the LCRS; and
- using a pipe-penetration through the secondary liner at the topographical low point of the pond to enable either gravity or pumped dissipation to achieve near-zero hydrostatic head transfer from the primary to the secondary liner, and clear demonstration of LCRS efficiency.

Construction of low permeability barrier

As previously discussed under "Monitoring data and discussion", the potential for leakage from an LCRS sump secondary liner exists because of the hydrostatic head in the sump necessary to maintain pump supply. One method commonly employed by the authors to mitigate this potential is shown in Figure 1 and requires construction of a sump foundation that has adequately low hydraulic conductivity to prevent migration of moisture outside of lined containment, should the secondary liner develop a leak. The design approach assumes that whatever caused the leak to occur will cause a similar leak in any synthetic liner option for mitigation, and that an earth liner of several feet thick will be more effective in countering external forces such as differential settlement.

The thickness of the foundation is dependent on both the time that the system is anticipated to require LCRS operation and the compacted hydraulic conductivity of the foundation soil. For example, if the time frame for pond operation is 20 years, and the compacted hydraulic conductivity of the foundation soil is 1×10^{-7} cm/s (or approximately one foot in 10 years), a two-foot thick layer of foundation soils would provide adequate mitigation against leakage developing on day one of operations.

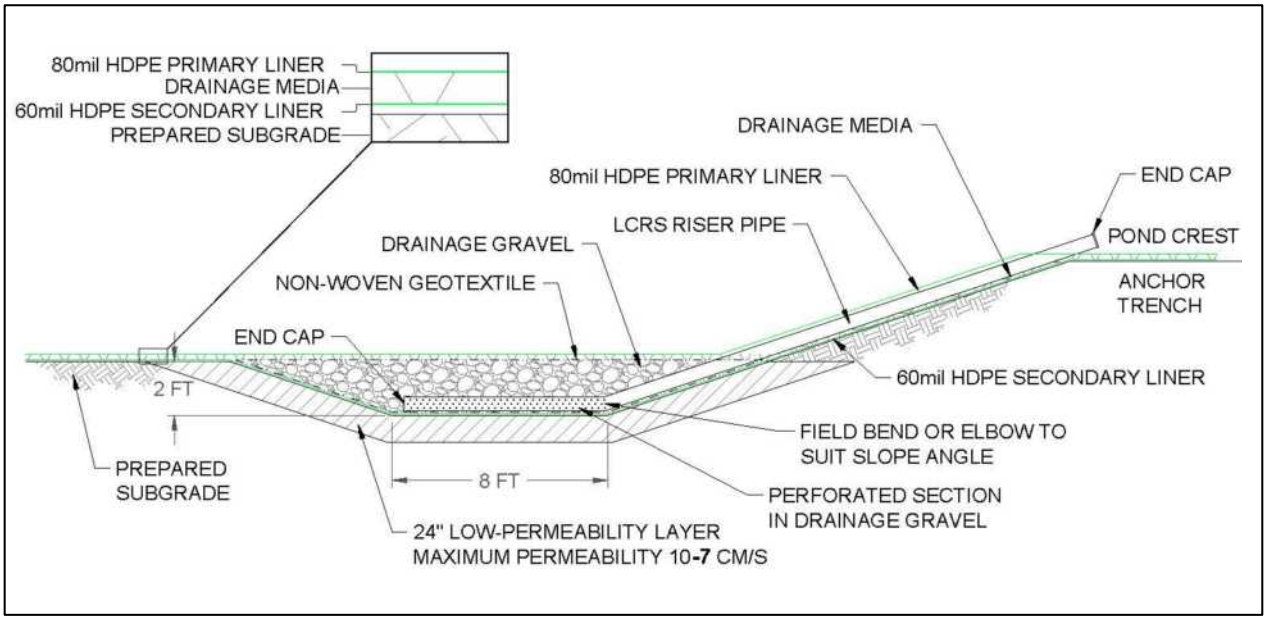

Figure 1: Typical LCRS details

Pipe-penetration through secondary liner

The benefits of a successful pipe penetration through the secondary liner include the ability to gravitate leaking fluids away from the low topographic elevation of the secondary liner as they occur, and in so doing, fully comply with the objectives of the LCRS and eliminate the risk of leakage from the secondary liner. Figure 2 shows a pipe-penetration gravity system flowing into an external lined sump (or external pond), for pumped conveyance back into the operating pond (or evaporation from the external pond).

The elements of this design include a pre-fabricated LCRS drain outlet installed in a prepared foundation, a pipe-in-pipe connection to an external sump/pond and methods of operation that will eliminate the potential of transference of hydrostatic head from the primary to the secondary liner (as dictated by Nevada regulations).

As shown in Figure 2, the prepared foundation grading is more steeply graded in the immediate vicinity of the LCRS drain outlet to facilitate gravity drainage within the LCRS toward the low-point inlet of the pipe-in-pipe system. Regardless of material type forming the foundation, it is necessary to have low tolerance on the dimensional finish to the LCRS foundation to ensure a neat and wrinkle-free surface on the secondary liner. The foundation shown consists of a prepared soil subgrade underlying a cast-in-situ mass concrete foundation block housing the pre-fabricated LCRS outlet, with smooth finish specified for the concrete surface, to allow wrinkle-free installation of the HDPE apron factory-welded to the LCRS outlet.

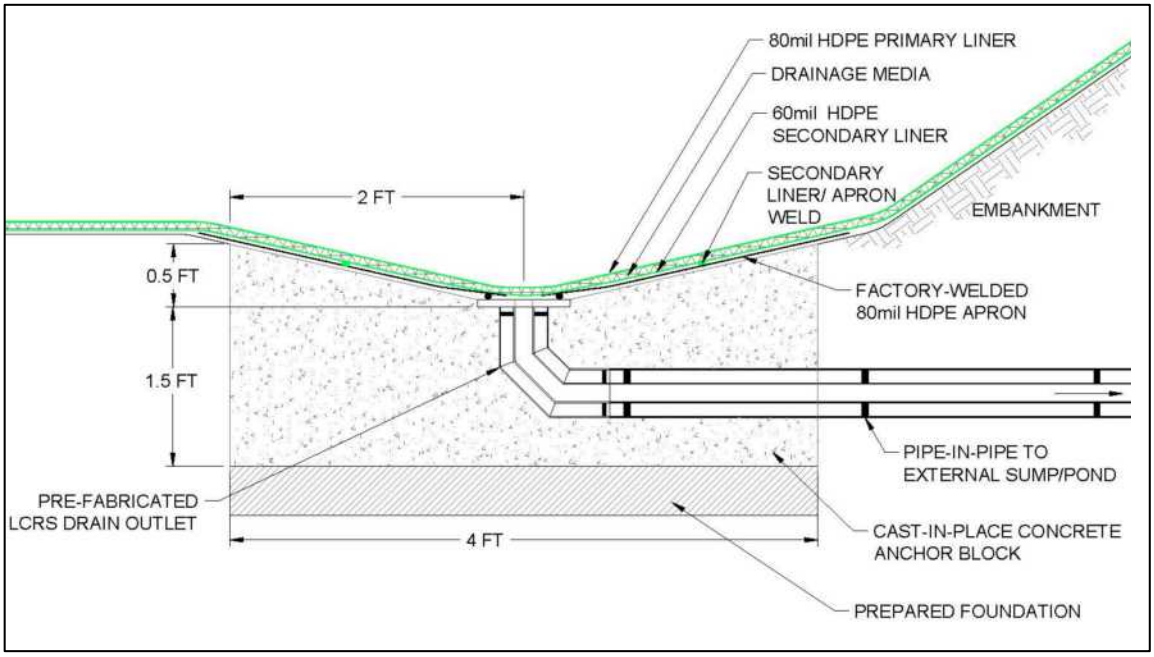

Figure 2: Pipe penetration through secondary liner

Figure 3 is a photograph of a pre-fabricated HDPE outlet incorporating a pipe-in-pipe configuration for secondary containment, which shows a trimmed, factory-installed HDPE apron. The apron both minimizes and facilitates field welding to the pond secondary liner. The inner and outer pipes are buried beneath the pond embankment and extend downstream into pumping/pond facilities.

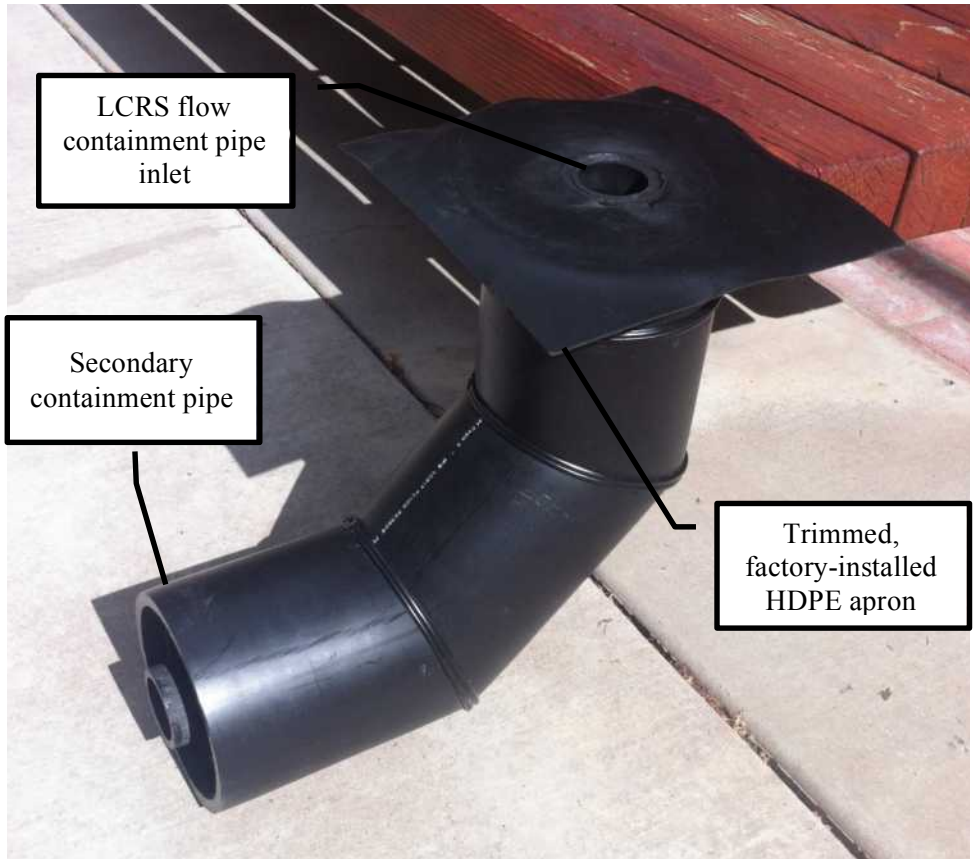

Figure 3: Photograph of pre-fabricated HDPE outlet pipe

Whether pumping from a downstream sump or gravitating into a downstream pond, as long as the water level in the downstream sump or pond is maintained lower than the secondary liner elevation at the HDPE outfall, the potential for, and leakage through, the secondary pond liner will be negligible, if not entirely eliminated.

Conclusions

Recommended operations and monitoring protocols will result in the following benefits:

- no transfer of hydrostatic head from the primary to the secondary liner;
- low to zero hydrostatic head development above the sump secondary liner;
- low to zero potential for leakage from the single-lined sump system; and
- remediation of potential for leakage from the single-lined sump system by providing 20 years of resistance via a low permeability foundation. This can also apply to any alternative LCRS systems.

In addition, secondary liner penetration as recommended will ensure negligible potential for transfer of hydrostatic head from the pond primary to the pond secondary liner, and negligible leakage from the sump secondary liner.

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Nevada Administrative Code 445A.350 through 447: Regulations governing design, construction, operation and closure of mining facilities.

Trends of large scale direct shear strength results for LLDPE-HDPE geomembrane/soil interfaces

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Abstract

The mining industry is developing at a fast pace in the South America region, and demands the construction of important and larger mining structures such as: tailings impoundments, heap leach pads, rock waste facilities, etc. Heap leach pads by design, must contain leached mineral solutions through properly designed engineered liner systems. As part of the geotechnical design of a heap leach pad, a geotechnical characterization of the geomembrane-soil interface must be conducted. This characterization is performed by large scale direct shear (LSDS) testing, using confinement pressures which simulate the weight of the ore stacking in the leach pad. The types of geomembrane mainly used in the mining industry are the high and linear low density polyethylene (HDPE and LLDPE, respectively), single side textured (SST), and of variable thickness (1.5 and 2.0 mm).

Many authors have studied the responses of geomembrane-soil interfaces (Koener et al., 2005; Koerner et al. 2007; Thiel, 2009; Parra et al., 2010; Breitenbach, 2011), and analyzed their strength envelopes considering linear and nonlinear geometries (Parra et al., 2012). This paper compiles LSDS test results obtained in Peru, and other data available in the existing literature, and analyzes their trends and range of values. Results from HDPE and LLDPE single side textured geomembranes are presented, with the textured side in contact with the underlying soil, and using confinement pressures of up to 800 kPa, which are equivalent to about 45 m of ore stacking. The results show a consistent trend in the residual shear strength envelopes of different types of geomembranes, regardless of geomembrane thickness. For high pressures, this trend could be non-linear. The values of adhesion range from 26 to 53 kPa, and angles of internal friction range from 16° to 26° for HDPE interfaces, and adhesion between 15 and 35 kPa, and angles of internal friction between 12° and 24° for LLDPE interfaces.

Introduction

The mining industry is developing at a fast pace in the South America region, and demands the construction of important and larger mining structures such as: tailings impoundments, heap leach pads, rock waste facilities, etc. The correct geotechnical design of these structures is extremely important due to their large size and the implications of their potential failure.

Heap leach pads must contain leached mineral solutions through properly designed engineered liner systems. The proper design of these systems requires proper characterization of the interface strength properties between geomembrane and soil. This characterization is performed by LSDS testing, using confinement pressures which simulate the weight of the ore stacking in the leach pad.

In Peru, leach pads are commonly located in areas where the irregular topography requires the design of grading with steep slopes and gradients. In light of this, the liner system becomes a potential plane where a translational or block sliding failure surface could occur. Therefore, it is extremely important to characterize the shear resistance of the liner system in the geotechnical design of a leach pad. The geotechnical design of a heap leach pad includes the completion of many geotechnical analyses and the characterization of materials which include: puncture tests, LSDS testing, slope stability and stress-strain analyses.

Slope stability analyses are performed considering circular, non-circular and translational slip surfaces. The circular and non-circular slip surface analyses evaluate stability associated with homogeneous and non-homogeneous soil conditions, respectively. Translational or compound slip surface analyses evaluate the possibility of landslides through the liner system of the leach pad.

Stress-strain analyses are usually required when the factors of safety from slope stability analyses have a low value (usually less than 1), either under static or seismic conditions. From these results, the designer must consider what type of geomembrane is most appropriate for the project (HDPE or LLDPE).

To ensure that the liner will not be perforated due to the gravel particles of the overliner, a puncture test should be performed, considering the same confinement to which the geomembrane will be subject during the mine operation. In order for this test to be representative, it is preferable to have reliable information about the anticipated or projected ore stacking height, the type and thickness of the geomembrane and the soil liner.

This paper describes the geotechnical design of a heap leach pad with emphasis on the liner system design, which covers laboratory testing and geotechnical analysis.

Liner system of heap leach pad

In Peru, the heap leach pads usually have a conventional liner system, consisting of a soil liner (low permeability soil) covered by a geomembrane which in turn is covered by a gravelly material (overliner), as shown in Figure 1.

The soil liner has to be clayey and its hydraulic conductivity must be less than 5×10^{-6} cm/s. The geomembrane is commonly made of polyethylene of high and linear low density (HDPE and LLDPE, respectively). These structures usually use geomembrane of a textured side and a smooth side (SST), where the textured side faces the clay soil liner and the smooth side faces the gravelly over liner material. The gravelly material generally has to have draining characteristics, and should not pinch the liner under confinement, which can reach pressures up to 4,000 kPa.

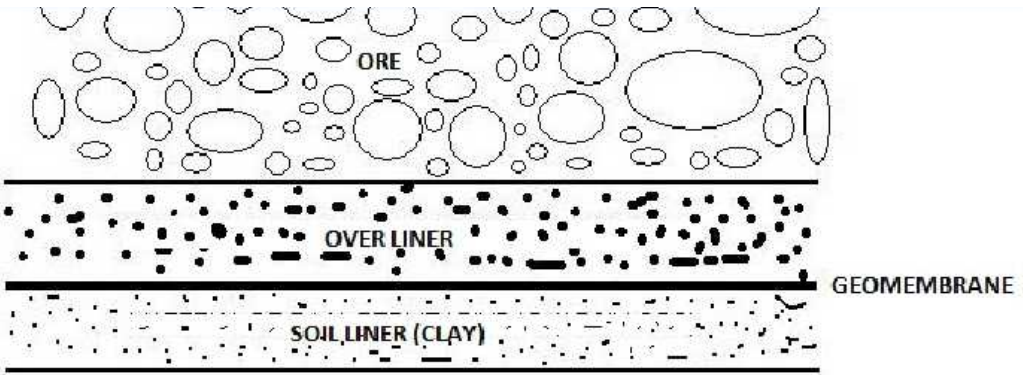

Figure 1: Conventional liner system

In cases where there is no clay material or where this cannot be placed (steep slopes), soil liner can be replaced by a geosynthetic clay liner (GCL), as shown in Figure 2. The GCL consists of a layer of bentonite confined by two layers of geotextile (woven or nonwoven). This geocomposite has a hydraulic conductivity of about 10^{-9} cm/s.

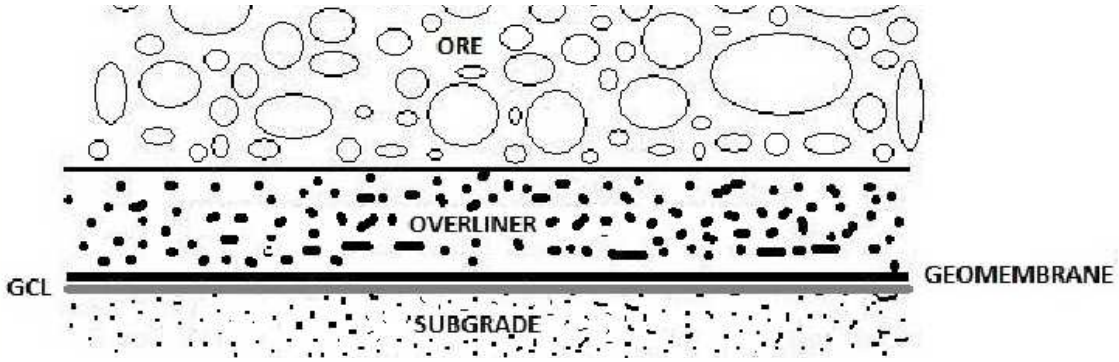

Figure 2: GCL liner system

Geotechnical characterization of liner system

General

This paper will focus on the geotechnical characterization of the conventional liner system. In this type of liner, there are two interfaces: geomembrane/soil-liner and geomembrane/overliner. Many authors have studied the behavior of geosynthetic interfaces (Koener et al., 2005; Koerner et al. 2007; Thiel, 2009; Parra et al., 2010; Breitenbach, 2011) and analyzed their linear and nonlinear geometry (Parra et al ., 2012). However, this paper shows LSDS test results grouped by type and thickness of geomembrane. Finally, these results are compared with each other.

Geomembrane/soil interfaces analyses

In the case of a heap leach pad, the geotechnical characterization of the interfaces is carried out by performing direct shear tests on a large scale box (30 cm × 30 cm), following the methodology of ASTM D5321 standard. These tests can be performed including all the materials that form the lining system: gravelly material, geomembrane and clayey soil. The shear strength values obtained by this test depend on the resistance values of the geomembrane/gravel and geomembrane/clay interfaces (two variables).

Parra et al. (2010) studied the difference of values of the large scale direct shear (LSDS) test results considering the presence and the absence of a gravelly material (overliner) on the geomembrane. In the test, the overliner which must be in contact with the smooth side of the geomembrane was replaced by rigid substrata (concrete). The results of this comparison for the case of an LLDPE geomembrane showed that it is more conservative (lower residual shear stress values are achieved) to consider the presence of a rigid substrata than it is to consider a regular overliner material, as shown in Figure 3.

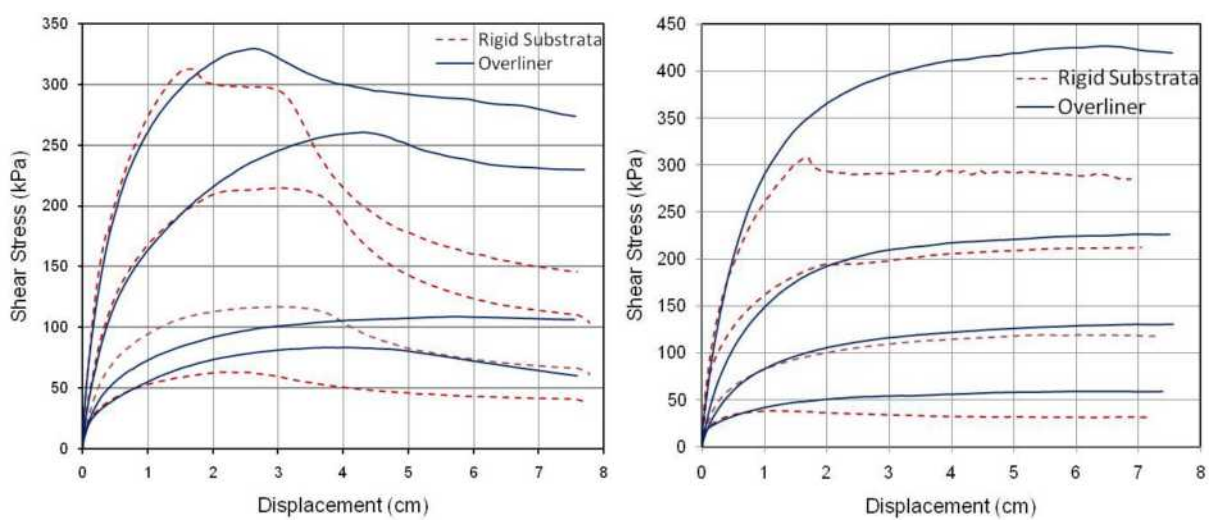

Figure 3: LSDS tests performed with overliner and rigid substrata (Parra et al., 2010)

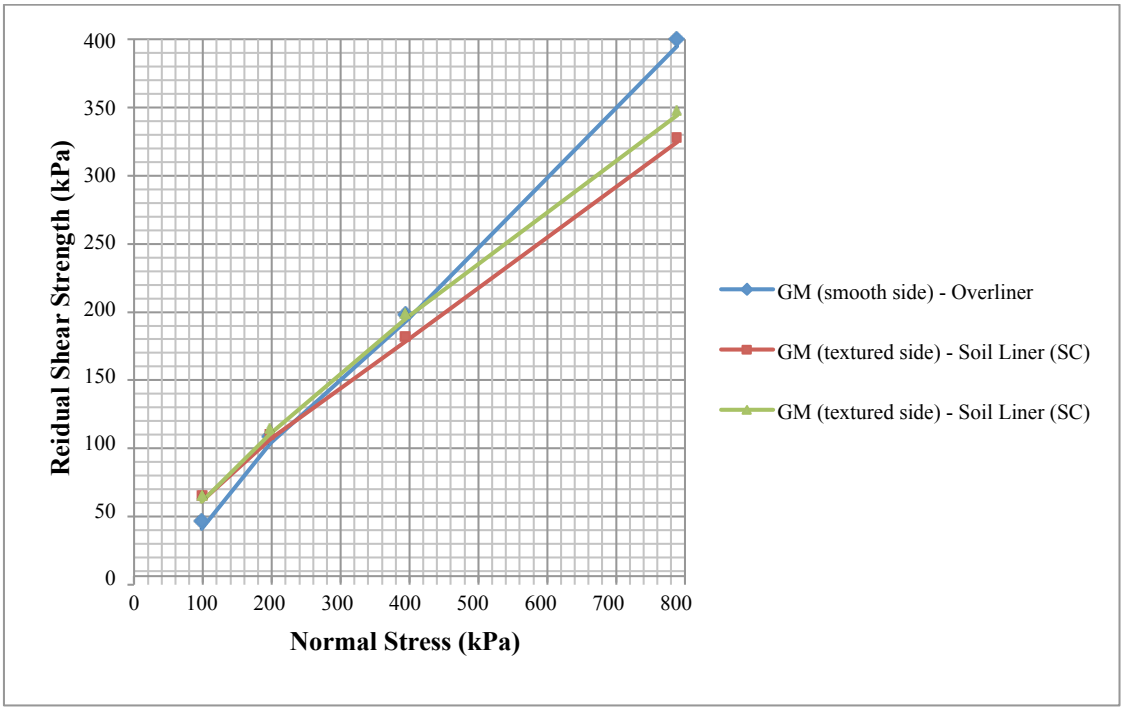

Figure 4: LSDS in Geomembrane – overliner and geomembrane – soil liner interfaces

Furthermore, comparing independently three results of LSDS tests from the interfaces of a liner system, it can be seen that the interface geomembrane/overliner (over the smooth side) has a higher residual shear strength than the interface geomembrane/soil liner (in contact with the textured side) for the same LLDPE 1.5 mm geomembrane (Ale et al., 2010) for normal stresses greater than 400 kPa, as shown in Figure 4.

From this, it appears that the geotechnical characterization and shear strength of the liner system of a heap leach pad depends on geomembrane/clay interface shear stress.

Review of LSDS test results

In the geotechnical design of a heap leach pad, the physical stability of the structure must be analyzed. In common practice, the slope stability analysis considering a translational or block-slip surface (through the liner system) uses the residual values of the LSDS test in geomembrane-soil liner interface.

Taking this into consideration, results from 83 LSDS tests have been analyzed for LLDPE and HDPE geomembranes, of 1.5 mm and 2.0 mm thicknesses. All of these tests have been conducted in interfaces with soil liner (clayey material). The soil liner in all tests has been placed with a density of 95% of maximum dry density from the standard proctor (ASTM D698), with the optimum water content. The samples are saturated and then confined for two hours with the test pressure, before starting the test. A compilation of these test results is plotted in Figures 5, 6, 7 and 8. Table 1 then shows the average trend of strength parameters of the plotted data; in addition, the upper and lower boundaries for shear strength

residual values are shown. In all cases the residual values of shear strength have been measured at 7.5 cm of horizontal displacement. These results are consistent with those reported by Koerner et al. (2005).

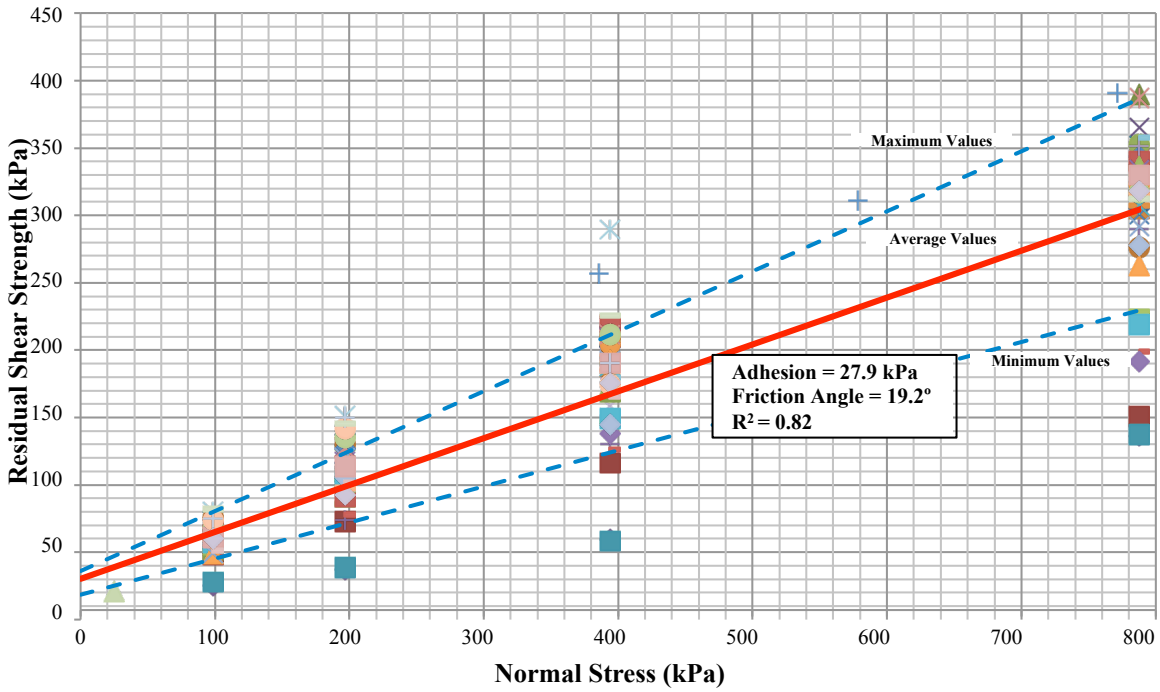

Figure 5: LSDS tests in LLDPE geomembrane, 2 mm (residual values)

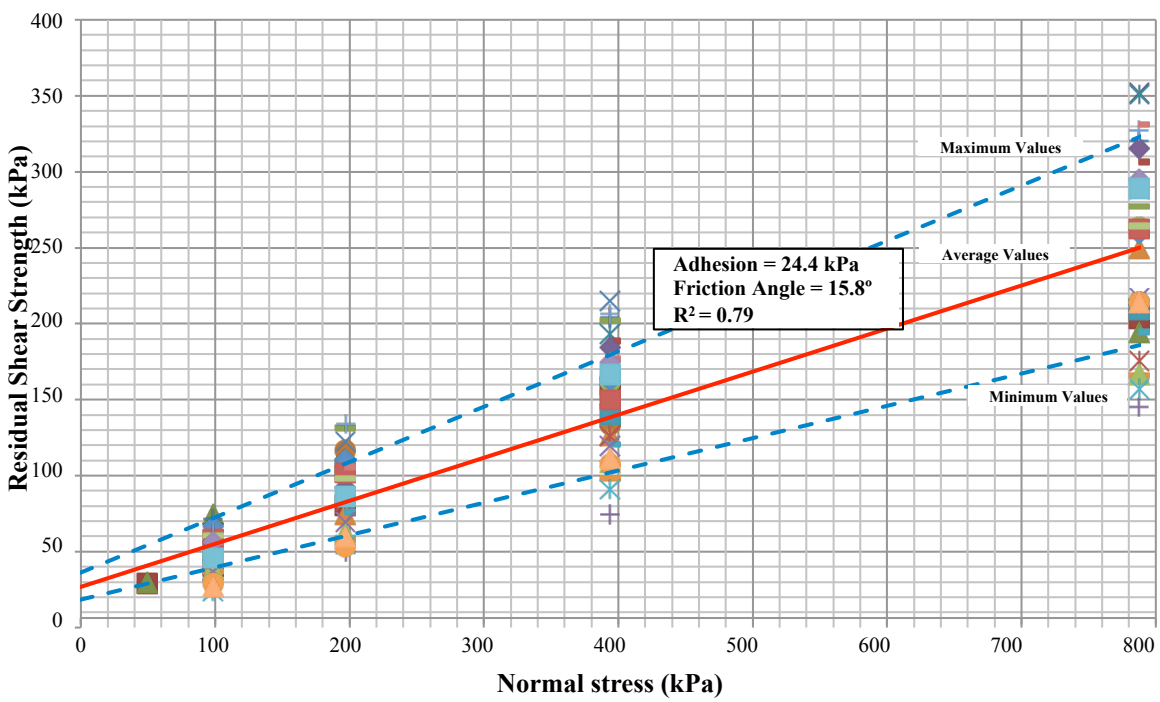

Figure 6: LSDS tests in LLDPE geomembrane, 1.5 mm (residual values)

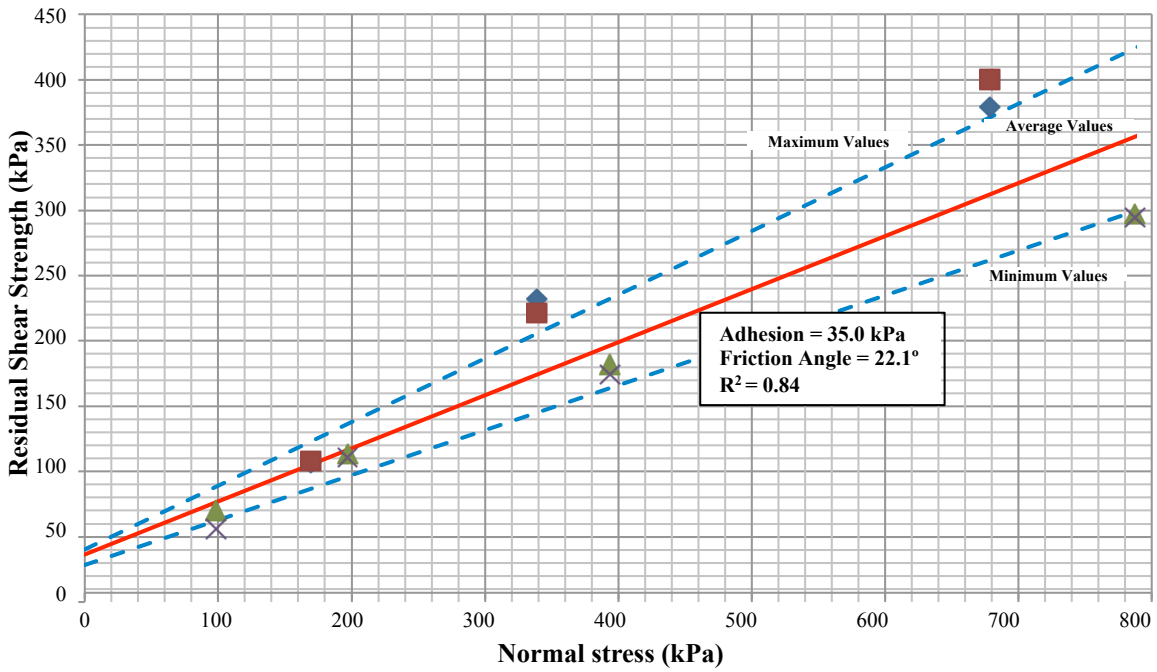

Figure 7: LSDS tests in HDPE geomembrane, 2 mm (residual values)

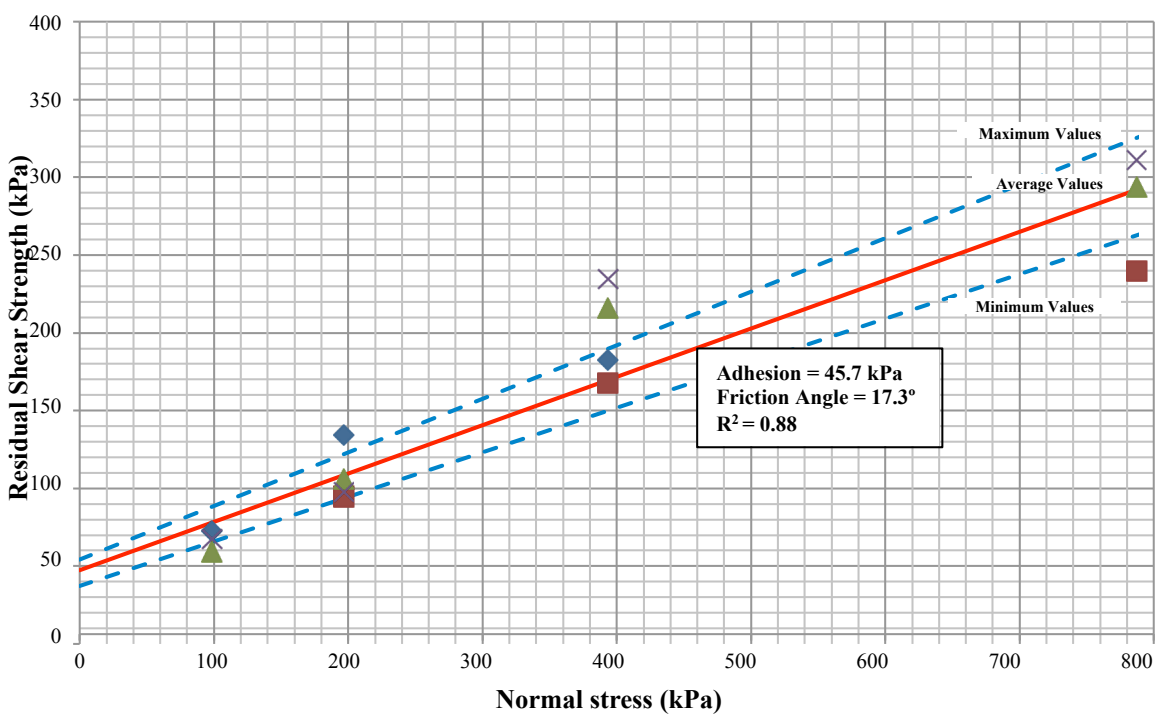

Figure 8: LSDS tests in HDPE geomembrane, 1.5 mm (residual values)

The available data from LSDS tests in HDPE geomembranes are much less than those from LLDPE, because in recent times, the use of LLDPE geomembranes in leach pads has been more widespread due to their flexibility, higher tensile break elongation, and resistance to puncture (Islam et al., 2011). For this

reason, the data shown in Table 1, for LLDPE (75 tests) may be more representative than those of HDPE (8 tests). Figure 9 shows all values listed in Table 1.

Table 1: Trends of shear strength parameters from LSDS tests

Geomembrane	Shear strength parameters					
	Average		Maximum		Minimum	
	Adhesion (kPa)	Friction angle (°)	Adhesion (kPa)	Friction angle (°)	Adhesion (kPa)	Friction angle (°)
LLDPE, 2.0 mm	27.9	19.2	34.0	24.0	15.0	15.0
LLDPE, 1.5 mm	24.4	15.8	35.0	20.0	15.0	12.0
HDPE, 2.0 mm	35.0	22.1	40.0	26.0	26.0	19.0
HDPE, 1.5 mm	45.7	17.3	53.0	19.0	35.0	16.0

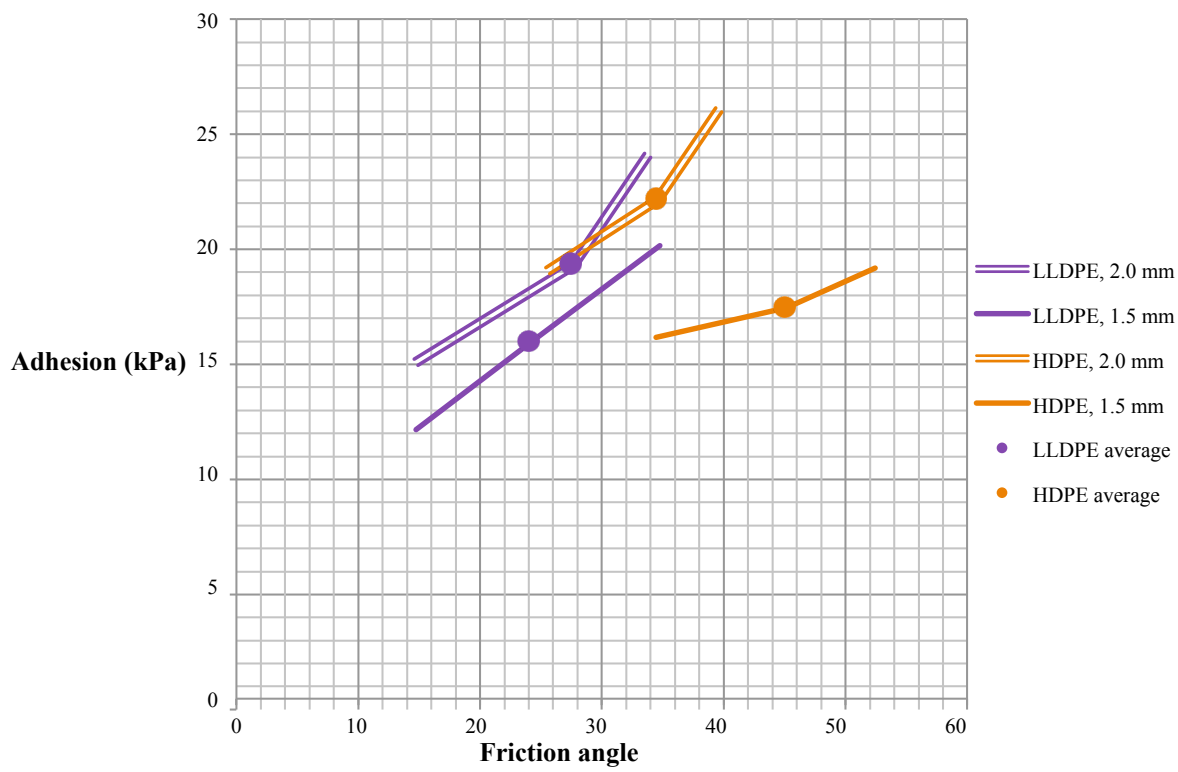

Figure 9: Trends of shear strength parameters from LSDS tests

Influence of shear strength of liner system on slope stability

In order to analyze the influence of shear strength of interfaces on the physical stability of a heap leach pad, a hypothetical case is conducted by limit equilibrium method (Spencer method), considering a heap height of 100 m, and the slope of liner system varying between 0% and 4%, as shown in Figure 10. The

analyses were performed inputting average, maximum and minimum shear strength parameters for a LLDPE geomembrane of 2 mm thickness. Table 2 summarizes the results of all of these analyses.

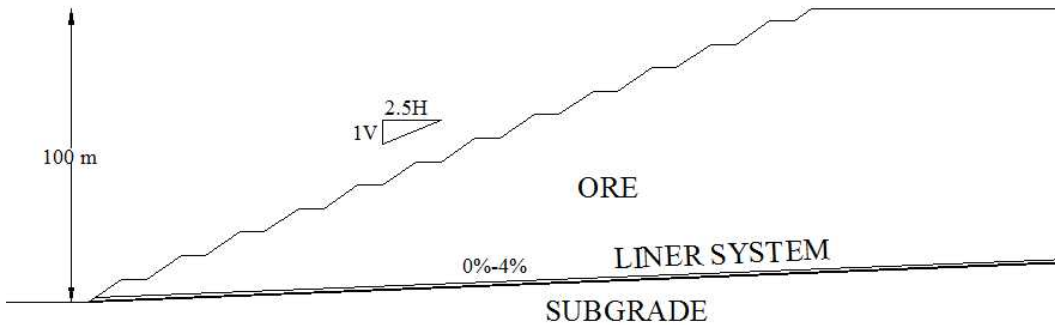

Figure 10: Hypothetical heap leach analyzed

Table 2: Factors of safety for slope stability using 2 mm LLDPE geomembrane

Slope	Shear strength parameters		
	Average	Maximum	Minimum
	Factor of safety	Factor of safety	Factor of safety
0%	1.73	1.93	1.52
1%	1.71	1.89	1.51
2%	1.70	1.88	1.49
3%	1.69	1.87	1.46
4%	1.67	1.86	1.45

Conclusions

Results from a database of LSDS test performed on HDPE and LDPE geomembrane materials of different thicknesses, as well as other data available in the existing literature, show that the shear strength of the smooth side geomembrane/overliner interface is larger than that of the textured side geomembrane/soil-liner interface for confinement pressures over 400 kPa.

The database was compiled from tests where the soil liner had been placed at 95% of maximum dry density (standard proctor) and optimum water content. The samples were also saturated and then confined for two hours at the test pressure, before starting the test. In all cases the residual values of shear strength were measured at 7.5 cm of horizontal displacement.

Results from the database also support the conclusion that the physical stability in the block-slip surface or translational failure mode of most heap leach pads depend on the shear strength of the interface geomembrane/soil-liner, rather than the geomembrane/overliner interface.

The linear shear strength envelopes from the different geomembranes and thicknesses tests showed R^2 values of 0.82, 0.79, 0.84 and 0.88 for geomembranes of 2 mm LLDPE, 1.5 mm-LLDPE, 2 mm-HDPE, and 1.5 mm-HDPE, respectively.

The residual shear strength of 2 mm LLDPE geomembranes ranged from 15 to 34 kPa in adhesion and from 15º to 24º for friction angle. The average shear strength parameters were adhesion of 27.9 kPa and friction angle of 19.2º. Whereas for 1.5 mm LLDPE geomembranes, the residual shear strength ranged from 15 to 35 kPa for adhesion and from 12º to 20º for friction angle, with average adhesion of 24.4 kPa and friction angle of 15.8º.

The residual shear strength of 2 mm HDPE geomembranes ranged from 26 to 40 kPa for adhesion and from 19º to 26º for friction angle. The average shear strength parameters were adhesion of 35 kPa and friction angle of 22.1º. Whereas for 1.5 mm HDPE geomembranes, the residual shear strength ranged from 35 to 53 kPa for adhesion and from 16º to 19º for friction angle, with average adhesion of 45.7 kPa and friction angle of 17.3º.

In a hypothetical case example of a heap leach pad 100 m high, and using the results from the database presented in this paper for 2 mm LLDPE, the factors of safety range from a maximum of 1.93 to a minimum of 1.52 for 0% liner system slope and under static conditions.

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Conveying and stacking systems design for heap leach applications

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Abstract

Heap leach construction is an important component in the operation of a successful heap leach. Many methods are used, including direct truck stacking, on/off stacking in thin lifts, and conveyor stacking. Conveyor stacking is very common and uses a string of portable conveyors that feed a stacking conveyor where ore is then stacked onto the heap. While conveyor stacking is very practical and has many advantages, most of the industry has accepted using conveying and stacking systems that have not been engineered or designed with heap leaching in mind. These conveying and stacking systems tend to be overly heavy, which has led to many difficulties in proper heap construction and operation, especially for multi-lift heaps. Items that must be considered when designing these systems include the terrain, ore/material type, tonnage throughput, moisture content, cell width, lift height, as well as others. Engineering companies usually do not take these issues into consideration when conveyor systems are being specified, and suppliers tend to supply traditional off-the-shelf designs which are not well suited to the heap leach process.

As with any sophisticated processing system, the heap leach stacking system needs to be optimized. A look at several conveyor stacking systems being used in the field reveals many problems and pitfalls with the current approach and a need for a serious shift in how the industry thinks about conveyor stacking system design. Issues from poorly designed conveying and stacking systems range from minor problems, such as lost productivity and increased down time as a result of conveyors getting stuck or damaged while being moved, to severe problems, including potential permeability issues in the heap. While mining companies have dealt with these risks and difficulties by various operating methods, most could be avoided by making a purpose-built conveying and stacking system. This system would include lighter stacking conveyors, wide stance low ground pressure tires on portable conveyors, and the selection of the appropriate stacking method (top stack conveying system or retreat stacking).

Heap leaching: Permeability is key to success

Heap leaching has been used for the recovery of metals from ores for many centuries, but it was not until the advent of high-tonnage, high metal recovery operations in the 1970s, that it became necessary to optimize all aspects of the process. One of the major factors in heap leach performance is permeability, of which there are four aspects:

- permeability of the top working surface;
- uniform permeability of the heap;
- permeability of any intermediate layers between lifts;
- permeability of the drainage layer below the heap.

The first three of these are most directly affected by the method used to stack the heap. If the heap is stacked carefully so as to minimize compaction and maximize uniformity of material placement, then the heap should perform as predicted by laboratory tests.

Truck stacking of heaps

Heaps can be stacked by dumping the ore directly onto the heap from trucks. Truck stacking is used for the following reasons:

- Truck stacking offers operational flexibility: for example, it allows different ores to be sent to different areas of the heap.
- When the stacking rate exceeds 1,500 tonnes/hr (36,000 tonnes/day), conveyors begin to get large and difficult to maneuver.
- When run-of-mine (uncrushed) ore is stacked, conveyor stacking systems are not an option. However, run-of-mine heaps are very difficult to construct in a controlled manner, and recovery is difficult to predict. Run-of-mine heaps usually fall into the category of "dump leaches", not heap leaches. In dump leaching, the object is to recover more value than it costs to do the processing. In heap leaching, the object is to recover a predictable amount which will show a predictable return on the investment.

Unfortunately, truck stacking compacts the top of the heap. Also, there is very little flexibility to add chemicals or moisture to the ore being stacked. Many precious metals heap leaches, and nearly all acid-based heap leaches such as copper heaps, are dealing with crushed ore which is high in clay. Truck stacking these types of ore almost always results in significant loss of values. Also, truck stacking is usually a higher cost method than conveyor stacking. For these reasons it should generally be avoided. Having said that, there are some very successful truck stacking operations such as the Cripple Creek

operation of AngloGold Corporation. These heaps are built in 30 m high lifts, and the heaps are now over 200 m high.

Conveyor stacking of heaps

There are four basic configurations for conveyor stackers:

- Discharge from a tripper on a bridge conveyor which straddles the heap, normally applicable for on-off small tonnage heaps (such as the 1970s Rancher's operation at Ortiz, New Mexico).
- Discharge from a tripper on a long traveling belt which is supported on bulldozer tracks running on the heap (the Rahco stacker system, normally considered to be the best configuration for heap construction, but only applicable to large-tonnage heaps with regular geometry).
- Retreat stacking, using a radial stacker which operates from the base of the lift being stacked (this is the most common design).
- Top advance stacking, which uses a radial stacker operating on top of the fresh ore being stacked.

These four basic methods are discussed in more detail in the following sections.

Bridge conveyor stacking: Thin layer heap leaching

Thin layer on-off heap leaching (with heaps 1 to 4 m high) is sometimes practiced when the ores are not very permeable, or where high grade materials can be effectively pre-processed on their way to multi-lift heaps (where recovery would be slower and less effective). Examples are the above-mentioned Ortiz mine, or the classic Pudahuel, Chile, thin layer copper heap leach. These thin-layer heaps are especially useful for acid heap leaches of copper, zinc and rare earths. Often these ores are high in clay. They leach rapidly (in less than one week) but the acid solutions break down the ore and generate even more clay, so leaching for longer than one week does not work.

The idea of thin-layer heap leaching is a variation along the continuum of leaching (on the fast end of the scale) in vats or on belt filters, to (on the slow end of the scale) heap leaching. (Incidentally, the term "vat leaching" is often misused. It does not refer to agitated tank leaching. It refers or shallow tanks in which coarse ore is bedded, with leach solution percolating through the ore in either upflow or downflow mode.)

For alkaline heap leaches, such ores can be agglomerated and stacked in high lifts. But for acid heap leaches, there are no agglomerating agents that can maintain permeability of stacked materials in high lifts.

Racetrack design

A very productive but more capital intensive alternative to the bridge conveyor layout consists of an oval "racetrack" leach pad constructed of asphalt or other traffic-stable material. The heap occupies the oval, except for a "slot" in which the ore is removed. A front end loader or bucket wheel excavator removes the ore in the slot and advances the slot around the oval. A stacker runs on top of the advancing heap and dumps fresh ore on the leading edge. A photo of the system in operation at Round Mountain, Nevada, is shown in Figure 1.

Figure 1: Round mountain "racetrack" for on-off heap leaching
The tracked stacker shown on the left of the field of view is stacking ore over the edge, advancing into the slot. The front end loader and trucks on the right of the field of view are removing spent ore, which is sent to permanent heaps (seen in the far background) for additional leaching. Ore is brought to the stacker via a conveyor which lies in the center of the oval (it can be seen in the near background). Equipment moves continuously around the oval, and the ore is leached for approximately one month before it is removed

Rahco-type stacker for permanent heaps

A photo of a Rahco stacker stacking a permanent heap is shown in Figure 2. The stacker in Figure 1 is the same type of stacker – these are versatile machines. However, they are large machines that turn gradually, so the leach site must be large, or of regular geometry. Because this type of stacker can climb slopes, it can be used to construct multi-lift heaps.

Figure 2: Rahco stacker at the Chuquicamata, Chile, copper heap leach
The tripper can be seen in the background, dumping ore over the edge. These stackers can climb moderate grades, and make broad turns, while building a flat heap

Retreat stacking

The retreat stacker is the most common type of conveyor stacking system. It is easy to operate when the heap has a single lift. Because of limitations on the stacker weight, this single lift is limited to about 8 m. There are many cases where a single lift heap is better than a multiple lift heap – for example when the ore is very high grade (heaps at Sterling, Nevada and Hassai, Sudan, each averaged over 10 grams gold per tonne), when the ore cannot remain permeable in very high lifts, or where excessive rainfall must be managed. The cost of extra leach pad is usually not prohibitive, so single lift heaps are sometimes the best design choice.

When multiple lifts are utilized, the retreat stacking system has severe drawbacks related to trying to move heavy equipment around on the previously leached surface, which is often water-saturated and clay-rich. Therefore, the retreat stacking system should be used for multiple lifts only when the ore is hard and clay free.

Figure 3: Retreat conveyor stacking system at Yatela, Mali
Elements of the stacker system include: the radial stacker; the follower conveyor; the cross conveyor; and the grasshopper conveyors. The stacker builds the heap by swinging from side to side (the shadow of the heap can be seen in the foreground). As the stacker retreats up-pad, grasshoppers are sequentially removed. The light gray material is a 0.5 m cover of crushed rock situated above the plastic liner

Top advance stacking

Top advance stacking involves a flat radial stacker. The wheels operate on the top surface of the heap. The stacker swings from side to side and dumps ore over the leading edge of the heap, which is at the same level as the wheels. A top stacking system is shown in Figure 4.

Figure 4: Top advance stacking of a 12,000 tonne/day heap
This heap at Ocampo, Mexico, is climbing up a steep ravine. It is constructed as a spiral heap. Ore comes in on grasshopper conveyors at the lowest corner of the heap (these can be seen in the right side of the photo). The stacker is shown placing ore in a 10 m high lift which is being advanced over an existing, previously-leached, lift

The ore that the stacker is operating on is the newly placed ore, and so it is generally dry and can support the load of the stacker. However, since it is freshly stacked, it is in a highly uncompacted state so it is important to make the stacker as light weight as possible. Fortunately, since the stacker works at and discharges at "ground" level, there is no need for a truss lifting mechanism, or for the extra structural strength needed to support the elevated truss. This is a critical design area which the process engineer needs to give attention to: conveyor manufacturers usually build elevating radial stackers, and so a non-elevating stacker tends to inherit the more complex and overweight structure of its cousins.

With top advance stacking, there is no limit to lift height. This is a significant advantage when compared to retreat stackers. Unlike the Rahco stacker, the top advance stacker can be maneuvered in tight and irregular areas.

By proper conditioning of the ore it is possible for almost any ore to be stacked in single lifts of 15 or 20 m, or even higher. This provides another challenge for the process engineer: what is the optimal lift height? Some truck-stacked heaps (for example, Anglogold's Cripple Creek operation) build individual lifts of 30 m.

But there can be a problem with high lifts: Do the ore fines segregate near the top of the heap? How long does it take the leach chemicals to stabilize throughout the ore? How long does it take the solution to saturate the lift and start exiting from the base of the lift? Conveyor-stacked heaps have excellent control over the ore quality: ore can be wetted on the conveyors, almost to the full operating moisture level. This

accomplishes two things: fines stick to coarser particles, and end up uniformly distributed throughout the heap; and chemical solutions can be added on the belts so that the heap is uniformly under leach as soon as solutions are applied to the top surface. The only height restriction may be related to geometry of the heap.

Conveyor stacking equipment is not designed with the process in mind

Surprisingly, although the critical importance of permeability is recognized by almost all production personnel, stacking systems designers do not usually start with this concept when designing production equipment. Why? Stacking systems designers are usually the stacking systems suppliers. They are mostly concerned with how easy is it to fabricate, sell, and use the equipment, rather than how the design of the equipment affects overall recovery. The metallurgists in charge of process design should be actively involved in the equipment design process, but they usually give this a very cursory review. It is common to see operators struggling with stacking equipment which gets bogged down in water-saturated, soft material. Often mobile equipment such as front end loaders or even cranes are brought onto the heap surface, to help the process. All of this leads to the formation of an impermeable or variably permeable drainage surface, with resulting loss of metal recovery. Proper equipment design could eliminate this problem. The process design professionals need to spend more time with the equipment suppliers, making sure the equipment is optimized for the process.

Deficiencies of conveyor stacking systems

The lack of interaction between the process design engineers and the equipment manufacturers has resulted in the widespread application of conveyor stacking systems which have significant design imperfections. These can be characterized as follows:

- insufficient or overly-sufficient equipment structural strength;
- excessive equipment weight;
- excessive ground pressure of tires or tracks; and
- poor design of the systems which are used for equipment movement on the heap.

Insufficient or overly-sufficient equipment structural strength and excessive equipment weight

The conveyor manufacturer is naturally biased towards making equipment too heavy. Weight does add to costs, but a bit of extra weight will result in higher quality, which is good for product reputation. This serves the equipment manufacturer well for most applications where conveyors are used, since the

equipment is running on a constructed surface such as a concrete slab, and weight is not an issue. The operating surface of a heap leaching poses unique challenges. The working surface is often irregular, and composed of soft or water-saturated material. This imposes severe stress on the equipment. This can cause collapse of the equipment. Unfortunately, the fear of collapse is often compensated for by over-design, which results in excessive equipment weight. Many existing stacking systems struggle with excessive weight, and resort to the excessive use of mobile equipment to aid in positioning conveyors. It is important that the conveyor designers should have to defend their design weights to the process engineers, and demonstrate structural calculations that yield a high strength to weight ratio.

Excessive ground pressure of tires and tracks

The ideal conveyor stacking package should be composed of equipment which travels over the heap without causing visible wheel or track ruts. The human foot exerts a pressure of about 5,000 kg/m^2. This pressure will not depress the top heap surface. A pickup truck tire exerts a ground load of about 18,000 kg/m^2. It is easy for a pickup truck to depress the top surface of a soft, wet heap to the point where the truck has trouble driving out of the depression. So, as an ideal goal for conveying equipment operating on top of a heap, designers should try to achieve a ground pressure of no more than 10,000 kg/m^2. Using this rule of thumb, large high-flotation tires can support a load of 2,000 kg. Larger or heavier loads should be supported on steel caterpillar-type tracks.

Poor design of systems for moving equipment

Stacking equipment must continually move on the heap, often several meters per day. This movement should be done by the equipment itself, using powered wheels or tracks. The goal should be to eliminate all auxiliary equipment such as bulldozers, front end loaders, or cranes from the top of the heap. The designer should remember that every time an endloader or a bulldozer moves on top of the heap, some recovery is lost. Equipment drives should be powerful enough so that the equipment can extract itself when it runs into depressions in the surface.

Conveyor stackers should be equipped with stinger conveyors and operator controls that allow continuous movement of the ore discharge stream. With good equipment and good training, the conveying system can be used to build a nearly flat heap surface with no assistance from mobile equipment such as bulldozers.

Summary

In summary, there are several different ways to place the ore onto heaps for leaching, and not all of these are optimal in all circumstances. Poor choices result in a small loss of recovery due to bad stacking equipment or methods. Such losses are often accepted in practice because they are not easily measurable, or because the equipment cannot be modified.

Sensitivity analysis of variably saturated flow and transport in a heap leaching operation

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Abstract

This paper describes a series of numerical simulations designed to investigate non-uniform flow and solute transport in a heap leach undergoing ore leaching. Details regarding unsaturated flow within heap leach piles are generally lacking, and pilot and field research in the past decade is providing more insight into non-uniform flow and leaching within these systems. To improve our understanding of flow behavior within a heap, this paper describes a conceptual model in which the heap consists of repeated sequences of layers that dip at the angle of repose as a result of the heap construction process of repeated dumping from ore trucks and spreading by bulldozers. This conceptual model is implemented in a numerical model of liquid flow and solute transport.

For the numerical simulations, the unsaturated hydraulic properties are modeled using the Van Genuchten constitutive relations and are specified to vary in association with the sorting which occurs during layer emplacement and compaction from truck traffic on top of each lift. A two-dimensional vertical slice through the pile is modeled using the Richards equation for liquid flow and the advection-dispersion equation for solute transport; these equations are solved using the MODFLOW-SURFACT® numerical code. Simulation results indicate that large portions of the cross-section can be bypassed by the bulk of liquid flow due to capillary barrier effects. While high flux rates can better access the bypassed zone, they can also lead to pore pressure and exit gradient issues of concern from a geotechnical perspective. This variability in flow regions affects results in the solute-transport problem by reducing the ore leaching efficiency (extending leaching time) in those regions bypassed by the bulk of the liquid flow. A number of the geometric parameters and physical properties are varied to assess the sensitivity of model results to factors such as heap size (small, medium, or large), lift thickness, compacted layer

thickness and properties, and particle size range (and consequently, the range in variability of K_{sat} and unsaturated moisture characteristics). The results suggest that characterization of the unsaturated hydraulic properties of heap rock materials can be used to help design heap leach piles and select better leaching flux rates or cycling of rates to improve metal recovery rates and leaching efficiencies.

Introduction

This paper describes an investigation of the heap leaching process from a variably saturated flow groundwater modeling perspective. We focus on heaps constructed of run-of-mine (ROM) ore using truck end dumping (approx. 300 ton capacity) in conjunction with dozer spreading. This method of construction results in a heap composed of a heterogeneous mix of materials in terms of clast or particle size, but with the heterogeneity exhibiting a characteristic structure that is repeated throughout the entire heap lift. Figure 1 is a schematic diagram that illustrates the scenario we are investigating.

Previous research has investigated the heterogeneous structure of materials within a heap (Kinard and Schweizer, 1987; Keane, 1998).Previous modelling of heap leach (McCord et al., 1997; Orr, 2002; Orr and Vessilinov, 2002) and waste rock piles (Fala et al., 2003) clearly show non-uniform flow effects. They also suggest that large portions of cross-section can be bypassed by the bulk of liquid flow due to capillary barrier and preferential flow effects associated with the material heterogeneity structure, which can affect leaching efficiency and geotechnical stability. Webb et al. (2008) performed a detailed field study investigating spatial variability in unsaturated and saturated flow within a heap at a variety of scales.

The objectives of this paper are as follows:

- Develop a conceptual model of heterogeneity structure within a heap constructed using run-of-mine ore and truck end dumping.
- Develop a numerical model of variably saturated flow and solute transport of a heap comprised of multiple lifts of this heterogeneous structure.
- Use the model to investigate capillary barrier and preferential flow effects within the heap and the potential impacts of this behavior on leaching efficiency and geotechnical considerations.

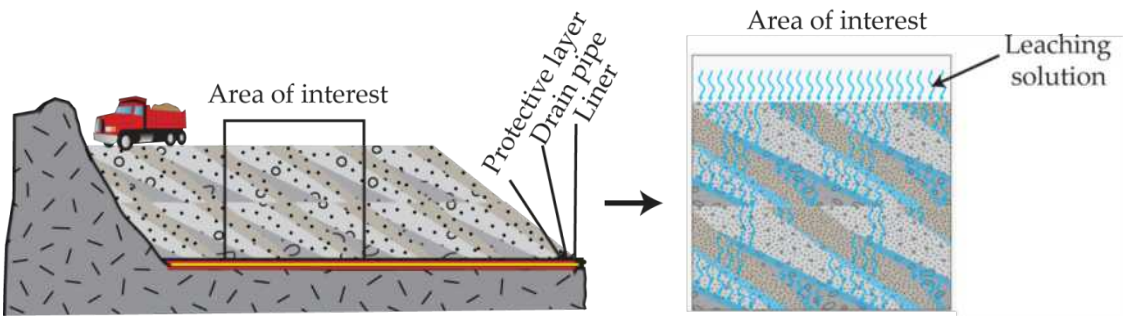


Figure 1: Schematic diagram illustrating heap construction, heterogeneity structure, and preferential flow

Methodology

As described above, the methodology involves first defining a conceptual model of the geometric configuration of material heterogeneity within a heap leach pad, then translating that into a quantitative framework in terms of representative material properties and appropriate boundary conditions that can be simulated using a variability saturated flow and transport code. We use the resulting model to illustrate non-uniform flow within the heap under a variety of raffinate application rates, and look at the results in terms of moisture distributions, leaching rates and efficiency, pore pressure distributions, and exit gradients at locations where seepage faces may develop.

Conceptual model

In the conceptual model, a heap leach pad consists of repeated sequences of layers that dip at the angle of repose (defined as 40 degrees for the model in this paper) as a result of the heap construction process, which involves repeated dumping from ore trucks (capacity approx. 300 ton) and spreading by bulldozers. Figure 1 schematically illustrates the process and the characteristic repeated sequence of heterogeneity that results from the construction process. To facilitate quantitative analysis of this large-scale conceptual model, it is hypothesized that a characteristic "unit cell" of heterogeneity exists and can be defined. Then we can construct a multi-lift pad by simply putting together and stacking these unit cells repeatedly. Figure 2 illustrates the prototypical unit cell, or depositional layer, which exhibits outward coarsening in the direction normal to the slope and upward fining in the direction parallel to the slope from the toe of the layer. The final aspect of this leach pad conceptual model is a compacted layer at the top of each lift, which results from truck and bulldozer traffic on top of the lift. In other cases, this compacted upper layer is loosened up by ripping, resulting in higher permeability; this paper does not consider that situation.

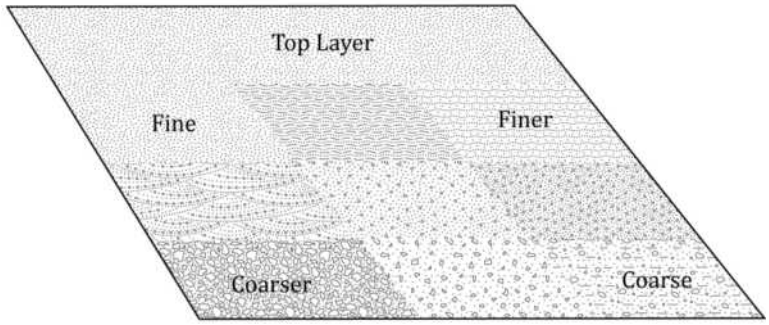


Figure 2: A segregated "unit cell," the fundamental building block for a hypothesized heap pile

Material properties

To simulate flow and transport through the leach pad, this conceptual picture of a unit cell was translated into physical hydraulic properties that can be input into the numerical model; the properties should vary in association with the sorting that occurs during layer emplacement and compaction from truck traffic on top of each lift. Those hydraulic properties consist of the soil water characteristic curves (SWCCs) that describe the moisture content-capillary pressure constitutive relationship and the saturated hydraulic and unsaturated hydraulic conductivities. We employed the Van Genuchten formula for defining the SWCCs (Van Genuchten, 1980):

$$S = \frac{\theta-\theta_r}{\theta_s-\theta_r} = \begin{cases} \left[1+(-\alpha\Psi)^{\beta}\right]^{-\gamma} for\ \Psi < 0 \\ 1\ for\ \Psi > 0 \end{cases} \tag{1}$$

Where

S = the effective relative saturation

θ_s, θ_r = the saturated and residual volumetric water content

ψ = the pressure head

α, β = the Van Genuchten unsaturated soil parameters

Conveniently, the Van Genuchten relationship can be used to calculate the relative permeability (HGL, 1996), which, multiplied by the saturated hydraulic conductivity, K, yields the unsaturated hydraulic conductivity:

$$K_{rel} = \frac{K_{unsat}(S)}{K} = \sqrt{S}[1-\left(1-S^{\frac{1}{\gamma}}\right)^{\gamma}]^2 \tag{2}$$

Where

K_{rel} = relative permeability

K_{unsat} = unsaturated hydraulic conductivity

γ = $1 - 1/\beta$

Table 1 below presents the values used for the saturated hydraulic conductivity and Van Genuchten parameters used in our analysis. This table also lists the transport properties for the solute transport model.

Table 1: The flow and transport properties on the heap leach model

Material type	K (m/day)	K (cm/s)	□ (1/m)	□ (–)	S_r (–)	$□_L$ (m)	$□_{TV}$ (m)	D* (m^2/day)
Top layer	0.22	2.55×10^{-4}	3	1.20	0.17	0.5	0.05	1.78×10^{-4}
Finer	0.43	4.98×10^{-4}	5	1.30	0.17	0.5	0.05	1.78×10^{-4}
Fine	1.73	2.00×10^{-3}	7	1.65	0.17	0.5	0.05	1.78×10^{-4}
Coarse	10.80	1.25×10^{-2}	10	1.80	0.17	0.5	0.05	1.78×10^{-4}
Coarser	43.20	5.00×10^{-2}	15	2.20	0.17	0.5	0.05	1.78×10^{-4}

In Table 1, K is the saturated hydraulic conductivity, α and β are the Van Genuchten parameters, S_r is the residual saturation, α_L is the longitudinal dispersion coefficient, α_{TV} is the vertical dispersion coefficient, and D* is the molecular diffusion coefficient.

Numerical model and boundary conditions

A two-dimensional vertical slice through the pile (shown in Figure 3) is modelled using the Richards equation for liquid flow and the advection-dispersion equation for solute transport (see HGL, 1996, for a general description of the Richards equation for flow and advection-dispersion equation for solute transport); these equations are solved using the MODFLOW-SURFACT numerical code (HGL, 1996).

The total model domain size is approximately 195 m high and 70 m across, representing a heap composed of several 10 m high lifts. The whole model domain is then discretized using rectangular grid cells with dimensions 1.2 m × 1.0 m in X and Z directions, respectively.

An initial concentration of unity is used for the solute transport part (see Figure 3c); a normalized concentration is used for the active domain (see Figure 3a for active and inactive model domains). Regarding boundary conditions for the transport model, no influx of solute mass is applied along the left and right model boundaries. Because the cells, specified as drain cells, are allowed to drain water from the model, solute mass is allowed to outflow from these cells; however, the concentration of the solute leaving the model domain is calculated by the model itself based on the total mass balance of the complete system.

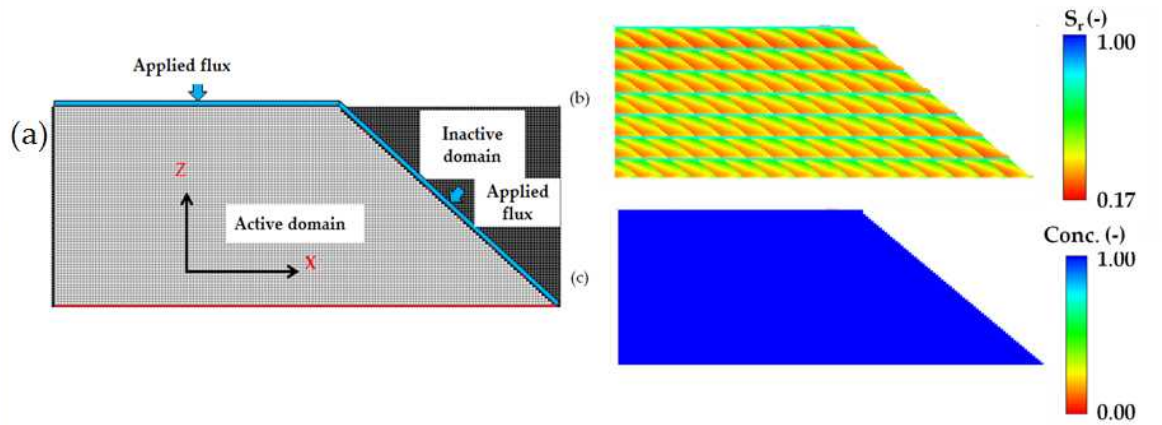


Figure 3: Model domain along with the boundary (a) and initial conditions for flow (b)and transport models (c)

Considered scenarios

Four scenarios representing four applied leaching solution application rates or flux rates (represented by q) are considered. These scenarios for the applied flux rates are based on the cumulative density function (CDF) (not shown here) of the probability density function for the saturated hydraulic conductivity field for the whole heap pile. Based on this CDF, the scenarios are as follows:

- Scenario 1: this corresponds to an applied flux rate of 9.17 l/h/m^2 (0.22 m/day), the minimum value of the CDF.
- Scenario 2: this corresponds to an applied flux rate of 51.25 l/h/m^2 (1.23 m/day) at 25% CDF.
- Scenario 3: this corresponds to an applied flux rate of 347.92 l/h/m^2 (8.35 m/day) at 50% CDF.
- Scenario 4: this corresponds to an applied flux rate of 760.42 l/h/m^2 (18.25 m/day) at 75% CDF.

The scenarios above are each further subdivided into two cases. In Case A for each scenario, flux is applied only along the top boundary; in Case B for each scenario, flux is applied both along the top and the sloping boundaries.

In general, Scenarios 1 and 2 represent a reasonable range of lixiviant irrigation rate (based on the cumulative distribution function of the saturated hydraulic conductivity). Scenarios 3 and 4 are higher than one would typically apply, but they were included here as a sensitivity analysis to demonstrate the impacts on geotechnical stability of the heaps as a function of irrigation rate.

Results and discussion

For the various scenarios described above, the flow and transport model is run for a one-year period, and the results are then compared for each scenario. Figures 4 and 5 show relative saturation (S_r) in the top two rows and solute concentration (Conc.) in the bottom two rows after one month and one year, respectively.

The relative saturation and solute concentration profiles in Figures 4 and 5 show that when a small flux rate is applied only along the top boundary, most of the flow domain below the sloping boundary is unaffected by the leaching process, and relative saturation and solution concentration in this unaffected zone are at their initial values (see Figure 3 b and c for initial values). Furthermore, even with this small flux rate, when the flux is applied along both the top and the sloping boundaries, relative saturation increases in the whole pile, and there is more mineral recovery (see the first column in Figures 4 and 5). For higher flux rates, Figures 4 and 5 clearly show that the finer soil material conducts water at a higher rate compared to the coarser material when the pile is not completely saturated, thus indicating a capillary barrier effect: a finer material on top of a coarser material must attain a higher saturation for water to flow into the underlying coarse material. This is clearly shown in plot 2A in Figures 4 and 5. The only difference between plots 1A and 2A is that a higher flux rate is applied along the top boundary in Case 2 as compared to Case 1. This higher flux rate and the capillary barrier effect have caused higher horizontal flow (mainly in the sloping fine soil material), resulting in higher porous media saturation even below the sloping boundary.

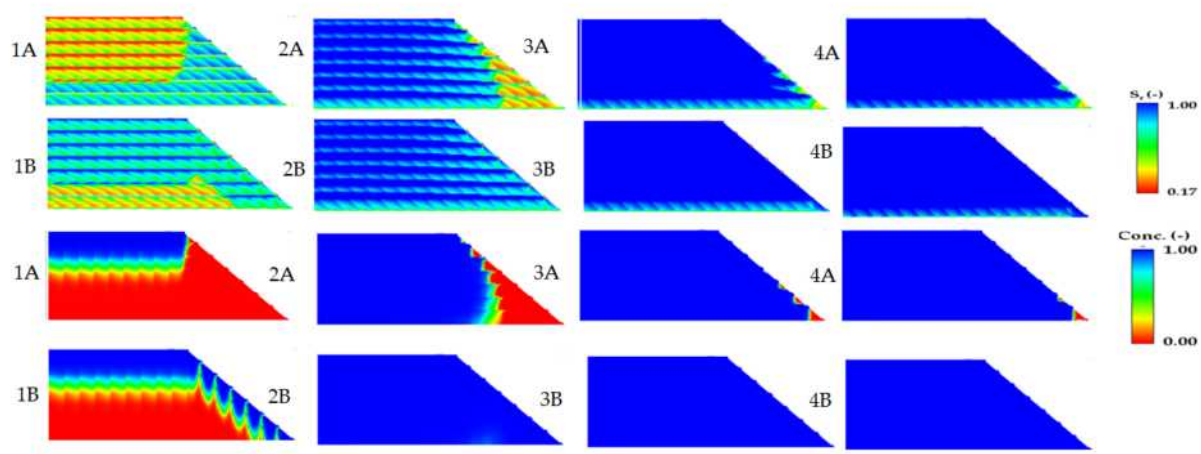


Figure 4: Relative saturation and solute concentration after 1 month for various scenarios. Scenario 1x represents lowest irrigation rate and 4x the highest rate; A considers application of lixiviant to only the top surface and B considers application to both the top and sloping surfaces

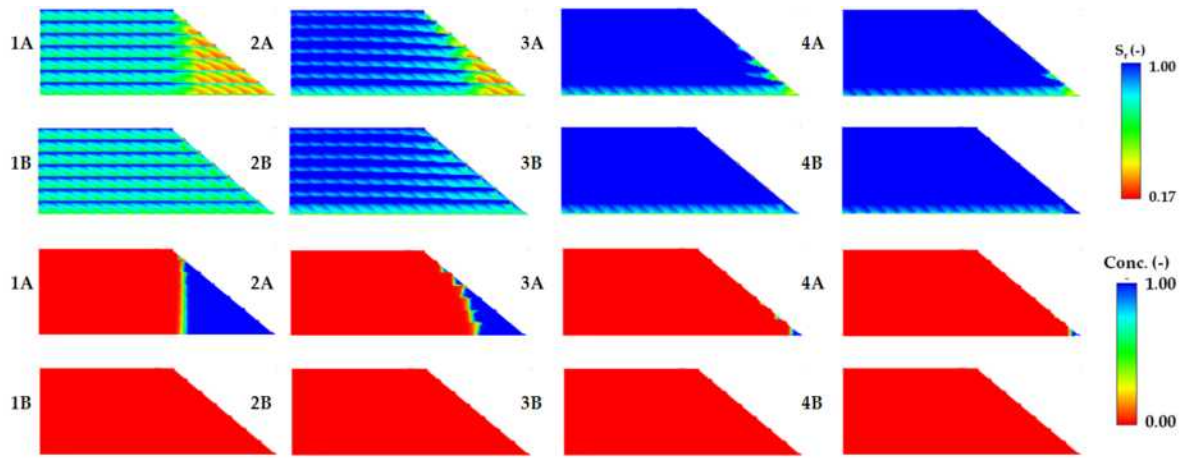
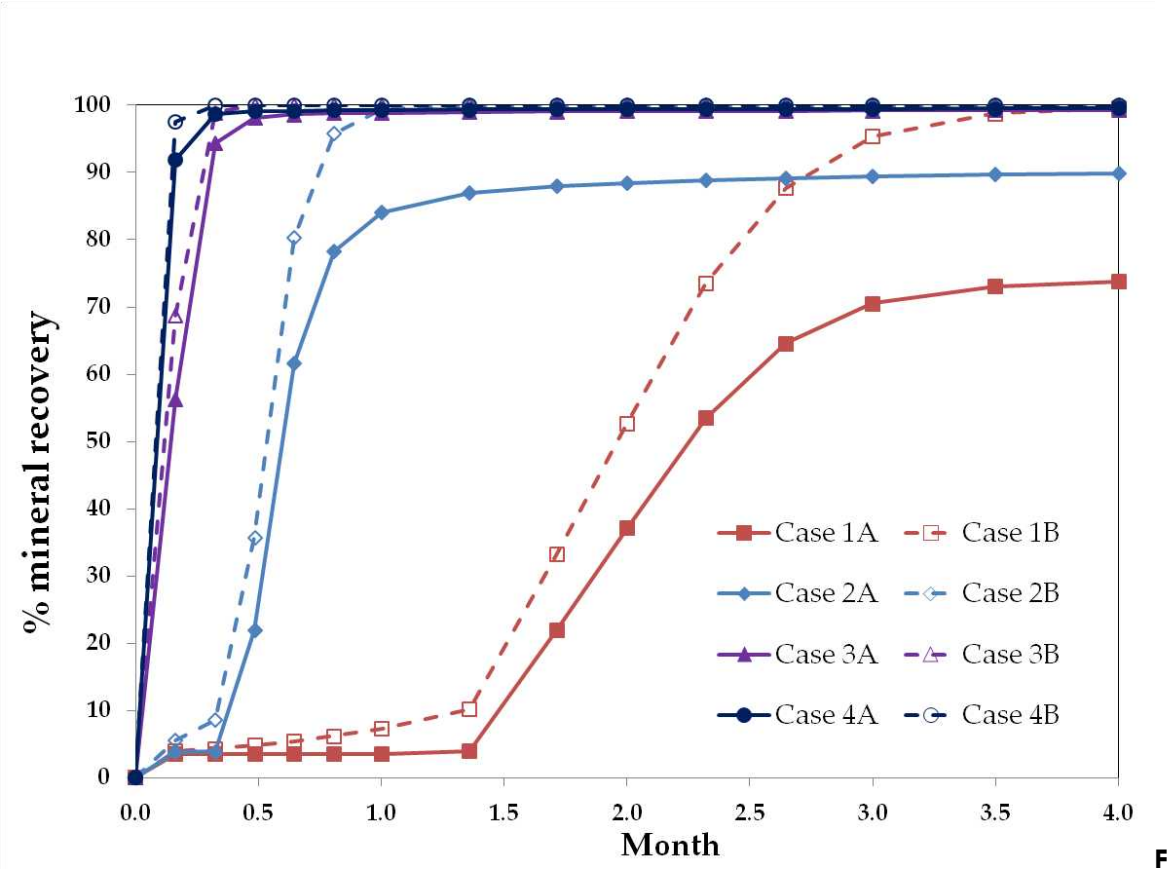


Figure 5: Relative saturation and solute concentration after 1 year for various scenarios

For the scenario with the higher flux rate or with flux along the top and sloping boundaries, there is more mineral recovery (see Figure 6). This is mainly because, with the aforementioned conditions (i.e., higher flux rate or flux along both top and sloping boundaries), a higher proportion of the media contributes to the matrix water flow. However, the percentage of mineral recovery approaches an asymptote, larger for a higher flux rate, with a given flux rate. We believe this is caused by constancy in flow volume. In other words, the flow domain that can contribute to water flow is contributing at its maximum capacity for the given conditions. Figure 6 shows that as the flux rate is increased, the increase in mineral recovery is not appreciable after a certain rate. With a two-fold increase in the flux rate from 8.35 m/day to 18.25 m/day, there is almost no change in the mineral recovery percentage (see Figure 6). Because total initial solute mass is the same for both cases, the constant mineral recovery with higher flux rate, and thus higher expenses, indicates existence of an optimum flux rate for maximum mineral recovery with acceptable expenditures.

Furthermore, the numerical models show that with higher applied flux rates, the change in mineral recovery for the case when flux is applied along both the top and sloping boundaries (Case B) is only marginally higher than the case in which flux is applied along the top boundary only (Case A). In our opinion, this is because, in Case A, the higher horizontal flow due to higher flux rate leads to near saturation or saturation of porous media (see Figures 4 and 5, relative saturation profiles for Scenarios 3 and 4). When the same flux is also applied along the sloping boundary, it does not change or only slightly changes the saturation of the porous media.



Figure

Figure 6: A comparison plot of percentage mineral recovery for each scenario

Although the discussion above suggests that the higher flux rate leads to higher mineral recovery, to a large extent, there is a drawback with the higher flux rate. A higher flux rate also leads to higher pore pressures and higher exit gradients. Table 2 lists the minimum and maximum values of the exit gradient and pore pressure for each scenario. It can be inferred from this table that the pore pressure and exit gradient for Scenarios 3 and 4 are substantially higher compared to the other scenarios. Table 2 also shows that Case B (flux along both top and sloping boundaries) leads to more uniformity in pore pressure distribution.

Although the main purpose of this article is to develop a conceptual understanding of the heap leaching operation under various leaching solution application rates, the model results compare qualitatively well with the published results. For example, Webb et al. (2008) conducted a field study at a heap leach facility in Nevada, USA, for 90 days. In their study, the leaching solution application rate is very low and the scenario closely resembles Case 1A of this study. The authors (Webb et al., 2008) have reported a high mineral recovery even with a low leaching solution application rate and near isolation of the heap pile material under the sloping side of the heap pile. The modeling results for Case 1A of this study have shown similar findings: (a) after approximately 3 months, the mineral recovery reaches a

maximum and a relatively high mineral recovery percentage (~75%) (see Figure 6); and (b) most of the mineral below the sloping boundary is unaffected by the leaching processes. Thus, although the modeling results are based on a conceptual heap leaching operation, they compare well with the field observations and bear some practical relevance.

Table 2: Maximum and minimum exit gradients and pressure heads after 1 year

Scenario	Min. exit gradient (–)	Max. exit gradient (–)	Min. pressure head (m)	Max. pressure head (m)
Case 1A	3×10^{-9}	1.5×10^{-3}	-3.33×10^{1}	3.45×10^{-1}
Case 1B	4.6×10^{-4}	7.8×10^{-1}	-3.32×10^{1}	3.45×10^{-1}
Case 2A	5.2×10^{-9}	5.1×10^{-1}	-3.31×10^{1}	4.32×10^{0}
Case 2B	3.6×10^{-3}	9.8×10^{-1}	-3.24×10^{1}	4.33×10^{0}
Case 3A	6.5×10^{-8}	9.6×10^{0}	-2.85×10^{1}	1.61×10^{2}
Case 3B	8.9×10^{-3}	1.1×10^{1}	-2.49×10^{1}	1.66×10^{2}
Case 4A	4.7×10^{-8}	1.8×10^{1}	-2.01×10^{1}	4.00×10^{2}
Case 4B	2.2×10^{-2}	2.2×10^{1}	-1.18×10^{1}	4.16×10^{2}

The conceptual model presented in this study can also be applied to waste rock drainage scenarios. In such scenarios, the applied flux rate can be replaced by either precipitation or the effective precipitation (precipitation minus evaporation). Thus, the modeling results from this study for a heap leaching operation can serve as an analogue to rock drainage scenarios. For example, Scenario 1, which has low applied flux, can be considered as waste rock drainage under dry conditions, Scenarios 3 and 4, which have higher applied flux rates, can be considered as waste rock drainage under wet conditions, and Scenario 2 can be considered as representing a waste rock drainage under intermediate conditions.

Conclusion

Based on the modelling results presented in the results and discussion section above, our conclusions are as follows:

- There is higher mineral recovery when flux is applied along the top and sloping boundaries compared to flux only along the top boundary.
- After a certain lapsed time, mineral recovery reaches an asymptote and no more leaching occurs.
- There is a large change in exit gradient when flux is applied only along the top boundary compared to the case when flux is applied along both the top and sloping boundaries.

- Although higher flux rate leads to higher mineral recovery, the higher flux rate is also associated with (i) higher pore pressure as compared to the other case, and (ii) higher exit gradient as compared to the other case.
- The percentage of mineral recovery has only marginally changed with higher flux rates.
- An optimal case exists with higher mineral recovery but with an acceptable maximum pore pressure and exit gradient when geotechnical stability of the heap pile is considered.

The main purpose of the present conceptual heap leach model is to provide a deeper understating of leaching operations and how these operations are influenced by the media heterogeneity and leaching solution application rates. Like Orr (2002) and Orr and Vesselinov (2002), the authors of this article believe that a thorough understanding of the complex flow processes within a heap is important for optimizing heap leaching operations.

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Water balance and cost evaluation for different scenarios of impermeable covers (raincoats) in heap leach pad operations

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Abstract

Currently, some mining operations which use heap leaching technology and are located in rainy regions use impermeable covers or raincoats on the top of the ore heap to reduce the amount of rainwater that gets into the heap. The raincoats are also used in those areas of the heap still under irrigation. Various experiences of heap leach pads on an industrial scale indicate that the entrance of rainwater into the system causes solution dilution, making metal recovery less efficient. It also produces surplus contaminated water that requires recirculation or treatment before it can be discharged into the environment. These two issues cause significant additional costs.

This paper presents an analysis of two different scenarios involving raincoat placement in heap leach pads. The first case is in a high precipitation tropical region in northern Brazil, where copper ore is processed; the second case is in a mountain range in the Andes in southern Peru, where gold is recovered. The water balance was developed considering differences in percentage of raincoats, treatment plant capacity, stormwater pond capacity, and raincoat pond capacity. The water balance results allowed researchers to determine, on a monthly basis, the operating flows to be stored in the stormwater pond and the flows which had to be purged out of the system and therefore had to be treated before they could be discharged into natural streams. The paper also presents a comparative analysis of capital expenditure (Capex) and operating expenditure (Opex) of different scenarios in the two cases. The cost evaluation indicates that the use of a larger quantity of raincoats reduces the total cost for the operating life of the heap leaching facilities, with significant savings to the project.

Introduction

Some years ago, the use of impermeable covers or raincoats in heap leach pad operations located in high precipitation areas was restricted to minimum areas of the heap for cost reasons; however, experience indicates that as the raincoat installation minimizes the entrance of rainwater into the system, long-term operating costs reduction are achieved. Moreover, raincoats offer an economic and efficient way to divert rainwater flow to a raincoat pond and finally discharge it into the environment without previous treatment, reducing process solution dilution, stormwater pond capacity, treatment plant size, and water treatment cost.

Two cases were analyzed for water balance simulation: the first is a copper heap leach pad located in northern Brazil; while the second is a gold heap leach pad in southern Peru. Both are in high precipitation regions. The hydrology in each region was evaluated based on precipitation and evaporation data from nearby weather stations. Water balance refers to the interconnections among the heap leach pad, the pregnant leach solution (PLS) pond, the intermediate leach solution (ILS) pond if any, the stormwater pond, and the raincoat pond.

Hydrology

Basic information was gathered from nearby weather stations through the Brazilian National Water Agency (ANA, in Portuguese) and the Peruvian Meteorology and Hydrology National Service (SENAMHI, in Spanish).

Precipitation and evaporation

Visual inspection of available precipitation and evaporation data allowed researchers to use a consistency analysis of jumps and trends, which determined that weather station records used had uniform distribution and consistent data. Tables 1 and 2 show monthly average precipitation and evaporation for each analyzed case.

Table 1: Total monthly precipitation (mm)

Month	First case				Second case			
	Max.	Aver.	Min.	% Annual	Max.	Aver.	Min.	% Annual
Jan	414.5	237.4	104.2	14%	511.5	219.7	7.7	23%
Feb	440.2	269.0	152.7	16%	406.4	220.2	81.5	23%
Mar	510.2	280.9	141.7	17%	461.5	200.6	0.0	21%
Apr	608.2	219.5	38.6	13%	301.3	66.7	0.0	7%
May	271.5	114.1	0.0	7%	79.4	15.4	0.0	2%
Jun	101.0	24.9	0.0	1%	47.4	5.9	0.0	1%
Jul	154.7	20.0	0.0	1%	62.8	7.8	0.0	1%
Aug	86.4	24.1	0.0	1%	198.7	17.8	0.0	2%
Sep	131.5	54.2	3.6	3%	75.6	20.8	0.0	2%
Oct	249.0	110.7	3.7	7%	133.3	31.9	0.0	3%
Nov	249.7	139.8	34.2	8%	313.4	44.2	0.0	5%
Dec	495.9	197.3	69.1	12%	279.5	98.7	0.0	10%
Total	**2,415.4***	**1,691.9**	**1,057.8***	**100%**	**1,603***	**950**	**455***	**100%**

*Total annual maximum and minimum precipitation is an annual historical record and is not obtained by adding the maximum values of each single month

Table 2: Total monthly evaporation (mm)

Month	First case				Second case			
	Max.	Aver.	Min.	% Annual	Max.	Aver.	Min.	% Annual
Jan	161.2	130.7	90.6	14%	196.9	93.6	34.8	7%
Feb	158.3	125.3	93.3	16%	121.2	66.8	15.0	5%
Mar	151.1	119.8	99.4	17%	125.0	71.3	23.2	5%
Apr	219.1	134.4	38.7	13%	175.9	92.7	16.9	6%
May	155.1	110.0	73.5	7%	217.0	123.6	32.0	9%
Jun	148.9	124.9	100.2	1%	260.2	131.8	41.1	9%
Jul	207.0	164.7	129.4	1%	259.2	141.1	0.0	10%
Aug	244.7	193.4	137.1	1%	247.6	141.3	0.0	10%
Sep	249.3	194.1	165.0	3%	265.5	136.6	0.0	10%
Oct	206.6	166.6	127.2	7%	246.5	146.2	59.0	10%
Nov	148.4	119.0	100.9	8%	214.9	134.1	36.9	10%
Dec	240.4	142.5	83.0	12%	179.1	121	63.2	9%
Total	**1,939.1***	**1,725.3**	**1,454.5***	**100%**	**2,347.2***	**1,400**	**817.5**	**100%**

*Total annual maximum and minimum evaporation is an annual historical record and is not obtained by adding the maximum values of each single month

Generated series

Precipitation and evaporation series with lengthy records of the analyzed cases were implemented for water balance. Synthetic series were obtained using the index sequential method (ISM). Figure 1 shows the variation of total monthly precipitation and evaporation for the first and second case.

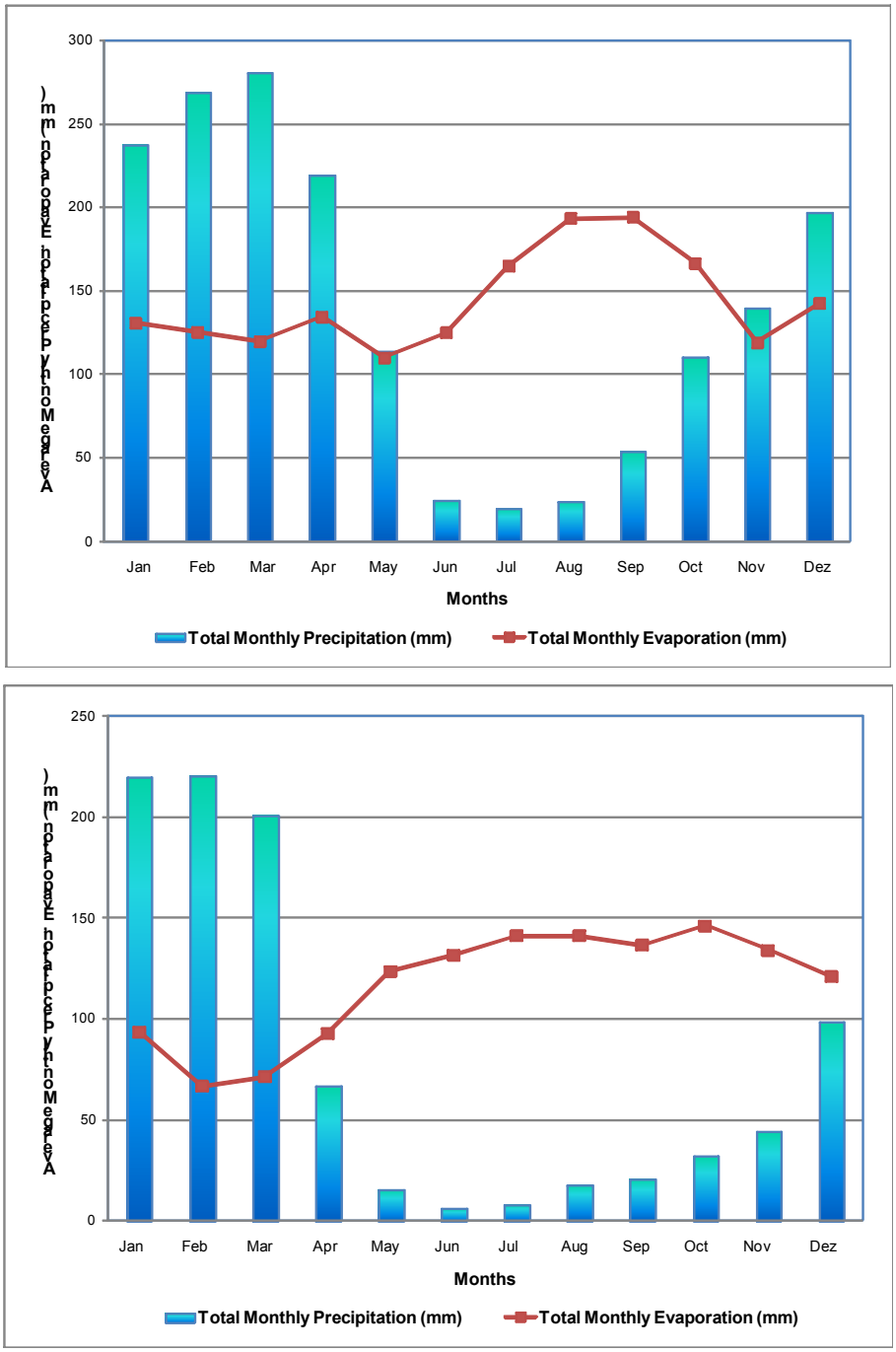

Figure 1: Total monthly precipitation and evaporation (mm) (first case above; second case below)

Extreme hydrological events

The maximum precipitation evaluation was performed based on extreme events at representative weather stations of each analyzed case. Data on maximum precipitation in a 24-hour period were fit to several probabilistic models. Based on various statistical indexes and hydrological criteria, the generalized extreme value index (GEVI) distribution was selected to provide uniform criteria because it presented the best indexes, according to the Kolmogorov-Smirnov goodness-of-fit test. Table 3 shows the maximum precipitation in a 24-hour period for different return periods of the analyzed cases.

Table 3: Maximum precipitation frequency in 24 hours (mm)

Return period	2 years	5 years	10 years	25 years	50 years	100 years	500 years
First case	95.8	119.5	135.2	155.1	169.8	184.5	218.3
Second case	30.3	39.9	46.3	54.4	60.4	66.3	80.0

Water balance

Water balance description

As with any other water balance model, the water balance was developed using a spreadsheet based on the following equation:

$$\text{Inflow} - \text{Outflow} = \text{Storage change}$$

Inflow comes from precipitation falling over the heap leach pad area and from fresh water for reposition. Outflow (discharge) corresponds to pad evaporation (from active areas under leaching, inactive areas, and losses due to heap irrigation), pond evaporation, and excess outflow of the pad-pond system previous to effluent treatment (detoxification).

Changes in storage capacity are associated with changes in the moisture content stored in ore voids and pond water level fluctuation. Recirculation flows between ponds (PLS, ILS, barren, raffinate, or stormwater) and the heap leaching area are considered as internal flow (do not generate inflow or outflow). The use of raincoats will minimize water entry into the system.

Parameters and simulation criteria

The water balance model depends on the ore production plan, the stacking plan in the heap, the raincoat installation area, ore properties, irrigation type, precipitation, evaporation, the size of the ponds, and their initial storage capacity. As water balance is a function of plant operation conditions, results obtained are directly related to operational parameters introduced in the model and are susceptible to changes. Table 4 presents parameters related to the conditions mentioned above.

In the two cases, use of a raincoat system has been considered. This offers an effective and economic way to separate and deviate rainwater flow to the raincoat pond, where water will be monitored for contamination and then discharged into natural streams or deviated to the stormwater pond in case non-permissible contamination levels are found. This minimizes process solution dilution, reduces stormwater pond storage, and diminishes water treatment cost.

Table 4: Parameters and design criteria

Parameter	Unit	First case	Second case
Daily production rate	t/day	9,400 to 16,000	4,500 to 8,500
Phases 1, 2, and 3 capacity	Mt	8.4 – 26.1 – 35.5	6.4 – 9.0 – 13.7
Phases 1, 2, and 3 extension	Ha	26.8 – 48.2 – 54.7	21.7 – 13.2 – 13.3
Operation period	months	156	118
Ore moist density	t/m^3	1.45	1.53
Application rate	$l/h/m^2$	10	12
Draindown time	hours	12	24
Typical lift thickness	m	5,2	8
Leach cycle	days	120	120
PLS pond capacity	m^3	19,120	15,000
Raffinate pond capacity	m^3	17,000	–
ILS pond capacity	m^3	–	15,000
Stormwater pond capacity	m^3	tbd	tbd
Initial ore moisture	%	19	5
Residual moisture content	%	25.6	7
Absorption, moisture retention	%	6.6	2
Evaporation factor of ponds	–	0.9	0.7
Evaporation factor of leaching area	–	0.65	0.5
Evaporation factor of non-leaching area	–	0.05 – 0.30	0.25
Irrigation losses	%	0.1	1
Raincoat coverage	%	30, 50, and 80	30, 50, and 80
Initial month of simulation	–	January 2016	January 2014

Water balance scenarios

There are four possible water balance scenarios represented. Scenario 1, the base case, consists of the heap leach pad without raincoat coverage, while scenarios 2, 3, and 4 involve placing raincoats on a varying percentage of the heap leach pad area: 30%, 50%, and 80%, respectively.

Pond sizing

The storage capacity of the pregnant leach solution (PLS) pond depends on leaching operating conditions. The stormwater and raincoat ponds were sized taking into account the following considerations:

- Stormwater pond. This pond was sized considering the largest volume for maximum precipitation contingency, determined for the most unfavorable monthly sequence in wet seasons, considering stormwater and raincoat ponds.
- Raincoat pond. This pond was sized considering scenarios 2, 3, and 4 (i.e., 30%, 50%, and 80% of total heap area covered by raincoats), with a raincoat efficiency of 90% (due to its exposure to rips and other defects during heap operation), a design storm event, and 2-hour periodic monitoring.

Contingency volume for extreme storms has been established according to inferred criteria (Van Zyl et al., 1988). Van Zyl et al. list two criteria: adding 24-hour and 100-year return period storm volume to volume fluctuations of an average year, and using water balance evaluations of historical records or total monthly precipitation and evaporation synthetics records. This last criterion was implemented due to existing lengthy records, which have led researchers to carry out a series of water balance simulations. In wet weather this criteria is the most critical.

The analysis also included breakdown or malfunction contingency duration (12- or 24-hour draindown; see Table 4) considered as acceptable and conservative, given the operation capability for responding and restoring operations in each case.

Water balance results

The evaluations were performed for the following maximum, average, and minimum variable values:

- operation and contingency total maximum volume;
- fresh water demand; and
- water discharge needs of pad-ponds system.

Because heap leach pads rise gradually, results depend on heap leach pad size from initial to final configuration. The total estimated storage for simulation scenarios is limited by the capacity of PLS and

stormwater ponds. Table 5 shows the water balance storage volumes based on the most critical hydrological situation for each case being analyzed.

Table 5: Total storage volume in water balance (m^3)

Scenario	First case		Second case	
	Operation volume + contingency	Stormwater pond volume	Operation volume + contingency	Stormwater pond volume
No raincoats	205,240	150,000	129,003	105,000
30% of raincoats	205,240	150,000	124,003	100,000
50% of raincoats	155,240	100,000	115,564	95,000
80% of raincoats	130,240	75,000	100,129	85,000

The demand for fresh water for a proper heap leach pad operation decreases as the percentage of raincoats over the heap increases, because of existing high evaporation in the areas under study. This trend is generated because the raincoats limit water losses from evaporation and the entrance of rainwater into the system. Larger water demands occur in the dry season. This explains why, during years with low precipitation, rainwater captured in the leach pad is not enough to maintain operations during the dry season of that year. Table 6 shows fresh water demands for the system in dry season, considered as the most critical hydrological situation. Figure 2 shows the time variation of the maximum fresh water demand for each scenario simulated, for both analyzed cases.

Table 6: Fresh water demands (m^3/h)

Scenario	First case			Second case		
	Max.	Aver.	Min.	Max.	Aver.	Min.
No raincoats	177.2	119.9	52.7	59.4	14.7	0.0
30% of raincoats	144.6	98.0	51.3	46.4	11.1	0.0
50% of raincoats	125.0	97.5	60.4	38.9	11.0	0.0
80% of raincoats	93.6	78.2	60.3	25.3	15.3	0.0

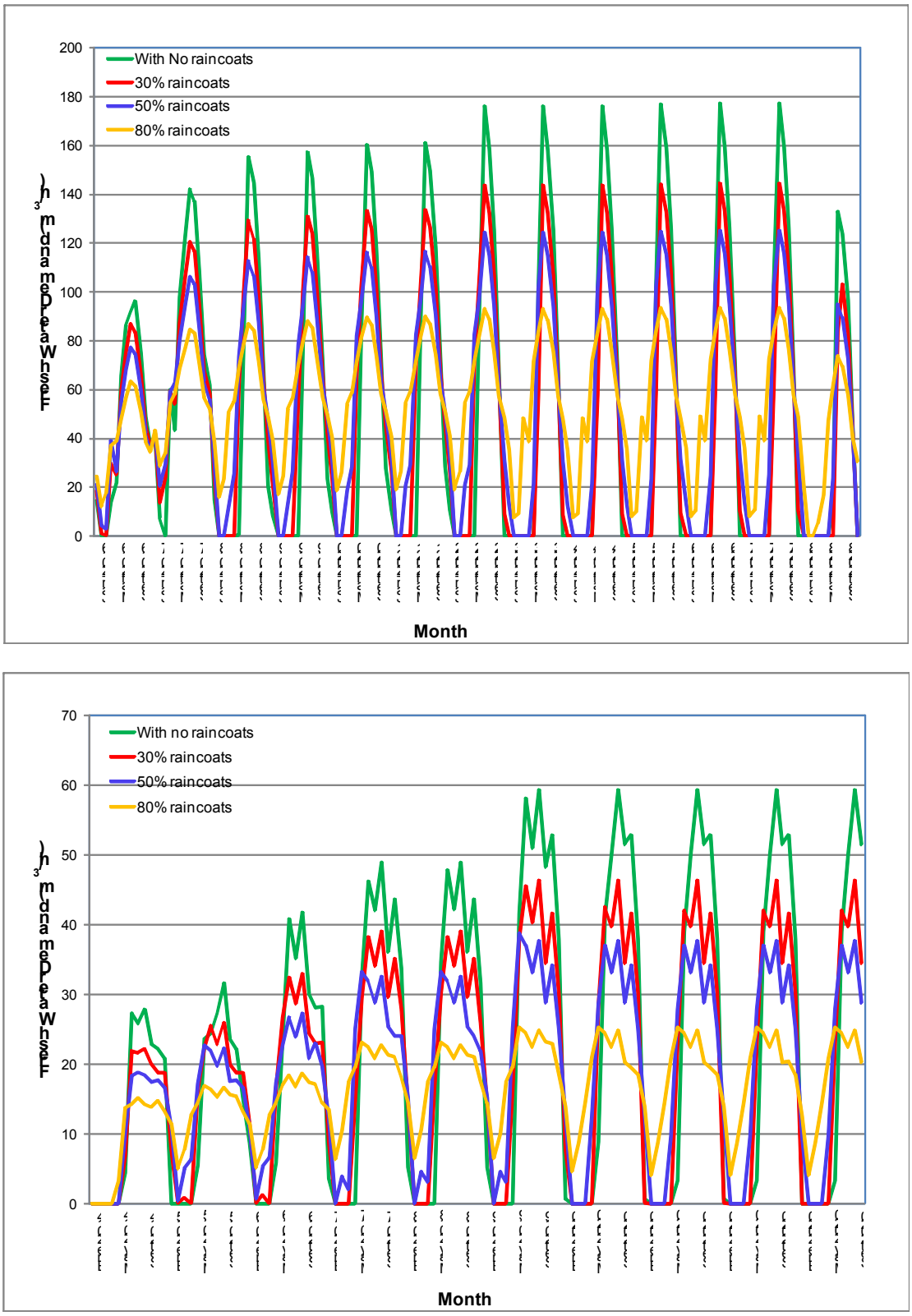

Figure 2: Maximum fresh water demand (first case above; second case below)

Purge water discharges estimated in the water balance show an increment each year as stacking of the heap leach pads increases. Water discharges from the stormwater pond determine the capacity of the contaminated water treatment plant. This is because, at the beginning of heap leach pad operations, the required capacity for the treatment plant is low; as the volume of the heap leach pad increases, it requires a larger plant capacity. Tables 7 and 8 show purge water discharges of water balance for each simulated scenario, for both analyzed cases. Figure 3 shows the maximum purge water discharges time variation for each simulated scenario for both analyzed cases.

Table 7: Purge water discharges (m^3/h) – first case

Year	No raincoats			30% of raincoats			50% of raincoats			80% of raincoats		
	Max.	Aver.	Min.	Max.	Aver.	Min.	Max.	Aver.	Min.	Max.	Aver.	Min.
1	41.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
2	116.9	0.0	0.0	66.9	0.0	0.0	46.6	0.0	0.0	0.0	0.0	0.0
3	366.1	123.9	0.0	260.1	36.9	0.0	189.4	13.3	0.0	33.8	0.0	0.0
4	364.5	125.5	0.0	258.8	60.9	0.0	188.3	28.7	0.0	53.3	0.0	0.0
5	362.2	123.8	0.0	256.8	58.7	0.0	186.6	21.6	0.0	51.3	0.0	0.0
6	361.5	123.1	0.0	256.2	58.1	0.0	186.0	15.9	0.0	49.8	0.0	0.0
7	476.5	122.9	0.0	284.9	57.9	0.0	191.6	15.0	0.0	49.4	0.0	0.0
8	538.3	216.4	59.9	379.9	133.2	7.8	274.4	88.5	0.0	89.1	0.0	0.0
9	538.2	216.7	59.9	379.9	133.2	7.8	274.3	88.5	0.0	89.0	0.0	0.0
10	537.8	215.9	59.6	379.5	132.9	7.7	274.0	88.2	0.0	88.8	0.0	0.0
11	537.4	214.1	59.3	379.2	132.6	7.5	273.7	87.1	0.0	88.6	0.0	0.0
12	537.3	213.8	59.1	379.1	132.5	7.4	273.6	86.2	0.0	88.5	0.0	0.0
13	566.1	247.8	84.1	403.1	150.0	20.8	294.4	103.9	6.6	118.5	0.0	0.0

Table 8: Purge water discharges (m^3/h) – second case

Year	No raincoats			30% of raincoats			50% of raincoats			80% of raincoats		
	Max.	Aver.	Min.	Max.	Aver.	Min.	Max.	Aver.	Min.	Max.	Aver.	Min.
1	96.3	14.4	0.0	36.0	2.4	0.0	10.2	0.2	0.0	0.0	0.0	0.0
2	106.5	19.4	0.0	75.8	6.9	0.0	23.2	1.5	0.0	0.0	0.0	0.0
3	108.8	16.1	0.0	64.4	4.4	0.0	35.5	1.0	0.0	0.0	0.0	0.0
4	189.0	50.3	0.0	122.2	25.0	0.0	87.5	7.3	0.0	0.0	0.0	0.0
5	188.0	43.7	0.0	122.0	17.4	0.0	84.0	5.4	0.0	0.0	0.0	0.0
6	188.0	45.0	0.0	122.0	19.1	0.0	84.0	5.7	0.0	0.0	0.0	0.0
7	268.8	82.2	0.0	177.4	45.5	0.0	124.7	21.0	0.0	0.0	0.0	0.0
8	268.8	84.6	0.0	188.6	47.3	0.0	124.7	22.0	0.0	0.0	0.0	0.0
9	268.8	87.3	0.0	188.6	50.6	0.0	124.7	23.5	0.0	0.0	0.0	0.0
10	268.8	81.6	0.0	188.6	45.7	0.0	124.7	21.7	0.0	0.0	0.0	0.0

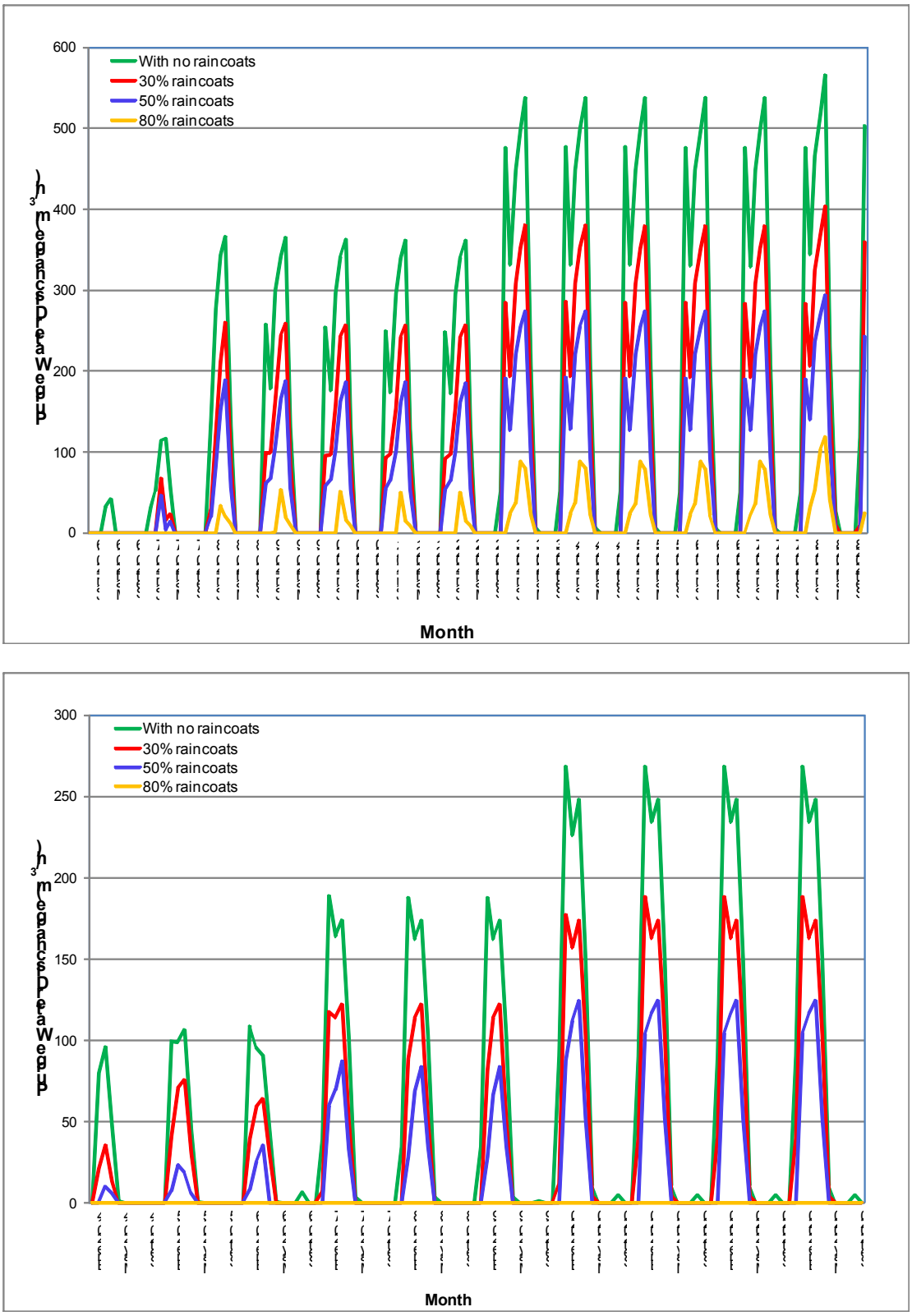

Figure 3: Maximum purge water discharges (first case above; second case below)

The stored volume in the raincoat pond is estimated considering a design storm for a 100-year return period, a heap leach pad covered area, and raincoat efficiency of 90%. Raincoat pond volume has a discharge time of two hours before monitoring. Table 9 shows raincoat pond storage capacities for each simulated scenario, for both analyzed cases.

In summary, the heap leach pads' water balance shows relationships between stored volumes in stormwater and raincoat ponds and water treatment (detoxification) plant capacity for the simulated scenarios. This is illustrated in Table 10.

Table 9: Raincoat pond stored volume (m^3)

Scenario	First case	Second case
No raincoats	–	–
30% of raincoats	25,000	8,000
50% of raincoats	35,000	13,000
80% of raincoats	55,000	20,000

Table 10: Water balance summary

Scenario	First case			Second case		
	Stormwater pond volume (m^3)	Raincoat pond volume (m^3)	Treatment plant capacity (m^3/h)	Stormwater pond volume (m^3)	Raincoat pond volume (m^3)	Treatment plant capacity (m^3/h)
No raincoats	150,000	–	500	105,000	–	250
30% of raincoats	150,000	20,800	400	100,000	8,000	150
50% of raincoats	100,000	34,600	300	95,000	13,000	100
80% of raincoats	75,000	55,300	100	85,000	20,000	–

Cost evaluation

The water balance was analyzed considering four scenarios (see Table 9). According to the obtained results, Capex and Opex were estimated for each scenario. The following aspects were considered:

- Stormwater and raincoat ponds construction cost, which was considered as Capex.
- Raincoat system per year. This corresponds to geomembrane used as raincoat as Opex. We assumed that 30% of geomembrane can be reused or recovered.
- Treatment plant per stages. The year when it needs to be acquired is indicated in Tables 7 and 8. In Year 1, plant cost corresponds to Capex; if the plant is acquired afterwards, it is considered a sustaining capital cost.
- Discharge volume is estimated per year according to Tables 7 and 8.

- For the first case (copper process), the estimated treatment cost was US$ 2.5/m^3 and the treatment plant cost of 100 m^3/h has been estimated at US$ 10 million.
- For the second case (gold process), the estimated treatment was US$ 3.0/m^3 and the treatment plant cost of 100 m^3/h has been estimated at US$ 2 million.
- Tables 11 and 12 show estimated costs (Capex + Opex) for average purge water discharges for each simulated scenario, for both analyzed cases.

Table 11: Estimated costs – first case

Description	No raincoats (US$)	30% of raincoats (US$)	50% of raincoats (US$)	80% of raincoats (US$)
Stormwater pond	871,693.9	871,693.9	444,262.5	384,521.4
Earthworks	476,594	476,594	254,812	224,321
Geosynthetics	395,100	395,100	189,450	160,200
Raincoat pond	0	196,277.8	275,844.4	473,246.2
Earthworks	0	154,428	210,144	368,981
Geosynthetics	0	41,850	65,700	104,265
Raincoat system	0	760,099	1,261,082	2,017,731
Year 1	0	271,496	452,494	723,990
Year 2	0	139,516	232,526	372,042
Year 3	0	69,014	115,024	184,038
Year 4	0	6,140	10,234	16,374
Year 5	0	5,972	9,954	15,926
Year 6	0	7,787	12,978	20,765
Year 7	0	233,218	388,696	621,914
Year 8	0	2,201	3,668	5,869
Year 9	0	2,940	4,900	7,840
Year 10	0	3,385	5,642	9,027
Year 11	0	5,922	9,870	15,792
Year 12	0	7,678	12,796	20,474
Year 13	0	4,830	2,300	3,680
Treatment plant and discharge volumes	65,552,735	44,394,095	32,288,795	10,000,000
Year 1	10,000,000	5,000,000	5,000,000	0
Year 2	25,000,000	20,000,000	15,000,000	0
Year 3	416,973	112,863	24,818	0
Year 4	505,534	157,272	66,878	0
Year 5	488,493	150,272	50,926	0
Year 6	472,877	148,040	39,517	0
Year 7	21,469,404	15,147,493	10,037,571	10,000,000
Year 8	1,167,100	590,224	326,889	0
Year 9	1,167,687	590,287	327,018	0
Year 10	1,163,536	5,873,629	324,882	0
Year 11	1,156,926	583,629	320,452	0
Year 12	1,153,575	582,024	317,633	0
Year 13	1,390,630	744,739	452,212	0
Total cost	**66,424,429**	**46,222,166**	**34,269,984**	**12,875,499**

Table 12: Estimated costs – second case

Description	No raincoats (US$)	30% of raincoats (US$)	50% of raincoats (US$)	80% of raincoats (US$)
Stormwater pond	2,047,752	1,857,689	1,848,618	1,840,965
Earthworks	1,889,965	1,700,660	1,692,394	1,686,427
Geosynthetics	157,787	157,029	156,224	154,538
Raincoat pond	0	422,700	463,688	499,126
Earthworks	0	404,623	423,966	451,345
Geosynthetics	0	18,077	39,721	47,781
Raincoat system	0	528,606	881,010	1,409,616
Year 1	0	193,635	322,725	516,360
Year 2	0	21,762	36,270	58,032
Year 3	0	123,201	205,335	328,536
Year 4	0	19,890	33,150	53,040
Year 5	0	9,302	15,503	24,804
Year 6	0	120,920	210,533	322,452
Year 7	0	39,897	66,495	106,392
Year 8	0	0	0	0
Year 9	0	0	0	0
Year 10	0	0	0	0
Treatment plant and discharge volumes	7,800,689	4,297,942	2,500,283	0
Year 1	2,047,573	7,426	761	0
Year 2	91,009	2,031,364	5,440	0
Year 3	2,089,555	23,689	2,004,517	0
Year 4	266,325	114,350	31,317	0
Year 5	1,217,854	84,419	25,626	0
Year 6	227,901	1,088,016	26,060	0
Year 7	454,861	229,563	94,448	0
Year 8	469,858	238,195	102,871	0
Year 9	477,817	245,532	106,243	0
Year 10	457,936	235,389	103,000	0
Total cost	**9,848,441**	**7,106,937**	**5,693,599**	**3,749,707**

Table 13: Total cost summary (US$)

Scenario	First case	Second case
No raincoats	66,424,429	9,848,441
30% of raincoats	46,222,166	7,106,937
50% of raincoats	34,269,984	5,693,599
80% of raincoats	12,875,499	3,749,707

Conclusions

- Fresh water entrance is required every month, even in wet year conditions.
- Earthworks and geosynthetics costs for pond construction (stormwater and raincoat) are very low compared with operating costs.
- The higher the raincoat coverage in the heap, the lower the total project cost (Capex + Opex).
- If water treatment or plant costs are higher than those considered in analysis, the differences between scenarios would be even higher; the best option would always be to cover as large a heap area as possible.

Recommendations

- The water volume in ponds should be kept as low as possible, and the entrance of fresh water should be regulated based on additional rainwater volume. This common practice in the mining industry has been one of the assumptions of this model.
- In heap leaching projects located in rainy regions, the use of raincoats is strongly recommended to minimize process solution dilution, reduce the need for stormwater pond storage and thereby the size of storage ponds, reduce treatment plant size, and reduce water treatment cost.

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Heap leach pad design in very aggressive terrain

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Abstract

Heap leach pad design in regions where terrain characteristics are particularly aggressive is carried out using the valley fill method. In Peru, most heap leach pad facilities are designed and constructed with the valley fill method; many of these facilities are in very aggressive terrain. Valley fill heap leach pads have also been used in very aggressive terrains in Argentina, Mexico, Colombia, Indonesia, and the Philippines.

Valley fill heap leach pad design in aggressive terrain results in high construction costs in most cases for several reasons. Earthworks represents up to 70% of total project costs in some cases. Other costs arise from the need for earthworks optimization and surface grading for soil liner and geomembrane placement; there is also significant settlement when massive fill platform is used or weak foundation remains, which requires a pre-camber to offset settlement. Difficulties in soil liner placement make it necessary to use geosynthetic clay liners (GCL) where soil liner cannot be placed. Geomembrane liner installation is complex on very steep slopes, and this complexity makes it necessary to ensure an efficient drainage collection system. Other costs are associated with the design of intermediate benches in areas cut for GCL installation and the reduction of leaching areas in first lifts in narrow valleys, which compromises initial mine production due to insufficient irrigation of the heap.

This paper presents experiences in designing and constructing heap leach pads in very aggressive terrain and considers technical criteria developed specifically for these conditions. These criteria have been successfully applied to heap leaching projects in Peru and also in Argentina, Mexico, and Colombia.

Introduction

The topography of some areas is very aggressive and unfavorable for heap leach pad design and construction. For example, mining projects in the Andes region typically operate at altitudes higher than 2,500 meters above sea level, where the only place available for a heap leach pad is usually a valley. In

such areas, the design of earthworks, liner systems, solution collection systems, and first lift stacking involve additional effort and specific design criteria that differs from those used in areas where terrain is much more favorable (for instance, in flat areas or at lower altitudes).

Based on our experience, several main aspects must be considered in designing heap leach pads in a region of very aggressive terrain: earthworks optimization, selection of areas for soil liner or GCL, pre-cambering design in massive fills or weak foundation, geosynthetics design and installation, intermediate benches, solution collection system design, and first lift issues.

Earthworks optimization

Earthworks optimization is one of the most important aspects of heap leach pad design in aggressive terrain; it helps to ensure proper stability for the entire facility and also helps avoid increasing construction costs. In very aggressive terrain, earthworks costs can represent up to 70% of total construction costs.

Foundation depth

Before starting the cut and fill work to grade the whole area for pad construction, a geotechnical engineer should define the foundation depth and determine excavation depth. Figure 1 shows a typical foundation plan, which is used by design engineers for generating the foundation surface and leach pad grading plans. This foundation plan is generated based on test pitting, which is carried out as part of the geotechnical field investigation program. The geotechnical engineer defines the appropriate foundation depth for reaching competent foundation soil in each test pit or trench, and then the design engineer uses this information to complete the final grading plan. The foundation plan must be optimized, especially at the base of the leach pad, to verify proper stability conditions for the whole heap.

After the grading plan is completed, the underdrain system design is developed. The final underdrain layout considers the final cut surface, which depends on a combination of cut surfaces between the foundation and grading.

Solution collection drainage plan

The leach pad grading plan must also include a drainage plan for collecting solution. This plan must direct drainage to the lowest spot of the platform in order to provide a clear idea of the required cut and fill zones. The grading plan should preferably ensure that solution drains to a single spot or the lowest spot of the pad platform at the toe of the heap. An adequate drainage system helps to ensure earthworks optimization by helping engineers accurately design cut and fill zones.

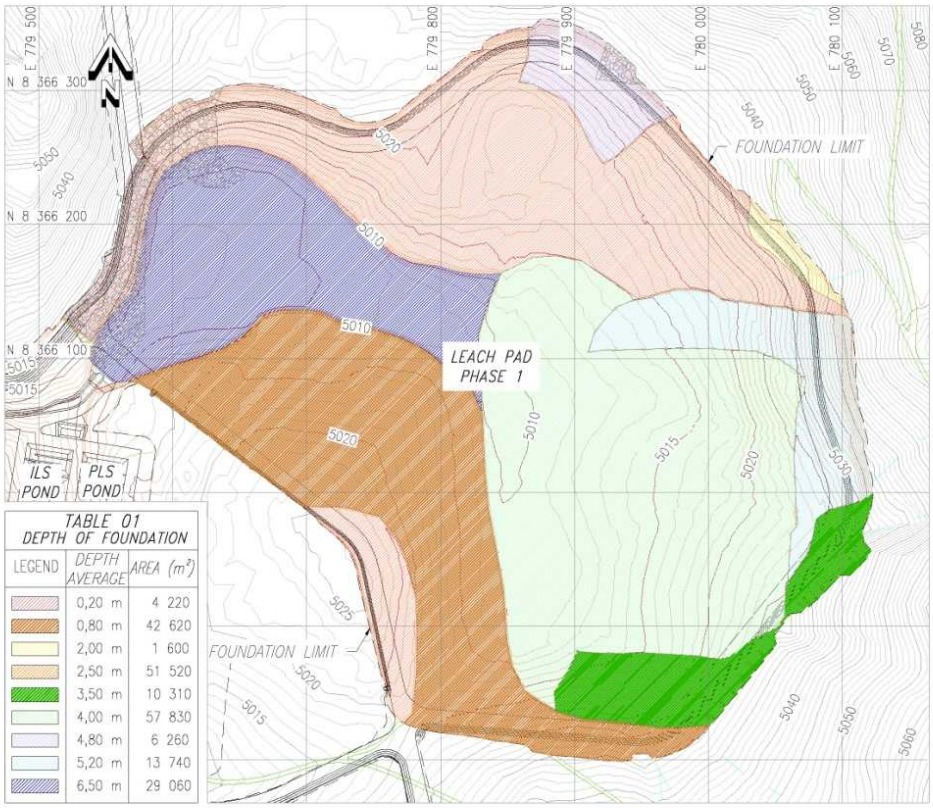

Figure 1: Typical heap leach pad foundation plan

Very steep slopes

Earthworks can also be optimized when the area has very steep slopes, as is common in aggressive terrain. In such cases, extra cuts are made to create a proper slope (usually less than 2.5H:1V or 2H:1V) to make it possible to place and compact low permeability soil (clayey soil).

In these cases, a well-shaped slope is preferable to massive cuts or excavations (Figure 2). This slope will be lined with low permeability soil or GCL, depending on how steep the final slope is, before a geomembrane is put in place. If GCL is placed, then its effect on heap leach pad stability needs to be addressed. A well-shaped slope reduces not only the requirement for earthworks but also the length of the construction schedule.

In most cases, very steep rocky slopes in aggressive terrain must be shaped or graded using blasting. As a consequence, is unlikely to get graded or regular surfaces in theses slopes, as shown in Figure 3. In such cases, irregular surfaces should be improved using shotcrete, mortar, soil-cement, or other products that improve the final grade. Depending on slope irregularity, another way to improve the surface is to use a protection system such as high-weight geotextile, geocomposite (geonet between two geotextile layers), or electro welded wire mesh before GCL and geomembrane installation. These methods reduce rocky surfaces with sharp edges or cavities that can damage the GCL and geomembrane through punctures or

deformation during ore stacking. Figure 4 shows some techniques used for grading irregular slopes and final surfaces for protecting the liner systems.

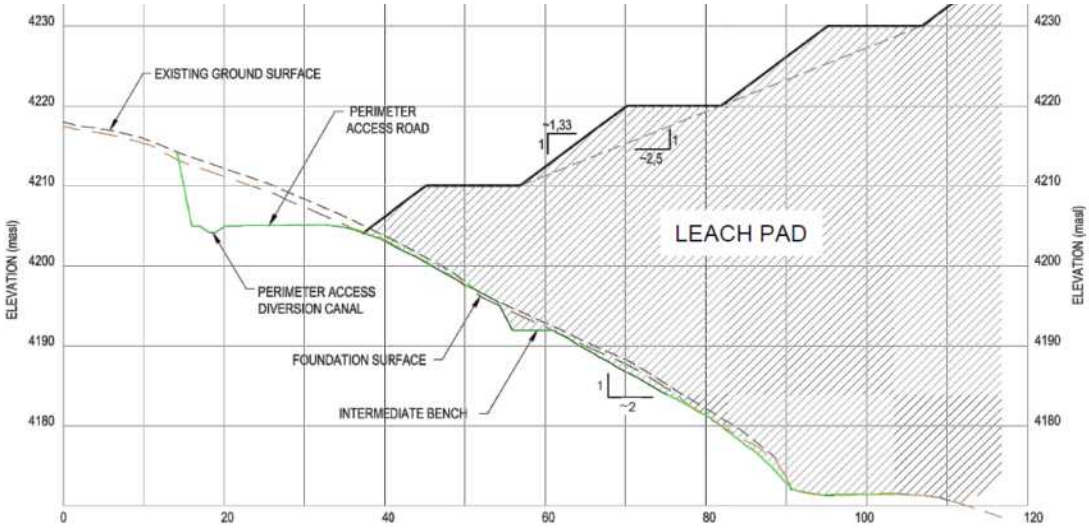

Figure 2: Typical grading (well-shaped slope) on a very steep slope in aggressive terrain

Figure 3: Very steep rocky slope with irregular shape after blasting

Figure 4: Typical grading and protection for geosynthetics installation on very steep slopes

Selection of soil liner or GCL

In deciding whether to use low permeability clayey soil liner or GCL in an area, the following general recommendations should be considered:

- Soil liner can be placed on slopes up to 1.5H:1V, but requires a winch system to pull the roller for compacting on the slope (see Figure 5), as indicated in Currie and Parra (2010). This is done mainly for safety reasons, even in cases where the roller can work on the slope without a winch system.
- Slopes steeper than 1.5H:1V must be lined using GCL.
- In some projects with safety restrictions on the use of winch systems or with untrained staff, soil liner must be placed on slopes lower than 2H:1V or even 2.5H:1V.

Figure 5: Soil liner compaction on a very steep slope

Surfaces with very aggressive terrain and very steep slopes involve additional aspects that should be considered in evaluating whether to use GCL or soil liner. When slopes are very steep – greater than 1.5H:1V or even 2H:1V, in some projects – it can be impossible or very difficult to place and compact low permeability soil (typically 300 mm). Other considerations include optimization of earthworks cost when there are restrictions or difficulties in obtaining soil liner, and reduction of the construction schedule.

Some projects in Peru have been developed where there were considerable restrictions on obtaining soil liner. These restrictions include a lack of available borrow sources close to the project; available borrow sources a long distance from the construction site, which increases hauling costs and,

consequently, the overall cost of soil liner placement; available borrow sources, but in wet conditions and a rainy environment, which makes them impractical to use. These restrictions often lead engineers to optimize earthworks with the objective of installing GCL in areas where the slope is not necessarily very steep. Table 1 compares the use of soil liner or GCL for earthworks optimization.

There are several technical aspects that must be taken into consideration when using GCL:

- GCL can be used on steep slopes with a previous verification of heap leach pad stability.
- GCL must not be used in platforms or lower zones in the leach pad because its shear strength is much lower than that of soil liner.
- Reduction of GCL shear strength through hydration must be considered during laboratory testing for shear strength determination. During heap leach pad operation, GCL is likely to be hydrated by solution leakage through the liner system or by underground water, making the shear strength even lower.

Figure 6 shows a typical project drawing for a region where soil liner availability is restricted; the design criteria call for installing GCL as much as possible, including in some areas of the leach pad with slopes lower than 2.5H:1V.

Table 1: Comparing low permeability soil liner and GCL

Low permeability soil liner	GCL
Generally the most economical alternative as long as slope is lower than 2.5H:1V or even 2H:1V and proper borrow sources are available close to the project.	More economical on irregular slopes after blasting and when soil liner is not readily available close to leach pad.
Can be used on slopes up to 1.5H:1V. Restricted in some places to 2H:1V or 2.5H:1V.	Can be used in any slope of leach pad with verification of overall geotechnical stability.
Can be placed directly on subgrade and structural fill, according to the grading design.	In steep rocky slopes with irregular shapes, must be placed in conjunction with geocomposite or geotextile to avoid punctures.
Can be placed on very irregularly shaped subgrades.	Because of length of roll, intermediate benches have to be designed and constructed for anchoring.
Does not require intermediate benches unless solution collection system needs them.	Vulnerable to hydration by phreatic level in foundation, decreasing its efficiency as protection.

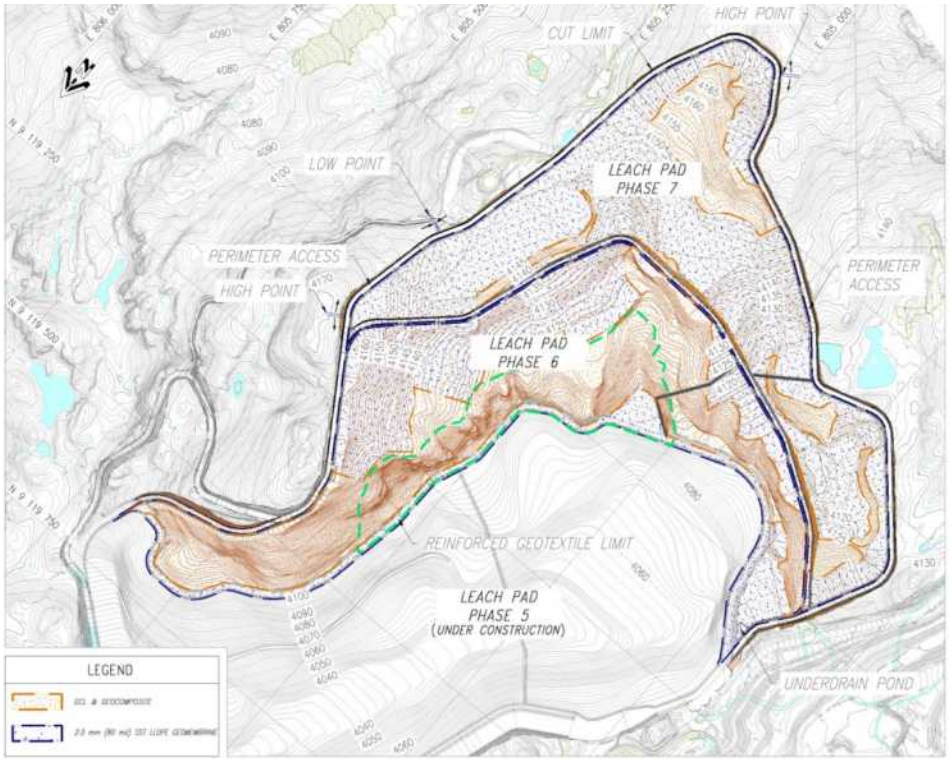

Figure 6: Optimized area for GCL utilization

Pre-cambering in weak foundation or massive fills

Pre-camber design as part of the grading plan can be considered in the following two scenarios:

- When the foundation is very deep in large areas of the leach pad, conventional design encourages removing all unsuitable soil and then backfilling with borrow material up to the final leach pad subgrade; this significantly increases construction costs. In this case, a deformation analysis is needed to estimate both the minimum practical thickness of unsuitable soil to be removed and replaced and the settlement to be expected during heap operation because of the remaining weak soil foundation. Settlement of the still weak foundation is used to configure a pre-camber as part of the grading plan.
- When a large fill platform is needed for proper stability or to increase the area available for first lift stacking, deep fill is needed for the platform and in order to reduce costs. In this case, a massive fill with 0.5 or 1 meter lift is used and compacted with large compaction equipment. As the fill is not structural, settlements (significant in some cases) are expected to occur during ore stacking (by ore weight); these are estimated based on a deformation analysis and used for pre-cambering in the leach pad platform as part of the grading plan.

In both cases, pre-cambering will help to maintain the integrity of the geomembrane and the efficiency of the solution collection system.

A case similar to the first scenario is located southern Peru. Here, there was 20 m of loose granular soil in a large area of the projected leach pad. Deformation analysis indicated that at least 4 m of this unsuitable soil had to be removed and replaced by a 3 m rockfill (as stabilization material) and a 1 m structural fill. At this condition, 0.6 m maximum settlement was calculated through a strain-stress PLAXIS analysis, using maximum heap height; this value was used for configuring pre-camber in the settlement area, with significant savings in capital expenditures (capex). Figure 7 shows a detail of the pre-camber as proposed in the engineering design; rockfill and structural fill are shown along with the final subgrade projection considering the expected settlement.

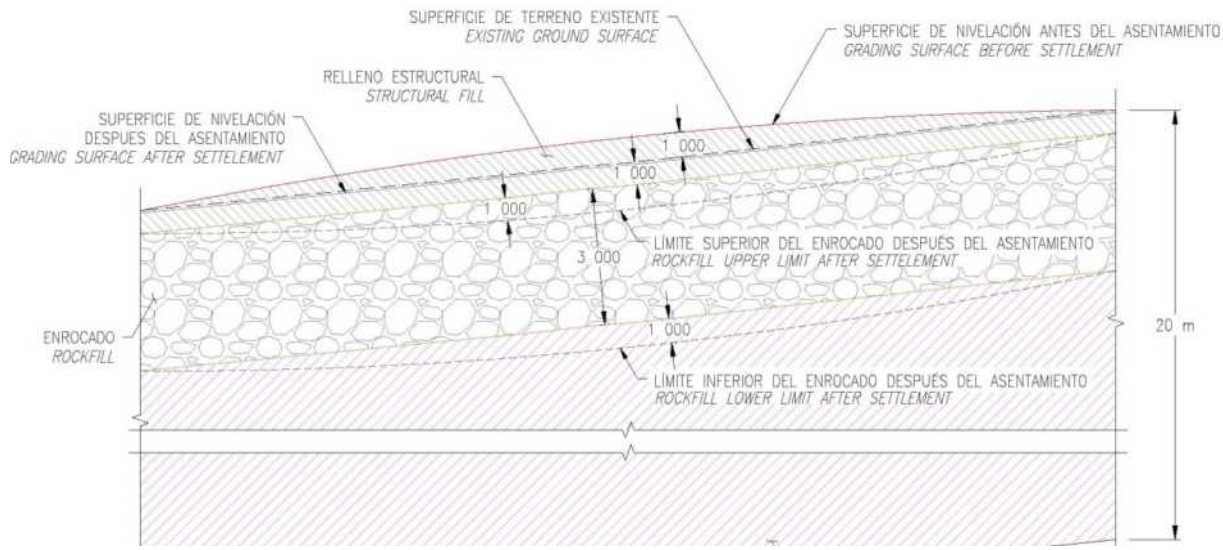

Figure 7: Pre-camber design in a structural fill over weak foundation

Geosynthetics design and installation

Geosynthetic installation in very aggressive terrain needs a proper design, as described below. In areas where the final slope after grading is steeper than 1.5H:1V (or lower in some places), GCL must be installed; the edge will be buried in an anchor trench and the roll deployed along the slope (see Figure 8). In order to protect GCL from punctures on rocky slopes, a protection geocomposite or another protection system must be used. The GCL and geocomposite must be installed in those areas of the leach pad where overall stability is not compromised. These zones will be identified as part of the design process and verified during construction.

Soil liner, GCL, or a zoned combination of both will be placed as a second containment layer on prepared subgrade. The first and main containment will be the geomembrane liner, which will be installed

over the soil liner or GCL. Based on our experience in very aggressive terrain, low linear density polyethylene (LLDPE) geomembrane, single-side textured, is recommended for heap leach pad application. The textured side must be placed in contact with the soil liner or GCL to improve the shear strength of this interface. LLDPE geomembrane shows better performance against punctures and higher interface shear strength when compared with high density polyethylene (HDPE). The optimum thickness of the geomembrane depends on its susceptibility to punctures. However, the following practical rule can be used: use 1.5 mm for heaps up to 100 m and 2.0 mm for higher heaps. If the heap leach pad is very deep, then a geotextile protection is better and cheaper than a thicker geomembrane (2.5 mm).

In very aggressive terrain with very steep slopes, geosynthetics installation is very complex. Because of the deployment of heavy rolls is performed downslope and the geomembrane welding is done many times with the operators hanged on the slope, it is extremely important that the installers are experienced in order to guarantee installation quality and safety. Figure 8 shows complex geosynthetics installation work.

Figure 8: Complex geomembrane and GCL liner installation on very aggressive terrain

Intermediate benches

As noted, because steep slopes cover large areas in many leach pads, intermediate benches are sometimes needed. Figure 9 shows a typical section of an intermediate bench. Bench layout is designed based on a maximum-length GCL roll (typically 45 m); however, when only a soil liner is used, the distance between benches is determined by the maximum length of the geomembrane roll, in order to avoid horizontal seams. Typical roll length is 150 m for 1.5 mm or 120 m for 2 mm geomembrane; however, longer rolls can be ordered before manufacturing to minimize benches. These benches provide the following advantages:

- The leach pad can be designed using a phased construction concept, incorporating as many benches as needed in each phase, depending on the capacity requirements for each phase.
- The solution collection system can be designed independently in each phase, making the system more efficient, with no connection to the previous phase; the solution can be diverted directly to the process ponds.
- Rainwater control is efficient and avoids the entrance of these flows into the heap leach pad, which would dilute the solution and make metal recovery at the plant more complicated and costly.

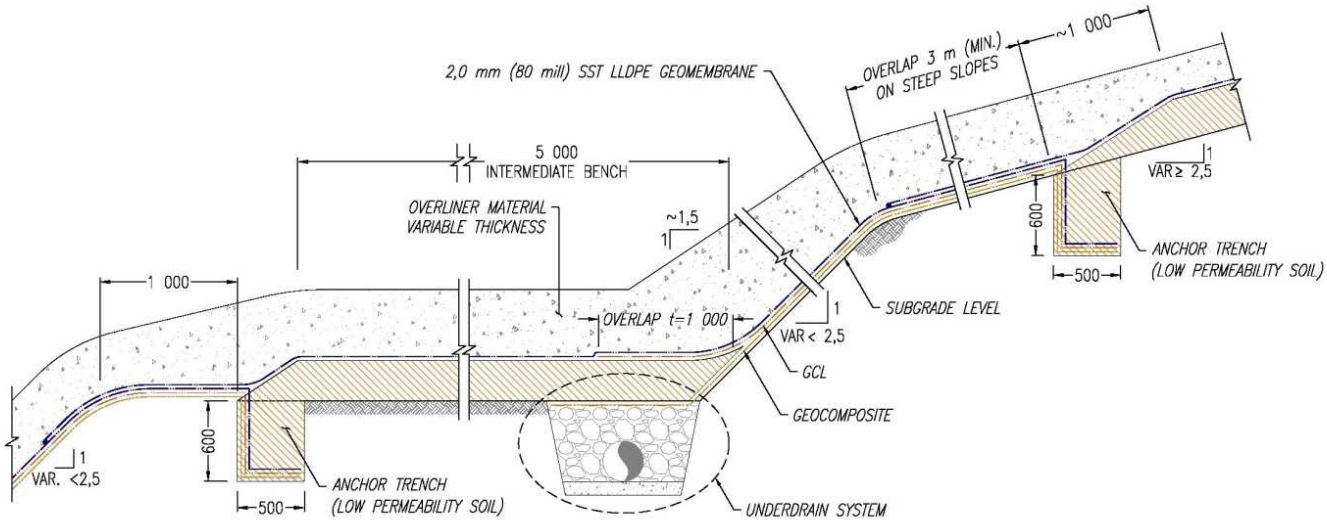

Figure 9: Intermediate bench: geosynthetics anchorage

Solution collection system

Solution collection system design in very aggressive terrain is governed by the following considerations:

- Collection pipes should be placed in areas where slopes are lower than 4H:1V. Installation in slopes steeper than 4H:1V is not necessary because collection will not be efficient.
- Intermediate benches should be used for evacuating solution from subsequent phases directly into the process pond, or even to a process plant if gravity makes this possible. This is a good practice, verified in several mining operations, and results in a more efficient system.

Figure 10 shows a plan view of a solution collection system in very aggressive terrain with intermediate benches.

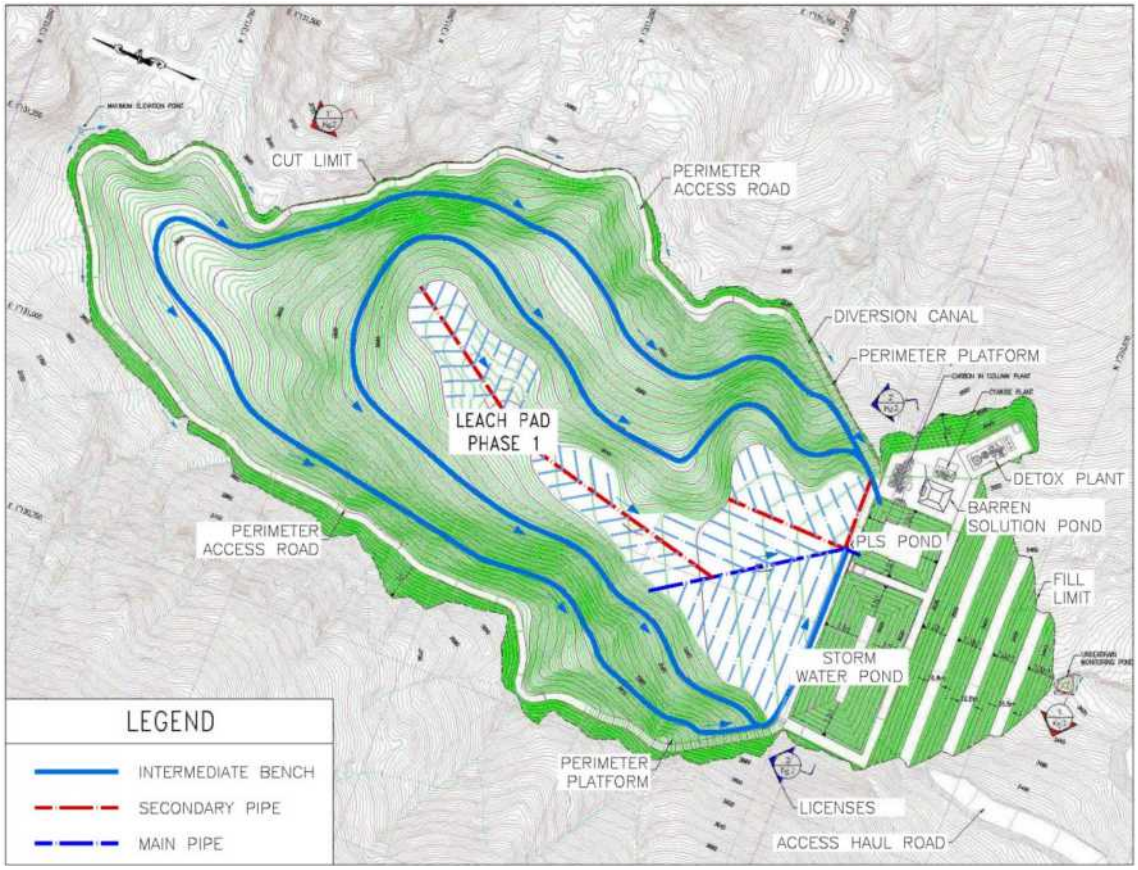

Figure 10: Solution collection system in very aggressive terrain

Issues with first lifts

In valley heap leach pad design, there are generally issues with first lifts associated with narrow valleys. During stacking, it is common to have a mismatch between the capacity (tonnage) and leach area of first lifts for initial irrigation. In this situation, operators usually have three options:

- Restrict ore stacking, and therefore production, based on the area available, and irrigate just the available area.
- Stack two or three lifts to obtain the minimum area for leaching, based on the ore leach cycle, and then start irrigation.
- Combine the two previous options.

In all cases, production will be restricted to the minimum leach area as long as the valley is progressively filled.

Figure 11 shows a view of an ore stacking plan, where problems in the first two lifts are observed; Figure 12 shows a typical section of a valley fill heap leach pad.

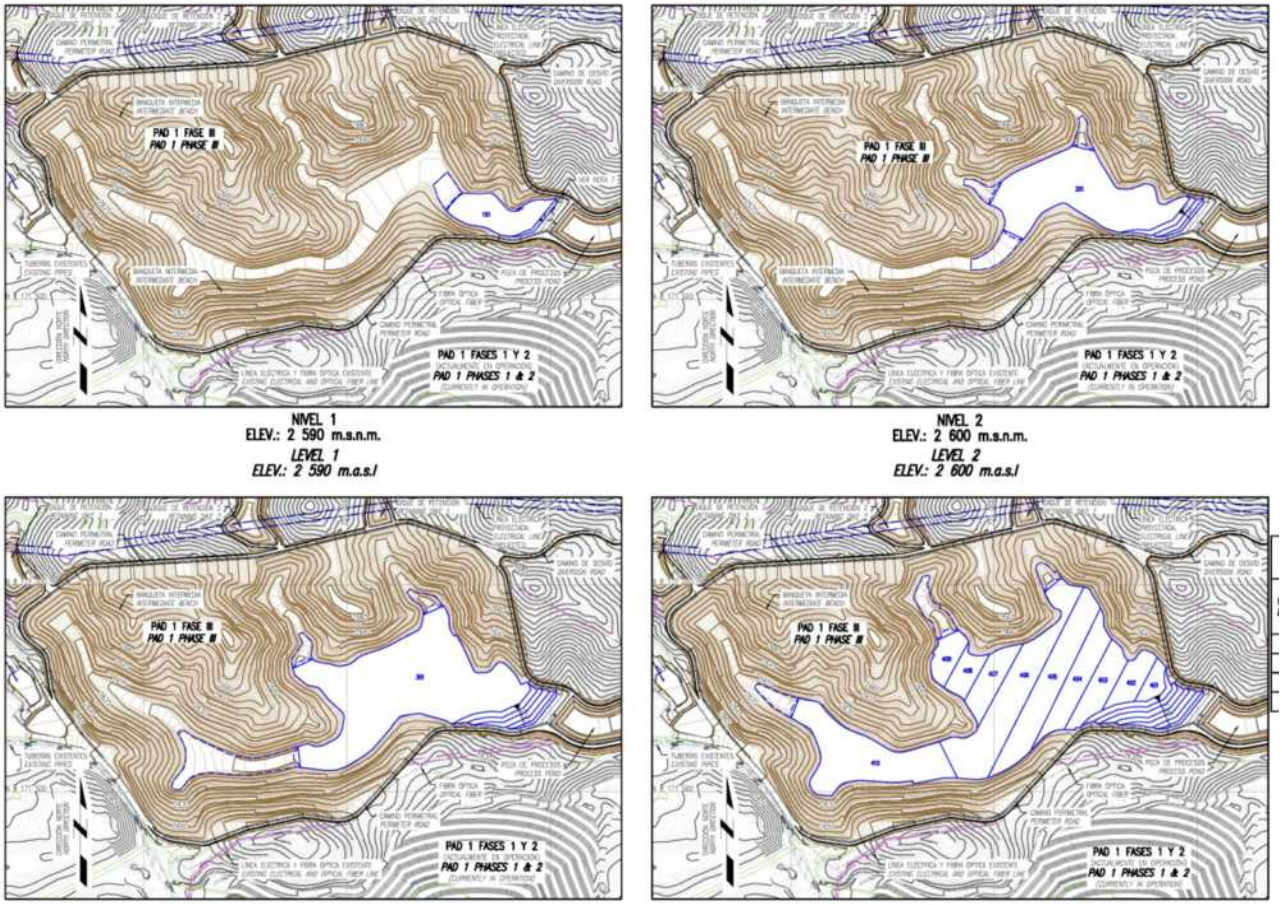

Figure 11: Ore stacking plan in a valley fill heap leach pad

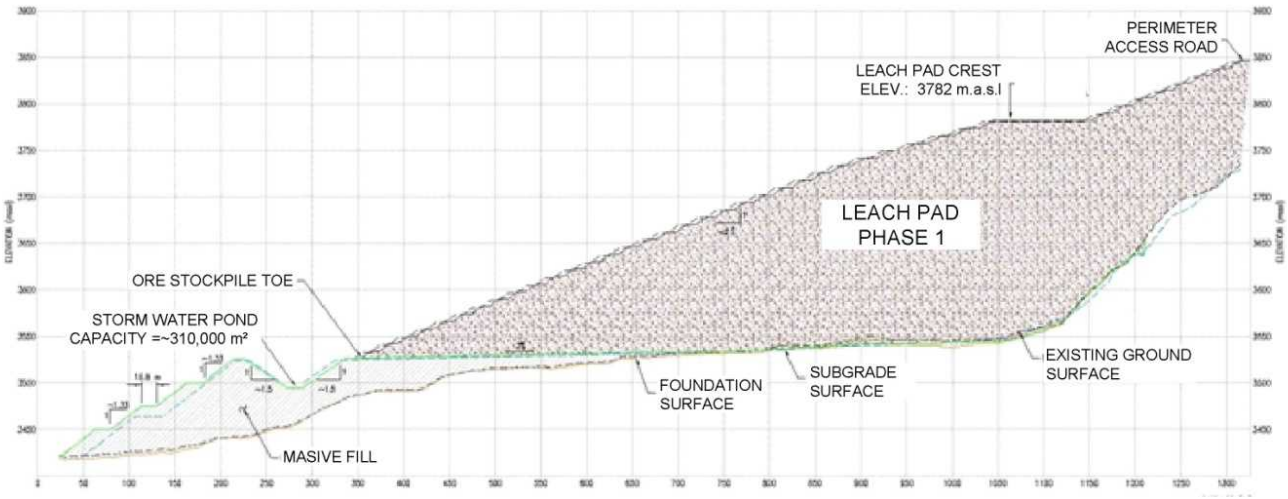

Figure 12: Typical section of a valley fill heap leach pad

Conclusions and recommendations

The following conclusions and recommendations are based on the developed work:

- When low permeability soil liner is not available close to the project or is limited because of high costs for obtaining or preparing the liner, it is important to perform a comparative cost analysis between soil liner and GCL.
- When using GCL is an option for a leach pad project in very aggressive terrain, earthworks optimization should be performed to reduce the overall cost of the project and the construction schedule.
- In areas with weak foundation or massive fills, a pre-camber design becomes very useful for avoiding potential damage to the geomembrane and solution collection pipes.
- GCL should be protected from punctures by geocomposite, geotextile, or another system.
- LLDPE geomembrane, single-side textured, is recommended because of its better performance against punctures and higher interface shear strength compared with HDPE.
- The installer's experience is extremely important for guaranteeing installation quality and proper safety conditions in very aggressive terrain.
- Intermediate benches allow phased construction, independent solution collection systems, and efficient rainwater control.
- Collection pipes should be placed in areas where slopes are lower than 4H:1V.
- In valley fill heap leach pads, there are usually first lift issues associated with valley narrowness and small leach areas. Restrictions on initial metal production must be considered by the mining operation.

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Design and construction review of a heap leach pad for safe operation

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Abstract

This paper presents a design and construction review of a heap leach pad located in northern Peru, based on several aspects of interest that have been observed and should be taken into account in future heap leach pad projects.

The review consisted of compiling engineering design and construction (quality dossier) information, focusing mainly on data related to soil liner characteristics and the interface with the leach pad geomembrane. This review was complemented with field and laboratory investigations. Available information indicated a high variability of soil liner in the borrow source area, corresponding with low shear strength silty soils. Moreover, there was a significant difference between the basic properties assumed in the design of the soil liner and the results of the quality dossier, which were similar to those obtained in additional tests performed as part of this review. Likewise, large-scale direct shear testing of the interface between the soil liner and the geomembrane in samples of both materials obtained in the field showed significant differences, with lower interface shear strength than that assumed in the design phase.

During the design phase, a generally stable heap leach pad condition was observed, but with pseudo-static factors of safety below 1, and with limited seismically-induced permanent deformation under the Makdisi and Seed (1977) criterion. On the other hand, stability analysis performed with shear strength parameters obtained by testing *in situ* conditions indicated unstable conditions of the heap leach pad, with relatively low static factors of safety and pseudo-static factors of safety also below 1, but with seismically-induced displacement at greater than permissible values under both the Makdisi and Seed (1977) and Bray and Travasarou (2007) criteria; this led to the design of a toe buttress to obtain long-term stable conditions.

It is necessary to perform a design review after construction, or even better, during construction, in order to verify, update, modify, or improve the original design, as needed. This review must be a mandatory process performed as part of mining operation start up, in order to avoid future heap leach pad instability problems, which may lead to environmental consequences (geomembrane failure), where remediation can be expensive.

Introduction

Nowadays, mining companies in Peru are required by the national mining authorities to perform a detailed design review of their mining-related structures in the country in order to safeguard the environment from any further instability problems that may have not been forecasted during the construction or design phases due to bad practices.

The authors of this paper were involved in reviewing such a project: the design of a heap leach pad that is currently under operation. A plant view of this facility is shown in Figure 1. The review work consisted mainly of analyzing soil liner properties during the design and construction phases.

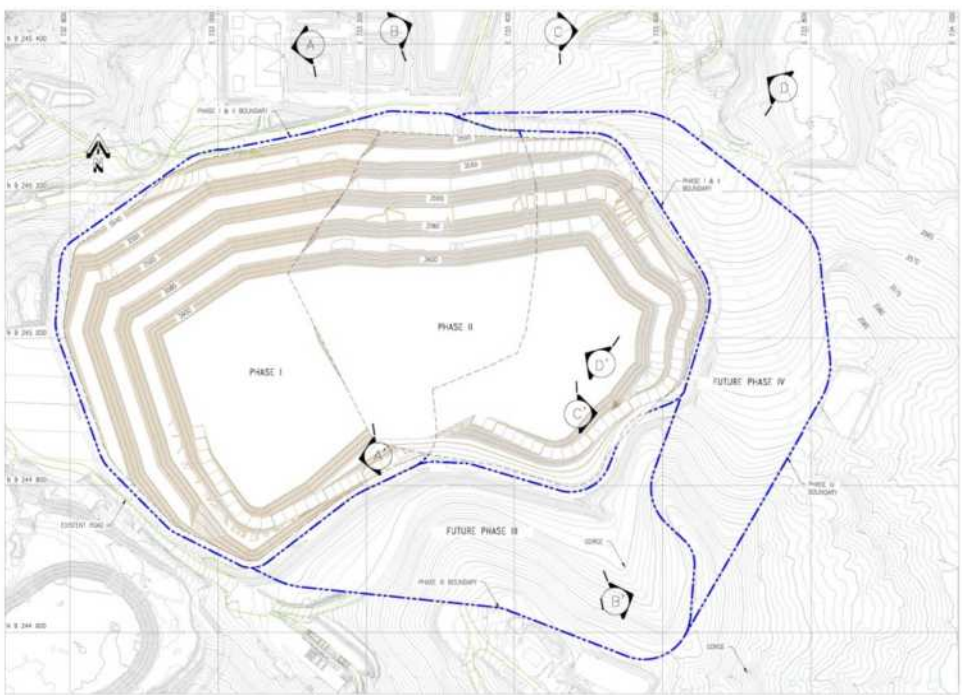

Figure 1: Plant layout of analyzed leach pad, Phases I and II

Soil liner properties were the main scope of the present paper since, in most leach pad projects around the world, interface characteristics are among the most important geotechnical constraints related to leach pad design. In this particular case, the interface consisted of a soil liner and single-sided textured geomembrane.

Construction phase data was obtained from a quality dossier provided by the mining company during the design review phase. The quality dossier conveyed laboratory information such as soil particle size and classification, plasticity index, liquid limit, standard Proctor dry densities, and the results of field tests, such as nuclear gage or sand cone tests for compaction verification. Additional data relating to distances between compaction control field tests and standard Proctor sample points was generated from the quality dossier in order to have an idea of the accuracy and frequency of compaction control.

Construction quality problems

The soil liner was sampled during the geotechnical investigation for revision of the leach pad design. One of the main issues observed during sampling was the soil liner's USCS classification (Unified Soil Classification System) which was mainly a low plasticity silt with sand (ML). This is generally not a proper material for soil liners, since it is likely to offer low interface strength. Another issue tackled during the geotechnical investigation was a comparison between the standard Proctor tests and density measured in the field, based on undisturbed soil liner sampling (shown in Figure 2) and testing.

Figure 2: Extraction of undisturbed soil liner sample for *in situ* block density and large direct shear testing on textured geomembrane and soil liner interface

Several large-scale direct shear (LSDS) tests were performed on textured geomembrane and soil liner interfaces on disturbed and undisturbed samples. These test results were compared to provide further insights on the differences in strength between the *in situ* and laboratory conditions for this kind of man-made soil material. It should be noted that these disturbed samples tested at 95% of standard Proctor density.

As shown in Figure 3, the tests performed on disturbed and undisturbed samples in this study show very low variability compared to tests performed by the designer, which are shown in Figure 4. Figure 3 shows a similar quality of strength across all soil liner tests from samples taken during the geotechnical investigation of the design review phase. Meanwhile, Figure 4 shows some randomness in interface strength over time, not only during the design phase, but mainly during the construction phase. In Figure 4, the strength differences between the design and design review envelopes are very clear; further research that may confirm this huge difference in behavior may account for future problems in leach pad slope stability.

The dossier data obtained at the leach pad was analyzed and compared to the initial data obtained at the design and design review phases. The outcome of this comparison was a huge variability in soil properties during construction time. Figure 5 shows differences of fines content during construction time; the soil liner may vary from silt to sandy or gravelly silt, depending on the gravel and sand content. Figure 6 shows the effect of this variability on standard Proctor points; many problems in controlling material used as soil liner during construction may be due to the high variability of the material obtained from the borrow area.

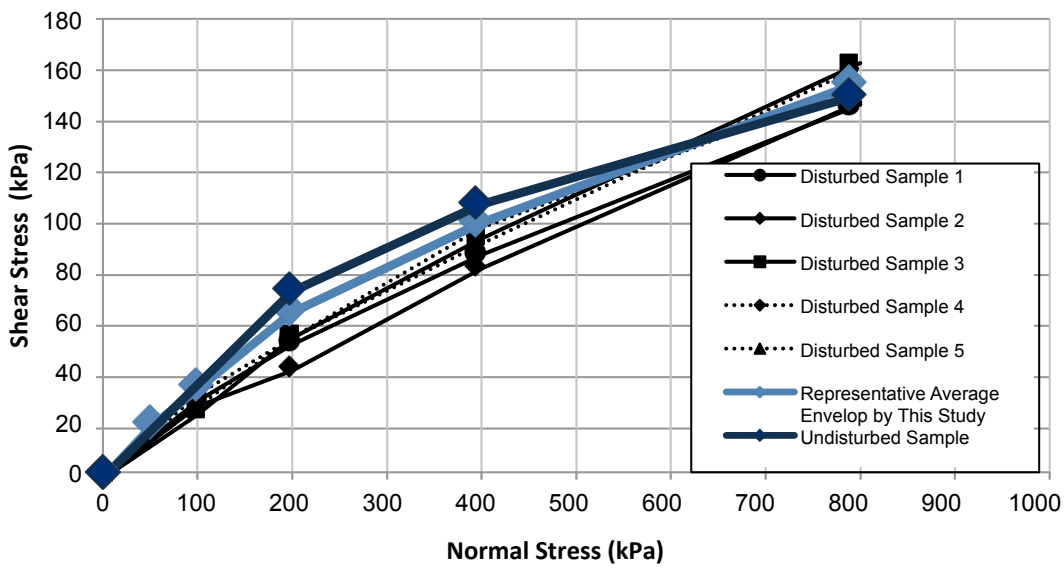

Figure 3: Soil liner shear strength interface envelopes by LSDS in this study

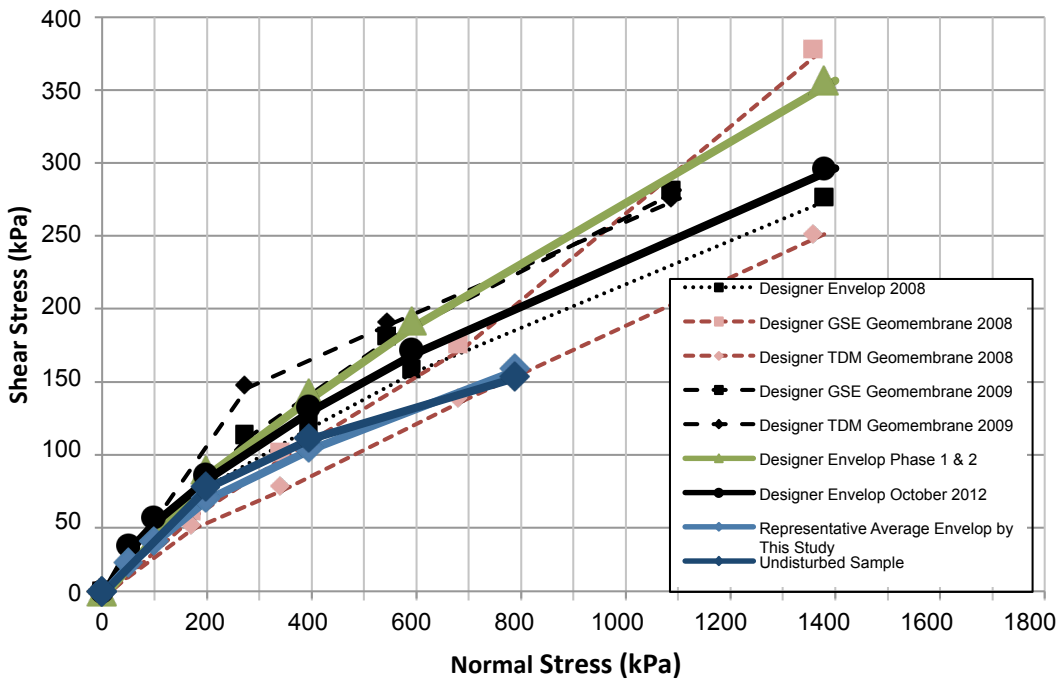

Figure 4: Soil liner shear strength interface envelopes by LSDS by designer. Green envelope was used for leach pad slope stability analysis during design phase

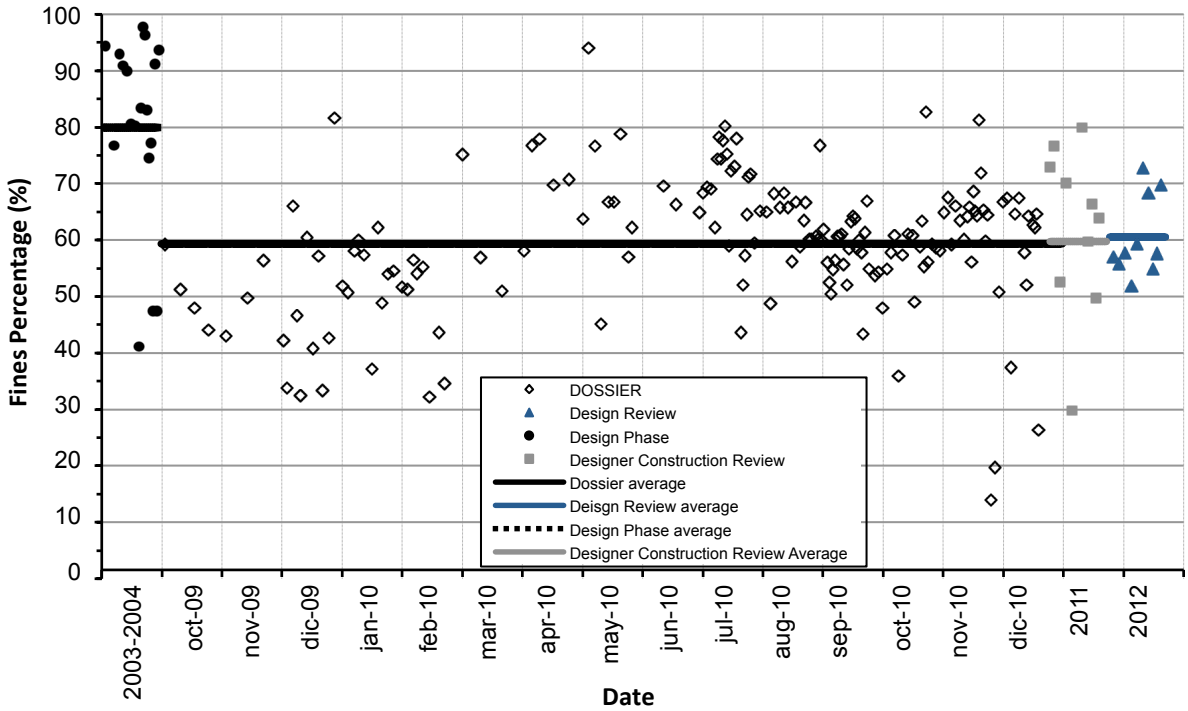

Figure 5: Fines content variation during construction

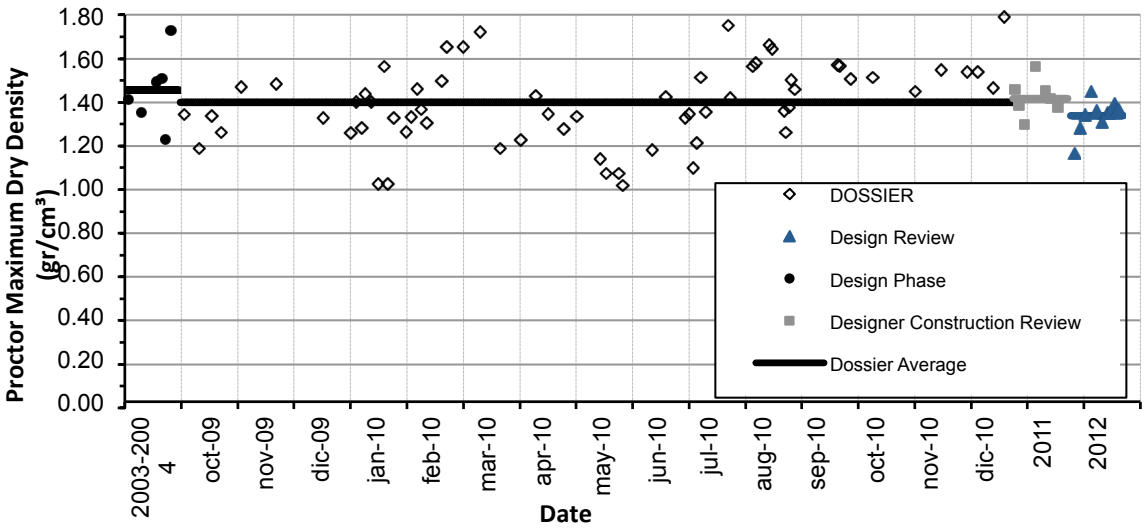

Figure 6: Standard Proctor maximum dry density variation during construction

Finally, another problem that arose while analyzing the dossier data was the distance between standard Proctor control sample points and the compaction control field test points, as shown in Figure 7. The average distance between control sample points and density field tests was about 180 m, which is an unacceptable distance from the laboratory control sample. Construction quality control (CQC) manuals in the industry recommend, at most, 250 to 500 m³ for performing field density tests, and 1,000 to 1,500 m³ for performing laboratory standard Proctor tests, which means a radio distance of around 30 to 40 m between the control sample and the field density test for a soil liner layer of 0.30 m. Since most soil liners are cast over rectangular cell shapes in the field, a maximum average distance of 150 m can be considered for shapes of 20 m in width.

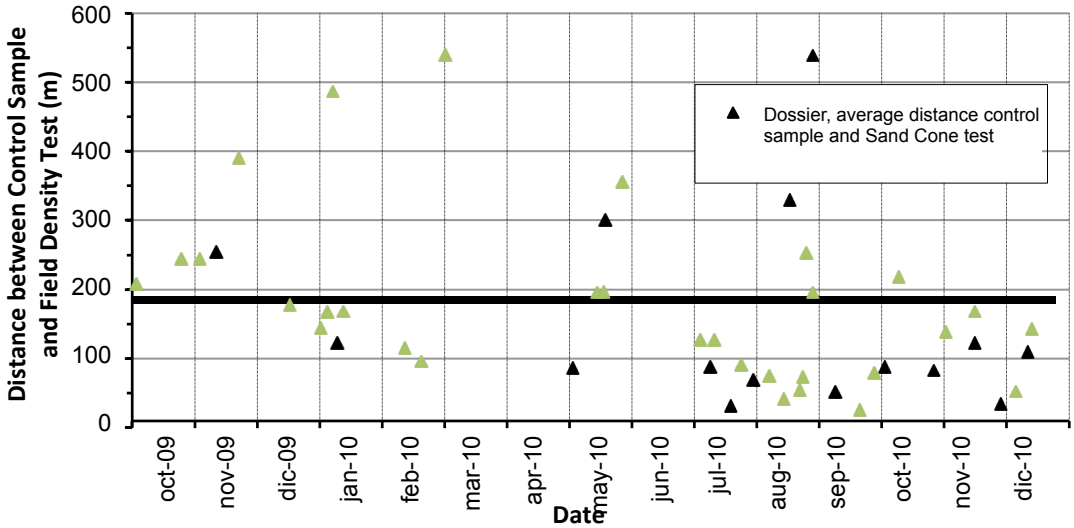

Figure 7: Distances between laboratory control samples versus field tests samples during construction time

Design and construction follow-up problems

In general, all construction work requires a construction quality assurance (CQA) engineer to be responsible for the control quality assurance of the structure involved. This leach pad had a permanent CQA engineer, who was aware of the design requirements for the soil liner and the properties that were obtained during the design phase.

Figure 7 suggests that the minimum amount of lab and field tests required by quality control manuals were not totally met during the leach pad construction. In addition, there were many significant differences between the soil liner properties at the design phase and the construction phase, as shown previously in Figures 5 and 6. One major difference observed is the relatively lower Proctor dry density presented in the soil liner during construction.

As a good practice, the CQA engineer must communicate these variability issues to the designer in order to test important parameters, such as permeability and shear strength, which may change due to the high variability of the borrow area material. Prior testing can help an operation avoid or prevent instability problems, which are easier to overcome in the early stages than at later phases of construction. A good comparison with another project where the soil liner borrow area material did not have high variability is shown in Figure 7; this shows a minimum difference in Proctor maximum dry densities during the construction time.

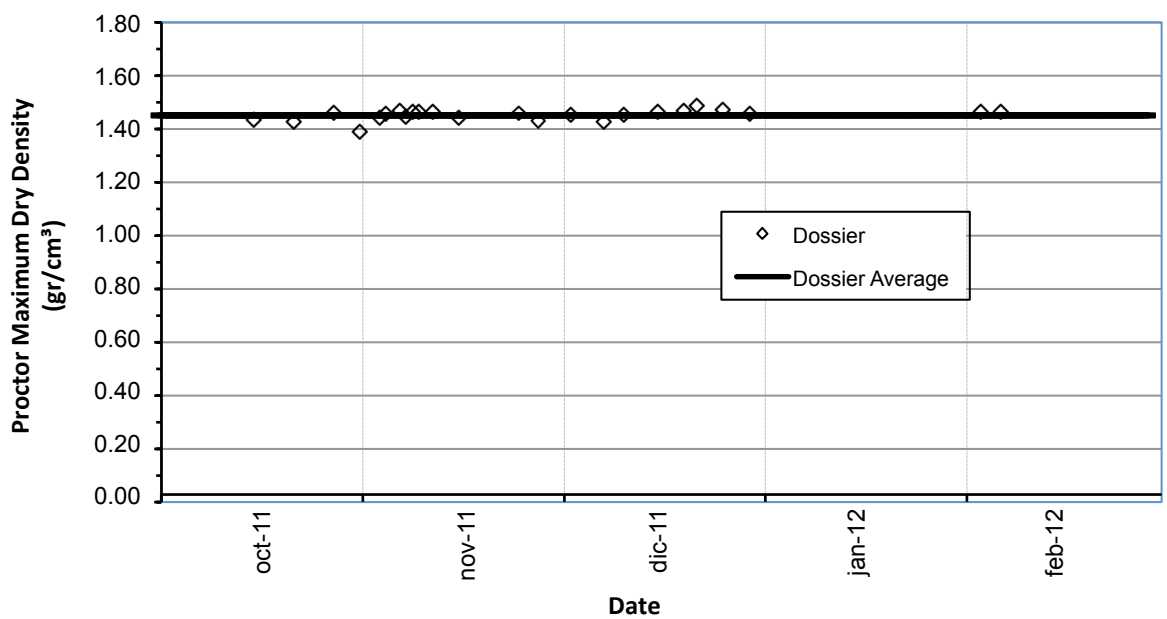

Figure 8: Standard Proctor maximum dry density variation during construction of a leach pad in central-southern Peru

Slope stability problems

The leach pad in this study was founded over an argillic tuff and residual soil without any major fault features; analysis of a circular failure under static and pseudo-static conditions, performed during the design and design review phases, confirmed proper leach pad foundation stability. However, block failure through the textured geomembrane and soil liner interface has developed from the toe of the leach pad following the soil liner path, and there are breaks at some points of the leach pad crest; this is why choosing a good borrow material for the soil liner becomes a critical issue for leach pad stability.

During the design phase, the leach pad was analyzed using the green strength envelope shown in Figure 4, and it was concluded that the leach pad was stable based on permanent deformation criteria (less than 30 cm) using the Makdisi and Seed (1977) method shown in Table 1. During the design review phase, parameters for ore and soil liner were updated by reviewing previous information and performing new tests, as shown in Figure 3. A comparison of the differences in the strength of the soil liner envelopes at the design phase and the design review phase is shown in Figure 4; these differences are based on the high variability of the borrow area material used as a soil liner. The results of the design review phase are shown in Table 2.

Table 1: Results of stability analysis of leach pad, design phase

Cross section	Static factor of safety	Yield acceleration (g)	Makdisi and Seed (1977) permanent deformation (cm)
A–A	1.30	0.080	30
B–B	1.33	0.088	25
C–C	1.38	0.085	25
D–D	1.55	0.110	20

Table 2: Results of stability analysis of leach pad, design review phase

Cross section	Static factor of safety	Yield acceleration (g)	Makdisi and Seed (1977) permanent deformation (cm)	Bray and Travasarou (2007) permanent deformation (cm)
A–A	1.22	0.082	40	53
B–B	1.20	0.072	54	65
C–C	1.31	0.123	14	31
D–D	1.38	0.113	20	36

As shown in Table 2, leach pad stability is endangered under permanent deformation as calculated by the Bray and Travasarou (2007) method, which is one of the most documented and studied updates of the Makdisi and Seed (1977) method, according to Bray (2007).

For static conditions, it is common in the industry to use a factor of safety of 1.5 in highly seismic regions such Peru; however, the designer chose a minimum static factor of safety of 1.3. Furthermore, this factor of safety was not met in two cross sections analyzed, which indicates not only a long-term seismic-related instability, but also a probable instability during the operation.

Results of factors of safety and permanent deformations obtained during the design review phase are critical for future phases of the project, since research data shows that soil liner and textured geomembrane liner lack the proper shear strength for static and long-term seismic stability. One proposed solution is to raise the static factor of safety and reduce permanent deformations for the leach pad, as shown in Figure 9, by lifting a buttress at the leach pad toe. This buttress can be constructed with spent ore and can be built in two phases, by first building a relatively small one to assure stability during operation, and later increasing the buttress size to ensure long-term stability for the closure stage. This solution will involve a large earthworks activity, since the buttress is as large as 200 m long and 16 m high; this activity does not consider any cost previously accounted during construction or operation.

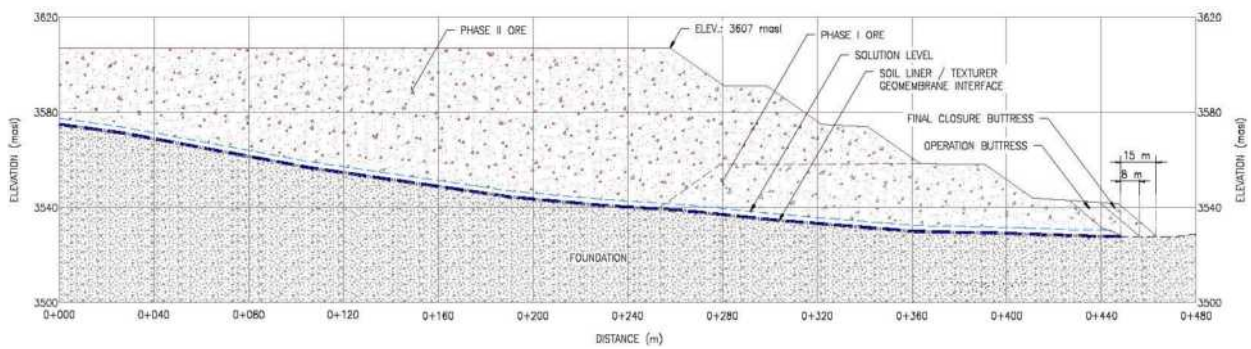

Figure 9: Leach pad cross section showing design configuration and the buttress proposed at toe

Other issues

The following issues have also been observed during heap leach pad construction and operation:

- Geomembrane quality should be verified before the product is shipped to the construction site. Once the geomembrane arrives at the site, problems detected before installation are very hard to resolve, and usually lead to construction schedule delays. Manufacturing quality assurance (MQA) is recommended for verifying the use of proper resin, measuring the minimum thickness based on project technical specification requirements, verifying quality control conformance testing and testing frequency, and so on.

- As part of the MQA, proper geomembrane texture needs to be verified; this should be similar to that used for interface shear strength testing during the design phase. If the texture is different or not as aggressive as expected, then LSDS must be performed and stability verified, if needed.
- Ore variability is very common in heap leach pads during operation, and ore properties are often not the same as assumed in the design phase, which is based on testing performed on a limited amount of samples; therefore, these properties should be verified frequently, depending on production rate and ore variability. The testing program should include global grain sieve analysis, plastic index, triaxial shear testing, staged hydraulic conductivity, point load tests, and geomembrane puncture tests.
- As engineering design is performed based on a limited field and lab testing program, an aggressive geotechnical monitoring instrumentation should be developed and installed in the leach pad and ponds. Vibrating wire piezometers, stand-pipe piezometers, inclinometers, vibrating wire settlement sensors, load sensors and strong motion accelerographs should be installed, depending on leach pad construction and operating conditions.
- Anomalous conditions of heap leach pad and related facilities have to be detected as part of daily, weekly, or monthly routine inspections and reported to the engineer of record. Annual safety inspections by a senior geotechnical engineer, including geotechnical instrumentation data review and analysis, should be part of the common operating practice.

Conclusions and recommendations

- A new frequency of soil liner testing must be established in order to identify material variability detected during construction.
- CQC and CQA crews must be able to identify potential problems in materials in general, and in soil liners in particular, in order to prevent potential leach pad stability problems.
- The CQA engineer should be aware of any changes in borrow area material, in order to prevent high variability of soil properties and strength differences during construction compared with those used for design.
- QC and QA in the toe area of the leach pad must be much stricter than in the rest of the facility, as the stability condition of the entire facility is mainly controlled by the shear strength of the soil/geomembrane interface in this area.
- Any high variability of material properties in borrow areas, especially where these are used for the soil liner, must be communicated as soon as possible to the designer, in order to ensure that this issue will not compromise leach pad stability.

In general, the strength properties of the soil liner and the geomembrane interface must be tested periodically during construction by performing LSDS, with a higher testing frequency for the material placed in the leach pad toe, in order to verify leach pad stability. The soil liner should also be tested for hydraulic conductivity. This is a very small investment that may prevent further problems requiring costly remediation after construction is completed.

- Bray and Travasarou (2007) present the most up-to-date method for calculating simplified seismic permanent deformations on earth structures; this is highly applicable to leach pad analysis and the failure mode it represents.

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Options for post-closure heap leach fluid management

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Abstract

Closure of heap leach pads in Nevada, USA and other semi-arid western United States climates typically includes regrading of the top surface of the pad to shed stormwater runoff, followed by placement of several feet of soil or growth media cover to eliminate the potential for meteoric waters to contact the spent ore. The cover also serves as a growth medium for vegetation, which limits downward migration of meteoric water through storage in the cover interstices and eventual release via direct evaporation or vegetation uptake and transpiration.

Regardless of the soil cover thickness and the vegetation component, some amount of infiltration is typically predicted by unsaturated zone modeling for:

- mine sites at elevations with relatively high annual precipitation and consequential infiltration; and/or
- mine sites where winter precipitation falls as snow and infiltrates into the heap system as spring snowmelt.

The use of soil covers at these sites (in predictive modeling and in practice) typically results in ephemeral effluent flows from the heap during and following spring snowmelt and sufficiently large precipitation events. Predicted and actual long-term effluent flows tend toward low to no flow for the rest of the year.

Closure operations typically include management of these flows in "evapo-transpiration" (ET) cells and evaporation (E) ponds, constructed by converting existing double-lined process water ponds according to a pre-determined design. Current State of Nevada guidelines for closure of E and ET facilities requires operation until the flows reduce to a "*de minimus*" value whereby non-degradation of groundwater can be demonstrated and final closure thus achieved.

Predictive modeling of commonly-used soil covers typically provides results that are not *de minimus*, requiring bonding for relatively low initial capital costs for process pond modification to ET cells, and long-term post-closure fluid management. In contrast, closure stabilization through installation of a low permeability cover requires bonding for relatively high initial capital costs and low short-term post-closure fluid management costs.

This paper describes methods of reducing long-term process fluid stabilization (PFS) costs via capital expenditure (for facility construction and PFS bond costs) by both modification of PFS requirements and by "design-for-closure" modification of capital design and construction actions.

Introduction

Closure of heap leach pads in Nevada and other semi-arid regions of the western United States typically includes establishing operational bench widths and lift heights to maximize efficiency during closure regrading, process pond design using water balances that take into account conversion of process ponds to evapo-transpiration cells (ET cells) or evaporation cells (E cells), and a cover layer of soil or growth media to eliminate the potential for meteoric waters to contact the spent ore. The soil cover also serves as a growth medium for vegetation, which limits downward migration of meteoric water through storage in the cover interstices and eventual release via direct evaporation or vegetation uptake and transpiration.

Regardless of the soil cover thickness and the vegetation component, some amount of infiltration of meteoric water is typically predicted by unsaturated zone modeling for mine sites at elevations with relatively high annual precipitation and/or mine sites where winter precipitation falls as snow and is introduced into the heap system as spring snowmelt. The result of using soil covers at these sites is ephemeral draindown flows from the heap, both in predictive modeling and in practice, during and following spring snowmelt and sufficiently large precipitation events. Predicted and actual long-term effluent flows trend toward low to no flow for the rest of the year.

Post-closure management of these ephemeral flows typically involves the use of ET cells or E cells, usually constructed by converting existing process water ponds to a pre-determined design. Existing State of Nevada guidelines for closure of E and ET cells (NDEP-BMRR, 2011) require fluid management system operation (and therefore bonding) until the flows reduce to a *de minimus* value whereby non-degradation of groundwater can be demonstrated and final closure thus achieved. As described above, predictive modeling of soil covers using unsaturated zone models typically provides long-term results that are ephemeral and not *de minimus*.The typical post-closure fluid management option therefore requires relatively low initial capital costs for process pond modification to E or ET cells, but may also require long-term post-closure fluid management costs. In contrast, closure stabilization through installation of a

low-permeability cover requires bonding for a relatively high initial capital cost, but only short-term post-closure fluid management.

The authors have assisted operators faced with high cash-flow obligations for payment of reclamation bonding, to achieve a more equitable balance between initial capital expenditure (for facility construction and PFS bond costs) by both modification of PFS requirements and by "design-for-closure" modification of capital design and construction actions, to effectively reduce initial cash flow liabilities (i.e., compared to typical PFS actions as described above).

These actions are described and discussed in the following sections:

- PFS requirements; and
- design-for-closure.

PFS requirements

State of Nevada reclamation regulations and guidelines for heap leach pad (HLP) draindown management require the determination of, and bonding for, process fluid stabilization (PFS) requirements. Nevada Administrative Code (NAC) 519A.068 defines process fluid stabilization to mean:

> binding, containing or otherwise treating contaminants in a fluid, including, without limitation, meteoric waters, that have intentionally or unintentionally been introduced into a heap leaching facility or tailings facility to prevent the contaminants from degrading the waters in this State through naturally occurring environmental conditions which may be reasonably expected at the mine site.

PFS typically includes:

- Interim fluid management (IFM), consisting of short-term (one to twelve months) recirculation pumping back onto the heap to actively evaporate surplus process fluid inventory and maintain compliance with permit limitations.
- Interim recirculation and evaporation of residual (seasonal) draindown flows until completion of construction and commissioning of long-term disposal facilities (e.g. ET cell) have been achieved.
- Management of long-term seasonal flows via the ET cell. As previously described, the duration of long-term PFS is contingent upon flows and chemistry reaching *de minimus* conditions that allow State of Nevada Water Pollution Control (WPC) permit retirement.

In addition to the above reclamation bond requirements, the US Bureau of Land Management's 2009 *Instructional memorandum IM 2009-153*, titled *Financial guarantees for notices and plans of operations* (BLM, 2009) provides guidelines for establishing funding mechanisms for potential long-term

post-reclamation obligations (PROs) deemed necessary to meet water quality objectives (i.e., for facilities potentially requiring long-term process fluid stabilization as described above), based on net present value "in-perpetuity" management costs.

PFS requirements and timeframes are typically determined using a water balance model such as the Heap Leach Draindown Estimator (HLDE) (JBR and Newmont, 2011). Longer-term post-closure fluid management requirements are typically determined based on a prediction of meteoric water infiltration through the soil cover and heap profile using computer models such as SVFlux, UNSAT-H, or HYDRUS-2D. Most modeling exercises result in the prediction of some draindown flow that at the end of the planned operation will require management because of its chemistry.

To demonstrate the potential relative contributions of both initial and long-term PFS costs, this paper evaluates a typical heap leach pad configuration under the following closure scenarios:

- **Option 1**: 0.75 meters (m) (2.5 feet) of alluvial growth media placed directly on the regraded top and sideslopes (i.e., store and release cover).
- **Option 2**: Geomembrane liner on the top surface and regraded sideslopes with 0.75 m of alluvial growth media placed above the liner.

Evaluation of options

HLDE modeling for a recent heap leach pad closure design project in Nevada predicted a covered infiltration rate of 1 to 2% of mean annual precipitation (MAP). Draindown flows immediately following cessation of operational leaching reduced from 315 liters per second (lps)(5,000 gpm) to around 0.6 lps (10 gpm) within 12 months, and further reduced to a consistent flow rate of around 0.10 to 0.15 lps (2 gpm) after installation of an earth cover as per Option 1 above. An initial minimum design flow rate of 0.125 lps is used for the ET cell, and this rate would theoretically require 8,100 square meters (m^2) (2 acres) of post-closure evaporation area to manage (refer to NDEP-BMRR [2011] and JBR and Newmont [2011]). The conversion of a process water pond(s) to an ET cell with a surface area of about 8,100 m^2 will therefore be sufficient to manage the effluent flow through the post-closure period. Post-closure PFS is assumed to be required for a maximum of 30 years (not too short, not too long). For Option 2, the predicted post-closure flow rate is zero, requiring no PFS. Capital, operating, and final closure actions for Option 1 and Option 2 are summarized below.

Option 1: Growth media on regraded top and sideslopes

This option includes:

- grading and compaction of the heap top surface and sideslopes;
- placement of 0.75 m of growth media on regraded top and sideslopes;

- revegetation of top and sideslope surfaces;
- 2 years of IFM;
- 30 years of post-closure draindown fluid management; and
- final closure of ET cell.

Option 2: Geomembrane liner on top and regraded sideslopes

This option includes:

- grading and compaction of the heap top surface and sideslopes;
- installation of a geomembrane liner on the regraded top surface and sideslopes;
- installation of a drainage system above the liner on top and sideslopes;
- placement of 0.75 m of growth media on top and sideslopes;
- revegetation of top and sideslope surfaces;
- 2 years of IFM; and
- final closure of ET cell.

Post-closure fluid management assumptions

For purposes of this discussion, the assumed long-term post-closure fluid management-related costs associated with the ET cells in Option 1 include:

- annual active operational maintenance for storing in the winter and spring and pumping from storage for evapo-transpiration in summer; and
- removal and replacement of ET cell media – once every 15 years.

Trade-off analysis

The results of a relative trade-off comparison between Options 1 and 2 are provided in Table 1. The actions for items in the first five rows are common to both options and result in a zero differential cost, as shown. Costs for Option 2 in the remaining rows have been subtracted from costs for Option 1 to obtain the relative cost differentials provided in the last column.

Table 1: Relative cost difference between Option1 and Option 2 for 30-year PFS period

Item	Description	Unit	Option 1: Top and sideslope grading + 0.75 m growth media	Option 2: Synthetic cover on top and sideslopes + 0.75 m growth media	Potential cost differential (Option 2 minus Option 1) (US$)
1	Interim fluid management	years	2	2	$0
2	Sideslope grading and compaction	ha	130	130	$0
3	Top surface grading and compaction	ha	65.6	65.6	$0
4	Growth media placement	m^3	1,492,000	1,492,000	$0
5	Revegetation	ha	196	196	$0
6	Process pond closure grading	m^3	0	118,500	$217,000
7	Convert process pond to ET cell (twice in 30 years, $1,000,000 each time)	ea	2	0	–$2,000,000
8	Synthetic liner supply and installation (assume US$10.75/$m^2$ for LLDPE)	ha	0	196	$21,070,000
9	Drainage system	m	0	70,700	$232,000
10	PFS (assume $200,000/yr)	years	30	0	–$6,000,000
					$13,519,000

The results indicate a potential $6 million dollar difference in PFS costs for Option 2 (synthetic liner) versus Option 1. If the required timeframe for PFS for Option 1 is increased for some reason, and the cost for synthetic liner installation can be lowered (e.g., by installing PVC liner that is rapidly deployed and covered with growth media), the differential could reduce to a value similar to that shown in Table 2 (i.e., Option 2 becomes cheaper than Option 1).

Table 2: Relative cost difference between Option1 and Option 2 for 60-year PFS period

Item	Description	Unit	Option 1: Top and sideslope grading + 0.75 m growth media	Option 2: Synthetic cover on top and sideslopes + 0.75 m growth media	Potential cost differential (Option 2 minus Option 1)
1	Interim fluid management	years	2	2	$0
2	Sideslope grading and compaction	ha	130	130	$0
3	Top surface grading and compaction	ha	65.6	65.6	$0
4	Growth media placement	m^3	1,492,000	1,492,000	$0
5	Revegetation	ha	196	196	$0
6	Process pond closure grading	m^3	0	118,500	$217,000
7	Convert process pond to ET cell (4 times in 60 yrs, $1,000,000 each time)	ea	4	0	–$4,000,000
8	Synthetic liner supply and installation (assume PVC at US$5.50/$m^2$)	ha	0	196	$10,780,000
9	Drainage system	m	0	70,700	$232,000
10	PFS (assume $200,000/yr)	years	60	0	–$12,000,000
					–$4,771,000

Designing for closure

Operators can reduce the PFS costs associated with the project by designing and constructing for operations in a way that will eliminate and/or facilitate closure actions, and consequent PFS costs. Relevant examples of this approach are:

- Gravity-draining pond leakage collection and recovery system (LCRS) routed to an external evaporation pond to eliminate head on pond secondary liners (per NAC 445A.435[2]) and minimize the necessity for active post-closure fluid management (i.e., monitoring and pumping).
- Using surplus cut from grading operations to form stormwater diversion berms upstream of process components that provide the required soil cover and growth media material for closure and reclamation.

- Providing a factor of safety for pond area design and construction that takes into account all storage and evaporation needs (i.e., increasing initially constructed pond areas for closure PFS – if you have to bond for it with cash).
- Designing an above liner drainage system to which vacuum-extraction can be applied, thus accelerating draindown timeframes (and reducing costs) for both IFM and annual PFM. This is applicable to both HLP and tailings storage facilities.

Conclusions

Operators faced with high cash flow obligations for payment of reclamation bonding can achieve a more equitable balance between initial capital expenditure (for facility construction and PFS bond costs) and long-term PFS by both modification of PFS requirements and by "design-for-closure" modification of capital design and construction actions. This is particularly valid for any site that may be required to consider long-term trust fund establishment.

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Heap leach pads over waste rock facilities, a different solution in mountainous terrains and limited space

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Abstract

The extraction and processing of gold and copper in Peru has a long history, so it is not surprising that Peru has some of the largest gold and copper operations in South America. However, gold and copper operations, in particular with heap leaching, face regional challenges such as mountainous terrain, rainfall and water management issues, environmental concerns, and limited space issues in the case of projects located in the proximity of communities where efficient use of space is required. Additionally, projects may have operational challenges such as constant growth due to market conditions or increased resources, or economic and budget restrictions. These challenges can generate a need for new facilities or expansions that have to be achieved in a short time frame, while at the same time addressing ermitting constraints and the need to reduce costs.

In this context, the Knight Piésold team sought new approaches to overcome these obstacles and explored the construction of leach pads on top of waste rock dumps to generate the required storage capacity from land previously thought unusable for a process facility.

This paper describes the critical points to consider during the initial study, further design, construction and operation of heap leach pads over waste rock facilities. The critical points primarily involve understanding the settlement process of the waste rock dump and its influence on the leach pad lining system and associated structures. The required analysis and proposed alternatives to implement solutions are also examined. Finally, the construction of the structure and subsequent monitoring during operation is assessed to evaluate whether the general performance is in accordance with the design intent.

Since early 2000, several heap leach pad facilities have been similarly designed and built in Peru based on this innovative concept (some of them are already close to the decommissioning phase of the heap), yielding project savings and creating useful capacity where none was thought to exist.

Introduction

The extraction and processing of gold and copper in Peru has a long history, so it is not surprising that Peru has some of the largest gold and copper operations in South America. Currently, Peru is the largest

gold producer and second largest copper producer in South America. As shown in Figure 1, most of current projects and potential future projects are located in the mountainous terrain of Peru (called "Los Andes", see brown area) that can reach an elevation of 6,700 MASL in a short distance, generating steep slopes. These topographic features are also shared with other mining regions in the world and present construction and operational challenges for heap leach pads such as construction on steep slopes and availability of enough area at the bottom of steep valleys to accommodate mine production rates and leaching requirements.

Figure 1: Physical Map Peru – Mining projects (INGEMET, 2013)

The mountainous terrain in Peru is characterized by heavy rains, and most of the water for the coastal region originates here, resulting in public and government environmental concerns, as most of the

projects are located in the proximity of communities. These conditions reinforce the need for efficient use of available space by the mines. Furthermore, the projects may have operational challenges during their development, such as constant growth due to changing market conditions, increased resources, or contrarily, economic and budget restrictions. These challenges can generate the need for new facilities or expansions that must be achieved in a short time frame, despite permitting constraints and the need to reduce costs.

Obtaining permits to operate mines in new areas is becoming complicated, often because of decision delays by government agencies due to negative public perception. It is also more difficult to buy new land as prices for land surrounding mining areas become inflated. The combination of these issues impacts potential mining projects, increasing costs and making it difficult to take advantage of windows of opportunity for metal prices. The Knight Piésold team sought new approaches to overcome these obstacles and explored the construction of leach pads on top of waste rock dumps to generate the required storage capacity from land previously thought unusable for a process facility.

As noted in the abstract, this paper describes the critical points that require consideration during the initial study, further design, construction and operation of heap leach pads over waste rock facilities. The critical points primarily require understanding the settlement process of the waste rock dump and its influence on the leach pad lining system and associated structures. The required analysis and proposed alternatives to implement solutions are also examined. Finally, the construction of the structure and subsequent monitoring during operation is assessed to evaluate whether the general performance is in accordance with the design intent.

Heap leach pads over waste rock facilities

The design, construction and operation of heap leach pads over waste rock facilities pose some challenges for each stage of the project. In order to facilitate the identification of critical points, as well as assess the advantages and limitations for the combination of these structures, we will review the subject from different points of view. First, we will evaluate the implications of the combination using different classifications for the structures; then we will review the main technical specialties involved during the design; and finally we will comment on each stage of the project.

Classification

Table 1 presents a summary with relevant classifications to analyze the combination of heap leach pads and waste rock facilities.

Table 1: Relevant classification for heap leach pads and waste rock dumps

Classification	Heap leach pad	Waste rock dump
ore/waste type of material	Coarse Fine grained Agglomerated	Hard rock Weathered or degradable rock Overburden
By preparation of the material	Run of mine (ROM) Crushed Agglomerated Chemical	Not prepared
By method of construction (placement of material)	By lifts (majority) Ascending	End dumping 1 single lift Platforms and lifts Ascending or descending Terraces and wrap around Buttresses and impact berms
By method for loading of material	Trucks Conveyors and stackers (retreat, most common) Conveyors and stackers (front loading)	Trucks Conveyors and stackers (front loading)
By configuration or operation	Reusable (on/off) dynamic or hybrid Permanent expanding pad schematic Valley fill (in heap pond)	Valley fill Cross valley fill Side hill fill Ridge crest fill Heaped fill

Adapted from Van Zyl et al., 1988 and British Columbia Mine Waste Rock Pile Research Committee, 1991

Based on this table, we present the following observations:

- The most important component for the combination of both structures (heap leach pads and waste rock facilities) is the waste dump material (and its foundation), since this will need to withstand the load imposed by the heap leach pad. The most efficient is hard rock, which provides a good foundation for the structure and results in smaller settlements. Using fine materials necessitates greater care during the geotechnical analysis (stability and settlements), imposes a possible limitation on the size of structures that can be placed on top of the facility, and requires additional monitoring during the construction and operation of the facility.
- The waste dump material is not normally specifically selected or prepared. This implies a need for investigation and research during the material characterization. Additionally, during the analysis, it is

necessary to account for the non-homogeneity of the material, which for example will require considering a layered soil profile in the settlement analyses.

- With regard to the construction method, the most important component is the waste dump, which preferably should be placed by lifts in order to reduce settlements, improve control of material and improve stability conditions. It is not advisable to combine high dumps placed in a single layer (and probably with a high dumping rate), given the great potential of the material for settlement accommodation, which could be intensified in high seismic zones like Peru.
- With regard to method for loading of material, our experience and recommendations are based primarily on the construction of waste dumps with trucks and bulldozers, which have a favorable effect on the arrangement of the material while placed in layers.
- With regard to configuration, the most efficient type (in terms of economic benefits and synergies) is a valley fill waste dump configuration, followed by a cross valley fill waste dump configuration. The valley fill configuration generates a flat area (ideal for the first lift loading) with an elevation that typically reduces hauling distances for the ore material. Other waste dump configurations and combinations are possible, but are not as efficient or economical due to increased haul distances or available areas for the heap leach pad.

Next we will review the main implications regarding different technical specialties and stages of the project. With regard to specialties, a combined structure requires specific considerations in relation to the geotechnical design. For the different stages of the project, the design and construction stages of the dump are the ones that require additional care. It is important to note that common recommendations and engineering practices for heap leach pads and waste rock facilities development (such as acid mine drainage evaluation, hydrology, geotechnical evaluation, geology, etc.), during all project stages (design, construction, operation, closure), should be maintained, whether you have a combined or separate structure; however, these are not part of this paper. Below are the main points to consider regarding technical specialties, followed by the main points for the different stages of the project.

Geotechnical considerations

The placement of basically loose, random material as a foundation for a heap leach pad is a specific issue to analyze and consider for all the stages of the project. The resulting settlement of the waste rock dump has significant implications on the behavior of the heap leach pad components. The settlement of a waste rock dump combined with a heap leach pad comprises the following components:

- settlement of the foundation produced by the weight of the dump;
- settlement of the foundation produced by the weight of the heap leach pad;

- settlement of the dump produced by the weight of the dump; and
- settlement of the dump produced by the weight of the heap leach pad.

Settlement of the dump is critical to the heap leach design as settlement of the dump surface will result in settlement of the leach pad liner and solution collection systems, which may affect positive drainage of the solution to the process ponds. Moreover, excessive localized differential settlements could compromise the integrity of the liner system. Finally, all the ancillary structures that are typically built as part of a heap leach pad (channels, ponds, tanks, process pipes, etc.) will have to take this issue into account. Special attention should also be applied to the design of rigid structures (such as steel pipes, tanks, etc.).

There are several methods for carrying out settlement analyses, from empirical methods to the latest computational methods. It is recommended that whatever methodology is applied is later calibrated with the results of the monitoring of the structure during construction or operation (for adjustment of the design or for use in future designs). Also, it is recommended that the non-homogeneity within the dump profile be taken into account by considering a layered soil profile in the settlement analyses (using data obtained from the geotechnical investigation).

Some of the results obtained to date (measured on site), include settlements that varied from approximately 0.1 m at the edge of the heap to an estimated maximum settlement of approximately 5 m, associated with a dump thickness approaching 100 m with an overlying heap of 120 m. Typically, it has also been found that the majority of the settlement generally occurs within the waste dump, with a minor percentage occurring within the underlying foundation soils. The historic performance of earth and rock fills also indicates that the majority of the settlement at the surface of the dump due to its weight will occur during its construction (prior to construction of the leach pad), but this should be verified by survey prior to proceeding with the construction. It is therefore recommended that the dump should be in place for a sufficient duration so that settlement in either the foundation or the dump due to the weight of the dump is completed prior to leach pad construction.

Seismic loading requires special consideraton, as it can cause additional settlement of the dump material or differential settlement. To determine the effect of an earthquake on the structures, typically a seismic response analysis is conducted, applying the methodology of one-dimensional propagation of the shear wave through a stratified medium. The analysis is conducted to estimate the magnitude of movement due to shaking during a seismic event.

In general, the main recommendations following a seismic analysis include considering a structural fill on the last meters of the dump to limit settlement or differential settlement that might be caused by such an event. The thickness of this material varies and depends on the size of the structure to be placed on top of the dump, limitations for movement (i.e. rigid structures have less tolerance), height of the

dump and material of the dump. This layer also serves as a filter between the waste rock material and the soil liner or prepared subgrade material for the heap leach pad.

Finally, it is important to mention that all the technical specialties involved in the project should be informed and knowledgeable on settlement issues in order to factor in their impacts during their design (civil design, hydraulic designs, etc.). Some of these aspects are discussed in the following section.

Recommendations for stages of the project

The main stages for a typical project include: exploration, design (conceptual, prefeasibility, and feasibility design), final design, construction, operation and closure. The main aspects for the construction of a combined structure during these stages are discussed below.

Design and final design

During the early stages of the project, the most important aspect is to determine if the combined option (heap leach pad over waste rock facility) is a viable option for the project and warrants further investigation. This is done during an alternatives analysis, where several options are compared from the technical, economic, operational, environmental and social aspects.

The combined option typically represents a viable alternative or option when some or most of the factors discussed in the first part of the paper are present (space restrictions, physical constraints, valley fill, favorable materials, etc.). If the combined option is selected, then the technical studies, site investigation and materials investigation (geosynthetics, pipes, etc.) should be planned and orchestrated as the required data for the design is gathered.

In order for the combined option to work, the mine plan should be reviewed to verify that the required amounts of material will be available when needed for construction of the dump. Additionally, it should be verified that the waste rock materials that are going to be covered by the heap leach pad do not have any potential economic value.

As most of the geotechnical aspects for the design have been covered in the previous section, the following paragraphs describe other aspects and components for a combined option design:

- During the design of the heap leach pad, the estimated settlement for the dump will be used to estimate the post-settlement surface configuration. The post-settled surface will then be used to establish the heap leach pad grading plan, solution collection system, and acceptable ranges of geomembrane properties required to withstand the anticipated vertical movements and resultant horizontal elongation. The anticipated settlements and the resultant strains in the geomembrane liner underlying the heap must be considered in selecting an appropriate liner material and thickness. The typical components for a liner and solution collection system are shown in Figure 2.

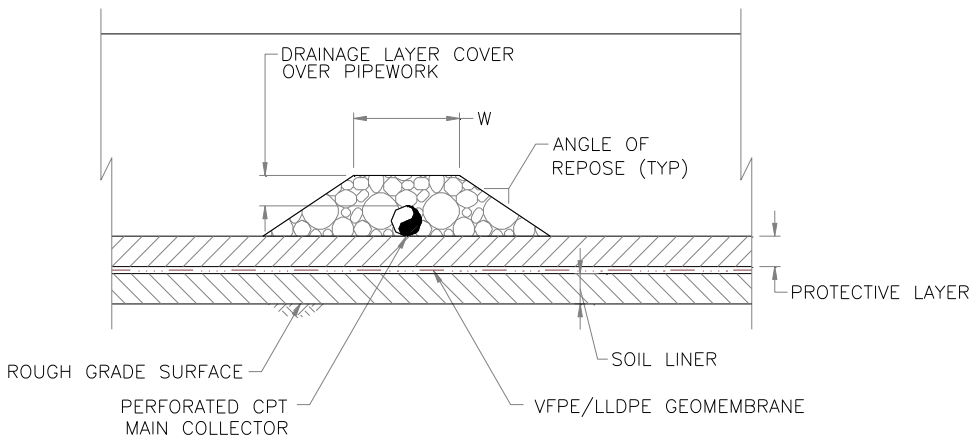

Figure 2: Typical liner and solution collection system

- A space should be left on the periphery of the dump for the construction of ponds and other structures. The ponds should be located in such a way that potential circular failures of the dump do not affect the pond, and so as to avoid shear deformations at the free face of the dump. This can be achieved by leaving a set back from the crest to form an "embankment" that works as a gravity dam, satisfying the minimum crest width to dam height criteria. The material to be placed on the side slopes of the pond should be specified, taking into account the equipment required for construction (compacted material is preferred). Other considerations for liner systems for ponds on compressible fills are discussed by Lupo and Morrison (2005). As a contingency measure, a contingency underdrain system below the ponds to separate the pond lining system from the waste dump fill may be considered.
- Due to the anticipated settlement of the dump, it is recommended that the connections of all the header pipes be designed with double wide couplers, for proper union and to reduce the potential for pipe separation at their junction due to expected settlements. To allow future inspection with cameras through the header pipes, some of these pipes should be extended to the leach pad perimeter as part of the design.
- Due to the settlements, the design of all the perimeter channels located on top of the dump should consider the use of flexible liners (geomembranes or concrete infilled geowebs) and measures to reduce infiltration (geosynthetic clay liners – GCL).
- Other considerations during the design include the evaluation of the interactions of potential acid generation material with the structures of the heap leach pad (if they are in contact) and the potential for gas or heat generation for some types of waste materials (sulfide waste materials), which are going to be covered by the heap leach pad (Pantelis et al., 2002).

Construction

Communication is likely the most important aspect for construction and operation of combined structures. The technical reports produced during the design phases are sometimes not enough to convey to the construction and operation teams the main recommendations for such facilities, which include additional recommendations and considerations compared to the typical ones considered for heap leach pad or waste rock facility projects. To assist in this effort, it is recommended that quality assurance and quality control procedures (QA/QC) for the construction of the dump be prepared, which would include laboratory testing to confirm the assumed properties of the materials. Periodic visits and presentations by the design team during the construction and operation of the facility should also be documented as part of the design and QA/QC procedures.

The most common problems observed during construction include the modification or addition of structures (field changes) that do not take into account the recommendations and limitations imposed by the settlement of the base material (which can lead to problems with rigid structures and/or connections not designed to accommodate settlement). Other problems include not following the specifications of the material on the final layer of the dump (material continues to be placed as a waste dump) and inadequate control of superficial water including ponding of water, which could cause the formation of superficial cracks on the final surface of the dump.

Prior to construction of the waste dump, the native ground within the footprint of the dump should be stripped of topsoil, organic materials and unsuitable materials, such as peats and soft clays, to provide an adequate foundation for placement of the waste, enhance slope stability and minimize settlement. For a combined structure it may be necessary to remove additional material to reduce settlements, which should be evaluated during the site investigation and included in the cost while comparing alternatives for the project (see previous section). Thus, not all the locations will be appropriate for the location of a combined structure.

Typically, the mine waste has to be placed in lifts with the thickness determined during the design phase. The top of the lifts should be superficially compacted by selectively routing the equipment over the surface. The final lift of the dump should consider a filter compatibility criteria and granular material of low plasticity in order to get a material that can effectively be compacted throughout the layer, but also does not allow vertical migration of fines content from the soil liner of the leach pad (see previous sections) into the underlying waste material.

Finally, settlement of the dump should be re-evaluated at approximately 90% completion of the structure in order to verify the initial assumptions of the design, to refine the settlement estimates with actual field data, and to allow settlements to be accommodated in the design of the leach pad.

Operation

As indicated above, communication of principal recommendations of the design to the operational team is considered an important aspect to avoid similar problems to the ones produced during the construction. Additionally, the main task during operation (specific for a combined structure) is the monitoring of settlements. For the monitoring of settlements during operation, the pad and dump surface should be instrumented to:

- Monitor actual settlements and compare them against design estimates, verifying design assumptions.
- Confirm that positive drainage of process solution to the edge of the leach pad and associated ponds is maintained.
- Verify resultant strains in the leach pad geomembrane liner underlying the heap based on actual vertical movements and resultant horizontal elongation of the geomembrane.
- Allow calibrations of the design models for future designs.

Settlement sensors should be placed in the areas where a greater amount of final settlements are expected and coinciding with the lower part of the pad. It is also recommended that instrumentation be placed so that the alignment of the sensors coincides with the alignment of the main collector pipes. The staff responsible for data collection must do the following as a minimum:

- record changes on the reading unit;
- record visual observations of changes in site conditions;
- verify the consistency of two or more readings;
- verify the repeatability of the data;
- note the presence of external factors such as details and progress of construction, environmental factors (temperature, rain, snow, sunny or cloudy) and seismic factors; and
- note construction or mining activities around the terminals and sensors for data collection.

The settlement system should include sensors located at a depth of stable ground that will not settle under the influence of the overlying structure. It is also good practice to compare the readings of the sensors with topographic landmarks to verify a consistent relationship.

The analysis of the data should be directed to two main aspects in the monitoring program: verify the instrumentation system performance and verify performance of the structure being monitored. Finally, the recorded data must be compared with the variations of the calculated settlements during the design, plotting settlement readings versus time and stack height graphs in order to assess trends and provide early detections of potential problems.

Conclusions

The construction of heap leach pad facilities over waste rock is an alternative to consider for a project if some or all of the factors described in this paper are present. These factors include mainly lack of space, physical constraints caused by steep slopes, and rigorous requirements for an efficient use of available space at the mine site.

In order to implement this type of leach pad/waste dump arrangement, a proper evaluation is required during the early stages of the project. This is done typically during the alternatives analysis, where several options are compared in terms of the technical, economic, operational, environmental and social aspects. There are advantages and cost reductions in the use of a combined structure; but also challenges, that can translate into costs that must be included during this evaluation for a proper assessment.

Not all locations are favorable for the construction of a combined structure and there is a combination of factors and configurations such us valley fill waste rock dumps that enhance its efficiency.

During all the stages of the project, settlements and their consequences are the most important aspects to take into account and have implications for each component of the facility and should be considered by all the technical specialties (civil, hydraulic, etc.). In this paper we have provided recommendations for the main project stages and have outlined also the main points to consider for each of them.

Since early 2000, several heap leach pad facilities have been similarly designed and built in Peru based on this innovative concept (some of them are already close to the decommissioning phase of the heap), yielding project savings and creating useful capacity where none was thought to exist.

Acknowledgements

I would like to thank Knight Piésold and its team of professionals who have developed most of the recommendations and major findings summarized in this paper over a period of several years.

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Influence of berms and channels in the stability of heap leaching

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Abstract

This paper describes an analysis of the stability of heap leach pads using berms and/or channels to increase the factor of safety against sliding along a basal liner. The constitutive model Hardening - Soil was used to calculate displacement of the heap leach pad materials along potential failure planes. The slope stability of the heap leach pad was calculated using the method of strength parameters reduction by finite elements. The study reviewed general concepts of the constitutive model Hardening – Soil and its parameters, and separated these parameters into two groups: parameters calibrated to the stress-strain curve of consolidated drained triaxial tests and parameters assumed according to the volumetric strain curve of the same test. The following materials were calibrated: the ore, the structural fill, and the low permeability soil. A heap leach pad was modeled using the finite element method and the commercially available software PLAXIS v. 8.2.

The location of berms and channels and their dimensions were analyzed to quantify their impact on the stability of these stabilization structures. In the pseudo-static stability analysis, which used a seismic coefficient equal to 0.15, the results are as follows: the channel increases this safety factor by 4.25%, regardless of the length of the channel. The depth of the channel is the most influential parameter; the optimum dimension is 1.2 m. As regards the failure surface, the volume of ore potentially mobilized by including one or two channels in the heap design was reduced by 33% and 60%, respectively.

The safety factor calculated by the limit equilibrium method is about 2% higher than the safety factor calculated by the finite element method. The failure surfaces for the two methods were also different. While channels affect the finite element failure surface, moving it closer to the slope of the ore, depending on the position and number of channels, this influence on the failure surface could not be modeled or replicated by the limit equilibrium method because, in all these cases, this surface was almost the same.

Introduction

Heap leaching is a process in which a pile of ore is irrigated with a leaching fluid at a carefully designed rate, thereby guaranteeing adequate extraction of the target metal or mineral as the leachate percolates. Leach piles are generally large in terms of area covered, height, and therefore, the volume stored. Thus, the slope stability of these structures becomes a critical consideration for heap leach facilities. The majority of metal mines are in earthquake-prone regions; accordingly, sliding under seismic as well as static loads is considered. It is a common practice to provide a geomembrane liner and a layer of low permeability soil at the bottom of the pile to limit seepage into the natural ground below (see Figure 1a). The stability of the pile is mainly influenced by the shear strength of the interface between the low permeability soil and the geomembrane, and potential failure is often due to sliding along this plane. An intuitive approach to reducing the possibility of sliding along this interface is to remove the surface continuity of the potential sliding block through the provision of berms and channels (Figure 7) at the base of the leach pad. Classical limit equilibrium methods of slope stability analysis for real cases with such provision have shown increased safety factors. The objective of this research paper is to analyze the influence of these structures − berms and channels − on safety factors and sliding surfaces through rigorous stress-strain analysis. The heap leach pile analyzed is shown in Figure 8. Figure 7 shows the berm and channel cross-sections considered.

Simulation and calibration of materials to constitutive Hardening Soil model

In this study, the soil behavior has been represented by a Hardening Soil (HS) model based on a theory of elasticity known as Hooke's law (Duncan et al., 1980, these principles applied constitutive models) and the theory of plasticity (Schanz et al., 1999, these principles applied constitutive models). This constitutive model simulates a state of isotropic hardening, implying that, in the triaxial stress corresponding to principal stresses σ_1, σ_2, σ_3, the hardening develops in such a way that the center and shape of the yield surface remain stationary.

The parameters of this model are as follows:

- Dependence of the stress state stiffness = *m*.
- Secant module for 50% of the maximum deviator stress = E_{50}^{ref}.
- Tangent module for the compression curve of the edometric test to the reference pressure = E_{oed}^{ref}.
- Modulus of elasticity in the discharge / recharge process = E_{ur}^{ref}.
- Parameters of the Mohr-Coulomb model = c', ϕ', ψ'.

Table 1: Parameters of the HS model

Calibrated Parameters	Assumed Parameters
Cohesion (c'), Friction angle (ϕ'), Dilatancy angle (ψ)	Plastic straining due to primary compression (*E_oed^ref*)
Secant stiffness with reference pressure (*E_50^ref*)	Elastic stiffness to the proces unloading / reloading (*E_ur^ref*)
Stress dependent stiffness according to the power law (*m*)	Lateral pressure coefficient at rest (*ko*)
Reference pressure (*P^ref*), Reduction at the interface (*R_inter*)	Poisson ratio (ν)

Calibration to Hardening Soil model

The model parameters were determined for each of the following materials: ore, structural fill, and low permeability soil. The calibration was verified through a simulation of a consolidated drained triaxial test using a finite element method as the procedure. The general scheme of the check is shown in Figure 1b.

(a) (b)

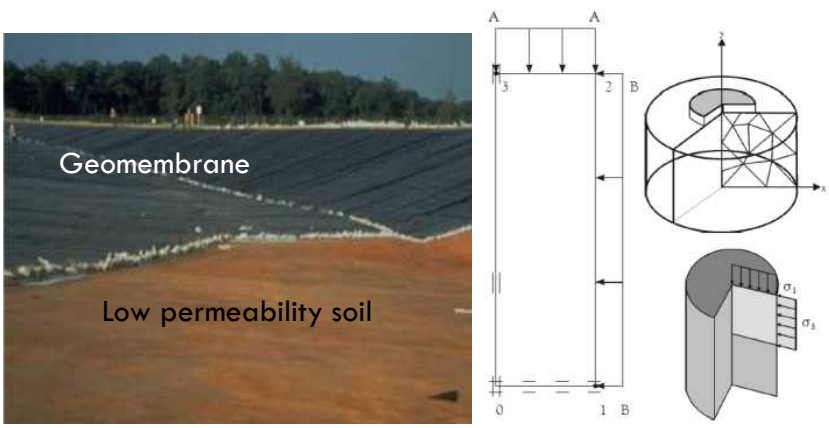


Figure 1: Low permeability soil and geomembrane (a); modeling of the soil by finite element (b) (Leal et al., 2009)

The parameters were separated into two groups: those that would be calibrated with available tests and those that would be assumed within some reasonable range. The calibration process for the first group of parameters involved simulation of consolidated drained triaxial compression tests using finite element software to match the actual test results. The general scheme of the calibration process is shown in Figure 1b.

Calibration of the ore

ϕ′, c′, ψ′

The values for the angle of internal friction, cohesion, and the soil dilatancy angle were obtained through the consolidated drained triaxial test (CD).

E_{50}^{ref} .y m

These parameters were determined from the stress-strain curve from the CD triaxial test, as shown in Figure 2. The parameter *E* is determined by the graphical hyperbolic relationship, taking as a reference pressure *200 kPa*; hence the E_{50}^{ref} corresponds to E_{50} on the stress-strain curve, in which the confining pressure is equal to the reference pressure, i.e., *200 kPa.*

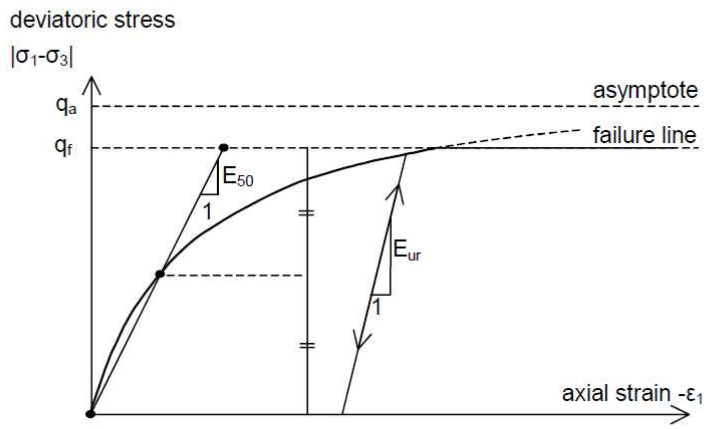


Figure 2: Stress-strain curve to obtain E_{50} for each confinement pressure (PLAXIS, 2004)

Having three values of E_{50} for each confining pressure makes it possible to obtain the value of *m*, which estimates the stiffness variation with respect to the state of stress. The confining pressure is plotted along the *X*-axis, while the stiffness modulus is plotted on the *Y*-axis. It is possible to adjust the curve to AX^B, where *A* has units of pressure and *B* is the value of the parameter *m* for given stiffness conditions. Therefore, results are shown in the following figure for *m = 0.62.*

P^ref (kPa)	E_50^ref (kPa)
100	19 650
200	30 900
300	41 860

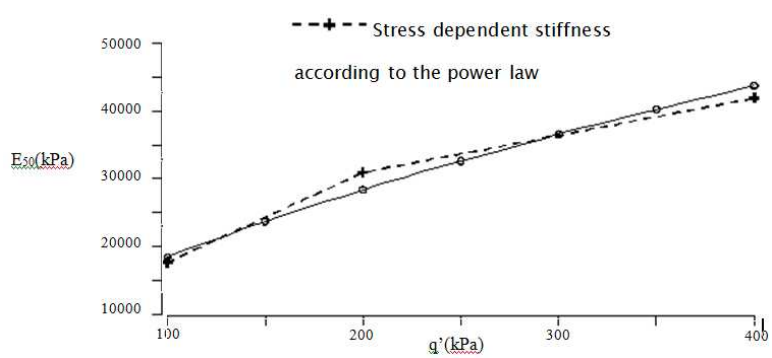


Figure 3: Dependence of the stiffness on the stress state

Permeability K_x, K_y

These coefficients were obtained from the hydraulic conductivity tests carried out according to ASTM D 5084. Isotropic permeability conditions were assumed.

k_x = 3880 m/day

k_y = 3880 m/day

The remaining parameters are obtained as follows: $K_0 = 1\text{-}\sin\phi$; Poisson's ratio for all cases is $v = 0.30$, except foundation soil, in which it is assumed to be $v = 0.25$; E_{oed}^{ref} is the tangent modulus of the oedometer test for a vertical load equal to P^{ref}; E_{ur}^{ref} is the modulus in a unloading-reloading process, which in the triaxial test would be the modulus in a process of removing and replacing deformations.

As explained above, some parameters were assumed because they do not have a great influence on the deformation and therefore do not significantly affect the displacement. The volumetric variation versus vertical deformation graphs are not shown, as volumetric variation will not be measured because of its small influence on deformation.

Model parameters of HS for the ore are presented on the left side on Figure 4; the right side of this figure shows the calibration curves. The behavior of the material is a good approximation of the curve from the numerical simulation of the triaxial CD test.

Parameters	Values	Units
γ_saturated	18.10	kN/m^3
γ_unsaturated	17.10	kN/m^3
c'	0.00	kN/m^2
ϕ'	37.00	°
ψ'	0.00	°
E_50^ref	30 000.00	kN/m^2
E_oed^ref	20 000.00	kN/m^2
m	0.62	-
E_ur^ref	90 000.00	kN/m^2
ν_ur	0.30	-
P^ref	200.00	kN/m^2
Ko	0.00	-
Kx	3 880.00	m/dia
ky	3 880.00	m/dia
Rf	0.90	-

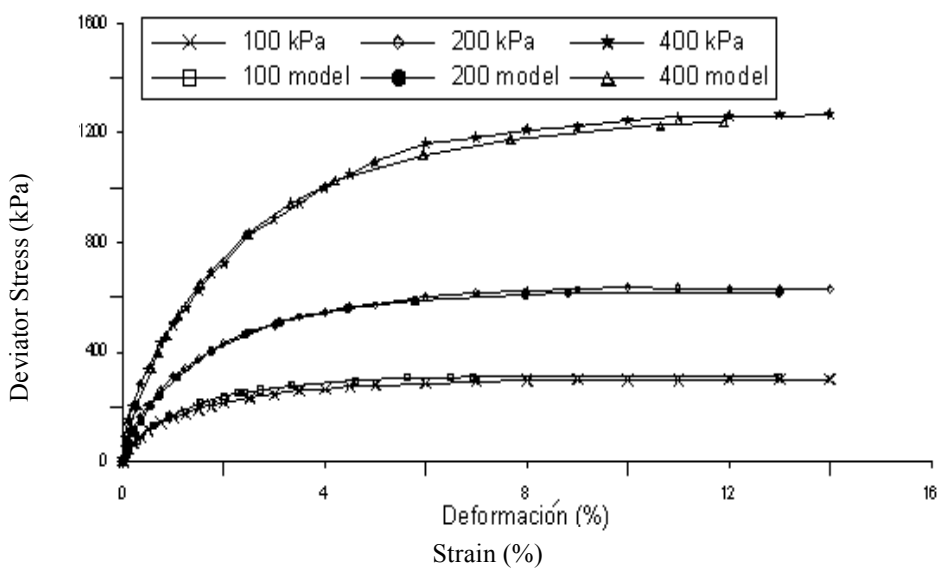


Figure 4: Model parameters for the ore and experimental graphs for the stress-strain model

Calibration of structural fill

The structural fill parameters, using the calibration procedure described above, are presented in Figure 5. There is a good comparison in curves between the triaxial CD test results and the numerical modeling by finite elements.

Parameters	Values	Units
γ_saturated	20.20	kN/m^3
γ_unsaturated	18.30	kN/m^3
c'	14.00	kN/m^2
ϕ'	35.30	°
ψ'	0.00	°
E_50^ref	59 350.00	kN/m
E_oed^ref	50 000.00	kN/m
m	0.31	-
E_ur^ref	178 050.00	kN/m
ν_ur	0.30	-
P^ref	200.00	kN/m
Ko	0.00	-
Kx	0.0003	m/dia
ky	0.0003	m/dia
Rf	0.90	-

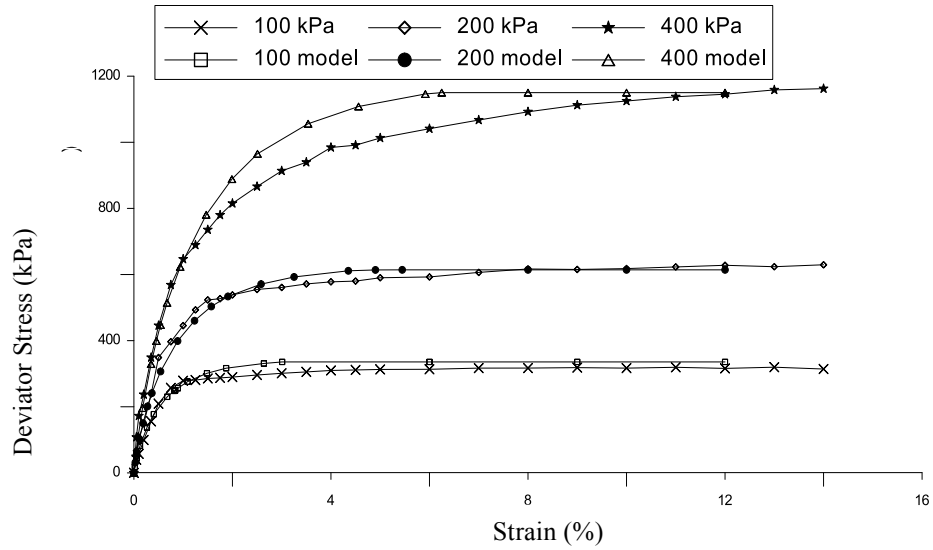


Figure 5: Model parameters for the structural fill and experimental graphs for the stress-strain model

Calibration of the low permeability soil

Heap stability is highly influenced by the resistance of the soil-geomembrane interface. Low permeability soil calibration is similar to the previous cases. This layer must be calibrated because the soil foundation has been modeled as an elastic material; if the low permeability soil layer is not considered, it will not be possible to calculate the stability or the strength parameters of the interface. This is because the foundation, having been modeled as an elastic material, has no strength parameters.

Reduction interface (R_{inter})

A large-scale direct shear test (ASTM D 5321B) was used to calculate R_{inter}. The geomembrane used was linear low density polyethylene (LLDPE) of 2.0 mm (textured side). The test results were as follows:

Adhesion = 15 kPa

Friction angle = 21.7°

The reduction in the interface between the low permeability soil and the geomembrane is the ratio of the tangents of the angles of friction between these two materials, according to the following expression:

$$\frac{tan(\phi\ large - scale\ direct\ shear)}{tan(\phi\ low\ permeability\ soil)} = \frac{tan(21.7°)}{tan(36.7°)} = 0.534$$

The results for this case were R_{inter} = 0.534. The parameters for this case are summarized in Figure 6.

Parameters	Values	Units
γ_saturated	20.60	kN/m^3
γ_unsaturated	18.10	kN/m^3
c'	14.00	kN/m^2
ϕ'	36.70	°
ψ'	0.00	°
E_50^ref	89 555.00	kN/m^2
E_oed^ref	80 000.00	kN/m^2
m	0.38	-
E_ur^ref	268 665.00	kN/m^2
ν_ur	0.30	-
P^ref	200.00	kN/m^2
Ko	0.00	-
Kx	0.00027	m/dia
ky	0.00027	m/dia
Rf	0.90	-

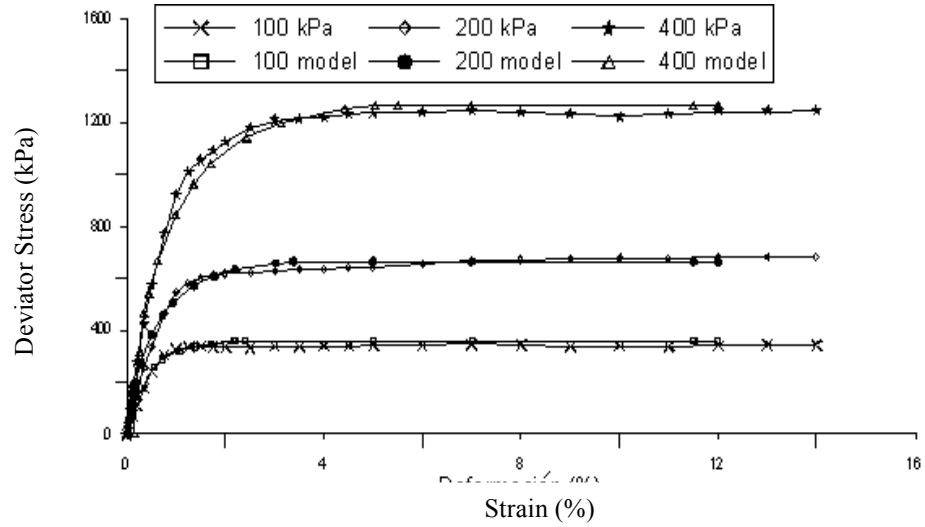


Figure 6: Model parameters for low permeability soil and experimental graphs for the stress-strain model

Soil foundation

Failure surfaces that affect the stability of a heap develop mainly along the interface between the low permeability soil and the geomembrane; this assumes that the foundation soil is sufficiently competent and the foundation soils are not involved in the analysis. For this reason, the properties of the soil foundation have been assumed for numerical analysis to be a rock material (see Table 2 and Goodman, 1989).

Table 2: Model parameters for soil foundation

Parameters	Values	Units
$\gamma_{saturated}$	25.50	kN/m^3
$\gamma_{unsaturated}$	24.50	kN/m^3
E	19,600,000	kN/m^2
ν	0.25	-
K_x	0.0003	m/dia
k_y	0.0003	m/dia

Heap stabilization

The method used to calculate the safety factor (SF) using PLAXIS software is the strength parameters reduction approach. The SF is defined as the factor by which these parameters must be divided in order to reach failure conditions (failure parameters), i.e., c'_f and ϕ'_f.

$$c'_f = \frac{c'}{SF} \tag{1}$$

$$\phi'_f = tan^{-1}\left(\frac{tan\,\phi'}{SF}\right) \tag{2}$$

Heap stability analysis

A heap leach pad located in the region of Cajamarca (in north-eastern Peru) with altitude between 3,800 and 4,000 m was analyzed using the methods described above. Figure 8 shows the section of this heap; its height measured between the crest and the toe is 130 m. The overall slope of the heap is 2.5H:1V and it has lift heights of 10 m.

Placement of the ore in 10 m lifts was simulated in order to represent more realistic displacements in the analysis, particularly in the area of interest, since it is known that the simulation of one single-phase ore placement does not suitably represent displacement distribution in the structure.

Berms and channels

Figure 7 shows the geometrical model of the berms and channels used and their standard dimensions.

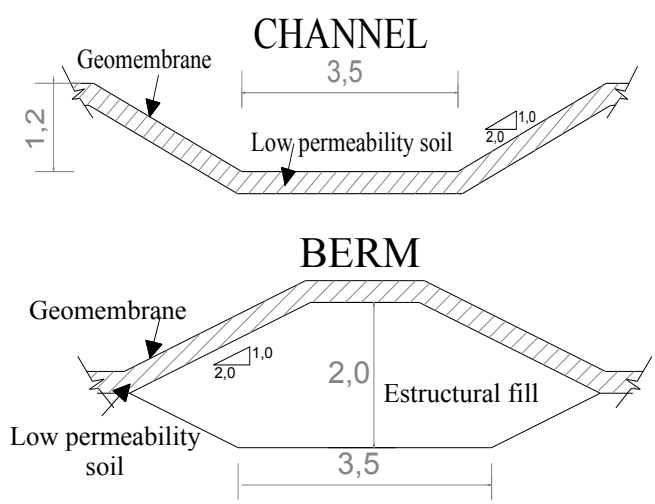
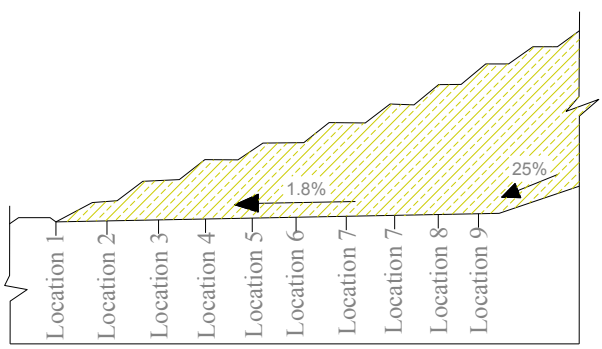


Figure 7: Standard dimensions in meters of berm and channel with low permeability soil (thickness 0.30 m)

Static and pseudo–statics analysis

Varying the location of berms and channels

In order to evaluate the effect of the location of berms and channels on the stability of the heap leach pad, stability analyses were performed using finite elements; the location of these structures was modified, as shown on the right side in Figure 8. Tables 3 and 4 show each SF for each location of berm or channel under static and pseudo-static load conditions (with seismic coefficient equal to 0.15) and increments in percentage of SF value. Figure 9 is the graph of the SF variation. These SFs are compared with those calculated without berm or channel, obtained through the limit equilibrium method: $SF_{static} = 1.674$, and $SF_{pseudo\text{-}static} = 1.058$.



Location	Values
1	16.0
2	37.2
3	58.2
4	78.0
5	97.9
6	116.9
7	136.1
8	155.2
9	174.4

Figure 8: Location of berms and channels

Table 3: Safety factor in different relative locations and increase in percentage of the initial safety factor (static)

Location with respect to the toe of the heap	X-axis	SF		SF Increment	
		Berm	Channel	Berm	Channel
1	16.0	1.667	1.666	-1.740	-2.180
2	37.2	1.679	1.682	-1.070	-0.910
3	58.2	1.691	1.692	-0.360	-0.310
4	78.0	1.699	1.701	0.120	0.220
5	97.9	1.706	1.713	0.500	0.970
6	116.9	1.712	1.724	0.850	1.610
7	136.1	1.713	1.731	0.940	1.980
8	155.2	1.716	1.735	1.110	2.220
9	174.4	1.720	1.740	1.360	2.560

Table 4: Safety factor in different relative locations and increase in percentage of the initial safety factor (pseudo-static)

Location with respect to the toe of the heap	X-axis	SF		SF Increment	
		Berm	Channel	Berm	Channel
1	16.0	1.050	1.052	-0.770	-0.570
2	37.2	1.064	1.065	0.550	0.630
3	58.2	1.072	1.073	1.290	1.400
4	78.0	1.077	1.080	1.750	2.040
5	97.9	1.084	1.089	2.460	2.930
6	116.9	1.086	1.092	2.620	3.250
7	136.1	1.089	1.098	2.910	3.750
8	155.2	1.088	1.097	2.800	3.690
9	174.4	1.090	1.103	3.020	4.250

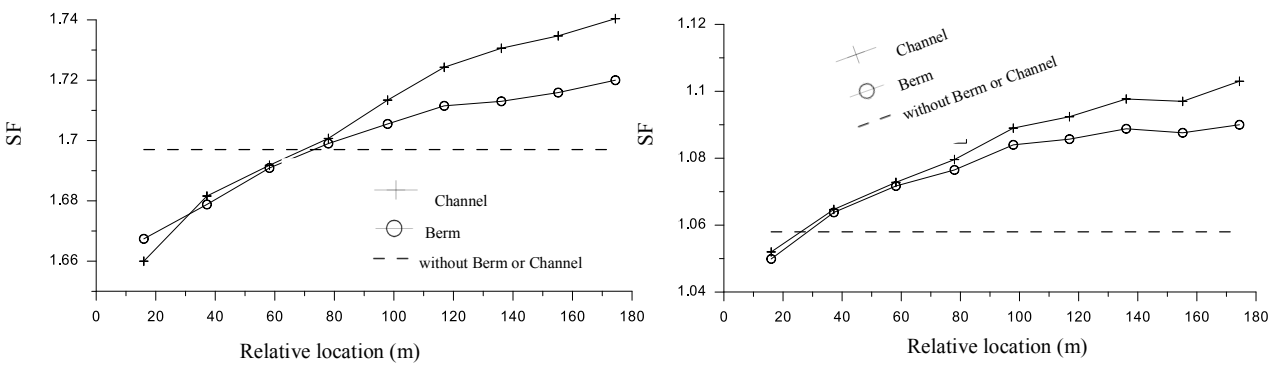
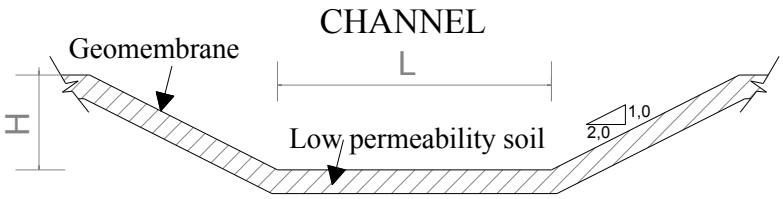


Figure 9: Safety factor relative to position under static load conditions (a) and pseudo-static load conditions (b)

As the results demonstrate, placing a berm or a channel about 78 m from the toe of the heap leaching pile means that SF_{static} is increased, while $SF_{pseudo\text{-}static}$ is increased with a berm or channel located approximately 30 m from the toe. The results also indicate that placement of a channel is more efficient than placement of a berm.

Varying the dimensions of the channel

In order to evaluate the effect of the channel dimension on the increment of safety factor in stability analysis, additional analyses were performed with a channel located in position 9. This was done using finite elements and by varying the dimensions of the channel (as shown in Figure 10) to 174.4 m from the toe of the heap. The variable dimensions are the length (L) and depth (H); the slope remains constant at 2H:1V. The thickness of the low permeability soil layer is 0.30 m. Table 5 presents the results obtained.



H (m)	L (m)
0.75	3.50
1.20	5.00
1.65	6.50

Figure 10: Varying channel dimensions

Table 5: SF in different combinations of channel dimension

H (m)	Static			Pseudo – Static		
L (m)	0.75	1.2	1.65	0.75	1.2	1.65
3.5	1.724	1.740	1.743	1.095	1.103	1.094
5.0	1.719	1.740	1.740	1.094	1.096	1.096
6.5	1.728	1.738	1.720	1.094	1.093	1.095

These results indicate that there are no significant SF increases with increases in channel length; however, there is a very slight trend of increases to SF with increasing depth, regardless of the channel length.

Sliding surface analysis

Displacement analysis

Figure 11 shows the distribution of displacements in the pseudo-static load condition. It shows that one face of the channel tends to contain displacement; this is probably one reason why the sliding surface tends to approach the heap leaching slope as shown in Figure 12, with the placement of a channel at the location indicated there.

This can be checked when a second channel is projected at a certain distance from the first channel. Incremental displacement may be critically affected, as discussed above. In this case, the sliding surface comes even closer to the heap slope, as shown in Figure 13.

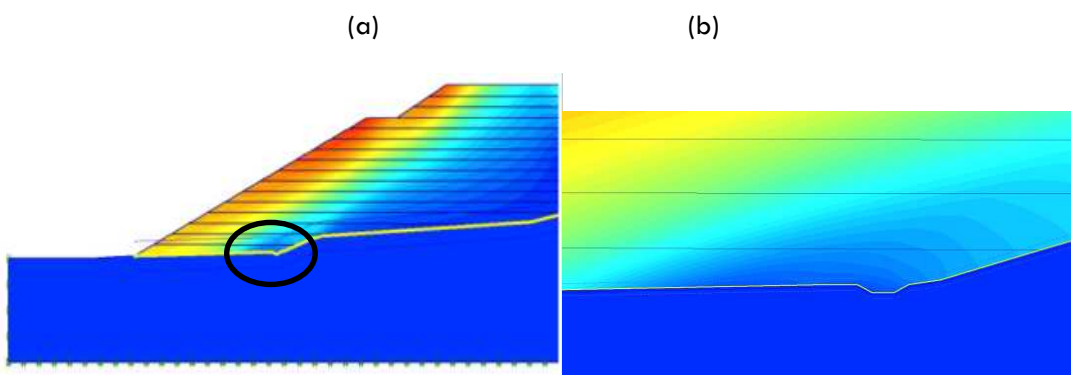


Figure 11: Distribution displacement in the pseudo-static condition: (a) general scheme, (b) detail

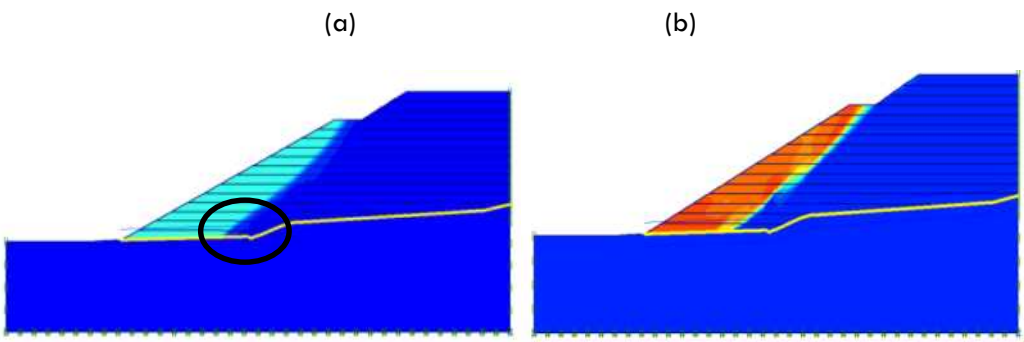


Figure 12: Failure surface with the placement of a channel in (a) static conditions and (b) pseudo-static conditions

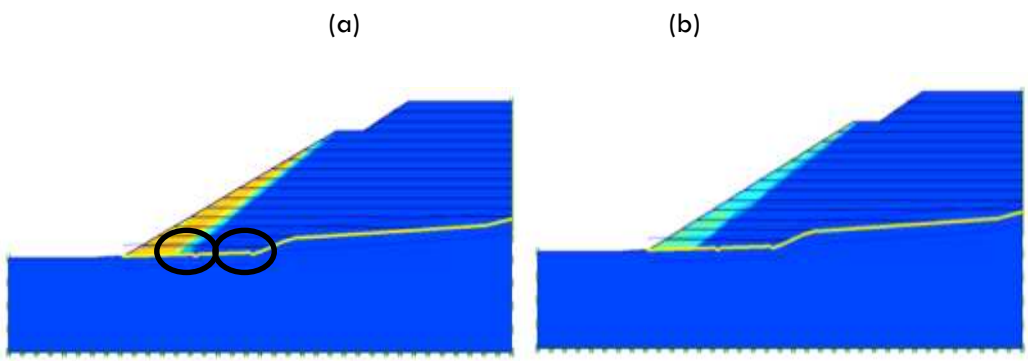


Figure 13: Failure surface with the placement of two channels in (a) static conditions and (b) pseudo-static conditions

Table 6 is a summary of the results obtained through finite element analysis in terms of stability and mass mobilized for sliding surface, considering one and two channels located in position 9.

Table 6: Summary of results — SF and mobilized mass for one and two channels

		Without Channel	One Channel	Two Channels	
				First case	Second case
Location of the chanel regarding foot of heap leaching (m)		-	174	70	105
				174	174
Safety factor	Static	1.697	1.740	1.768	1.765
	Pseudo - Static	1.058	1.103	1.127	1.116
Increase of safety factor	Static	-	2.56	4.18	4.00
	Pseudo - Static	-	4.25	6.52	5.48
Mobilized mass in case of a potencial failure regarding To failure surface without berms or channels (%)	Static	100	66	60	40
	Pseudo - Static	100	66	60	38

Analysis comparing finite element and limit equilibrium methods

Table 7 presents a comparison between the safety factors obtained from the placement of one and two channels through the finite element and limit equilibrium methods.

Table 7: Safety factor values by finite element and limit equilibrium methods

Condition Analyzed	Methods of analysis	Without Channel	With Channel
Static	Limit Equilibrium	1.674	1.740
	Finite Elements	1.697	1.740
Pseudo - static	Limit Equilibrium	1.085	1.148
	Finite Elements	1.058	1.103

Figure 14 shows the failure surfaces under both conditions tested (static and pseudo-static), calculated using the limit equilibrium method. It can be seen that these surfaces are almost the same.

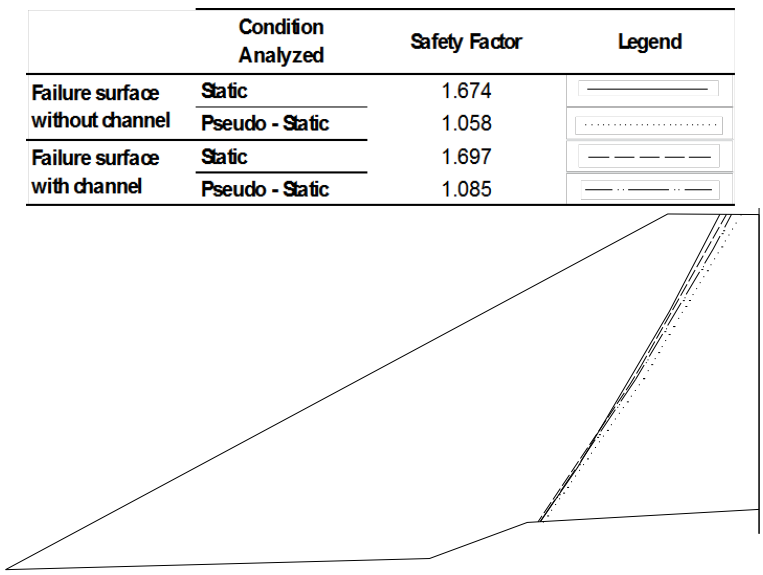


Figure 14: Failure surfaces and safety factors by limit equilibrium method

Similarly, Figure 15 shows the sliding surfaces by finite element.

	Condition Analyzed	Safety Factor	Legend
Failure surface without channel	Static	1.592	
	Pseudo - Static	1.058	
Failure surface with channel	Static	1.740	
	Pseudo - Static	1.103	
Failure surface with two channels	Static	1.765	
	Pseudo - Static	1.116	

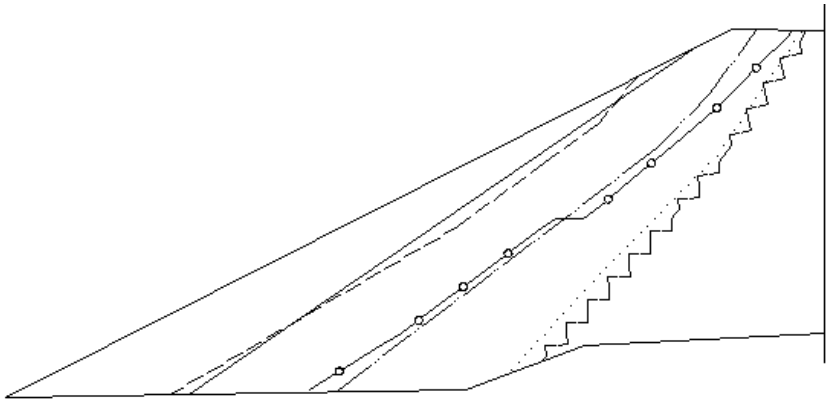


Figure 15: Failure surfaces and safety factors by finite element method

Conclusions

Analyses performed lead to the conclusion that the greatest influence on the safety factor of a slope at a heap leach facility is the design of a channel with standard dimensions, instead of a berm. Likewise, the failure surface is closer to the slope of the heap leach facility; this mobilizes a minor amount of mass when a channel is designed and increases the safety factor. Projecting a second channel produces a greater increase in the safety factor and significantly reduces the mobilized mass in the failure surface. Limit equilibrium methods are valid for calculating the stability gained by including berms and/or channels; however, failure surfaces are better represented using the finite element method. A basic constitutive model, such as Mohr-Coulomb, can be used for analyzing SF. The total load due to stacking of the ore can be calculated in a single-phase calculation, not taking into account the existence of low permeability soil; this results in a significant savings in computational time equal to almost a quarter of the time that would be spent without these simplifications.

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The importance of geomechanics to lateritic nickel heap leaching

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Abstract

The concept of using a heap leach process to recover metals is not new but the majority of heap leach projects focus on the recovery of copper and gold where there is a low percentage of overall mass loss arising from the metal recovery. As a result, there is usually little need for detailed analysis arising from changes in overall volume, shear strength, mass, porosity and permeability during the leaching process. However, the changes that occur during leaching become significantly more important when considering the modeling and design of heap leach projects that seek to leach nickel from lateritic ore. Detailed analyses are necessary in order to understand the effects of ore decrepitation on heap permeability and stability.

A theoretical approach to modeling lateritic nickel heap leach facilities requires the application of both saturated and unsaturated soil mechanics when viewing the problem from a geomechanical perspective. The basic volume-mass relationship needs to be revised to account for liquor density. The loss of mass during leaching results in changes to the volume-mass relations, (e.g., degree of saturation or saturation ratio, water or fluid content, void ratio and porosity, liquor density, and specific gravity). Consequently, numerical simulation of seepage and other processes become more complex. The results of case studies indicate that as decrepitated laterite ore approaches saturation, collapse often occurs. The saturation ratio needs to be carefully controlled to provide sufficient flow, and hence dissolution of minerals, without initiating structural collapse. Load-percolation and load-permeability tests, classification tests and specialist soil mechanics tests (e.g., water retention or soil-water characteristic curves) are required in order to calculate the saturation ratio in the heap.

Introduction

The concept of using the heap leach process in order to recover precious minerals is not new, but the vast majority of heap leaching undertaken to-date has been on copper and gold ores. An example of heap leaching is shown in Figure 1. The metals recovered from these projects are typically leached from hard rock ores where there is minimal need to consider the effects of changes in overall volume, shear strength, mass, porosity, and permeability as the leaching process progresses. However, changes that occur during leaching nickel from lateritic ore become significantly more important when considering the modeling and design of heap leach projects. Changes in overall volume, shear strength, mass, porosity and permeability become critical to analyses performed to simulate the effects of ore saturation ratio and decrepitation on heap permeability and stability. These factors are closely related to the success of a heap leach project.

Figure 1: Example of heap leach operation

A theoretical approach to modeling a lateritic nickel heap leach facility requires the application of both saturated and unsaturated soil mechanics from a geomechanical engineering perspective. In many cases, only saturated soil mechanics principles are given consideration. However, studies to-date would indicate that the structure of agglomerated laterite nickel ore often collapses if the material comes too close to saturation. The saturation ratio of the heap leach material needs to be carefully controlled to ensure sufficient flow through the heap leach material, and hence the dissolution of metals. These conditions need to be met without initiating collapse of the heap material. This paper outlines an approach that has been successfully implemented to characterize laterite nickel heap leach materials and support the design of a heap leach facility.

An appropriate soil mechanics theoretical context heap leach

Since the publication of *Soil mechanics for unsaturated soils* (Fredlund and Rahardjo, 1993), and *Unsaturated soil mechanics in engineering practice* (Fredlund et al., 2012), the consideration of many soil mechanics problems has included consideration of unsaturated zones. Seepage, volume change and shear strength problems can now be solved using extended constitutive equations from those originally proposed by Terzaghi in 1943. Fredlund and Rahardjo (1993) noted that while the importance of unsaturated soils was raised by Terzaghi, little focus was placed on the development, and subsequent evidential proof using controlled experiments that verified the constitutive equations for seepage, volume change and shear strength problems. In recent years, it has been recognized that previously applied analytical techniques do not fully account for the effects of mass loss during the heap leach process. In addition, the implication of having pore fluids with densities significantly higher than water was not taken into consideration.

It is necessary to draw upon the theories of unsaturated soil mechanics in order to more accurately represent the process of lateritic nickel heap leaching. The basic volume-mass relationship in soil mechanics need to be carefully examined and updated to accommodate the physical and chemical processes associated with heap leaching. Essentially every textbook on soil mechanics contains the assumption that the pores in a soil are filled with water at a density, ρ_w of 1.0 t/m^3 (or 1000 kg/m^3). However, the pore fluid associated with a laterite nickel heap leach project is likely to be a combination of acid, water and dissolved metals. There is also change in pore fluid concentration over the leaching cycle. The pore fluid will typically have a varying density which is significantly greater than that of water. The pore fluid density can be denoted as ρ_f, and must replace the water density variable, ρ_w, used in volume-mass equations. The change in pore fluid density can have a significant effect on laterite nickel heap leach analysis.

The basic volume-mass relationship commonly used in geotechnical engineering, (i.e., $Se = wG_s$), must take into consideration a variation in the pore fluid density. Porosity, n, void ratio, e, volumetric water (fluid) content, θ_f and saturation ratio, S, are defined in terms of volume designations, and therefore remain unchanged when the pore fluid is not pure water. Similarly, the equation for dry density, ρ_d, remains unchanged since this variable relates the mass of solids to the total volume. However, the bulk density of a soil and the above-mentioned basic volume-mass relationship must be examined and derived from first principles with the inclusion of pore fluid density being a variable.

The re-derived basic volume-mass equation was presented by Mundle et al. (2012):

$$G_s w_f = S_r e \rho_f \tag{1}$$

The density of the pore fluid now appears in a re-derived form of the "Basic Volume-Mass Relationship". If the pore fluid is water, the density of the pore fluid is 1.0 t/m^3 (or 1000 kg/m^3) and can be omitted from the equation. In all other cases, the density of the liquor must be taken into account. The revised volume-mass equation now incorporates gravimetric liquor content, w_f, which is defined as the mass of liquor divided by mass of solids. The gravimetric liquor content must be measured in order to satisfy the revised volume-mass equation.

Mundle et al. (2012) provided examples that showed the errors that can be encountered if the pore fluid density is not accounted for in calculations. In an example presented, the following laboratory measurements were assumed to be measured; namely, the specific gravity of solids was 2.65, void ratio was 0.90 and the water to liquor ratio was 0.34. Without accounting for the liquor density, a saturation ratio of 1.0 was estimated. Mundle et al. (2012) indicated, however, that for a liquor density of 1.2 t/m^3, a saturation ratio of 0.83 was calculated, resulting in a potential error in the saturation ratio which is the key variable used in decision-making.

Prior experience with lateritic materials would suggest that when a degree of saturation level of 100% is reached, the agglomerated material has a propensity to collapse. Associated with the material collapse is a rapid decrease in the coefficient of permeability and a loss of strength. It is advisable to conduct the leaching of lateritic nickel ore under unsaturated conditions. The adoption of unsaturated soil mechanics principles is considered essential when developing a theoretical approach to the analysis of lateritic nickel heap leach systems.

The benefits of using unsaturated soil mechanics principles can be listed as follows:

- Unsaturated soil mechanics provides a sound theoretical basis for estimating overall volume change, seepage and stability.
- There can be a smooth transition of theoretical models between the unsaturated and saturated heap leach conditions.
- Unsaturated soil mechanics' principles provide a more robust approach in geotechnical engineering and allows for the modeling of changes in strength, overall volume change and changes in the coefficient of permeability.

Lateritic nickel heap leach materials

Laterite nickel ore is typically clayey in nature, with 100% of the particles passing the 75 µm sieve. In many cases, the material is highly plastic and has a low coefficient of permeability when compacted. To achieve the most suitable leaching performance, the ore needs to be agglomerated, usually with acid, to create particles up to ~25 mm in diameter (Figure 2). The agglomerated particles have an increased overall porosity and a macro-scale hydraulic conductivity.

Figure 2: Example of agglomerated laterite nickel ore

Placement of the agglomerated material on to a heap (or into molds in the laboratory) creates a loose structure. As more material is placed, the material nears the base of the heap is subjected to overburden loading which reduces the porosity (and void ratio) and increase the saturation ratio, even before leaching begins. As leaching is carried out, the material near the base has the lowest hydraulic conductivity, and hence will get wetter more rapidly, potentially resulting in full saturation and ultimately, the collapse of the agglomerates (Figure 3). This material then controls the performance of the heap, and warrants further consideration.

Figure 3: Laterite nickel ore after loading and saturation

Fluid retention curves

Geotechnical engineering problems have recently evolved to include the application of unsaturated soil mechanics. The implementation of unsaturated soil mechanics has largely been made possible through use of a fluid retention curve (FRC), frequently referred to as a soil-water characteristic curve (SWCC). The FRC is considered to be the key soils information required for the modeling of water (and liquor) flow through both saturated and unsaturated materials. Fredlund and other researchers have published a large number of papers on the use of the SWCC curve in geotechnical engineering practice. As a result, the profile of unsaturated soil mechanics has risen significantly.

The FRC is used to characterize the unsaturated behavior of a soil. It's the unsaturated permeability function and the fluid storage function can be derived from the FRC. The FRC is considered to be a tool of primary importance when simulating the wetting and drying behavior of soils and artificially produced materials. Even though the FRC is hysteretic in character, methodologies are available for the estimation of permeability and water storage properties that can be used in transient analyses. Williams (2006) presented a summary figure illustrating the key features of the FRC, reproduced in Figure 4.

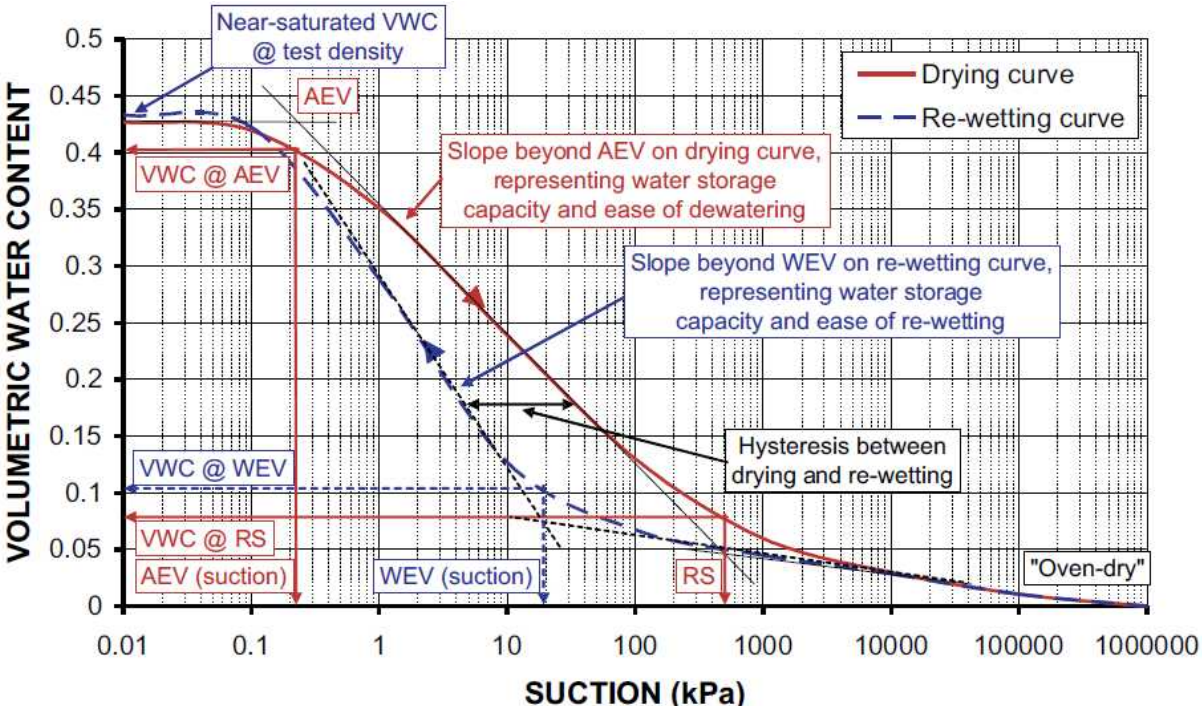

Figure 4: Key features of drying and wetting SWCCs (Williams, 2006)

The following key features are evident in Figure 4:

- The intercept on the y-axis, represents the porosity of the material.
- The air-entry value (AEV), is the term given to the point at which the material is unable to remain saturated, (i.e. the point where air begins to replace the water in the largest voids and the material starts to drain). The air-entry point is important in the context of the heap analysis since it is the point that defines when desaturation begins to occur.
- The slope of the curve represents the "difficulty" of removing water from the material. A flatter curve requires a greater increase in soil suction to remove water from the material (e.g., clay) and a steeper curve suggests that the material will readily release water (e.g., sand).
- The water-entry value (WEV) is similar to the AEV but is the point at which water re-enters the material upon wetting.

Heap leach theoretical flow model

Heap leaching requires the application of a pregnant liquor solution for the dissolution of metals. The dissolved metals are collected at the base of the heap for processing. The leaching process needs to be achieved without inducing heap instability, which can be rapidly brought about if the material saturates.

Upon saturation, the material collapses and produces a saturated zone (i.e., phreatic surface) near the base of the heap. Seepage through the heap is of paramount importance and the changes in seepage flow over time needs to be considered as an inherent property of the ore. Pore fluid flow can continuously change as leaching progresses. The unsaturated hydraulic conductivity of a material is linked to the FRC (Figure 5). The AEV has a strong influence on changes in hydraulic conductivity. Figure 6 shows that the point at which hydraulic conductivity begins to reduce is the AEV.

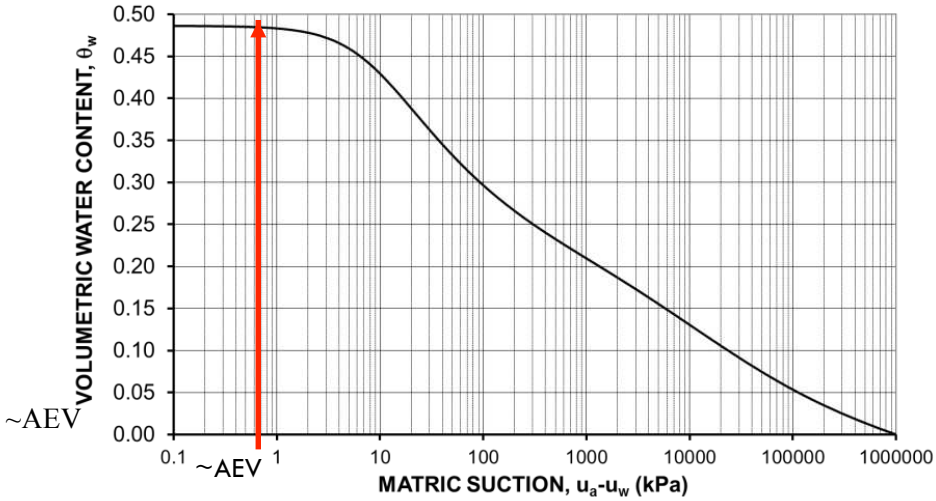

Figure 5: Typical fluid retention curve

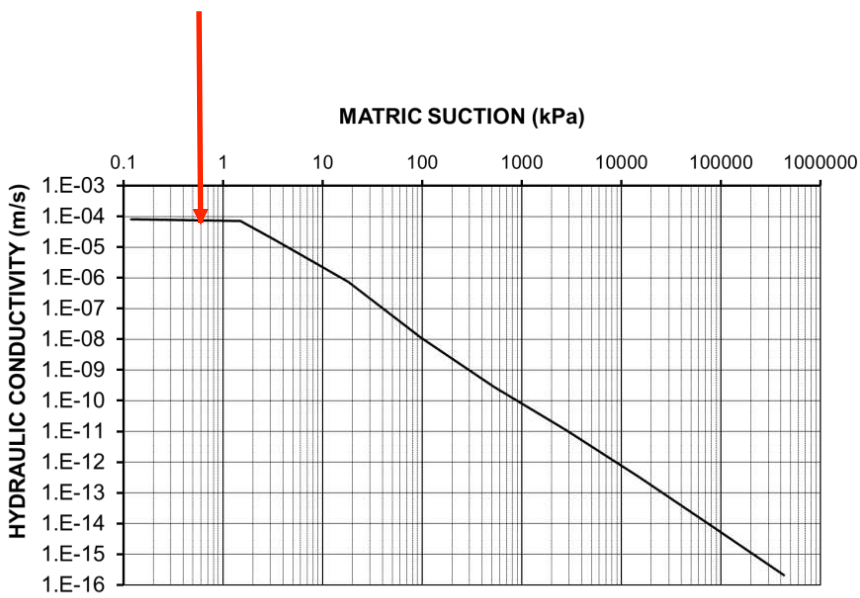

Figure 6: Associated hydraulic conductivity function

Figure 6 shows that the hydraulic conductivity of the material decreases significantly once the matric suction exceeds the AEV of the material. For heap leaching materials, the AEV is of significance as the agglomeration process alters the initial FRC for the material. The AEV reflects whether the material is sandy or gravelly in character. Coarse-grained materials have alow AEV while clayey materials have a high AEV. However, it should be noted that the FRC for the material does not remain constant. In addition to the hysteretic character of the FRC, the heap leach material also changes during the leaching process.

In consideration of Equation 1, the soil mechanics framework reveals the complexity of geomechanics when applied to laterite nickel heap leaching. In saturated soil mechanics, the specific gravity of the solids is constant, the saturation ratio is 1.0 and the pore fluid is water. Only one of the two remaining components (i.e., e or w) is required to calculate the other volume-mass properties. Hence, relatively simple laboratory tests or field tests can be used to characterize the material. In unsaturated soil mechanics, the saturation ratio is unknown and hence two components are required to fully characterize the volume-mass properties of the material. In laterite nickel heap leaching, the components of Equation 1 vary significantly over time. The end result is increased complexity in laboratory testing. In addition, the tests are time-consuming and require careful technician attention. Following are the changes in properties that need to be understood.

- The fluid content increases as liquor is applied to the heap.
- The specific gravity of the material decreases over time due to the dissolution of the metals (dominated by iron and nickel).
- The pore fluid density increases as metals dissolve.
- The saturation ratio increases due to changes in the other components, particularly the fluid content.
- The void ratio initially increases as metals are dissolved, creating more void space, with the potential for rapid decreases in permeability due to collapse upon saturation .

The rate at which each of the components listed above changes is challenging to estimate. The relative impact of small material changes depends on the material and the FRCs.

The drying FRC curve is usually estimated or measured for design purposes. Nevertheless, it is the wetting curve which is of most importance in the simulation of the heap leach process. There is unlikely to be a drop in the saturation level once leaching begins (i.e., until drain-down occurs). The materials start relatively dry, with a saturation ratio of perhaps 0.5 to 0.6. As liquor is applied to the heap, the low unsaturated hydraulic conductivity results in an increase in the fluid content rather than having the liquor pass through the material. This process will continue until the unsaturated hydraulic conductivity

increases to a point equal to the application rate, as shown in Figure 7. At this point, equilibrium has been reached and leaching is occurring.

The mechanics of heap leaching, however, result in dissolution of the material, and hence the hydraulic conductivity, typically a function of a constant saturated hydraulic conductivity now varies with the changing saturated hydraulic conductivity.

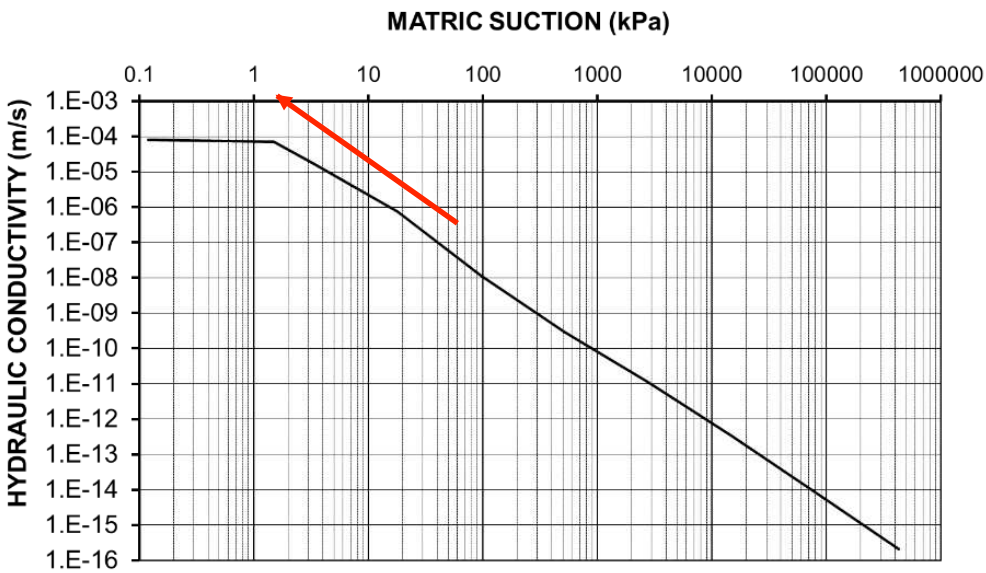

Figure 7: Increasing unsaturated hydraulic conductivity

It is important to know the point at which flow equilibrium is initially established. As the saturation ratio increases, it is possible for the material structure to be lost and structural collapse may occur. For material with little storage capacity (e.g., a steep FRC), small changes in fluid content will result in large changes in saturation and could potentially result in structural collapse. For materials with higher storage capacity (e.g., a flat FRC), changes in the saturation ratio may be less and hence these materials will be less susceptible to collapse. Nevertheless the higher storage materials are still likely to operate at a lower unsaturated hydraulic conductivity and hence need to be subjected to a lower application rate. The lower application rates will require longer cycle times (Figure 8).

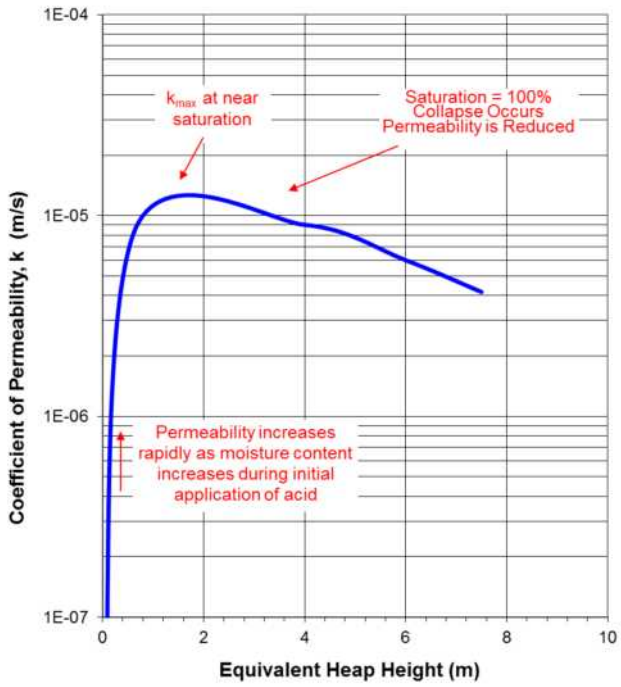

Figure 8: Effect of application of liquor

Link to metallurgical column testing

Metallurgical column testing is typically carried out well ahead of geotechnical testing. The metallurgical column tests define the potential recovery from candidate heap leach materials, and provide information on the optimum application rate for the liquor. The data gathered during this process can be used to support predictions (or estimations) of the geomechanical behavior of the materials. The liquor is regularly sampled and assayed during a typical metallurgical column test. The test results allow for an estimation of the reduction in specific gravity of the ore to be calculated (mass loss due to dissolution) by identifying the relative masses of dissolved metals in the liquor. Once the liquor density can be calculated, two of the five components of the volume-mass relationship are known. A relatively simple modification to the testing arrangement (i.e., the addition of a scale beneath the column), can provide information on the remaining three components.

If the mass of the column and the application rate are known at all times during the leaching period, the total volume of the column can be tracked and divided into its subcomponents; that is, the volume of solids (reducing over time), volume of liquor (typically increasing over time, calculated by subtracting the outflows from the inflows) and the volume of voids (increasing due to dissolution which is calculated by subtracting the volume of solids from the total column volume). With these sub-components known, the saturation of the column can be calculated by dividing the volume of liquor by the volume of voids.

Figure 9 presents the results of a column volume balance carried out by the authors. The change in saturation that occurs at about day 3 is attributed to a slump in the ore, indicated by the rapid drop in total column volume, initiated by the application of liquor. The increase in the saturation ratio between days 3 and 7 is attributed to the accumulation of liquor in the pore space due to a reduction of permeability of the material. The saturation ratio subsequently stabilizes to about 65%. If the application rate were to be increased, the saturation ratio might increase concomitantly and trigger a second collapse, significantly reducing the permeability of the ore and rendering the leaching process ineffective.

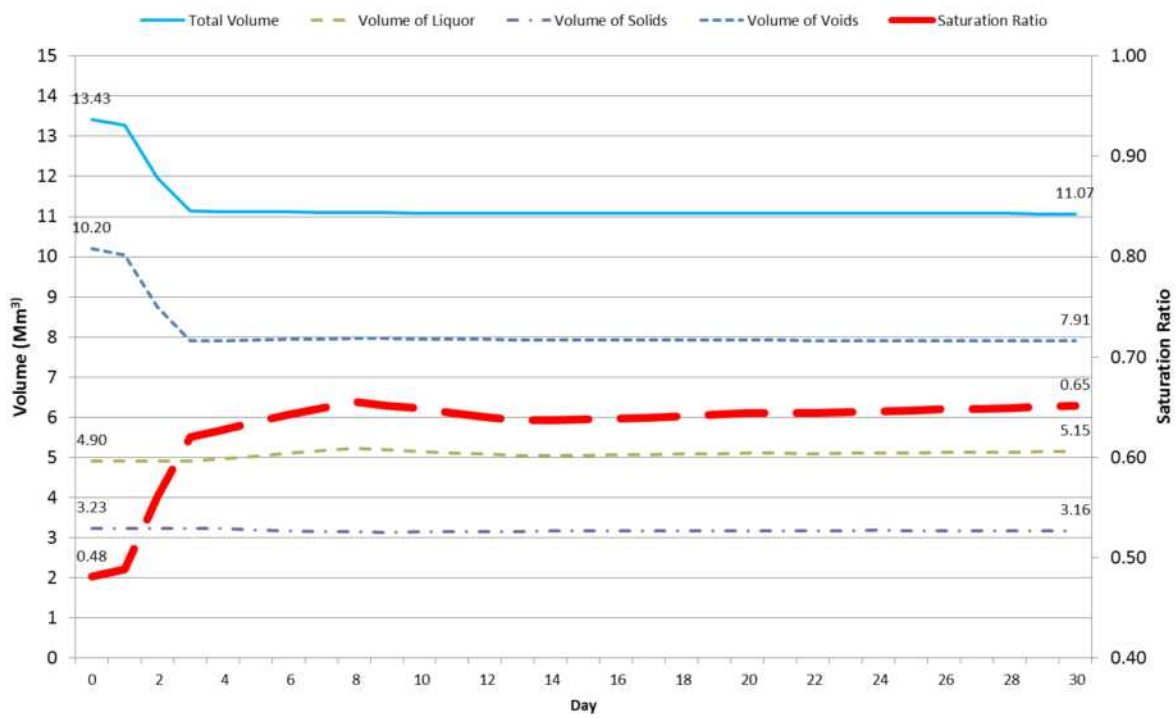

Figure 9: Example of column volume balance

Summary and conclusions

It is important to understand the geomechanics of the materials when designing a laterite nickel heap leach facility. Small changes in saturation can significantly impact the heap performance, either by inducing collapse or increasing the cycle times. A theoretical saturate-unsaturated soil mechanics framework can be used to predict changes in saturation. Laboratory testing must be carried out to provide appropriate characterization of the materials in question (i.e., FRCs). From this information reasonable estimates of permeability and fluid storage can be estimated. These estimates can be compared with calculations from existing (but slightly modified) metallurgical column tests. Modifications to the columns allow for the tracking of inflows, outflows and mass balance during the testing process.

Applying additional rigor to the tracking and estimation of the saturation ratio within laterite nickel heap leaching can result in increased accuracy in describing the leaching process. The heap leach simulations can result in greater optimization and increased confidence in the simulation of the leaching process. This could reduce cycle times, allow greater application rates, and the optimization of the heap height. All of these factors can translate into lower costs of operation and increased revenues.

Modeling of the heaps is, however, complex since each of the five components of the volume-mass relationship are continuously changing during the leaching process. Predicting the general performance of the heap leach operation, based on column testing can be completed but predicting localized impacts and ensuring that dissolution occurs evenly across the heap is challenging. Numerical simulation models can be used provided that the geomechanics of the materials are sufficiently well understood through extensive laboratory testing and possibly field trials. Up-front testing of heap materials can be costly but can also lead towards greater success of the project.

The following recommendations are made to assist in understanding the geomechanical performance of decrepitating ores under consideration for heap leach projects:

- Install balances beneath metallurgical columns to provide an indication of mass over time. The measurements should be combined with the regular assays of the pregnant liquor with the intent to develop a volume balance for the column that allows for the prediction of the saturation ratio.
- Carry out small-scale laboratory tests to identify the impact of the saturation ratio on the permeability of materials at various stages of leaching.
- Combine unsaturated soil mechanics theories with heap leach concepts for the better understanding of the impact of liquor density on the heap.
- Use the outcomes of the above measurements and predictions to optimize the heap height, application rate and cycle times.

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Effect of strongly acidic leachates on hydraulic conductivity of a needle punched geosynthetic clay liner

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Abstract

In the past few years, geosynthetic clay liners (GCLs) have been considered for deployment as hydraulic barriers to contain aggressive leachates, as well as to operate under environmental and geotechnical conditions for which they were not designed. For example, the use of GCLs as liners in metal ore heap-leach processing imposes both aggressive leachates and excessive stresses on the GCL. This paper addresses some of the issues associated with the impact that reaction of extreme acidic leachates (up to 0.5 M /l sulfuric acid) may have on hydraulic conductivity of GCLs. Under standard conditions (i.e., 35 kPa effective stress), the ratio of the hydraulic conductivity (k) permeated with a range of H_2SO_4 concentrations for non-prehydrated specimens to the k values based on the permeation of DI water (k/k_w) ranged between 4 and 42. At 200 kPa effective stresses, acid concentrations ≥0.125 M resulted in k values $< 4 \times 10^{-11}$ m/s, (ratio k/k_w ranging from 4 to 23) indicating manageable performance of GCLs under heap-leach conditions, at least in the short term. Loss of GCL performance is strongly dependent on leachate ionic strength at low effective stress, but at 200 kPa, this effect was less evident.

Introduction

The use of geosynthetics in various mining operations is now widespread. While they have not been accepted as readily as in the general construction market, growth in the mining industry is occurring as operators begin to understand the advantages associated with the use of these materials. For many applications, such as load support and retaining structures, the design and application is easily transferred to the mining field, albeit normally with far higher loading criteria. Examples of these types of applications are reported in Bouazza et al. (1995), Lupo and Morrison (2007), and Hornsey et al. (2010).

Containment of mine wastes and remediation of mine operations potentially subject geosynthetics to operating conditions beyond their design capability. Thus, it is not a simple matter of transferring the

technology from the tried and tested applications common to waste containment facilities such as landfills, where information is widely available and well established, to the mining industry. The long-term performance of geosynthetics subject to extreme ranges in leachate properties generated from the various ore extraction processes and the harsh environment to which geosynthetic materials are therefore exposed are primary issues that still need to be addressed by research. Nevertheless, the rapid growth in mining exploration and operation in the past decade has led to a sharp increase in the use of a wide range of geosynthetic materials by the mining industry for all types of applications. Smith (2008) reported that, from 1987 to 2008, more than 60 square kilometers of geomembrane liners were installed in leach pads alone. In addition to geomembranes, which are extensively used in evaporation ponds, heap leaching, and disposal of tailings (Breitenbach and Smith, 2006; Thiel and Smith, 2004), other major containment applications include geosynthetic clay liners (Lange et al., 2007; Bouazza and Rahman, 2007; Benson et al., 2008). Geosynthetic clay liners (GCLs) are thin (typically 5 to 10 mm thick) manufactured hydraulic barriers comprised of a thin layer of bentonite bonded to layers of geotextiles and/or a geomembrane. Interest in the use of GCLs as the secondary liners for leach pads has increased markedly in the past decade. Just as for geomembranes, the application of GCLs in mining generally pushes the performance beyond recommended limits typical for other environmental and engineering applications.

Exposure of the GCL to high overburden and traffic stresses and excessive temperatures as well as high salinity and extreme pH of the leachates and liquors may not only affect the geosynthetic components but can also negatively affect the performance of the bentonite component. The ability of bentonite to maintain a gel state with low hydraulic conductivity can be seriously impaired when exposed to leachates of excessive ionic strength (>0.3 M), elevated temperatures (>60°C), and either strongly acid or strongly alkaline pH (Gates et al., 2009). Acid attack of clays has been used to advantage industrially (Fahn, 1979; Gates et al., 2002), but less is known directly regarding the effect of strongly acid pH on performance of GCLs.

The primary purpose of this study was to evaluate the compatibility of a geosynthetic clay liner to acidic solutions with extreme pH. The assessment of the hydraulic performance of the specimens reported in this paper was based on the measurement of hydraulic conductivity when exposed to 0.015 M, 0.125 M, and 0.5 M H_2SO_4 (leachate pH = 1.48, 0.66, and 0.25, respectively). The results of this study indicate that the hydraulic performance of bentonites and the GCLs can be negatively impacted by sulfuric acid solutions, but that applying increased effective stress from 35 kPa to 200 kPa to the GCL ameliorates the hydraulic conductivity.

Materials and procedures

Geosynthetic clay liners

A commercially available needle punched and thermally locked GCL was used in this study. It is referred to herein as GCL1; powder bentonite formed the core of the GCL. GCL1 contained a minimum of 5.26 kg/m^2 activated (undisclosed beneficiation) powdered sodium bentonite sandwiched between a non-woven polypropylene geotextile cover layer and a woven geotextile carrier. The cover and carrier geotextiles had reference mass per unit areas of 0.25 kg/m^2 and 0.15 kg/m^2, respectively. The mass per unit area of GCL1 was 5.66 kg/m^2 with an initial unhydrated thickness of ~6.5 mm. Its hydraulic conductivity to water was ~1.1×10^{-11} m/s at 35 kPa effective stress; the bentonite component had a swell index (SI) of 23 mL/2g.

Permeant liquids

A series of sulfuric acid solutions with concentrations of 0.015, 0.125, and 0.5 M, diluted from the 98% sulfuric acid reagent (from Merck Chemicals, Melbourne, Australia), were used in this study. The resulting pH range was intended to mimic extreme acid conditions. While most acid mine leachates have a usual value of pH~3, pH values < 1 may occur when water evaporates from acidic pools, thereby increasing the concentration of hydrogen ions. In addition, the sulfate scrubbers in coal power plants may produce high concentrations of sulfuric acid (Fernández et al., 1997). Such acid is routinely used to acid-wash mineral materials (Souza et al., 2007; Kuan et al., 2010), and its excessive concentrations can potentially result in very low pH, especially initially, when dissolved metals concentrations may be low (Miessler and Tarr, 1998). The pH, electrical conductivity (EC), and ionic strength of these acid solutions are shown in Table 1.

Table 1: Properties of H_2SO_4 solutions used in the present study

H_2SO_4 concentration (M)	Target pH	Measured pH	EC (mS/cm)	Calculated ionic strength (M)
0.015	1.5	1.48	7.47	0.045
0.125	0.6	0.66	66.9	0.375
0.5	0	0.25*	197.6	1.5

Note: *Accuracy limited by measuring range of the pH meter. EC is electrical conductivity

Hydraulic conductivity test

The hydraulic conductivity tests were conducted using flexible-wall permeameters in general accordance with ASTM D 5084 and ASTM D 6766. The hydraulic conductivity tests adopted in this program used the constant flow method with flow pumps and a standard flexible wall permeameter. The influent and effluent flows were controlled by pressure/volume controllers capable of accurately inducing either pressure or flow. The controllers were connected to a computer allowing automatic monitoring and logging of inflow and outflow. The pressure gradient across the sample was monitored using the inflow and outflow pump whereas the cell pressure was controlled by a third flow pump. A constant pressure was set in the effluent pressure line and a constant flow was applied to the influent line; the hydraulic gradient induced could then be observed using the differential pressure recorded by the inflow and outflow pumps. The pedestal and cap were manufactured from Teflon. All tubes were acid resistant and the fittings were stainless steel to resist corrosion by acidic solutions.

GCL specimens were trimmed from a larger sheet by cutting around a steel cutting ring with an inner diameter of 76 mm using a sharp knife. To limit loss of bentonite during cutting, a small amount of permeant solution was sprinkled on both the inside and outside edges of the cutting ring. After removal from the ring, excess geotextile fibers were removed from the edges of the specimens with sharp scissors. Then the initial mass and thickness of the GCL specimens were measured. Each specimen was then placed in a flexible-wall permeameter, the permeameter was assembled, and the specimen was back-pressured with acidic solutions. All tests involving interaction with acidic solutions were conducted on non-prehydrated specimens (i.e., specimens were directly exposed to acidic solutions). Hydraulic conductivity tests were also conducted with DI water as the permeating liquid to establish a hydraulic conductivity reference baseline. The hydraulic gradient used in all tests in this study was ~250, which, while higher than recommended (30) in ASTM D5084, is acceptable as reported by Shackelford et al. (2000).

The tests were continued until chemical equilibrium was achieved in the system. Namely, the ratio of inflow and outflow EC and pH (respectively, EC_{out}/EC_{in} and pH_{out}/pH_{in}), were within 1± 0.15 as described by Shackelford et al. (1999). To obtain real-time EC and pH values, two sets of pH and EC sensors were installed in both inflow and outflow lines. A summary of the results obtained from the hydraulic conductivity tests is given in Table 2.

Table 2: Results of hydraulic conductivity tests

Effective stress (kPa)	Test parameters	Test duration (day)	Pore volume of flow (PVF)	Hydraulic conductivity	
				k (m/s)	k/kw
35	DI water	22	3	1.1×10^{-11}	1.00
	0.015 M	61	25	4.6×10^{-11}	4.20
	0.125 M	43	21	1.6×10^{-10}	14.55
	0.5 M	34	23	4.6×10^{-10}	41.80
200	DI water	71	3	1.5×10^{-12}	1.00
	0.015 M	149	21	2.0×10^{-12}	1.35
	0.125 M	114	21	6.5×10^{-12}	4.35
	0.5 M	67	25	3.4×10^{-11}	22.65

Discussion

The hydraulic conductivity of the GCL permeated with water reported in Table 2 was within the range typically reported for GCLs permeated with low ionic strength liquids, such as deionized water (e.g., Shackelford et al., 2000; Bouazza, 2002). Figure 1a shows a typical variation of the hydraulic conductivity against pore volumes of flow (PVF) of the acid solutions. In general, more than 20 pore volumes of flow (PVF) needed to pass through the specimens when permeated with sulfuric acid solutions to achieve equilibrium (see also Table 2). The breakthrough to the acid solutions occurred at 6 pore volumes of flow (PVF) for 0.5 M H_2SO_4 (at 35 kPa). Figure 1a indicates the presence of a hydration dominant zone present up to PVF 6, followed by a transition zone between PVF 6 to 10, and then a zone where the effluent pH and electrical conductivity (EC) indicated most of the exchange occurred. The hydraulic conductivity tests were not terminated until the volumetric flow ratio of inflow (Q_{in}) to outflow (Q_{out}), Q_{in}/Q_{out}, was within 1.0 ± 0.15, and the chemical equilibrium of each test was achieved before termination; namely, the ratios of both pH and EC in both inflow and outflow, pH_{out}/pH_{in}, EC_{out}/EC_{in}, were in the range of 1.0 ± 0.15. As indicated in Table 2, tests at effective stress of 200 kPa lasted at least twice as long as those at 35 kPa under both water and acidic conditions. Greater ionic strength required fewer days to reach chemical equilibrium, in general agreement with other studies regarding hydraulic conductivity tests with leachates having elevated ionic strength (Lee and Shackelford, 2005; Shackelford et al., 2010). Figure 1b indicates also that the ratio of pH between outflow and inflow, pH_{out}/pH_{in}, was greater than 1 and then approached 1 as the number of PVF increased. The ratio of EC between outflow and inflow, EC_{out}/EC_{in}, was lower than 1 and then increased to 1 as the number of PVF increased.

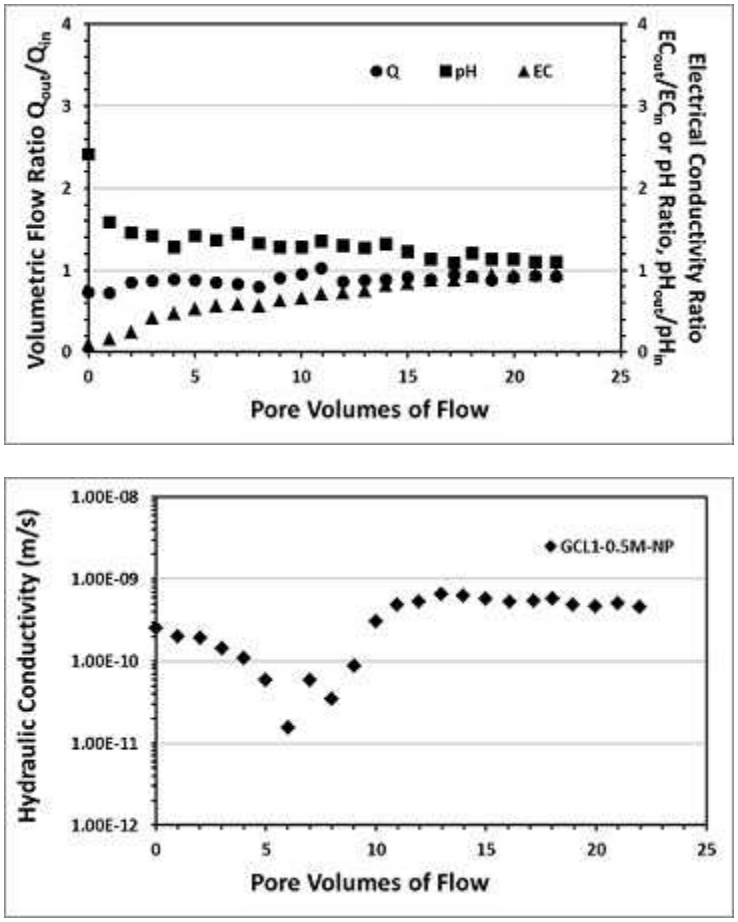

Figure 1: Typical hydraulic conductivity (a) and termination criteria versus pore volume of flow (b) under 0.5 M H_2SO_4 at 35 kPa for non-prehydrated GCL1 specimens

The hydraulic conductivity values for the GCL permeated with the acid solutions varied between 1.6 to 4.7 × 10^{-10} m/s at 35 kPa and between 6.5 × 10^{-12} to 3.4 × 10^{-11} m/s at 200 kPa (Table 2 and Figure 2); these values were higher than the initial hydraulic conductivity value for DI water at both effective stresses. The comparison between the hydraulic conductivity values obtained at 35 kPa and 200 kPa, respectively, are shown in Figure 2. The hydraulic conductivities values at 200 kPa effective stress were found to be lower (13.5~25× lower), over the range of acid concentrations investigated, than the values obtained at 35 kPa. However, while they still increased with increasing acid concentrations, the increase at 200 kPa was half of that at 35 kPa. It seems that the hydraulic conductivity is strongly dependent on the leachate ionic strength at low (35 kPa) effective stress. In contrast, it is relatively independent to ionic strength at high (200 kPa) effective stress, < 5× increase at ≤ 0.125 M (Figure 3 and Table 2). The effective stress appeared to be more significant than the effect of ionic strength in the present study. Use of higher effective stresses tend to mask the impact caused by the permeant liquid (Lee et al., 2005). The low hydraulic conductivity values obtained at 200 kPa in the present study indicates a compression of the

specimens, which indicates a decrease of the void ratio as well as a decrease in the size of the hydraulically active pores (Petrov et al., 1997; Katsumi et al., 2008).

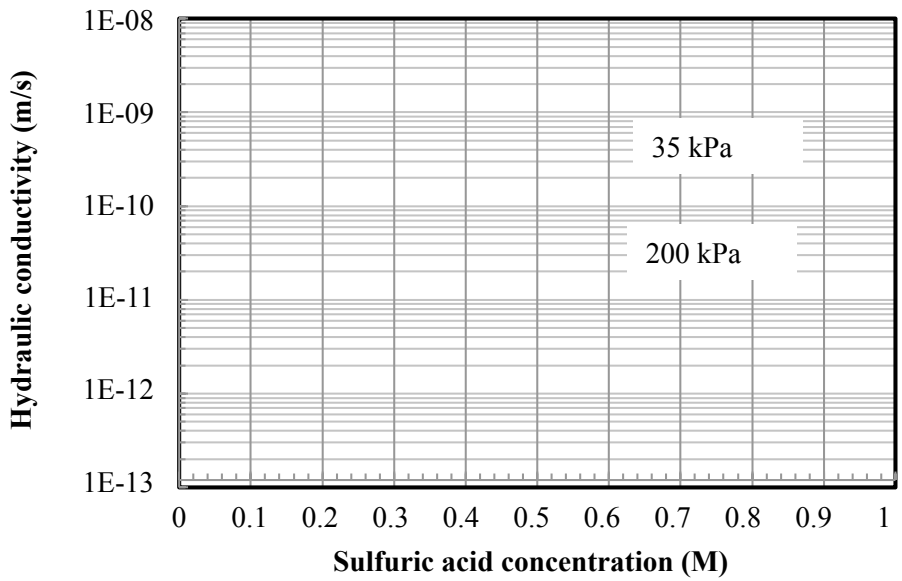

Figure 2: Variation of hydraulic conductivity versus acid concentrations

At low confining stress, the hydraulic conductivity of GCL1 is still acceptable (4.6×10^{-11}) at 0.015 M H_2SO_4 (k/K_w~4). A lower acid concentration (i.e., 0.015 M) had sufficiently low ionic strength (0.045 M) to suggest that the modest effect cannot be related to the soluble ion concentrations, but instead is probably related to the resulting changes to bulk and crystalline swelling caused by H^+ for Na^+ exchange (Liu et al., 2013). At high acid concentrations (i.e., 0.5 M), the dominating ionic strength causes the collapse of the diffuse double layer (van Olphen, 1977; Mitchell, 1993) resulting in a significant increase in hydraulic conductivity. Since dissolution of smectite will undoubtedly occur over the long-term (Komadel et al., 1990; Gates et al., 2002), bentonite swelling is expected to also be affected by the damage to the clay structure. Furthermore, the multivalent cations (Fe^{3+}, Mg^{2+}, Al^{3+}) released from the structure of smectite result in further increase in ionic strength and can impact swelling and hydraulic conductivity (Liu et al., 2013).

The values of hydraulic conductivity based on permeation with the acid solution relative to the values of hydraulic conductivity based on permeation with DI water, or k/k_w factor, ranged from 4 to 42 under 35 kPa effective stresses, indicating an adverse impact on the hydraulic performance of the GCL to even 0.125 M H_2SO_4. The same applies to samples tested at 200 kPa at relatively higher concentrations ($\geq$ 0.125 M, 4 to 23 increase in k/k_w). These effects are clearly depicted in Figure 3.

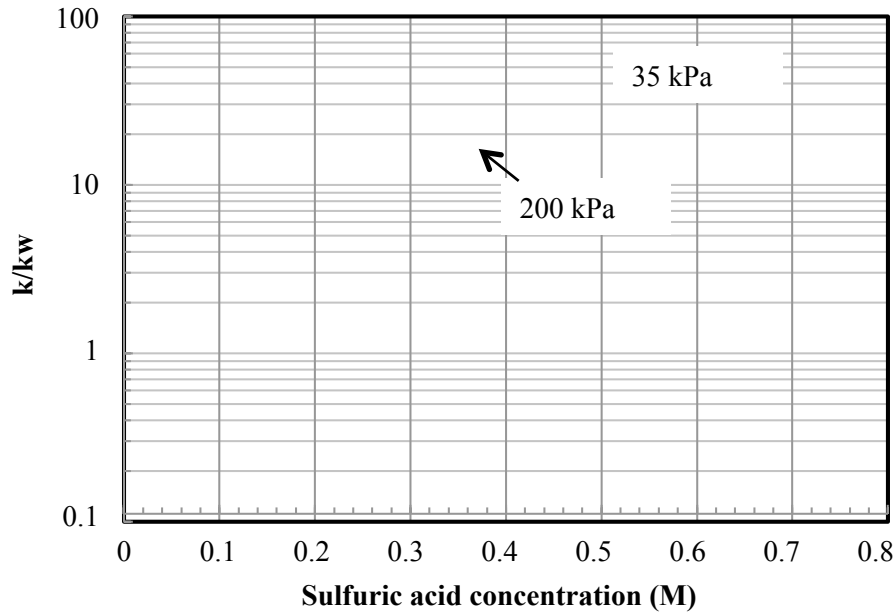

Figure 3: Ratio of hydraulic conductivity under acidic condition (k) to water condition (k_w), k/k_w, for all specimens with different concentrations of H_2SO_4

Conclusion

The hydraulic performance of geosynthetic clay liners permeated with strongly acidic solutions was evaluated using a flexible-wall permeameter. Results indicated that an increased acid concentration (ionic strength) resulted in the increase of the hydraulic conductivity for all the tested specimens. The values of hydraulic conductivity based on permeation with the acid solutions relative to the values of hydraulic conductivity based on permeation with DI water, or k/k_w factor, ranged from 4 to 42 under 35 kPa effective stresses, and 4 to 23 under 200 kPa effective stress, thus indicating an adverse impact on the hydraulic performance of the GCL to concentrations ≥ 0.125 M H_2SO_4.

The hydraulic conductivities values obtained at 200 kPa effective stress were found to be lower (13.5~25× lower) than the values obtained at 35 kPa. The work reported in this paper indicates that the hydraulic conductivity was strongly dependent on the ionic strength at low (35 kPa) effective stress. In contrast, it was less dependent on ionic strength at high (200 kPa) effective stress. The high effective stress (200 kPa) tended to mask the influence caused by the acidic solutions.

Acknowledgements

The technical support provided by the Department of Civil Engineering, Monash University, for this project is gratefully acknowledged. The second author is grateful for the funding provided by the China

Scholarship Council to support his PhD studies. We also thank the GCL manufacturer for providing GCL specimens for this project.

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Characterization and in situ monitoring of large scale heap leach fluid dynamics

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Abstract

The efficiency of solution and air flow dynamics significantly affects metal recovery in heap leach operations. Quantifying these processes however presents extreme challenges. Laboratory and column scale measurements of physical and hydraulic properties of ore provide valuable information in aiding heap leach design and leaching operations; however, these measurements may not always be directly transferable to leach pad scale conditions. Consequently, the ability to operationally monitor cribs and heaps can be useful to quantify the effect of varying crush conditions, ore types, irrigation rates and, in the case of metal sulfide ores, aeration schemes on solution and aeration efficiency.

Depending on the desired monitoring parameters, monitoring systems can be designed to assess phreatic level surface, in situ solution content, oxygen content and temperature, and air injection pressures and air velocities. Sensors can be installed during heap construction or via coreholes or drive points to depths limited only by the achievable maximum depth of the chosen drilling method. Sensor installation can also be coordinated with a heap pad characterization program to determine in situ ore and permeability related properties at depth. Finally, field tracer tests can be used to determine solution transport properties and assess irrigation efficiency, the influence of macropores and preferential flow and solution retention times.

The selection and application of appropriate monitoring methods depends on the type of leaching operation (i.e. precious metal [cyanide] leach versus base metal [acid] leach), the leach pad size and duration of leaching. Cyanide leach solutions are less corrosive and heap temperatures typically approximate ambient conditions, which allows for more flexibility in sensor selection. In addition, air injection and oxygen concentration monitoring are not typically important, so monitoring instrumentation can focus on phreatic level and solution content/distribution monitoring. Conversely, acid leach heap

operations can benefit greatly from in situ temperature and oxygen monitoring, but these technologies require robust sensors to ensure long-term viability in elevated temperature and corrosive environments.

As a case study, we present the methodology and monitoring results from a large-scale copper heap leach pilot project. The pilot heap was designed to collect solution from different sectors of the heap to evaluate the efficiency of solution movement. Instrumentation was installed after the placement of ore material using drilling methods at various locations and at different depth intervals to monitor temperature, gaseous oxygen, solution content and capillary pressure. Monitoring data was collected in real time and transmitted to a control room with telemetry. Data collected from the monitoring program were used to validate initial field scale characterization and laboratory and column test results, and support industrial scale heap leach pad design and operation.

Introduction

In situ monitoring of solution and air flow within an active heap leach facility can provide useful information about the efficiency of the leaching process and create significant opportunities to improve metal recoveries. For example, solution content monitoring can determine when and whether the ore has been adequately wetted, while oxygen content and temperature monitoring can determine the distribution of air and temperature within the ore profile and whether aeration is sufficient/efficient. Laboratory and column scale measurements of physical and hydraulic properties of ore can provide valuable information in aiding heap leach design and leaching operations; however, these measurements may not always be directly transferable to leach pad scale conditions. Consequently, the ability to operationally monitor cribs and heaps can be useful to quantify the effect of varying crush conditions, ore types, irrigation rates and, in the case of copper sulfide ores, aeration schemes on solution and aeration efficiency. Moreover, in situ temperature, solution content, capillary pressure, solution chemistry, oxygen content, and air pressure and velocity measurements can be used to validate laboratory and field characterization tests and allow for real time adjustment of heap leach operations to increase leaching efficiencies.

In this paper, we provide an example of a 500,000 ton copper sulfide leach pad that was instrumented and monitored to investigate large-scale heap leach fluid dynamics under varying irrigation and aeration schemes, and describe a conceptual model of hydraulic flow in the heap derived from the monitoring data. The 18 m high heap had an approximately 90 m × 90 m leaching area. Prior to stacking, aeration lines were placed within an underlying gravel drainage layer to provide oxygen to the leach ore for enhancing bio-assisted leaching. Two air-line grids were established that allowed for the air source to originate from the east or from the west of the pad. Data collected from the 1.3 year monitoring program were used to validate initial field scale characterization and laboratory and column test results, and support industrial scale heap leach pad design and operation.

Methods

Instrumentation

The heap leach monitoring system consisted of:

- Nine 30 m by 30 m solution collection modules constructed beneath the gravel drainage layer.
- Nine boreholes (one per module) instrumented with temperature, oxygen/air piezometers, raffinate content, and capillary pressure sensors at 3 m intervals and at 0.5 m below ground surface (bgs).
- A phreatic surface sensor was installed 0.5 m above the gravel drainage layer, and suction lysimeters (for solution sampling) were installed at approximately 6 and 12 m bgs.
- 16 boreholes instrumented with temperature and oxygen/air piezometer sensors were installed at 3 m intervals.
- Raffinate content and capillary pressure sensors were installed at approximately 5, 11, and 16 m bgs.

A plan view of the heap as well as the location of instrument boreholes and solution collection modules relative to the leaching core are provided in Figure 1. The instrument boreholes were spatially located to provide data radiating from the center of the leaching core; 21 instrument boreholes were located within the leaching core and four instrument boreholes were located outside of the core to assess lateral migration of solution and air.

Instruments were installed in boreholes drilled with a 15 cm inside diameter hollow stem auger. All downhole instruments were attached to the outside of 2 inch diameter PVC pipe and the PVC was lowered inside the core barrel. The annulus between the monitoring instruments and the borehole wall was backfilled with:

- 10/20 mesh graded sand mixed 50:50 with ore surrounding the suction lysimeters;
- Leach ore surrounding the advanced tensiometers and raffinate content sensors;
- 3/8 inch gravel surrounding the air piezometers, oxygen sensors, and temperature sensors.

Backfill materials were added directly to the annulus of the borehole. A tag line was used to determine the correct backfill depth for each material. Bentonite chips were added above each backfill layer using a tremie pipe to seal between the instrument arrays and prevent surface and subsurface water, air, and/or heat from preferentially migrating down the borehole. The bentonite chips were hydrated after placement. All sensors were wired to data loggers for automated data collection; data loggers were connected to the central control room via telemetry.

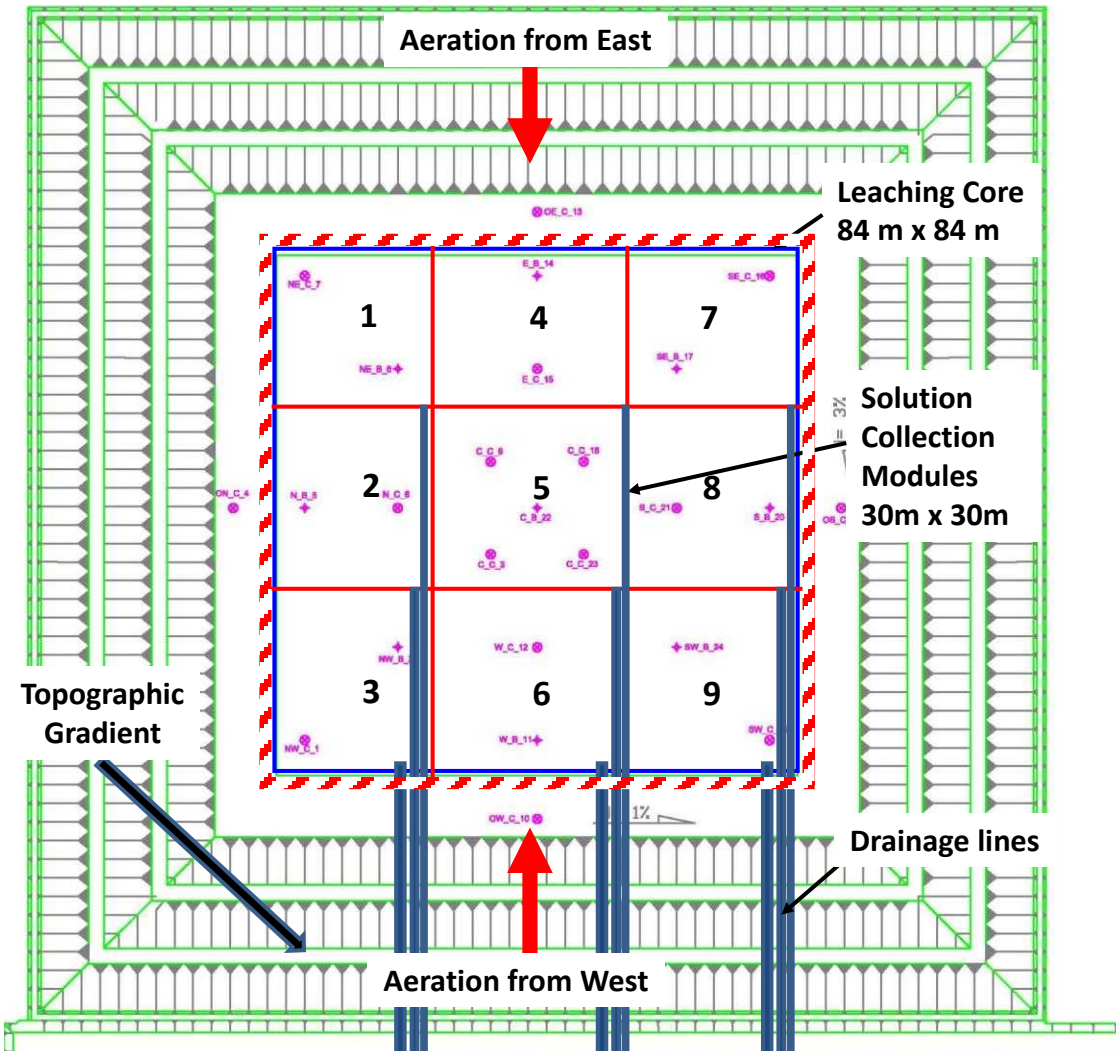

Figure 1: Plan view of heap instrument well locations and solution collection modules

Post test coring

Following completion of testing, sonic coring of 52 coreholes located within and around the leaching core of the heap was performed to collect samples for physical and hydraulic property testing. Geologic logging of the cores was also performed to estimate ore material texture and degree of oxidation (color). Particle size distribution (PSD) tests were performed on each 1.5 m interval.

Laboratory hydraulic analyses were performed on twelve composited cores selected to represent the range of particle size, core bulk density, predominant color characteristic (oxidized or un-oxidized ore), and module performance data. Consolidation permeability tests were conducted to determine the relationship of saturated and unsaturated hydraulic conductivity (Ksat and Kunsat) and air permeability to overburden pressures representing approximate sample depths within the heap. Details of the hydraulic test methods are described in a companion paper (Milczarek et al., 2013). The results of hydraulic

property analyses were then correlated with particle size distribution data and sample depth to develop a predicted spatial distribution of heap hydraulic parameters across the heap.

Results

Estimated solution budget and drainage

An accounting of solution entering the heap and the estimated overall heap solution content is presented in Figure 2. The predicted volumetric solution content assumes an average initial water content of 0.12 cm^3/cm^3 (0.07 g/g) and that the difference between the irrigation and drainage equates to solution storage with correction for surface and heap evaporation (from elevated in situ temperatures and increased water holding capacity of air space). Predicted solution contents increased with the onset of double grid aeration, then decreased during the drainage phase of the study when irrigation was stopped (Figure 2). Predicted solution content increased again with the restart of irrigation and continued to steadily rise for the remainder of the project, to approximately 0.216 cm^3/cm^3 (21.6% volumetric), by the end of operation. The continued increase in storage represents solution going into storage within the leaching core and also represents any solution that moved laterally outside the core.

Figure 3 presents average normalized drainage, defined as drainage divided by irrigation, for module groupings along the north-south direction. Doubling the east aeration rate resulted in a significant drop in the normalized drainage from eastern modules 1, 4, 7 (0.76 to 0.18), a smaller decrease in the middle modules 2, 5, 8 (1.08 to 0.82) and a large increase in normalized drainage from western modules 3, 6, 9 (1.01 to 1.60). The effect of aeration direction is also observed during the alternating aeration period. Normalized drainage along the east side modules (1, 4, 7) decreased when the aeration source was from the east and increased when the aeration source was from the west. Similar behavior is observed with the west modules (3, 6, 9); normalized drainage from the west modules decreased during aeration from the west and increased when aeration was from the east. Note that normalized drainage from the middle modules (2, 5, 8) remained relatively stable during alternating aeration and that during the no aeration period, normalized drainage trends from the north-south module groups resembled the single east side aeration normalized drainage trends.

It is possible that damage to solution collection drainage pipes could account for some intercommunication of module solution or damage to aeration pipes could have resulted in irregular aeration distribution throughout the heap. Nonetheless, the correlation of aeration direction and rates to module drainage rates is strong, and is believed to be primarily caused by entrapped air within the leach ore and aeration back pressure within the drainage layer, as discussed further below.

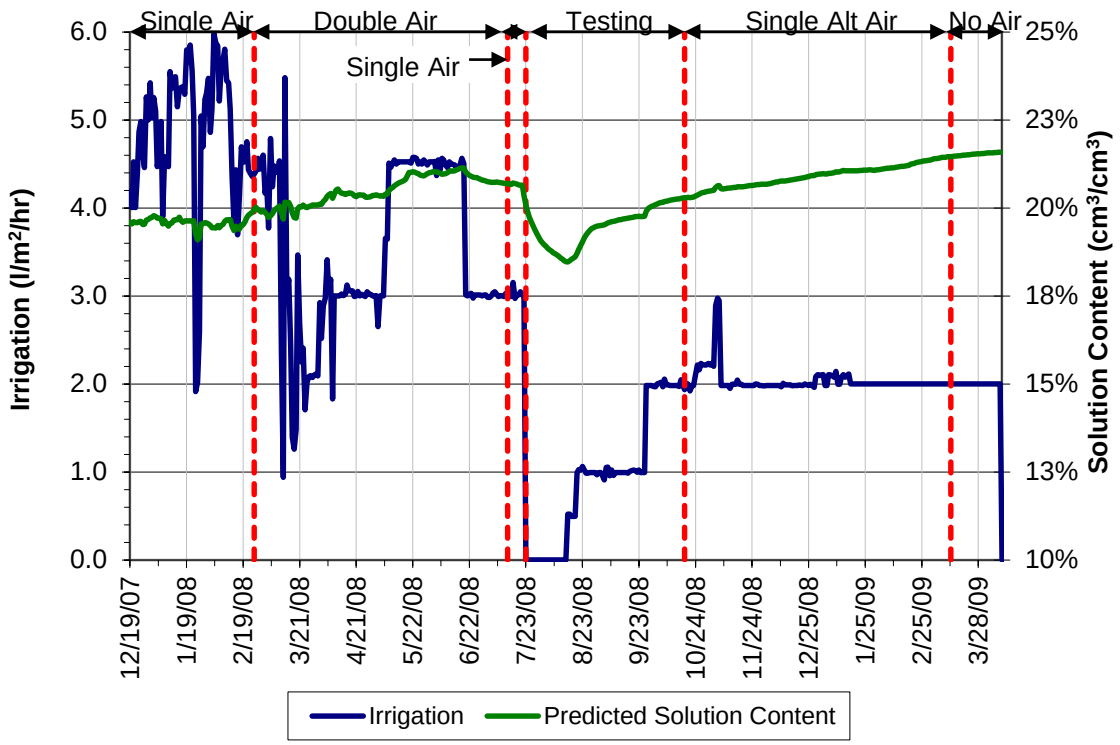

Figure 2: Irrigation rate over time and estimated solution budget

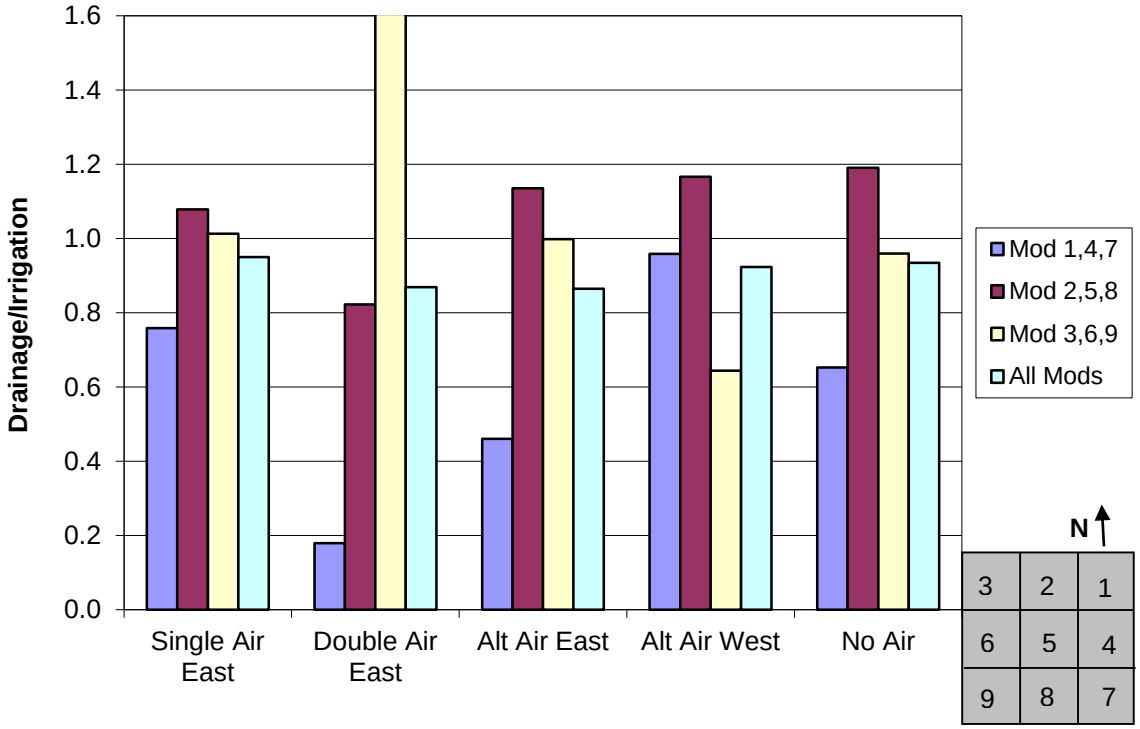

Figure 3: Average normalized drainage for module groups during different aeration schemes

Capillary pressure, phreatic surface, and temperature

Figure 4 provides soil water pressure potential (capillary pressure) contours interpolated from advanced tensiometer measured capillary pressures at 4 m bgs at various time periods representing single and double grid aeration from the east side. Capillary pressure data was also collected at 10 m and 15.5 m bgs but is not shown here due to space constraints. At all times capillary pressure was variable both laterally and with depth. Negative capillary pressures (< 0; orange and red) represent desired unsaturated leach conditions and very negative pressures (i.e. < –30 cm) indicate dry zones that may not be receiving sufficient solution. Conversely, positive capillary pressures (blue) may represent areas of entrapped air that reduce downward solution flow and result in perched solution conditions (pseudo-saturation). Figure 4 shows evidence of solution mounding at the 4 m bgs interval in Module 9 during both single and double aeration periods (mounding was also observed at 15.5 m bgs). Capillary pressure gradients can cause lateral solution movement following from high to low (more negative) capillary pressures. Of note, Module 9 showed below average drainage rates whereas adjacent, and upgradient, modules 5, 6 and 8 showed above average drainage rates, suggesting that solution was flowing laterally from Module 9 into these adjacent modules.

Capillary pressures generally increased as aeration went from single east to double east aeration (Figure 4). Capillary pressures subsequently decreased as aeration went to alternating aeration and then to no aeration. Specifically, the doubling of air injection rates resulted in increased capillary pressures which is most likely due to decreased leach ore hydraulic conductivity as air and solution competed for flow paths. As aeration decreased or was alternated from east-west, competition of air flow with solution flow was reduced and capillary pressures also decreased.

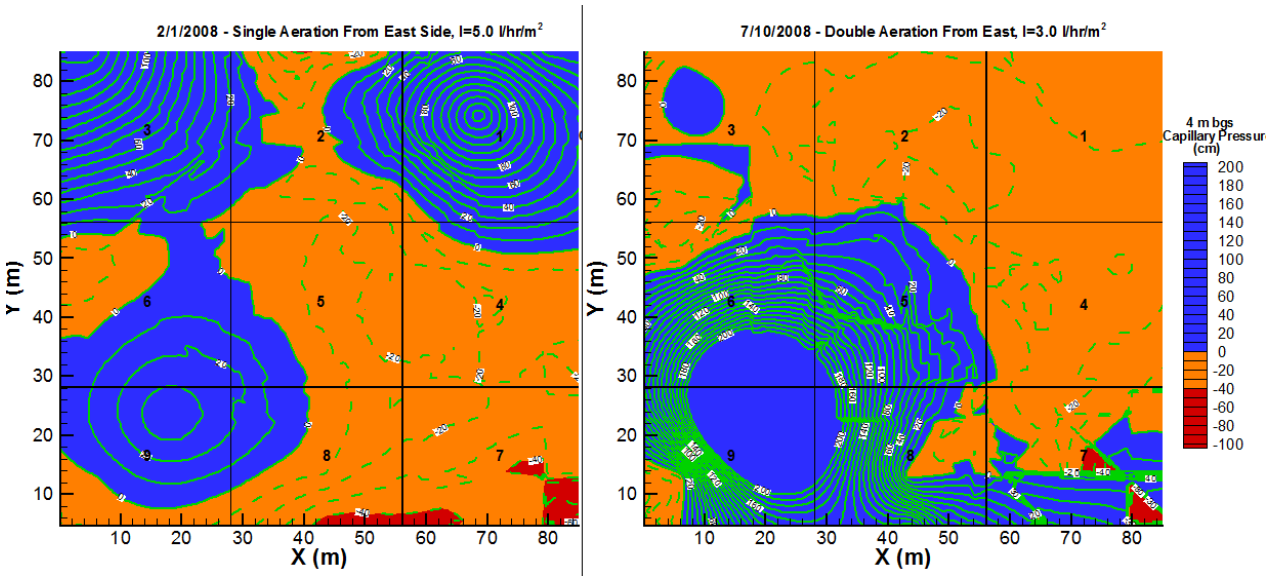

Figure 4: Capillary pressure contours at 4 m bgs at different time periods

Phreatic level monitoring 1.5 m above the drainage system liner (0.5 m into the leach ore), indicated the phreatic surface gradient corresponded to changes in aeration regimes (date not shown). Phreatic level changes also corresponded to changes in drainage rates; in general, as phreatic levels decreased, drainage rates increased and vice versa. The most probable mechanism for increased phreatic levels (and decreased drainage) is aeration back pressure within the underlying drainage layer and/or air entrapment in the leach ore above the drainage layer. This effectively reduces the leach ore permeability and solution pressure in the overlying leach ore must increase to allow the solution to drain. When aeration rates/pressures are reduced, entrapped air and back pressure dissipates and more pore space is available for solution movement. This results in increased leach ore permeability, which also increases the drainage rate and decreases the phreatic level.

Temperatures increased during the early period of heap operation and generally stabilized as double aeration from the east began (Figure 5). At the onset of increased irrigation (4.5 l/m^2/hr) on 5/7/2008 temperatures began to decrease and continued to decrease until irrigation was stopped on 7/23/2008. At this time temperatures began to increase and then generally stabilized during the alternating aeration scheme. Temperatures then decreased after aeration was stopped. Within the core of the heap, temperatures were within the optimum temperature range for bioleaching (30°C to 60°C) for the majority of operation. The increase in heap temperatures with the cessation of irrigation also coincided with an increase in oxygen (data not shown).

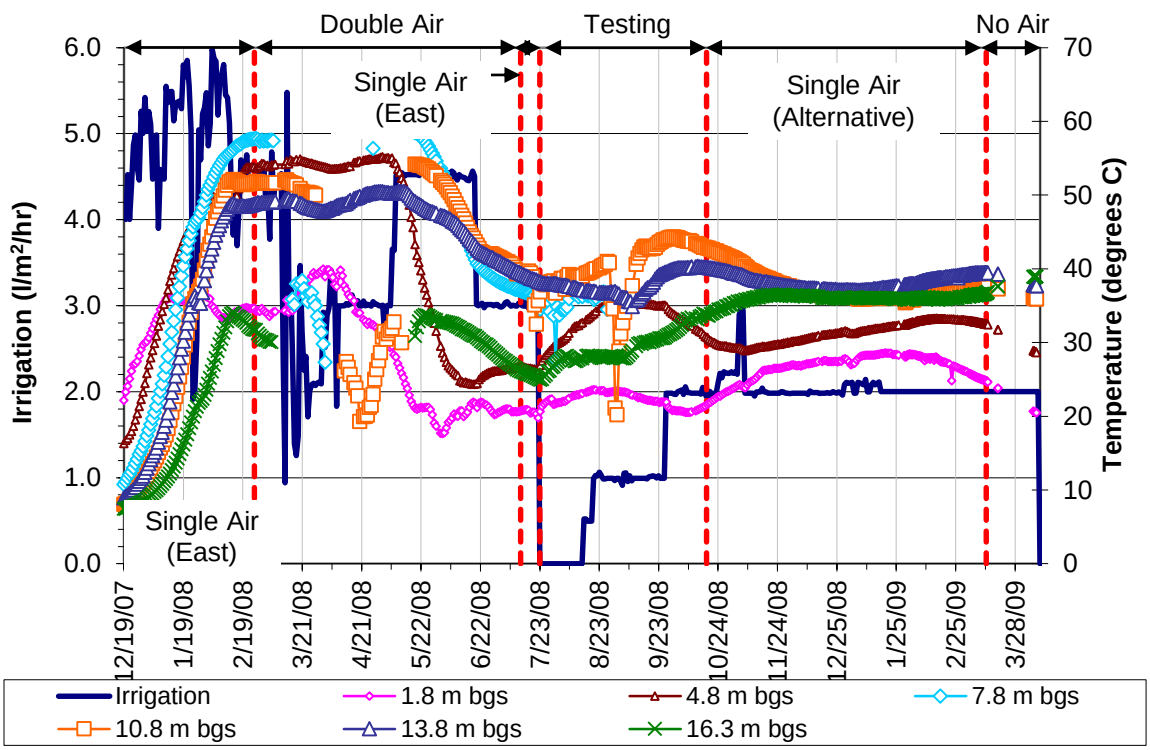

Figure 5: Leaching core temperature for Module 5

Post leach ore sampling and testing

Average PSD for all post-leach core samples and the average PSD for pre-leach ore material (after crushing and prior to agglomeration) indicated that leaching and decrepitation of the ore significantly increased the amount of material passing all mesh sizes, with material passing the #100 mesh increasing on average from 11.4 to 21.1% for pre-leach ore to post-leach ore (Figure 6). The post-leach ore was also more "well-graded", whereby the distribution of particles is more evenly distributed between sieve sizes. Well-graded ore material is easier to compact (higher bulk density), which reduces the leach ore permeability under in situ heap pressures.

The average percentage passing the #100 mesh versus the module percolation as a percentage of total heap percolation showed a trend of decreasing percolation with increasing percentage passing the #100 mesh (Figure 7). Excluding Module 9, which behaved as an outlier, a reasonable relationship between percentage passing the #100 mesh and percolation is obtained ($R^2 = 0.66$). As discussed above, Module 9 experienced solution mounding at the 4 m bgs and 15.5 m bgs depths; in addition, the 10 m bgs intermediate depth showed more negative pressures (much drier). There was an observed increase in fines at 3.5 m and 15.5 bgs in Module 9 which could be due to increased solution contact and ore decrepitation at those depths, whereas the intermediate depths showed much less decrepitation, most likely due to lateral movement of solution away from Module 9 into the adjacent modules.

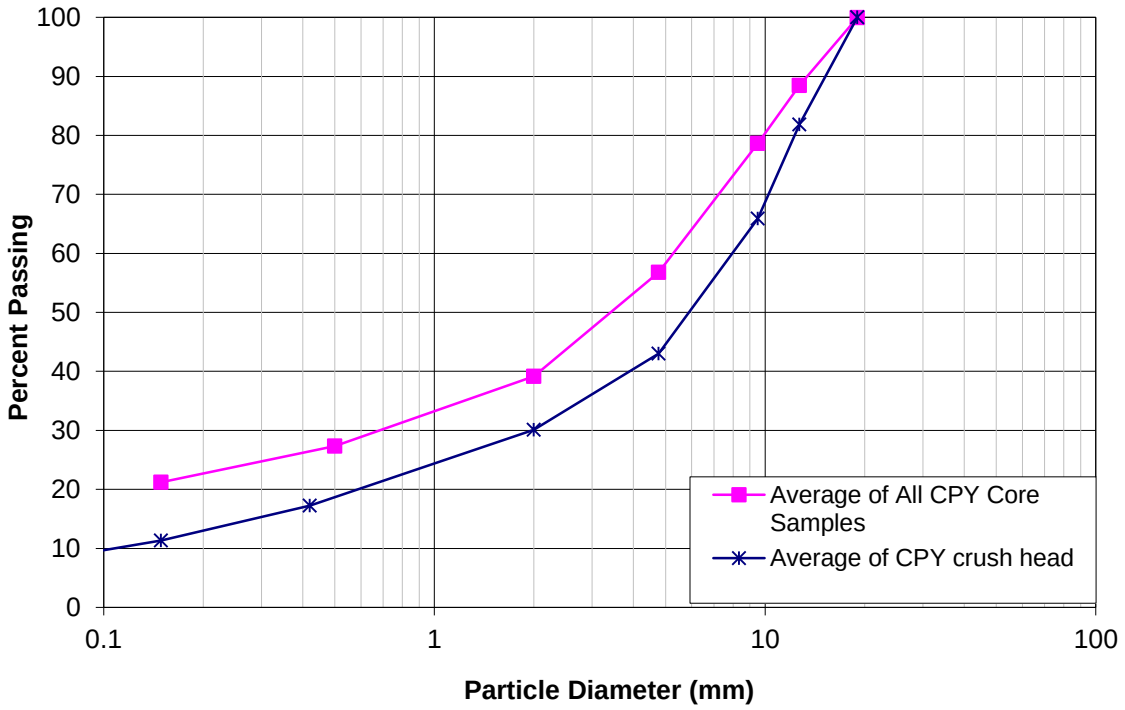

Figure 6: Average particle size distribution for all post-leach samples and crush head samples

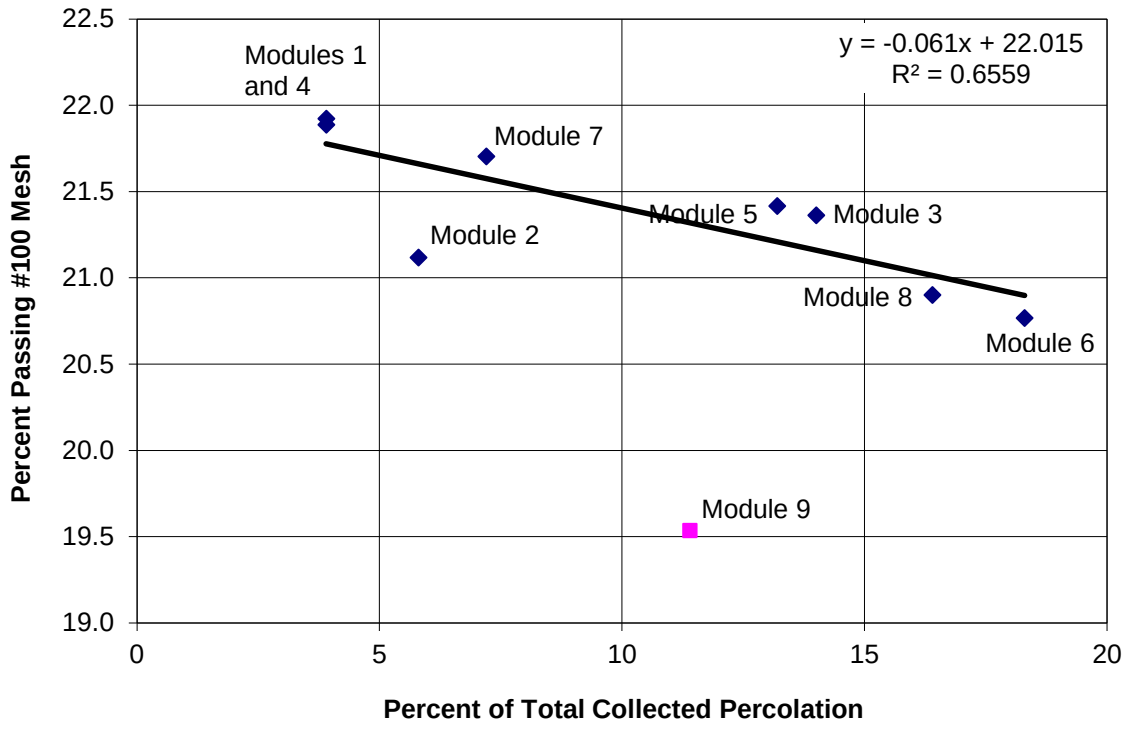

Figure 7: Average percentage passing the #100 mesh versus module percolation as a percentage of total heap percolation

Corehole logging indicated variable ore oxidation with the presence of jarosite (yellow ore) precipitates indicating oxidizing conditions and absence of precipitates (grey ore) indicating reducing conditions. The average percentage of total core that was yellow (oxidized ore) and percentage that was gray (reduced ore) is presented in Figure 8. Other precipitates (i.e. goethite) observed were generally less than a few percent within each module and are not presented. The average percentage of oxidized ore exceeded the average percentage of reduced ore in all modules, except for Module 9 and in coreholes located outside the perimeter of the modules (Figure 8). The greatest oxidation levels were observed in the eastern and western modules (except for Module 9) and also Module 5. Modules, 2, 8 and 9 showed the least oxidation. Higher oxidation in the eastern and western modules may be due to the proximity of these modules to the aeration source, whereas Module 9 most likely received less solution (and acid reagent) due to mounding and lateral flow from above 4 m bgs (see above). Lower oxidation levels in Modules 2 and 8 could be due to these modules being farther from the aeration source; however, Module 5 showed high oxidation levels which indicates there were potentially greater aeration losses from the north and south leaching core boundaries next to modules 2 and 8. Monitoring along these boundaries showed generally drier ore conditions. Finally, oxidation outside the leach core was also observed, indicating lateral movement of solution had leached ore outside of the irrigated core.

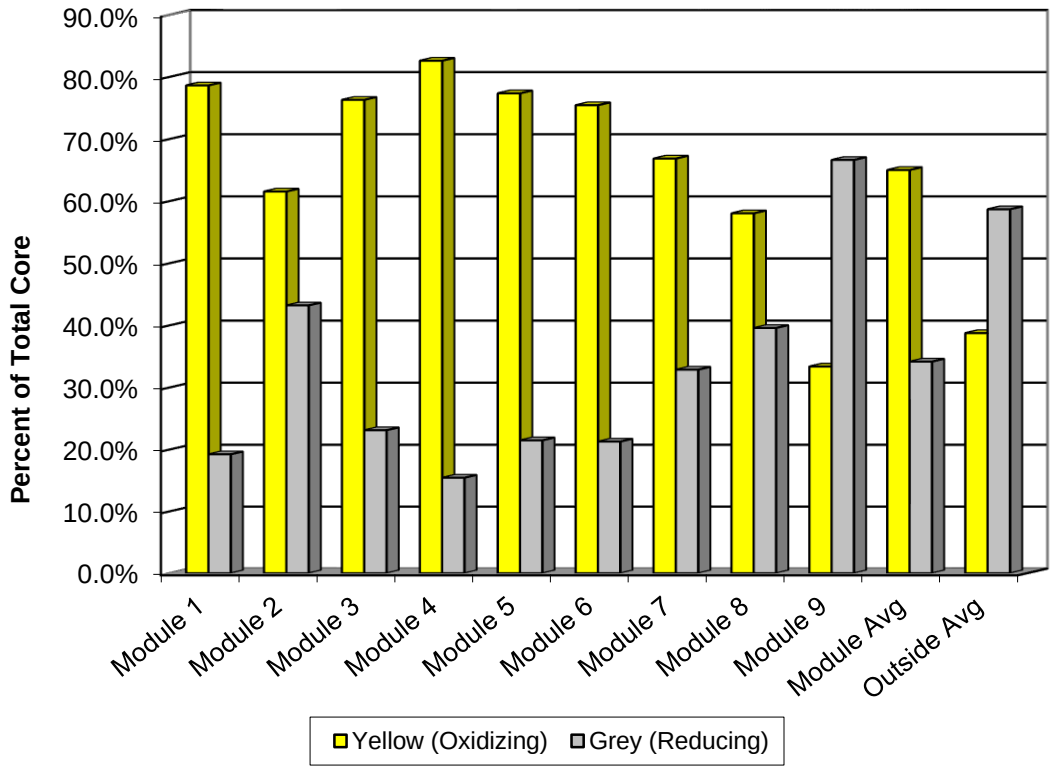

Figure 8: Module average percentage of yellow (oxidized) and grey (reduced) ore

Corehole samples selected for hydraulic property testing indicated that ore permeability decreased with increasing fines and bulk density values. Consolidation-permeability measured Ksat values showed strong correlation to a PSD indicator value which is described by:

- Fraction retained by the #4 mesh
- Fraction passing #100 mesh

The PSD indicator is a measure of the sorting of the leach ore with larger values representing poorly-graded material with a large fraction of gravel sized particles and smaller values representing well-graded material with higher percentage of fines. Multi-variate analysis of the average corehole measured PSD indicator and the estimated bulk density at each sample depth interval within individual modules (44 to 55 samples/module) resulted in a predicted Ksat (from the consolidation permeability data) of:

$$Log(Ksat) = 0.066xy + 0.011x^2 - 0.529x - 11.748 + 9.894y - 2.065y^2 \qquad R^2=0.76$$

Where x is depth in meters and y is the PSD indicator.

As was observed with the measured data, the predicted Ksat typically decreased with depth. Areas of increasing Ksat are predicted over module profiles due to increases in the PSD indicator or bulk density. Modules 1, 4 and 7 showed a lower predicted Ksat than other modules (due to the greatest amount of fines/decrepitation), which agreed with the observed data. However, the predicted Module 9 Ksat values did not agree with the poor percolation observed from this module. As discussed above, Module 9 appeared to have suffered from air entrapment and reduced permeability.

Conclusions

The in situ monitoring system allowed real-time monitoring of solution content, temperature, gaseous oxygen content and capillary pressure at various depths and locations in the heap. This allowed better understanding of the movement of solution and air in the heap leach facility and ultimately the efficiency of the leaching process. The initial air permeability of the heap was sufficient to develop high temperatures; however, in situ gaseous oxygen contents were observed to be variable and declined rapidly with a subsequent decline in in situ temperatures. Loss of aeration efficiency in the leaching core may have occurred due to reduced leach ore air permeability, which caused air to move laterally outside the core where ore conditions were drier and air permeability greater.

Individual module drainage rates were highly variable and decreased in proximity to the side where air was injected into the heap. Based on the drainage rate and phreatic level data, it is believed that aeration back pressure within the drainage layer caused solution mounding within the leach ore above the drainage rock. During aeration from the east, elevated phreatic levels at the leach ore/drainage layer interface caused lateral solution flow to down-gradient modules and reduced drainage from the eastern modules. This occurred most significantly during double grid east aeration and to a lesser extent during single grid east aeration. Phreatic level changes at the drainage layer were not large enough to result in solution moving from west to east. However, the movement of aeration to the west from the east reduced aeration back pressure in the eastern modules and increased drainage from the eastern modules and decreased drainage from the western modules. There was the potential for intercommunication of module solution resulting from damaged drainage collection piping or irregular aeration line distribution; however, the strong correlation of module drainage rates and the heap phreatic levels indicate that aeration back pressure within the drainage layer predominantly controlled solution flow behavior.

In addition, entrapped air at various locations higher in the leach ore may have resulted in increased capillary pressure and reduced hydraulic conductivity, which resulted in solution movement, as evidenced by low drainage rates from Module 9 and elevated drainage rates in adjacent modules. Module 9 also showed poor leach recovery below 4 m bgs where the high capillary pressure conditions were observed.

Solution was also observed to move outside of the leach core as evidenced by the capillary pressure data, high oxygen levels and presence of oxidized leach ore in the outside coreholes.

In situ monitoring indicated a solution and air flow system that is highly dynamic, multidimensional, and significantly influenced by the aeration system. Data collected during the in situ monitoring system allowed for a more complete understanding of the leach recovery dynamics.

Acknowledgements

This work would not have been possible without the support and collaboration over many years of BHP Billiton, Barrick Gold, Freeport McMoRan Copper and Gold, and Newmont Gold, under which our methods have been developed and tested.

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Milczarek, M., Banerjee, M., Yao, T-m and Keller, J. (2013) Ore permeability methods of evaluation and applications to heap leach optimization. In *Proceedings of the Heap Leach International Conference 2013*, Vancouver, British Columbia, Canada.

Ore permeability methods of evaluation and application to heap leach optimization

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Abstract

Efficient recovery of mineral resources from ore heap leaching requires good ore permeability to optimize metallurgical recovery. Solution and air (in the case of copper sulfide ores) need to move freely through the heap for adequate reagent-ore contact to occur. The primary factors that influence leach ore permeability include ore/rock behavior resulting from blasting and crushing processes, ore lift height and lixiviant irrigation rate. In the case of acid leaching, chemical decrepitation from agglomeration and raffinate contact (chemical crushing) are also factors. Whereas the generation of "fines" significantly affects ore permeability, the overall gradation of the material and how it consolidates (compacts) within the heap and over leaching time-frames also greatly influence permeability. Solution and air permeability typically decrease under increasing heap heights, and air permeability decreases under increasing irrigation rates. Depending on the ore type, ore consolidation may result in minor reductions in ore permeability, or may significantly decrease the permeability, thereby causing increased leach times, incomplete recovery and reduction of the economic value of the process.

A variety of physical and hydraulic laboratory property tests can be used to evaluate the solution and air permeability of leach ore material prior to placement on a pad. These tests are designed to emulate the effect of physical and chemical crushing on the physical and hydraulic properties of the ore, and the effect of construction methods (heap lift height) on the consolidation and permeability of the ore material. The primary physical and hydraulic parameters of concern are:

- the full particle size distribution resulting from ore processing and reaction with the lixiviant;
- consolidation of ore under different overburden pressures and changes in particle size distribution resulting from leaching; and

- the saturated permeability, soil water characteristic curves and air permeability under different irrigation rates.

These data are typically interrelated and can be reasonably related to ore specific physical properties. In addition, there is evidence that the bulk particle size distribution of post-leached ore samples and fully crushed (< #10 mesh) pre-leach ore samples can be related to the hydraulic characteristics of individual ore types.

Hydraulic property parameters developed from lab testing can then be used within a geometallurgical classification scheme to predict the hydraulic performance of different ore materials under different operational strategies. Typical geometallurgical classification schemes may incorporate factors such as rock type, degree of alteration, and resistance to physical and chemical crushing. A case study which compares different ore types based on geometallurgical classifications to laboratory derived hydraulic properties using the aforementioned test schema will be presented as an example of how quantifying the hydraulic properties of different ore types can be used to guide and optimize heap leaching operations.

Introduction

Ore permeability has long been recognized as a critical factor in heap leaching performance. Poor ore permeability results in decreased metal recovery and increased leach recovery time. Heap leaching practitioners generally attribute poor permeability to a variety of factors including a large proportion of fine particles (< #100 mesh) within the ore matrix, the migration of fine particles deeper into the heap and the loss of porosity and permeability due to ore consolidation, compaction or decrepitation. Finally, heterogeneous permeability is inherent in any porous media; ore blasting, processing, heap construction and irrigation practices affect the permeability distribution and can result in low or delayed metal recovery due to poor ore-solution contact and solution channeling through preferential flow paths.

Ore permeability can be improved by agglomeration to bind fine particles to coarser particles which provides a more uniform distribution of particle sizes (McClelland, 1986; Lastra and Chase, 1984). This in turn increases the amount of large pores to facilitate solution flow and also typically improves solution distribution within the ore. Whereas gold ore agglomerates are typically stable because of the use of cement, agglomeration of base metal ores with sulfuric acid or other binders is frequently unstable due to ongoing chemical decrepitation during the leaching process (Lewandowski and Kawatra, 2009).

In the case of bio-assisted leaching, both solution and air need to move freely through the heap for adequate reagent-ore contact to occur. High temperature bio-assisted heap leaching (Dew et al., 2011) has even greater needs for aeration efficiency. There are a number of factors that influence leach ore permeability, including: ore/rock behavior under physical crushing, chemical decrepitation from acid

agglomeration and raffinate contact (chemical crushing), the nominal crush size, heap height and lixiviant irrigation rate. It is very challenging to accurately characterize the effect of these factors on permeability, and to identify spatial and temporal effects on permeability within large industrial heaps.

Over the last decade, we have developed several laboratory and field methods to improve our understanding of solution and air flow behavior in heap leach materials. Laboratory methods, which are the focus of this paper, include:

- using large diameter cores;
- directly measuring the hydraulic conductivity function under irrigation with the lixiviant; and
- using flexible wall methods to determine solution and air permeability under variable bulk density and irrigation conditions to mimic the effect of overburden pressure and chemical decrepitation.

Leach ores have been tested from a number of sites in North and South America, with reasonable agreement between measured leach ore permeability and heap leach performance.

In this paper we provide an example of a laboratory permeability testing program that simulated effects of operational conditions on ore hydraulic properties and correlations of permeability and ore physical properties which allows for mapping of ore body permeability.

Methods

Eighteen samples with different mineral associations and alterations were selected for hydraulic property testing. The following briefly describes the leach ore material hydraulic testing procedures used in this study.

Anticipated ore permeability classification

The eighteen samples represent a variety of geologic facies and alteration from one mine pit. Prior to the hydraulic property testing, the ores were grouped into three expected permeability types: “Bad”, “Regular” and “Good” based on known geometallurgical properties such as mineral type, particle distribution under crush, alteration, rock quality designation, rock strength and clay type and fraction.

Hydraulic and physical property test overview

The eighteen ore samples were processed as follows: 50 kg of drill core of each leach ore sample was crushed to 1.27 cm diameter (25 mm tertiary crusher setting). The <1.27 cm crush samples were screened and divided into 12 size fractions with screen sizes of >22.2 mm, >19 mm, >16 mm, >12.5 mm, >9.5 mm, >6.3 mm, >#4, >#14, >#35, >#65, >#100, and <#100. Based on the target sample weights needed for the various physical and hydraulic property tests, sample fractions were split using a universal

type splitter (Versa-Splitter) and reconstituted to the target sample weights with identical particle size distributions. Each fraction was split a minimum of three times before reconstitution.

Each of the <1.27 cm diameter (whole) samples was thoroughly agglomerated in two batches in a mixer with 5% to 12% raffinate content by sample weight until all the fines agglomerated to the larger particle sizes. Then 19 or 7.5 kg of H_2SO_4 per ton of sample material, depending on the ore-type, was added to the batches; after thorough mixing, the samples were cured for at least three days.

Physical property screening tests were conducted on sample material to include:

- 1.27 cm crush sample particle size distribution (PSD) on before (pre-test) and after (post-test) hydraulic property testing to determine the gradation of the material size fractions and examine the effect of decrepitation.
- Specific gravity to determine the mineral density before hydraulic property testing.
- Atterberg limits to determine clay characteristics after hydraulic property testing.
- Specific surface area (SSA) was calculated from pre-test and post-test PSD data.

The following hydraulic property tests were conducted on the 1.27 cm crush samples over a range of bulk densities representative of conditions from the top to the bottom of 20 m ore lift height:

- Consolidation-permeability tests to estimate changes in saturated hydraulic conductivity (Ksat) at pressures mimicking various heap heights.
- Direct irrigation measurements (unsaturated hydraulic conductivity [Kunsat]) to determine solution content and air porosity at different irrigation rates and ore densities.
- Air permeability tests during the Kunsat measurements as a function of solution content, air porosity, and repacked ore sample density.

Hydraulic property testing methods

Consolidation-permeability tests were conducted in 15 cm diameter by 30 cm high, dual wall permeameters on the 1.27 cm crush samples. 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. Dual wall permeameter procedures for flexible wall permeameters are specified in ASTM D5084-03 (American Society for Testing and Materials, 2003). The sample is packed to an initial relatively low bulk density and then saturated overnight with raffinate leach solution by upward infiltration. Subsequent hydraulic conductivity tests are performed over a range of sample consolidation created by increasing the flexible wall membrane pressure to mimic increasing lateral earth pressures within the heap.

Dual wall permeameters were packed to initial bulk densities between 1.53 and 1.64 g/cm^3. Sidewall stresses of 0 kilopascal (kPa) to 140 kPa were applied to simulate vertical pressures of up to 420 kPa

depending on the sample. Ksat tests were run via upward infiltration for 8 hours using a constant head of approximately 3 cm.

The simulated heap height is calculated from an assumed relationship between lateral earth pressures applied to the side-walls and the overburden pressure (and heap height). The "at rest lateral earth pressure ratio" or "coefficient of earth pressure at rest" (K_0) is the ratio of horizontal pressure (or stress) to vertical pressure. K_0 is a function of the ore's angle of shear resistance (or effective angle of internal friction) φ' and in its simplest form can be calculated as (Bishop, 1959):

$$K0 = 1\text{-}\sin(\varphi') \tag{1}$$

For sandy gravel material such as leach ores, φ' is expected to range from 35 to 50 degrees, such that K_0 may range from 0.23 to 0.42 (Bowles, 1988); thus K_0 is assumed to average 0.33 for the tests. Consequently, overburden pressures within the heap can be simulated up to approximately 20 m.

Direct irrigation (Kunsat) tests were performed using the dual wall permeameter and a range of bulk density values representing different heap heights as determined from the dual wall consolidation permeability Ksat results. Raffinate solution was applied to the surface of the ore through an evenly distributed network of irrigation points at two different rates, a low rate (≈ 1 to 2 $l/m^2/hr$) and a high rate (≈ 6 to 13 $l/m^2/hr$). A constant suction of approximately 30 cm was applied to the bottom of the core by a wick. Water content and tensiometer sensors were also buried within the leach ore to continuously measure water content and matric potential (capillary pressure) at two depths in the ore column.

Air permeability was determined during periods of irrigation and solution drainage (i.e. no raffinate irrigation) by injecting air into the bottom of the dual wall cell during the direct irrigation experiments. Air permeability is determined by measuring the pressure drop across the bottom to top of the cell with several air flux rates (9.4×10^{-3} to 0.45 cm/sec) used to confirm the consistency of measurements.

Specific surface area calculations

Specific surface area (SSA) is defined as the total surface area of the particles per unit mass. The clay fraction largely determines the SSA of a material because the SSA of a clay particle is upwards of 100 times greater than that for a sand particle. Specific surface area for each ore sample was estimated based on the relative mass of the different particle size diameter groups. For the estimation, all particles were assumed to be spherical, allowing the SSA to be calculated from the particle size distribution using the summation equation:

$$SSA = \frac{6}{\rho_s} \sum \frac{c_i}{d_i} \tag{2}$$

Where ρ_s is particle density and c_i is the mass fraction of particles of average diameter d_i. This estimation method likely underestimates SSA because particles have irregularities and are not smooth

spheres. For example, clay may be platy in shape which would produce a larger surface area than a sphere, and the estimation does not consider expanding type clays. Nonetheless, errors associated with the estimation method are likely to be relatively consistent for all samples, allowing comparative analysis of SSA to be valid.

Correlation analyses

Correlation analyses were conducted by fitting power functions to the relationships between the 1.27 cm crush Ksat values at estimated heap heights of 3 m, 6 m, and 10 m compared to:

- estimated post-test 1.27 cm crush sample SSA;
- estimated pre- and post-test 1.27 cm percent passing #100 mesh; and
- PSD indicator which is calculated as the percent retained by the #4 mesh/percent passing the #100 mesh (> #4 mesh/< #100 mesh).

Results

Particle size distribution

All of the ore samples tested showed some signs of decrepitation (chemical crushing) during the hydraulic property testing. Table 1 compares the observed change in the percent of material passing the 1.27 cm, #4 mesh (4.75 mm), and #100 mesh (0.15 mm) before and after the hydraulic property testing. The anticipated "Bad" ore types showed the lowest PSD indicator values and with the exception of M9, the anticipated "Good" ores showed the greatest values. Two anticipated "Regular" ores also showed low PSD indicator values. Of note, the initial PSD was a poor predictor of the final PSD after agglomeration and leaching; this is due to variable resistance to decrepitation from acid between the different ores.

Table 1 also presents PSD indicator values for pre- and post-test material. The PSD indicator relates the proportion of large particles (> #4 mesh), which increase permeability by creating large, clast supported pore sizes, to small particles (<#100 mesh) which decrease permeability by reducing the average pore size diameter. Previous test work by the authors (data not published) indicates the post-test PSD indicator is well correlated to ore permeability since it is representative of the material gradation, which is not reflected in the percent passing the #100 mesh. Previous work also shows that post-test ore samples with PSD indicator values lower than 2.0 show unacceptable permeability and values greater than 3.0 show acceptable permeability for heap leaching; ore samples values between 2 and 3 can show either poor or good permeability. Comparison with Ksat testing shows that for this group of 18 samples, all of the samples with post-test PSD indicator values greater than 3.0 showed acceptable permeability.

Table 1: Change in percentages of material passing the 1.27 cm sieve, # 4 mesh, and #100 mesh and pre- and post-test PSD indicator values

Sample	Expected ore permeability group	Change in percent material passing (Post-test – Pre-test)			Pre-test PSD indicator	Post-test PSD indicator
		12.7 mm 1.27 cm	4.75 mm #4 mesh	0.15 mm #100 mesh		
M16	Bad	27	28	23	5.35	0.85
M17	Bad	18	27	29	10.95	1.11
M22	Bad	5	8	20	1.65	0.70
M7	Bad	–2	–5	14	4.65	2.21
M4	Regular	8	11	11	8.67	2.56
M5	Regular	6	3	8	9.34	3.93
M8	Regular	4	8	10	7.01	2.79
M10	Regular	5	3	8	16.09	5.17
M14	Regular	2	1	10	9.01	3.31
M18	Regular	–1	2	7	16.42	6.27
M2	Regular/Good	1	3	9	6.21	3.11
M13	Regular/Good	2	2	6	7.88	4.14
M1	Good	–4	–1	10	16.82	4.38
M3	Good	2	10	7	11.68	4.31
M6	Good	2	2	8	13.05	4.77
M9	Good	0	4	11	4.54	2.22
M15	Good	–2	–3	1	7.40	7.00
M20	Good	–7	–2	7	10.41	5.08

Saturated hydraulic conductivity testing results

Ksat was measured as a function of estimated heap height (applied equivalent pressure) and ore consolidation (expressed as bulk density) with the dual wall consolidation-permeameters. Ore consolidation was variable as observed by the ore bulk densities. Figure 1 shows the ore sample consolidation under various equivalent heap heights; under 10 m of equivalent heap height bulk densities ranged from 1.72 to 2.05 g/cm^3.

For the purposes of analysis, we have defined acceptable permeability as 100 times greater than the nominal irrigation rate of 6 l/m^2/hr. The 100× permeability safety factor allows for sufficient air permeability and also spatial variability in permeability under field conditions. Figure 2 shows that the permeability criteria of Ksat values of at least 100 times greater than the target irrigation rate is met by eleven samples, at all of the simulated estimated heap heights (from 0 m to about 18 m). Anticipated

"Bad" samples M7, M16, M17, and M22 showed the least favorable performance with acceptable Ksat values at estimated heap heights less than 4 m. Anticipated "Regular" sample M4 and "Good" sample M9 only showed acceptable Ksat values at estimated heap heights less than about 8 m and 6 m, respectively.

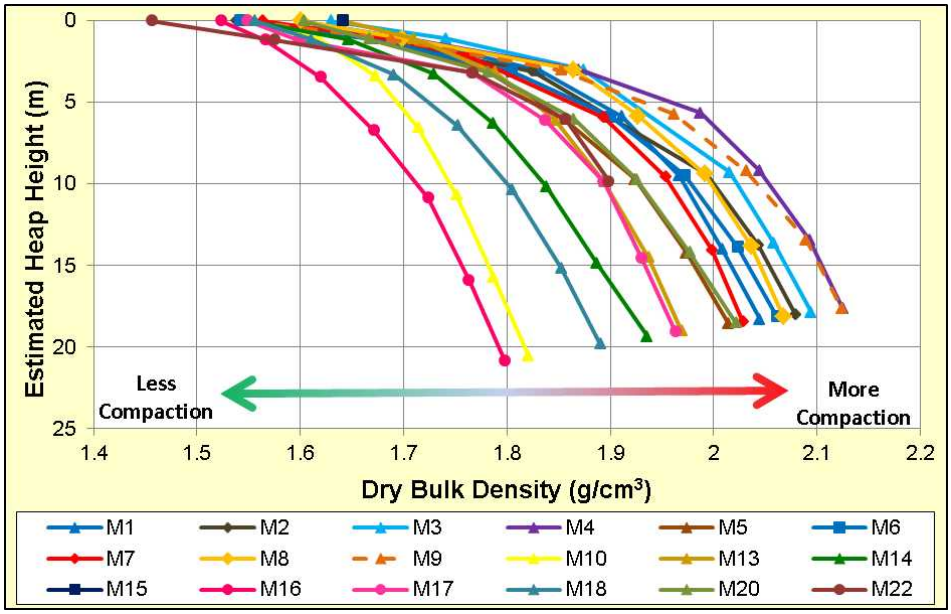

Figure 9: Changes in ore bulk density over estimated equivalent heap heights

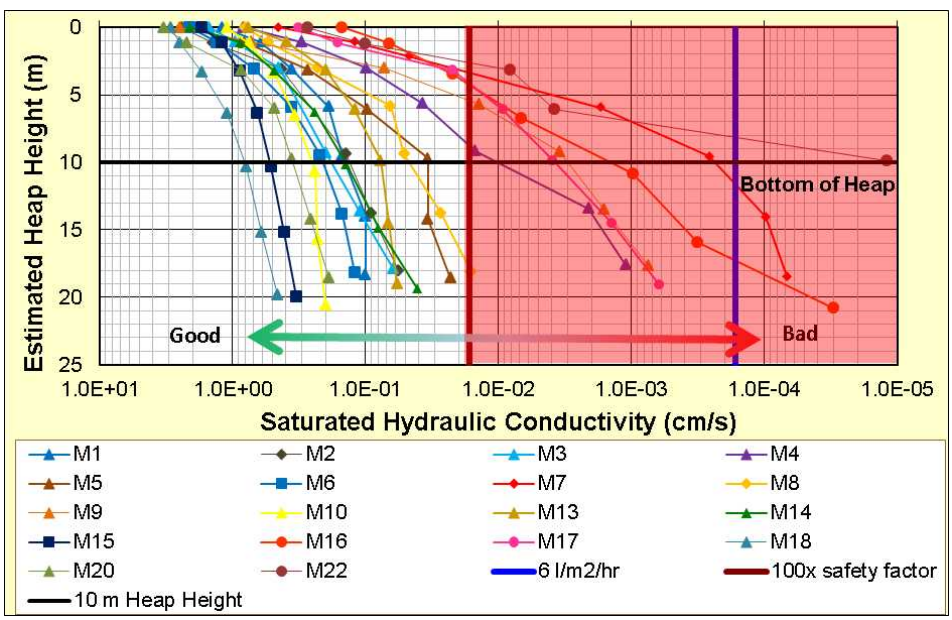

Figure 10: Changes in Ksat over estimated equivalent heap heights

The consolidation-permeability results are consistent with a greater percent of post-test fines in samples M7, M17, M16, and M22 and higher plasticity and slightly greater percent of post-test fines in M4 compared to the remaining samples. However, the maximum bulk density did not correlate highly to

Ksat, because pore size distribution also contributes significantly to permeability. For example, sample M4 had the highest maximum bulk density and intermediate performance; M16 had the lowest maximum bulk density and the third worst performance. Relative changes in bulk densities were most pronounced in samples that were well graded, poorly graded samples (samples with a high percent of large or small particles) showed the least consolidation.

Unsaturated hydraulic conductivity and air permeability testing results

Under direct irrigation testing in the dual wall consolidation-permeameters the volumetric solution contents were observed to increase by 1% to 6%, resulting from an increase from low irrigation rates (1 to 2 $l/m^2/hr$) to high irrigation rates (6 to 13 $l/m^2/hr$). The "Good" ore samples generally showed greater air porosity and small changes in air porosity with increasing heap height conditions, which indicates that the pore size distribution for these samples does not change significantly under consolidation pressures. Higher initial volumetric solution contents and larger increases in solution content were observed in the remaining samples under increasing irrigation rates. Of note, the anticipated "Bad" samples M7, M16, M17, and M22 all showed ponding (and zero air permeability) under high irrigation rates at estimated heap heights of 6 m or less. M4 showed ponding under high irrigation rates at estimated heap heights above 9 m. These results are consistent with the saturated hydraulic conductivity test results.

For the purposes of analysis, we have defined acceptable air permeability as 100 Darcies at the target irrigation rate. 100 Darcies is believed to allow sufficient air permeability and interconnectivity of air pore space for efficient aeration. Figure 3 through Figure 5 show the measured air permeability plotted versus the estimated equivalent heap height for the anticipated "Bad", "Regular", and "Good" ores, respectively. Significant changes in air permeability (from around 600 to 0 Darcies) were observed under low and high irrigation rates and variable estimated heap heights for anticipated "Bad" samples M7, M16, M17, M22, "Regular" sample M4, and "Good" sample M9. These samples also failed the Ksat permeability criteria of greater than 100× the nominal irrigation rate. Samples M1, M6, M10, M13, M14, M15, M18, and M20 showed air permeability greater than 100 Darcies at estimated heap height greater than 10 m under both high and low irrigation rates.

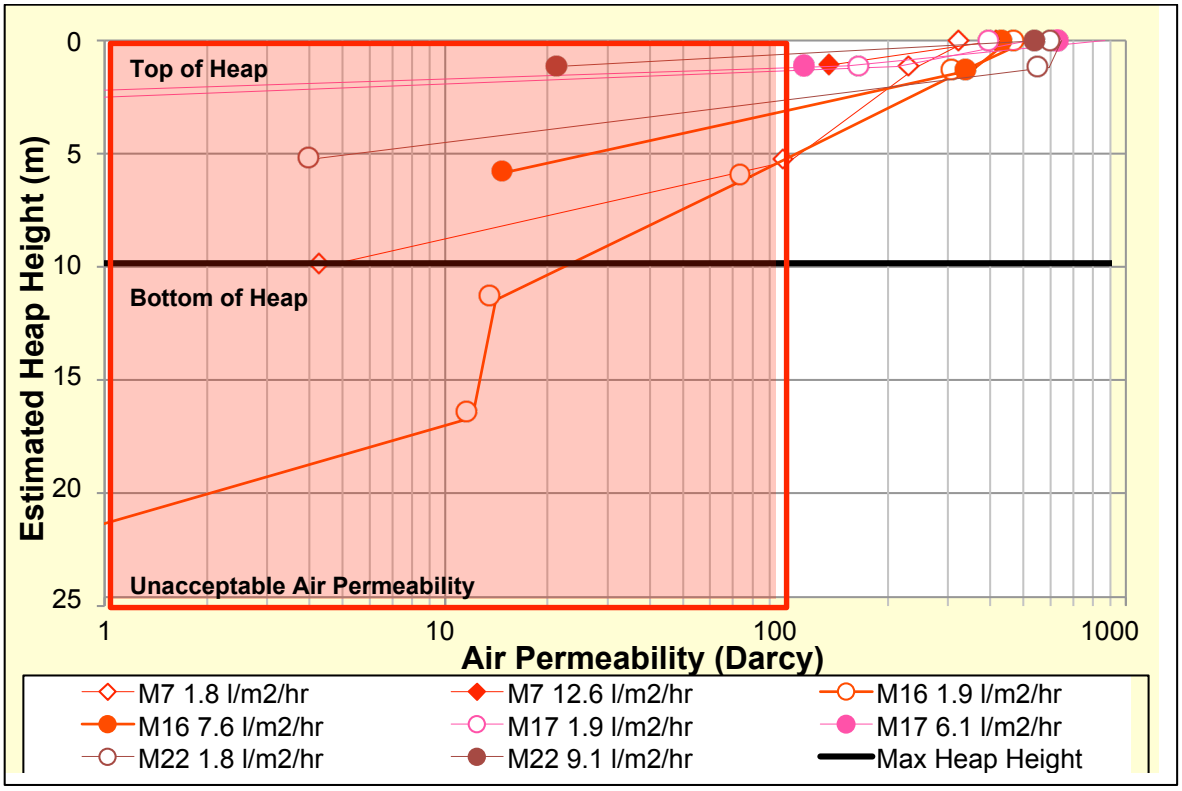

Figure 11: Air permeability versus estimated heap height for "Bad" samples

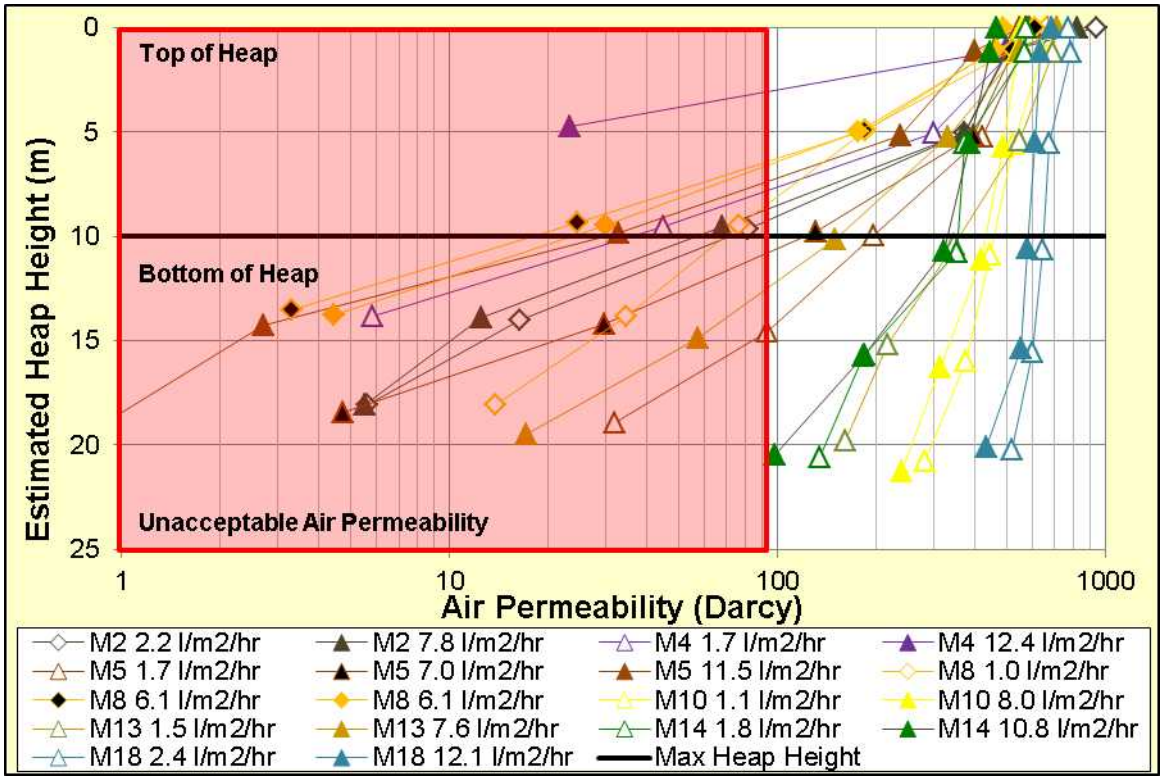

Figure 12: Air permeability versus estimated heap height for "Regular" samples

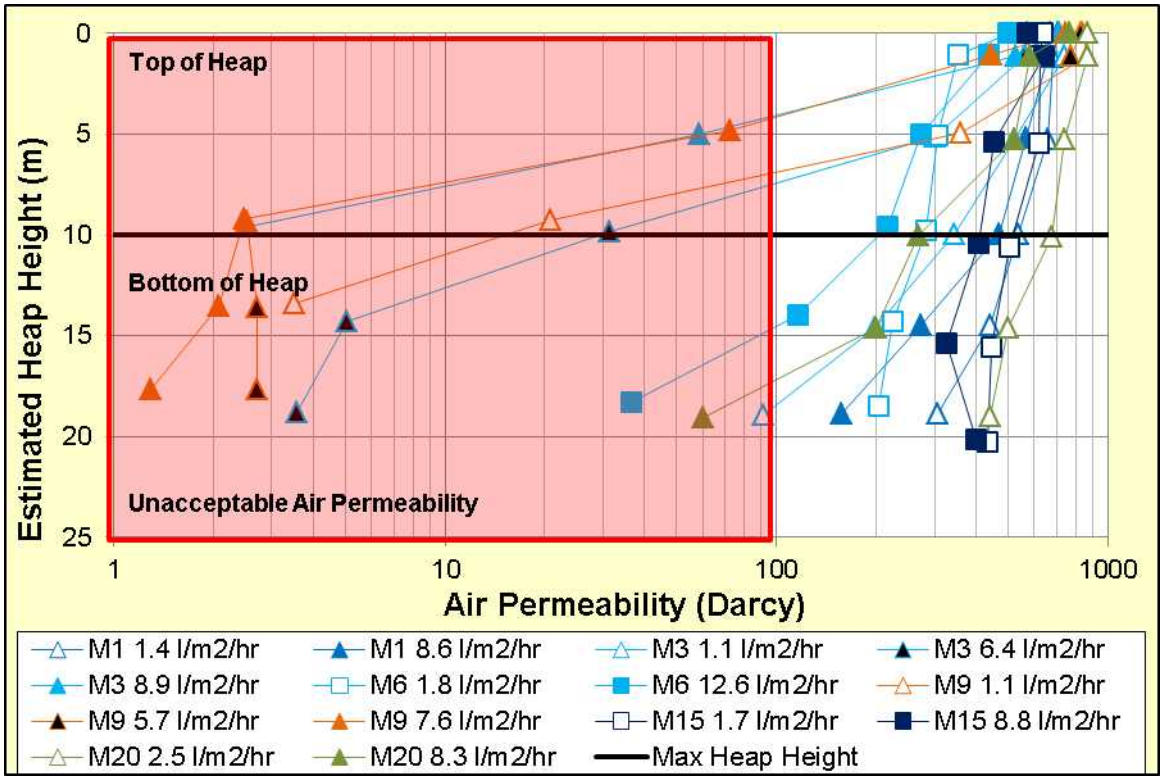

Figure 13: Air permeability versus estimated heap height for "Good" samples

Of note, the air permeability of samples M2, M3, M4, M5, M7, M8, M16, M17, and M22 typically decreased by 50% or more at the high irrigation rates compared to the low irrigation rates. This indicates that the pores contributing to air permeability (active air porosity) in these samples consist of relatively small pore diameters that become filled with solution at higher irrigation rates with a concomitant reduction in air permeability.

Correlation analyses

Correlation analyses were performed to evaluate potential relationships between Ksat values for the ore samples under different heap heights and estimated SSA, the PSD indicator, and the percent passing the #100 for the pre-test and post-test < 1.27 cm crush samples. Table 2 summarizes the results of the correlation analyses of Ksat values at estimated heap heights of 3 m, 6 m, and 10 m.

Low correlations were observed between the Ksat values and estimated SSA at all estimated heap heights (Table 2). Low correlations were also observed with the pre-test percent passing the #100 mesh. R^2 values for Ksat and the post-test percent passing the #100 mesh greatly improved to 0.88, 0.80, and 0.81 at estimated heap heights of 3 m, 6 m and 10 m, respectively. This result is expected since decrepitation causes large changes in PSD before and after hydraulic property testing.

Table 2: Ksat correlation analysis (R^2) results[1]

Test parameter[2]	K_{sat} < at 3 m	K_{sat} < at 6 m	K_{sat} < at 10 m
Estimated SSA (post-test)	0.41	0.50	0.49
Passing #100 mesh (pre-test)	0.50	0.49	0.63
Passing #100 mesh (post-test)	0.88	0.80	0.81
PSD indicator (post-test)	0.89	0.75	0.78

[1] R^2-values from fitting a power function relationship

[2] Passing #100 mesh, PSD indicator values and SSAs were calculated from the PSD of the respective sample

The post-test PSD indicator value was also highly correlated to Ksat, but correlation values were slightly lower at the 6 and 10 m estimated heap heights than correlation values with the percent material passing the #100 mesh. Overall, the post-leach values for percent passing #100 mesh and the PSD indicator can reasonably predict the measured Ksat for the ore samples.

Correlation analyses were also performed on the measured air permeability of the 1.27 cm crush ore samples at estimated heap heights of 6 m and 10 m under both low and high irrigation rates. Correlation was moderate to strong (R^2 = 0.74 and 0.86) for air permeability under low irrigation rates. Correlation with air permeability was poor at high irrigation rates (R^2 = 0.54 and 0.42). Under low irrigation rates the correlation increased with heap height, whereas under high irrigation rates the correlation decreased with heap height. These results indicate that the measured Ksat values can reasonably predict air permeability under low irrigation rates, but that additional data (i.e. moisture retention characteristics) are needed to predict air permeability at high irrigation rates.

Conclusions

Eighteen ore samples were tested for saturated and unsaturated hydraulic conductivity and air permeability characteristics under a range of estimated heap height conditions. The laboratory testing program was used to validate and improve the geometallurgical classification scheme used to predict the ore permeability and thereby optimize heap leach operations. Acceptable permeability is defined as Ksat being greater than 100 times the nominal irrigation rate (6 $l/m^2/hr$) and air permeability greater than 100 Darcies.

All of the samples showed some decrepitation during testing. Decrepitation significantly affected the permeability of some of the samples, whereby they had a low percent passing the #100 mesh before hydraulic property testing, but produced “Bad” permeability performance. Measured Ksat values met the acceptable permeability criteria at 10 m equivalent heap heights for all of the samples except anticipated

"Good" sample M9 (< 6 m heap height), "Regular" sample M4 (< 8 m heap height) and "Bad" samples M7, M16, M17, and M22 (< 5 m heap height). These samples showed similar poor air permeability behavior. Air permeability for all "Bad" samples and some "Regular" and "Good" samples decreased by 50% or more at the high irrigation rate (12 $l/m^2/hr$) compared to the low irrigation rate (2 $l/m^2/hr$).

Post-test PSD indicators ranged from 0.70 to 7.0. Samples with PSD indicators less than 3 did not meet the acceptable solution or air permeability criteria under high irrigation rates at estimated heap heights up to 10 m. The post-leach percent passing the #100 mesh and the PSD indicator values showed the strongest correlations to the measured Ksat values for 1.27 cm crush samples. The poor correlation of solution and air permeability with pre-test percent passing #100 mesh and PSD indicator values is consistent with results from a large number of ore samples from other properties we have tested (data not-published), and serve as a caution to heap leach operations that rely on pre-test < #100 mesh values to predict permeability.

Acknowledgements

This work would not have been possible without project support and collaboration over many years from BHP Billiton Base Metals, Barrick Gold, Freeport McMoran Copper and Gold, Newmont Gold and Dirk van Zyl of the University of British Columbia, from which our methods have been developed and tested. Many thanks also to all the workers in the GeoSystems Analysis, Inc. laboratory for their diligence and commitment to excellence.

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Heap leach drainage improvements at Freeport-McMoRan's Safford Mine

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Abstract

Optimizing solution leaching metal recovery from run-of-mine and crushed ore stockpiles is dependent on maintaining ore permeability and drainage through the stockpile. Various factors can impact permeability including host rock type, mineralization, particle size, densification with increasing stockpile height, chemical and physical ore decrepitation, etc. In more challenging ore deposits and processing conditions these factors may cause stockpile permeability to decrease to the point that metal recovery is inhibited and geotechnical stability is compromised. This paper discusses novel methodologies to promote leach stockpile solution drainage which maximize metal recovery from the ore and control critical variables that affect slope stability, and which have been implemented at Freeport-McMoRan's Safford Mine.

Introduction

Freeport-McMoRan's wholly owned Safford Mine is located in Graham County, Arizona, approximately 13 km north of the town of Safford and 270 km east of Phoenix. Safford is located in a desert environment with rainfall averaging 25 cm per year. Summertime temperatures routinely exceed 38°C, while winter nighttime temperatures dip below freezing. The mine began operation in 2007 and produces copper from crushing, heap leaching, and solution extraction/electrowinning (SXEW).

The ore at Safford is relatively low grade (0.42% TCu), is a refractory oxide with <70% copper recovery, and the gangue mineralization is characterized by high acid consumption. Three stage crushing is required to liberate copper mineralization and achieve optimum economic recoveries. The crush size and ore characteristics result in generation of fine particles which can be problematic in any stockpile

leach operation, but the unique combination of mineralization and acid requirements in the Safford ore results in considerable ore degradation that creates challenging, variable permeability conditions in the heap. Safford has instituted a number of operational practices that minimize ore compaction; utilizes monitoring technologies to measure ore permeability, solution levels and pore pressures in the heap; and has implemented a program of wicking to drain solution from areas of the heap that have the most significant deficiencies in permeability. This paper focuses on the implementation of the extensive wicking program that was initiated at Safford in June 2012, and the results that have been obtained.

Geology

The Safford Mine includes two ore deposits, Dos Pobres and San Juan, which have leachable oxide mineralization overlying primary sulfide mineralization. The predominant oxide copper minerals are chrysocolla and copper bearing iron oxides, with the predominant, deeper copper sulfide being chalcopyrite. The deeper primary sulfide deposit is not included in leach reserves.

The majority of ore to date at Safford has been mined from the Dos Pobres oxide deposit. The most important host rocks of this deposit are volcanics that are composed primarily of fine grained andesite. These Laramide volcanics (56−70 million years) comprise approximately 75% of the deposit. The deposit mineralization is associated with intrusive rocks varying in composition from tonalite to granodiarite porphyry that have been emplaced in the andesitic volcanics (Langton and Williams, 1982). Chalcopyrite and bornite were emplaced during mineralizing events associated with the intrusives, ultimately oxidizing in the upper regions of the deposit that are currently being mined and processed. Alteration is evident throughout the deposit forming chlorite-epidote-ablite-calcite assemblages on the volcanic pile. Biotite is evident in the volcanics and the intrusive. Deuteric alteration of the groundmass to sericite and clay is also evident with the latter being an important factor for ore selection and blending. Hornblende, plagioclase, chlorite, biotite and especially calcite in the deposit all consume significant quantities of acid. The deposit has undergone extensive alteration and many of the alteration minerals are more susceptible to degradation in an acidic leach environment than unaltered gangue.

The oxidation products of the primary chalcopyrite-bornite Dos Pobres deposit have created a significant near-surface leachable resource. Copper in the oxidized portion of the deposit is contained within copper bearing iron oxides, chrysocolla and tenorite. Other minor copper minerals present in the leachable ore reserve include chalcocite, cuprite and native copper that occur primarily in mixed oxide zones encountered at depth in the deposit at the transition between the oxide and sulfide zones.

Process description

Run-of-mine (ROM) ore from the two open pits is transported by haul truck to the primary C-160 jaw crusher, where the size is reduced to P_{80} −152 mm. The crushed ore is then conveyed to a coarse ore stockpile, reclaimed and conveyed to an "open" secondary screening plant. Oversize and middlings from the screens are fed to two MP1000 secondary crushers while undersize is fed to the tertiary crushing circuit. Secondary crusher product is combined with the secondary screening plant undersize and conveyed to a surge bin which in turn feeds four open circuit banana screens and four MP1000 tertiary crushers. Tertiary screen undersize and tertiary crusher product are combined on a single discharge conveyor, sampled and transported to a fine ore storage bin. The final crusher product achieves a P_{80} of 14 mm and typically has a fines content of 5% to 10% minus 75 micron (200 mesh).

Crushed ore from the fine ore bin is conveyed to one of two rotating agglomeration drums for acid curing. Water or raffinate solution is introduced to the ore in the drums, followed immediately by high strength (96−98%) sulfuric acid. Ore moisture content is increased in the agglomeration process from approximately 3% up to 7−9%, depending on ore characteristics. Acid addition to agglomeration is driven by calcite and copper content in the incoming feed, which is determined by blasthole assays. An ore blending program controls clay and calcite content as well as copper grade. Typical sulfuric acid addition to agglomeration ranges from 17.5−22.5 kg/ton. Each 1% of calcite content consumes 10 kg/ton sulfuric acid. Experience at Safford has demonstrated that the acid demand for calcite in the ore must be satisfied in the agglomeration process to ensure optimal copper leaching in the subsequent heap leaching process. The combination of high acid addition requirements in agglomeration and predominance of altered minerals in the gangue increases the potential for chemical degradation of the ore feeding the leach pad.

The agglomerated ore is transported via overland conveyor to the heap leach pad, where it is transferred to a series of portable grasshopper conveyors and a stacking conveyor system. Ore is discharged from a radial stacker onto the leach pad to create a lift height of 4.9 m. Ore is placed in rectangular cells on the leach pad that are a nominal 128 m in width. The conveyor stacker system retreat stacks across the length of the cell and as the placement advances, irrigation piping and drip irrigation tubing are placed on top of the lift to initiate the leach process.

The leach pad at Safford is constructed in a natural arroyo drainage that has a centralized solution collection system that discharges from a single point at the base of the leach pad. The leach pad has been constructed in two phases: the original Phase I, constructed for the 2007 start-up, is 762 m wide by 2,292 m long and was stacked with ore through August of 2012. A Phase II expansion was constructed for 2012 ore delivery and is constructed contiguously on the west edge of Phase I with an approximate dimension of 488 m wide and 2,292 m long. Solution from Phase II drains directly into the centralized

solution collection system in Phase I. The leach pad base follows the natural sloping drainage topography from north to south across the length of the pad while the uppermost surface has been leveled to accommodate the stacking system. This results in a stacked heap that is deeper on the downhill side of the pad in comparison to the upper area that has a higher base elevation.

Leach pad construction from the bottom up consists of compacted native soil, a 30 cm clay layer, 1.5 mm HDPE (high density polyethylene), a network of perforated drainage pipes, a 0.6 m minimum layer of minus 38 mm screened overliner gravel, and a ROM layer that ranges in depth from 0.6 m to over 10 m. The ROM layer was installed to provide additional protection for the liner and drainage system and to smooth out the natural elevation contours in the arroyo, so as to create a relatively flat gradient for the conveyor stacker system to operate on.

The gravel overliner drainage layer atop the synthetic liner continues to be free draining in the Safford leach pad, a critical factor for the recent success with wick drain installation in the leach pad. The central solution collection system in the drainage layer consists of two, 610 mm diameter DR 11 solid wall HDPE pipes that are fusion welded and perforated. Solid wall HDPE was selected for this critical centralized leach pad drainage to avoid possible failure by pipe collapse or coupler failure at joints as the pad load increased over time. The large diameter, solid wall pipes are fed by a lateral collection network of perforated, corrugated PE drain pipes.

Leaching

The surface of each stacked ore lift is irrigated using drip irrigation at an application rate of 5−6 L/hr/m^2. Sulfuric acid concentration in the raffinate has ranged in the recent past from 9 to 13 gpl. The leach cycle time is approximately 120 days, during which time about 75−85% of the recoverable copper is extracted, the remaining being left for subsequent residual leach cycles. The surface of the stacked ore is broken into modules of 128 m by 61 m. The flow and pressure of the raffinate solution feed to each module is controlled and monitored via a wireless mesh system to maintain process setpoints as total leach system hydraulics change over time. This close control of flow operating parameters minimizes both drip line plugging and variability in raffinate flow distribution over the surface of the heap.

The leach solution percolates from the surface of the heap through the underlying lift, dissolving copper from the mineral species and steadily losing acid concentration as it flows vertically through the ore column to underlying lifts. Gangue minerals continue to react and consume acid in the solution. At the base of the heap, the solution enters the bottom layer of gravel and drain collection pipes, and is transported out the base of the pad as PLS (pregnant leach solution) containing copper for extraction in subsequent downstream processes.

At the conclusion of the primary, 120-day leach cycle, irrigation piping is removed and the lift is allowed to drain for five to ten days (dependent on weather conditions). The surface is then prepped to provide a suitably level surface for the stacker system operation, and prior to ore placement with the stacker, the area immediately in front of advancing ore stacking is ripped with a dozer to break-up surface compaction.

Deterioration and/or compaction of the leach surface is a concern at virtually all stockpile leach operations and is especially critical in acid consuming ores that have elevated fines content, clays and clay-like altered minerals that are susceptible to further alteration and degradation with increasing acid contact. Ores such as these require attention to operating practices that limit the impact of the most problematic mineral types and physical compaction of the ore. At Safford, gangue mineralogy is identified in the geologic model and is further defined with blast hole sampling and analysis. Ore is blended to control the variability of problematic ore minerals, including swelling clays and clay-like minerals and significant acid consumers such as calcite. Despite close attention to blending, there continues to be variability in the mineral content of the stacked material, and despite excellent agglomeration and stacking practices, there will always be inherent heterogeneity in the particle size distribution in the stacked ore lifts.

The surface of underlying lifts on which fresh, agglomerated ore is stacked is treated with care to avoid mechanical damage and compaction from surface traffic. Only those vehicles critical for the maintenance of the conveyor stacking system are allowed on the leach pad; and the majority of these have been converted to low-ground pressure tracked vehicles, use double-bead low pressure tires, or are limited to driving on mats that spread the vehicle weight more evenly on the leached ore surface. Conversion to more exclusive use of low-ground pressure vehicles has been an evolutionary development at Safford; early lifts did not receive the same level of compaction avoidance that current operational practices dictate. Also, factors such as rain events during the lift draindown period, higher variability in the combination of clay minerals reporting to the leach pad, or chemical and mechanical degradation during the leach cycle can still cause zones or lenses of the lower permeability material at the upper-most surface of the leach pad. While these factors can be mitigated by dozer ripping, there are times when an overstacked area has less ideal conditions of ore permeability at the interface between lifts.

SXEW

The PLS from the leach pad flows into a PLS collection tank and is then transported to a conventional solution extraction (SX) facility. The SX plant at Safford consists of three extraction stages and one strip stage. The extraction stage is piped to have the flexibility to be operated in virtually any configuration of parallel or series-parallel operation. PLS flows to the plant typically range from 4,100 to 5,200 m^3/hr.

Raffinate from SX is reconstituted with acid and make-up water as necessary to achieve operating setpoints and is pumped back to the leach pad to continue the leach cycle.

Rich electrolyte from the SX circuit is pumped to an electrowinning tankhouse where copper cathodes are produced as a final product. Cathode copper production totaled 79,400 tonnes in 2012.

Heap permeability factors

Throughout its life cycle, the crushed ore is under the constant action of chemical and mechanical forces. The process of agglomeration and leaching alters the chemistry, composition, permeability and possibly the strength of the ore. The transportation over multiple conveyor transfer points and stacking of the ore impart mechanical force on the ore, as does steadily increasing weight as additional lifts are placed above. Ore located at the surface of each lift is subjected to isolated vehicle and foot traffic during irrigation system installation, the highest acid concentrations in the raffinate, and exposure to weather events. At the completion of the primary leach cycle, this same surface is then subjected to equipment operation to remove piping and prepare the surface, dozer operation to rip the compacted surface, the ground pressure from the radial action of the stacker, movement of the conveyor string feeding the stacking system, and routine vehicle traffic associated with maintenance of the conveyor stacking system. Safford employs a number of operating practices focused on limiting equipment impacts that cause mechanical degradation and compaction, but it is impossible to eliminate all mechanical energy associated with conventional leach pad operation.

Chemical reactions with water and acid change the mineral characteristics of the ore, weakening it in the process, and the mechanical forces break the weakened ore. The process of ore breakdown is referred to as degradation or decrepitation, which is manifested in the change in particle size distribution of the ore before and after leaching. At Safford, the fines content (particles smaller than 75 microns or 200 mesh) of pre-leach ore typically falls within the range of 5% to 10%; but the fines content of the post-leach crushed ore increases to a range of 15% to 25%. The upper end of the range, at 25%, greatly exceeds the values from the hydrodynamic column testing during the design stage of the project. Data from heap drilling also suggests a trend of increasing fines content in the first two or three leach cycles, after which the fines content remains relatively stable.

Saturation of crushed ore

Ore decrepitation leads to reduction of its in situ permeability, and potential saturation of the ore heap. The in situ permeability of crushed ore in the Phase I heap at Safford was measured in a large number of dissipation tests using cone penetration (CPT) and in field slug tests. The measured in situ permeability of the crushed ore varied over five orders of magnitude, from 10^{-1} to 10^{-6} cm/s, but mostly fell into a tighter

band of 10^{-2} to 10^{-4} cm/s. In comparison, the hydrodynamic column tests conducted during the design phases showed typical post-leach permeability of 1×10^{-2} cm/s or higher. While some of the in situ tests showed permeability on the order of 1×10^{-2} cm/s comparable with the value measured in the laboratory columns, the majority of the tests showed permeability of less than 1×10^{-2} cm/s. A minority of the in situ tests showed values less than 1×10^{-4} cm/s with some tests values lower than the equivalent heap irrigation rate of 1.4×10^{-4} cm/s (5 L/hr/m^2). The variability in the measured permeability is consistent with the variability in the fines content. The large drop in permeability of leached ore reflects both the increase in fines content and the clay mineralogy of the fines. The variations in permeability also reflect the presence of compacted and low permeability inter-lift boundaries. The combination of chemical and mechanical forces discussed previously contributed to the variable formation of zones or lenses of material, with elevated fines content that were susceptible to equipment compaction and decreased permeability. The reduced permeability zone or lens created by the compaction of the inter-lift boundary added to the complex variation of permeability within the heap.

It is commonly assumed in the industry that, as long as the permeability of the crushed ore is larger than the irrigation rate, the ore will remain unsaturated and the leach solution will be able to percolate through the pad unimpeded. At Safford, the equivalent irrigation rate is 1.4×10^{-4} cm/s (5 L/hr/m^2). Thus, according to the common assumption, most of the pad should remain unsaturated and should function as designed. However, numerical modeling of solution transport through the ore heap indicates that parts of the pad with sufficiently large permeability contrast can become saturated even though the permeability is higher than the irrigation rate. The numerical analysis thus demonstrated the unexpected finding that parts of the heap leach pad could become saturated even if the permeability of the crushed ore is generally higher than the irrigation rate. Saturated zones at Safford leach pad were indeed observed in geotechnical boreholes and the CPT results, due to both in situ permeability of crushed ore lower than the irrigation rate and large permeability contrast across the pad.

When a lift or several lifts become saturated, the stacking of additional crushed ore above the saturated lift will cause the fluid pressure to rise. The fluid pressure will rise because the pressure from the weight of the newly stacked ore is initially carried partly by the pore fluid. The pressure rise will not be uniform across the pad:

- Vertically, the cells being stacked and irrigated will contain lifts of variable saturation, while lifts that are not fully saturated will not show as much response as the saturated lifts.
- Horizontally, the cells are being stacked and irrigated sequentially and the saturated lifts in a newly stacked and irrigated cell will have higher pressure than those in the adjoining cells.

The difference in pressure at different parts of the pad creates hydraulic gradients, and the gradients cause the solution to drain from areas of high pressure to areas of low pressure. As the solution drains, the high fluid pressure will begin to fall. The rate of fall depends on how easily the solution drains which, in turn, depends on both the value and the variation of permeability of the crushed ore. In areas where the drainage is impeded by either low permeability zones or high permeability contrast boundaries, little pressure dissipation might occur during the standard leaching cycle, such that the pressure will accumulate when the next lift is being placed. The accumulation can eventually lead to conditions where the fluid pressure exceeds the hydrostatic value. The CPT results confirmed these conditions in areas of the heap.

Mitigation of heap saturation

Since start-up in late 2007, a total of 11 lifts have been stacked on the Safford Phase I leach pad and the maximum depth of ore and overliner reaches 61 m at the deepest location on the leach pad. As the leach pad neared its current height, solution lag time through the heap became increasingly difficult to predict and did not follow the assumptions in the production model that had generally been predicting solution reporting time from the heap accurately. Some seep areas developed on the heap side slopes and in late 2011, a resistivity survey suggested that areas of solution saturation existed within the heap. In early June 2012, a localized side-slope movement occurred on one of the cells that was related to multiple lift solution saturation and the build-up in pore pressure in that location of the heap. As a follow-up to this event, a series of monitoring and remediation activities were undertaken. CPT rigs were mobilized to site and began locating areas in the heap with variable permeability, particularly at interface boundaries between lifts in the heap, and measuring pore pressure associated with saturated zones above or between these areas of low permeability. The information from the CPT rigs was used to position horizontal drills to contact saturated zones and drain solution from some portions of the heap. Additional vertical piezometer wells were drilled to increase monitoring of phreatic surfaces in the heap and calibrate CPT results. The most significant advance however was the decision to bring in wicking rigs and place vertical wick drains throughout the Phase I leach pad.

Wick drains

Wick drains, commonly known as prefabricated vertical drains or PVDs, are prefabricated strips of flexible polypropylene plastic containing channels and surrounded by a geotextile filter fabric. Wick drains are typically used in construction applications to consolidate compressible soils rapidly in advance of building projects that require a consolidated foundation. They have also been used in mine tailings applications for dewatering and consolidation. Water in the soil, under pressure in excess of hydrostatic,

flows through the filter fabric of the wick drain and into the channels of the plastic wick drain core where it can flow vertically either up or down and out of the soil (American Drainage Systems, 2006). Wick drains may be installed to depths exceeding 60 m, but due to the density of wick drains, the water only has to travel a relatively small distance horizontally before contacting a wick drain and flowing unimpeded vertically out of the soil.

Wick drains are installed using a wick rig, which consists of a tall mast assembly typically mounted to a tracked excavator or crane. A mandrel, which is typically a steel H-beam or square tubing, is mounted on the mast assembly and is forced vertically into the soil in a smooth continuous motion. A spool of wick drain material is attached to the mandrel and the wick is driven vertically into the soil along with the mandrel. When the mandrel reaches its maximum depth of penetration, the wick drain is released to remain in the punched hole and the mandrel is extracted. A 30 m wick drain can be installed approximately every 90 seconds, although transit time between wick drain locations limits daily productivity on the Phase I leach pad to approximately 200 holes. As mast height and corresponding wicking depth increases, the surface preparation becomes increasingly important to provide a stable platform for movement of the wick rig and attached tall mast assembly.

At Safford, wick drains up to 38 vertical meters have been installed using a CAT 385 excavator. Wick spacing is typically on a 7.6 m square grid across the surface of the heap which is based on the raffinate irrigation rate. Welded blocks on the mast provide a physical block to limit penetration depth which keeps the mandrel and wick a minimum of 5.2 m above the synthetic liner. A larger wick rig with a 61 m mast and ability to achieve wick drain depths of 55 m is currently being prepared for use at Safford on deeper areas of the Phase I leach pad. To date, approximately 12,500 vertical wick drains have been installed on the Phase I leach pad to depths up to 38 m, and additional wick installation is planned using the larger rig.

As the Phase II leach pad steadily increases in height with additional lifts, wick drains will be installed on the interior of the Phase II leach pad to proactively ensure that metal recovery is maximized by routinely maintaining or reestablishing hydraulic connectivity between permeable areas of the leach pad. This preventative wicking will counteract the potential for future formation of low permeability ore zones and inter-lift boundaries that contribute to restricted solution flow in the heap. This will allow solution to contact underlying ore over multiple leach cycles to achieve ultimate metal recovery. This preventative wicking will also promote overall heap drainage and stability.

Wick drains are also being installed preventatively on 7.6 m centers on every second lift on the outer perimeter or shell of the Phase II leach pad to promote interlift percolation. This is done to ensure a drained shell which will improve geotechnical stability on the exposed, outer 122 m perimeter of the leach pad. An additional feature on Phase II is the installation of perforated drain pipes every 7.6 m

between lifts in the outer 122 m of the leach pad. This drainage layer will limit the height to which a phreatic surface can rise to ensure that a saturated zone will not form above the interface boundary between lifts.

Benefits from wick drain installation

The quantifiable value in wicking has been the release of trapped PLS in confined layers. Wick drains have reduced pore pressures and lowered the phreatic surface in zones of saturation in the heap, which has been confirmed by CPT and piezometer readings. The wick drains have reestablished hydraulic connectivity between more permeable zones in the heap, which has allowed leach solution to contact ore that has residual copper value. An additional, critical benefit of draining solution is that it increases factors of safety and slope stability on the heap. Lastly, preventatively wicking the Phase I and Phase II leach pads will ensure a drained condition in the heap to allow the placement of additional lifts in the future while maintaining residual leaching and acceptable factors of safety for geotechnical stability.

The impact of releasing trapped solution from the heap via the wick drains has been profound from a production standpoint. Between the start of wick drain installation in late June of 2012, and the end of May 2013, over 13,600 tonnes of additional copper have been recovered from the Safford Phase I heap. Revenue from this increased production has far exceeded the cost of increased heap monitoring and wick drain installation. A portion of this increased production is attributable to temporary increases in PLS flow that rapidly drained from horizontal wells and vertical wick drains into the downstream SXEW recovery system. However the largest impact to production has resulted from an increase in residual leaching due to the reestablishment of hydraulic connectivity between permeable areas in the heap. The penetration of low permeability zones and lift interface boundaries by the wick drains allowed ore to be contacted with “fresh” leach solution which restarted a dormant leaching process and produced significant quantities of soluble copper.

Summary

Ore deposits with mineral characteristics and high reagent or acid consumption that lead to ore decrepitaton can create physical conditions on leach stockpiles that lead to challenges in maintaining ore permeability and geotechnical stability in multi-lift, permanent heaps. The Safford Mine has addressed this challenge by employing an innovative strategy of CPT monitoring to target problem areas, and by proactive installation of wick drains to reestablish continuity between permeable areas of the heap. This combination has resulted in large increases in metal production and has led to preventative installation of wick drains and horizontal perforated pipe drains to promote optimal leach conditions in the heap and to maintain a fully drained perimeter shell that improves geotechical stability and factors of safety.

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Lessons learned in 30 years of laboratory liner interface strength tests compared to real world heap leach pad strengths

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Abstract

The engineering design of geomembrane-lined fill structures, such as heap leach pads, solid waste landfills and tailings impoundments, typically involves the use of one or more low permeability composite liner systems. A composite liner system generally includes a low permeability subgrade material in direct contact with a flexible geomembrane liner. In recent years, bentonite geosynthetic clays liners have seen more frequent use as the low permeability subgrade material, where clayey soils are not available or where the construction time is more efficient along steep sided valley slopes.

This paper will present a 30-year retrospect of the laboratory measurement of liner interface shear strength using the direct shear method, initially based on small-scale direct shear box analysis (ASTM D3080) and evolving into large-scale direct shear box evaluations (ASTM D5321). Discussions will be presented on the effects of shear box size, as-placed moisture/density, plasticity, fine content, preload consolidation, and pore water pressure dissipation under high normal stress conditions (typical of ore heap fill loads placed on a heap leach pad liner). The apparent effect of high-load deformations or "dimpling" on a micro scale into the planar elastic geomembrane liner surface in contact with the overlying and underlying fill rock particles will be addressed, as well as excess pore pressure conditions in overly wet composite liner clayey soils. The laboratory testing discussions will be compared to real world slope failures on liner systems, since slope failures are the true test for determining liner interface shear strengths. The intent of this paper is to apply 30 years of laboratory testing and real world heap leach pad design and construction knowledge to help separate the interface liner strength "facts" from the "fiction" with respect to composite liners under high fill load structures.

Introduction

The 1980s to the present day have shown a rapid growth in geomembrane-lined facilities throughout the world to meet government and industry requirements for the containment, recirculation, and/or treatment, and release of leachate and process solutions to closure. These lined fill structures include liquid and solid waste landfills, tailings impoundments and heap leach pads. Heap leach pads were first used in the mining industry in the 1970s with clay liners; however they quickly changed from 1979 onward to geomembrane liners, so that now gold, silver, and copper mine heap leach pads are the highest geomembrane lined fill structures in the world.

These lined fill structures require special design and operational considerations to prevent instability and leakage. The engineer relies on experience and field performance of similar lined fill structures, as well as laboratory testing of the interface shear strength under simulated construction conditions, for guidance in the design of stable liner systems. These composite lined heap leach pads typically include an overlying drain fill cover to minimize hydraulic heads on the liner system and protect the geomembrane liner surface from punctures or tears during subsequent controlled ore heap fill placement operations and an underlying soil layer, as shown in Figure 1.

The drain fill cover also minimizes exposure to the climate and human or animal activity before the relatively level ore lifts can advance uphill and bury the pad liner, as shown in Figure 2. As lined fill structures are constructed higher than ever before, or site conditions significantly vary from previous historic lined fill experiences, engineers must rely more heavily on the simulated laboratory strength testing of liner systems in order to predict actual shear strength conditions for the life of operations to closure.

a) b)

Figure 1: a) Soil liner underlying geomembrane b) Drain fill cover overlying geomembrane

Figure 2: Controlled stacking of ore lift on a geomembrane lined heap leach pad

This paper focuses on the lessons learned over 30 years in the practical application of large-scale laboratory direct shear test strengths at the composite liner interface relative to high fill load conditions. In particular, the limitations of laboratory testing will be compared to studies of fill slope failures on geomembrane liners.

Background

General

By the mid 1980s composite geomembrane liner systems had become the preferred choice of engineers and environmental regulators for liquid and seepage containment incorporating a low permeability soil in direct contact with the overlying geomembrane liner. However, the overall reduction in permeability for the "water tight" single or multiple liner systems beneath waste and process fills had a drawback, in that it also reduced the strength of the liner systems for stabilized fill slopes. The challenge for the engineer then became how to test and accurately predict the interface liner strength, to ensure that unprecedented fill heights on the various types of geomembrane liner systems would remain stable under differing site conditions.

Lined fill slope failures

Past slope failures on geomembrane-lined fill structures such as solid waste landfills, heap leach pads, and reclamation cover fill caps have shown that liner induced slides generally occur by wedge failure at the planar geomembrane liner interface contact with fine-grained soils or geotextiles. Most of the engineering community first became aware of the potential for low planar interface strengths in geomembrane liner systems following the Kettleman Hills landfill slope failure in Northern California in 1988 (Mitchell et al., 1990). However, the mining community was aware of low strength planar liner slide failures more

than a decade earlier in the 1970s, but mining engineers at that time had no case history liner experience or reliable test data to solve these slide problems.

Starting in the early 1970s geomembrane liners were used to line large evaporation ponds. Large uranium tailings impoundments were being lined by the mid 1970s. These lined facilities required liquid containment of water and tailings slurry by pipeline hydraulic deposition. Valley leach pads were being constructed by the late 1970s, and backfilled with drain fill cover and ore lifts for leaching within a contained basin. The valley leach pads had internal process solution ponds for potential ponding at high hydraulic heads on the liner system during operations. Some of the lined impoundments included a sand and gravel drain fill cover placed above the exposed liner surface for drainage and protection from climatic conditions.

The importance of knowing the friction strength at the geomembrane liner to soil interface contact became apparent when the protective liner cover fills began to fail on the steeper side slopes of the impoundments during construction or subsequent operations. Dozers would slide backward on cover fill slopes during dozed fill placement or haul trucks would slip on the lined access roads, while bringing ore fill to the valley bottom for controlled fill lift placement. In one case, the water pool in a lined tailings impoundment eroded the drain cover with wave action at the base of the slope, causing the upper slope cover fill to slide downward and tear the liner. By the mid to late 1980s, several large slope failures had occurred beneath geomembrane lined landfills and leach pad fills along relatively flat liner bottom grades. The slides generally occurred at the low permeability soil or geotextile interface contact directly beneath the geomembrane liner for fill loads, or at the drain fill contact above the geomembrane liner in the cases of dynamic loading by dozer and truck traffic activity.

As a general summary of the past history of liner slide failures, the land fill industry became aware of low interface liner strengths by the late 1980s in well documented studies of the Kettleman Hills landfill slide in California. However, it is unfortunate that neither the mining industry nor the landfill industry reviewed or studied liner slide case histories that extended back to the 1970s. Several major landfill slope failures occurred between 1988 and 1997 in North America, Europe, Africa, and South America (Koerner and Soong, 1999). Mining liner slide failures started in the 1970s, with the lesser known heap leach pad slope failures occurring between 1985 and 1993 at several mine sites in North America, South America, and Australia (Breitenbach, 1997). The knowledge gained from the past slope failures on heap leach pads has shown the importance of stabilizing both initial low lift load and ultimate high multiple lift load ore fill along the downhill pad toe limits to prevent unstable wedge slip failures on the geomembrane lined pad surfaces

Leach pad slope failures have continued to occur to the present day, primarily during placement of the first ore lift at startup of operations (generally high ore lift placement at angle-of-repose slope in the

downhill direction on downhill sloping geomembrane liner grades or underlying wet of optimum moisture subgrade soils). The concept of changing weak planar geomembrane liner surfaces to more stable non-planar surfaces by "micro-dimpling" loads (Breitenbach and Swan, 1999) and "macro-dimpling" stabilization grades (Breitenbach and Athanassapoulos, 2013) have proven to be useful engineering tools for applying laboratory interface liner test strengths to the stabilization of lined fill structure slopes.

Laboratory direct shear testing

A modified direct shear box apparatus became the primary tool in the early years of liner design for determining liner strengths, and it remains popular to the present day. The engineering challenge that still remains is how to accurately apply the laboratory direct shear test results to real world site conditions during actual fill placement operations.

The direct shear box was originally intended as a quick and simple test for measuring the effective shear strength of relatively free-draining, non-cohesive sandy soils. The first direct shear tests were conducted a long time ago by Coulomb in 1776. Specific laboratory direct shear test procedures for engineered soil fills were established by Lambe in 1951 (Lambe and Whitman, 1969). The commonly accepted shear box size for non-cohesive sandy soil testing to the present day is a 4-inch by 4-inch (100 mm by 100 mm) square box. A direct shear box with applied loads is schematically shown in Figure 3.

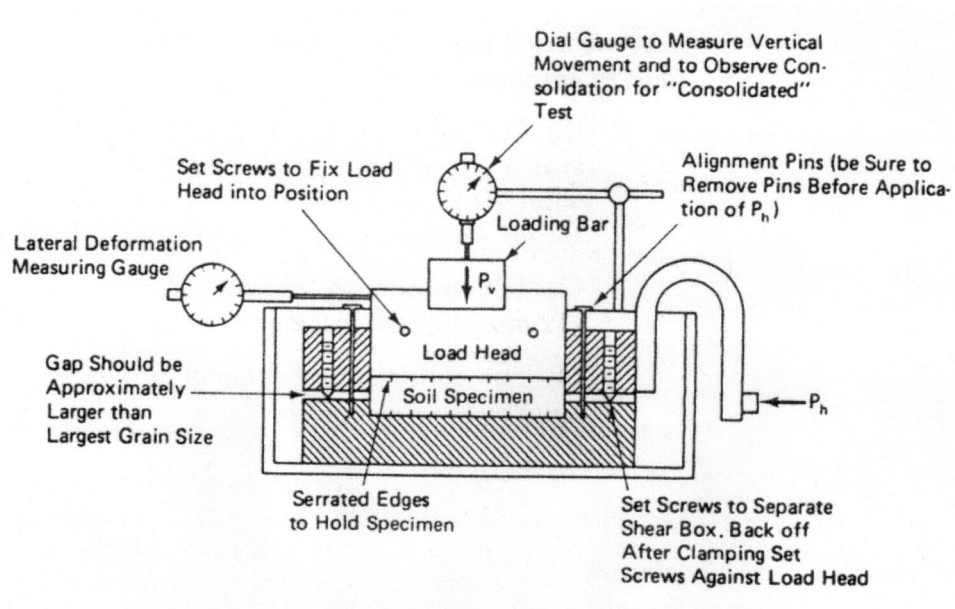

Figure 3: Direct shear apparatus (Bowles, 1978)

The 4-inch square direct shear box quickly became the "unofficial" test method for determining liner interface strengths, and remained so throughout the 1980s and into the 1990s, due to the planar weakness characteristics of the smooth geomembrane liner surface. Other types of liner strength test

methods were also adopted to a lesser extent, including circular ring shear tests and liner "pullout" shear tests. The 4-inch square box size was found to be inadequate in the late 1980s for leach pad interface liner design. It was found to give inaccurate test results that varied by as much as 10 degrees peak friction strength for the same composite liner geomembrane and soil materials tested by the top three liner testing laboratories at that time under the same sample preparation and test procedures.

Bigger 6-inch and 12-inch (150 mm and 300 mm) square direct shear box tests became available in the late 1980s and early 1990s to accommodate the testing of composite liner designs, which incorporated drain fill rock sizes of typically 3/4-inch (19 mm) sieve size or larger. A typical composite liner system is shown in Figure 4. A schematic example of an interface direct shear liner test is illustrated in Figure 5.

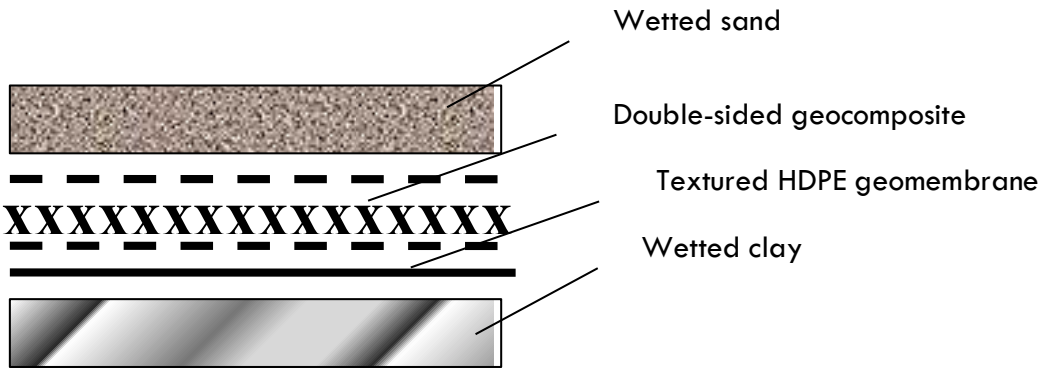

Figure 4: Typical composite liner system with leak detection

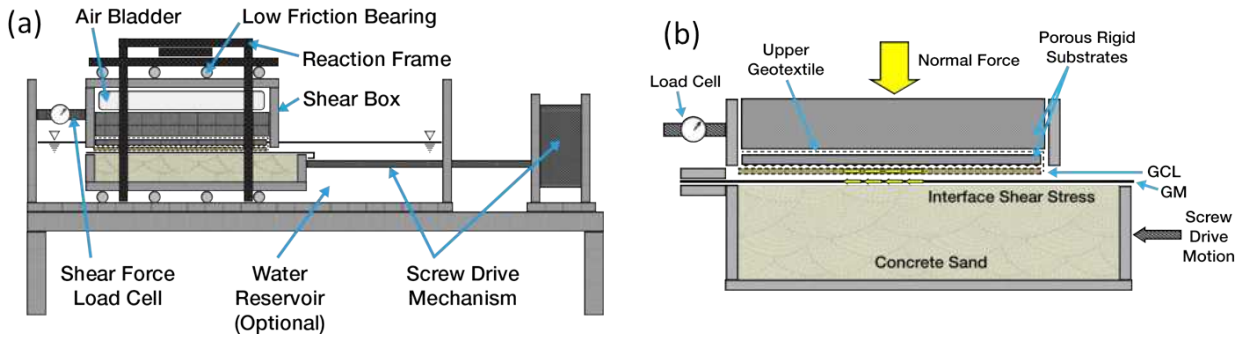

Figure 5: Schematic example of an interface direct shear liner test

The 12-inch square direct shear box test procedures for the liner interface strengths were officially established in November of 1992 (ASTM D5321). Although the 12-inch square shear box and established test standards produced more consistent test results compared to the 4-inch square shear box, other factors needed to be considered in applying simulated laboratory test strengths to the actual site conditions during construction and operation of the lined facilities.

Factors influencing interface test shear strengths

Published information on geomembrane liner interface shear strengths in textbooks generally presents a brief description of test sample materials (soil overliner/underliner classification, liner type, liner thickness, smooth or textured liner surface) along with the peak or residual friction angle and apparent cohesion interface strength test results in a table for easy reference. However, in order for the engineer to effectively utilize this information as a reference guideline for assumed design strengths on a particular project with similar liner and fill materials, there are many more factors to consider.

As an example, the underliner soil compaction could be either standard dry unit weight (ASTM D698) or modified dry unit weight (ASTM D1557), and may have been remolded near optimum moisture content to a typical range of 90 to 100% compaction. If the underliner soil compaction procedures are not reported along with the liner test strengths, the test results at 90% of the lower standard dry unit weight test method versus 100% of the higher modified dry unit weight test method could easily vary the published test strength values by more than 20%. If the lower standard dry unit weight test is placed at 2% points wet of optimum moisture content and the higher modified dry unit weight test is placed at 2% points dry of optimum moisture content, then the published test strength values could vary by another 20% or more.

These examples are just some of many reasons why the published strengths are highly variable for similar soil materials and liner types. In fact, there are so many factors influencing the laboratory interface direct shear strengths that site specific material tests are recommended as a requirement, along with appropriate laboratory test equipment and procedures, for final engineering analyses and designs.

Several interrelated factors that have the greatest influence on the interface shear strengths measured in laboratory tests are discussed by Breitenbach (2011). The following sections of this paper present a general chronological sequence of events by which the various factors discussed by Breitenbach were considered by the authors in determining the practical application of laboratory liner interface test strengths to the design of high fill load structures.

Small versus large direct shear tests

Laboratory interface shear strength tests for the various types of geomembrane liner systems and fill structures initially started with the small-scale 4-inch sized direct shear box. The authors found the small scale direct shear test results from the 1980s to be so highly variable for similar liner and soil materials that the test laboratories were asked to rerun the tests. The underliner soil material was being typically tested at optimum moisture and 95% of standard compaction (ASTM D-698) in combination with an overliner consisting of clean sand and gravel cover fill, so that excess pore pressures would not be considered a problem in testing. The test reruns differed by several degrees of frictional strength, along

with variable apparent cohesion values, using the same test equipment, materials, and procedures. A review of published literature at that time also showed variable test strengths for similar soil and geomembrane liner materials.

Subsequently in 1990, the primary author sent identical samples of geomembrane liner, underliner bedding fill, and overliner drain fill materials to three separate qualified laboratories in California, Texas, and Georgia as an experimental test program for small scale liner direct shear testing. Detailed instructions were provided with the identical samples to conduct the same sample preparation, loading, and testing. The purpose of the testing was to verify the reliability of the 4-inch direct shear box in determining accurate test strengths. The underliner soil material consisted of low permeability clay compacted to 95% of maximum standard dry unit weight at optimum moisture content. A 60-mil smooth HDPE (high density polyethylene) geomembrane liner was provided to each laboratory from the same manufacturer. The overliner soil material consisted of a clean sandy gravel placed in a single loose lift and wetted to simulate a protective drain cover fill above the liner surface. Three confining pressures were specified to simulate an equivalent 50, 100, and 150 feet (15 to 46 m) high ore heap fill. A constant shear displacement rate was also specified at 0.04 inches (1 mm) per minute.

The test results were surprisingly variable, with a difference of 19.8, 24.0 and 30.5 degrees in "effective" peak interface friction strengths and varying apparent cohesion values in the range of 100 to 900 psf. Needless to say, following the experimental 1990 test program, it was decided that all further liner interface strength tests would be performed using the large scale 12-inch sized shear box, which had an order of magnitude larger test surface area.

Further validation came from the subsequent performance of several high load laboratory tests conducted simultaneously in 6-inch and 12-inch square direct shear boxes. Several gold and copper heap leach projects were being designed in 1993 to ultimate fill heights of 600 to 800 feet (183 to 244 m) above the geomembrane liner surface. The ore fill densities were significantly higher than for landfills at about 110 pcf (pounds per cubic feet density − 1.75 tonnes per cubic meter) for moist ore versus about 75 pcf for a typical landfill density. The 12-inch square shear box equipment (actually 12-inches by 14-inches across the bottom box) could accommodate the design test loads to approximately 200 feet (61 m) in equivalent ore fill height, while the smaller 6-inch square shear box equipment could accommodate the higher loads beyond the 200 feet height. The same soil and geomembrane liner materials and test procedures were used for both the 6 and 12-inch square shear boxes. The end result, with a minimum of three test points plotted at different normal confining stress loads per test in each size shear box, was near identical in interface friction and apparent cohesion strength test values. Additional verification came from subsequent testing of different soil materials and liner types in 1993 and 1994 on several other high fill projects that continued to show near identical strengths in the simulated 75 to 750 feet (23 to 229 m)

equivalent fill range with the 6-inch size box alone. These test results proved that the larger 12-inch box can be utilized to accurately predict liner interface strengths with step load time delays as needed to minimize excess pore pressure conditions.

Restrained versus unrestrained liner

The direct shear tests can be conducted with the geomembrane liner anchored to the top test box along the opposite side of the lateral driving shear force, which restrains liner movement during testing. The restriction allows the liner material itself to elongate and resist shearing, contributing to a higher peak interface friction strength. The less flexible liners like HDPE have a higher material break strength than the more flexible liners like PVC (polyvinylchloride) or PPE (polypropylene) during elongation from direct shear testing. The end result from past tests conducted from 1990 to 1992 on HDPE, LDPE (linear low density polyethylene), and PVC liners is that the HDPE liner would show about one to two degrees of additional peak interface friction strength with the liner restrained (12 to 15% one-dimensional elongation to break typical). The more flexible liners show no significant increase in restrained liner strength due to their high elongation to break characteristics (400 to 800% two-dimensional elongation to break typical).

The geomembrane liner was also allowed to "free-float" without anchorage to the direct shear box and be unrestricted during testing. The unrestricted direct shear tests do not utilize the liner material strength, and therefore are the most conservative in test strengths. Considering the design loads and driving forces of high fill structures, the geomembrane liner tear strength in a one to two mm (40 to 80-mil) liner thickness becomes insignificant so that the unrestricted test is preferred for slope stability analyses.

Dry versus wet of optimum moisture

As the 1980s liner designs progressed into the 1990s, regulatory requirements for composite liners became more stringent with soil hydraulic conductivity (permeability) requirements as low as 1×10^{-7} cm/sec or less (recognizing that geomembrane liners are not 100% leak proof from punctures, tears, and weak seams). The engineer was faced with designing low strength composite liner systems and double or triple liner systems with the potential for very low strengths at the soil to geosynthetic and geosynthetic to geosynthetic material interface contacts. Some engineers compounded the strength problem by specifying above optimum moisture in the underlying liner fills, so that the local borrow soils could meet the regulatory requirements for permeability. The specifications were based on laboratory permeability tests, which indicated that the soil fill permeability can potentially drop an order of magnitude at a given dry unit weight by increasing the moisture content several percent points wet of optimum moisture (Hermann and Elsbury, 1987; Estornell and Daniel, 1992).

The tradeoffs in wetter soil at a "lower permeability" in the laboratory versus lower strengths and associated desiccation cracks in the field were well known by engineers with liner construction experience, but apparently took several years to reach published literature (Daniel and Wu, 1993). The end result of this turmoil is that some of the published literature of direct shear test results on composite liner systems from the early days to the present day may have been performed under partial saturation or excess pore pressure conditions. Most of the wet of optimum direct shear test results involving low permeability soils are essentially meaningless for determining the effective shear strength without knowing the excess pore water pressure conditions at the induced shear test failure along the geomembrane liner interface. The reported and published direct shear test strengths under wet of optimum conditions are likely between a total stress and effective stress strength, which leads to a discussion of the apparent cohesion in the next section.

Cohesion versus no cohesion

The standard practice from the 1980s into the 1990s for conservative design strengths is to assume no cohesion in the slope stability analyses for liner interface strengths. The cohesion intercept decreases to zero for a clean and fully drained sand or gravel fill and increases in value for low permeability or wet of optimum high plasticity soils. The opposite effect occurs for the frictional strength with high friction strengths typical for fully drained sand and gravel fills and lower friction strengths typical for finer grained silty and clayey fills. The friction strength can approach zero with the cohesion intercept at its maximum value for a relatively saturated and fine grained soil approaching unconsolidated and undrained (total stress) conditions with quick loading and shearing rates.

In most cases, the laboratory will apply a normal confining stress load to the prepared test sample and immediately begin the shear test. If direct shear liner tests are conducted under "quick" loading and shearing conditions or the sample has been remolded to wet of optimum moisture, then the effective stress conditions may be unknown due to the potential for partial pore pressure conditions in the test sample at failure. Under these conditions, the apparent cohesion should be discarded for a conservative friction strength estimate.

The true effective stress strength (fully drained for no excess pore water pressure) can be determined by allowing the confining stress to consolidate the sample before shear testing, in combination with a sufficiently slow strain rate to failure. Note that the friction strength increases and the apparent cohesion decreases as load consolidation time is allowed prior to shear testing. Allowing as little as 12 to 24 hours of load consolidation time before testing increases the interface liner strength by several degrees. There may be some effect from deformation of the flexible geomembrane liner surface with time under loading (see section discussing low versus high fill load strengths).

More flexible versus less flexible liner

There is some debate to the present day about whether or not a difference in friction strengths exists between the more flexible versus less flexible liners. Some confusion likely originated in the early literature on published strength test results conducted in the 1980s using the small scale 4-inch square shear box, or the early direct shear tests that continue to the present day on liner systems with wet of optimum low permeability soils. Experience gained from a combination of more than 10 years of observations of large scale liner test fills, liner installations, large scale 12-inch square shear box test results (performed at optimum or dry of optimum moisture content), and a review of known pad liner slope failures all point to only one conclusion. There is no debate.

The HDPE and MDPE (medium density polyethylene) liners are semi-crystalline and exhibit the lowest interface friction strength characteristics, associated with being the least flexible of all the liner types on the market. The PVC, CSPE (clorosulfanated polyethylene or HYPALON) and PPE liners exhibit the highest interface friction strength characteristics, associated with being the most flexible liners. The LDPE liners are generally a few degrees higher in interface friction strength relative to the HDPE/MDPE liners, and a few degrees lower in interface friction strength relative to the more flexible PVC and PPE type of liners.

The more flexible liners apparently allow “dimples” or slight undulations to more readily occur on the interface surface, even under relatively low fill cover loads, which increases the shear strength test results. The dimpling or uneven geomembrane liner surface under consolidation loads apparently causes the slide failure to shear through a portion of the underlying or overlying soil materials for an overall increase in shear strength along the geomembrane liner interface contact.

Smooth versus textured liner surface

Textured geomembrane liner sheets (roughened surface with variable protrusion patterns) for the HDPE, MDPE, and LDPE liners were available in the late 1980s and first used on leach pads in the early 1990s for increasing the smooth surface friction strength. A textured sheet of HDPE liner can be comparable in interface strength to the more flexible PVC and PPE smooth liner sheets.

There are other design considerations that should be noted in selecting textured liners for increased strength purposes. The textured liner sheets have some installation difficulties in tacking, grinding, and welding of the seams across the textured rough surface for a water tight seam. Most textured sheets have a 6-inch (150 mm) wide smooth surface edge for hot shoe wedge welding, but impoundment corners and occasional angled sheet overlaps require cross cutting and seaming through the textured surface area. Therefore, the textured sheets are typically designed for placement only in limited fill traffic areas or along the critical perimeter and downhill toe locations of the lined leach pad facilities for fill slope

stability. The liner installer deploys the textured sheets in a parallel pattern as much as practical to avoid the cross cut extrusion welded seams.

The textured liner sheets can be single or double sided, but there are a variety of textured patterns and protrusion shapes to select for design, depending on the liner manufacturing process. The deeper and more sharp-edged texture patterns may have the best interface strength, but can weaken the cross cut seams and the liner sheet material itself.

Direct shear test results to date indicate minimal texturing is required for adequate liner strengths at the liner contact with fine grained soils and geotextiles. More testing is required to make a judgment about the effects of texturing on the more clean and coarse grained drain fills placed above or below the geomembrane liner.

There may be certain clean drain cover fills that exhibit lower rather than higher strength characteristics on textured sheets with the larger rock particles essentially riding over the textured protrusion points for less surface contact to resist sliding. This phenomena may occur particularly for rounded rock particle shapes and for poorly graded rock size distributions. Finer rock particles, such as sand sizes, can nestle within the textured protrusions for greater interlocking contact strength. In one case, the composite liner test results on smooth and textured HDPE sheets showed an increase of only two degrees friction strength difference with shear movement occurring along the underliner fine grained soil contact for the smooth sheet on the first test, followed by shear movement along the overliner gravel drain fill contact for the second double sided textured liner test.

Laboratory test results and a review of slope failures on lined facilities show that the more granular overliner fills have higher shear strengths compared to the less granular fine grained underliner soils. Numerous direct shear tests have been performed on a wetted sand or gravel overliner drain cover fill and a composite fine grained clayey or silty underliner bedding fill placed generally within 1% wet or dry of optimum moisture content in contact with all types of geomembrane liners and liner thicknesses. The liner surface after shear failure consistently showed shear movement at the geomembrane to underliner soil contact. Therefore, the single sided textured sheet for composite liners should be placed facing downward in contact with the fine grained and more plastic soils.

Low versus high fill load strengths

The “dimpling” effect of the liner interface under loading, as discussed earlier, is related to several factors including the plastic deformation of the soil (moisture content, density, gradation, and plasticity), the potential for any localized differential settlement of the subgrade under loading, the flexibility of the geomembrane liner, and the fill load conditions (incremental versus ultimate load and rate of loading).

The dimpled surface allows the liner interface strength to approach the strength of the underlying or overlying soils by becoming non-planar in nature, similar to the effects of a textured liner sheet.

It is interesting to note two things related to load strengths and dimpled liner surfaces. First, the underliner bedding fill should show an increase in shear strength corresponding to both an increase in granular soil contact (reduction in fines passing the No. 200 sieve size) and a decrease in plasticity. Second, the maximum dimpling effect for any given liner occurs with large rock sizes allowed in the overliner fill in combination with finer silt and clay underliner materials placed at a low density and wet of optimum moisture content. The fill load strengths gained (rocky cover fill materials dimpling the geomembrane liner into the soft foundation soils) is lost by the reduced underliner soil shear strength in addition to increasing the risk of liner rock puncture. Therefore, textured sheets or very flexible liners are preferred beneath fill loads for increasing liner interface strengths, rather than designing for a dimpled liner surface.

Under low fill load conditions of typically less than five feet, such as a liner cap on a landfill, the thicker and less flexible HDPE/MDPE liners do not have enough normal loading on the liner surface to dimple the liner sheet. Therefore, the less flexible liners will show the lowest interface friction strengths compared to thinner sheets of the same liner type or more flexible liners. The "micro-dimpling" effect was first studied by the authors on several mine projects having different underliner and overliner sample materials in the late 1980s through 1992 by gluing mostly PVC liners to relatively rigid wood blocks. The purpose of the tests were to find the lowest planar strength of the flexible liner without the added strength of dimpling. The peak planar test strengths of the PVC liner with the wood block substrate were found to be in the range of 14 to 19 degrees frictional strength, compared to about 22 to 26 degrees frictional strength with a composite liner and drain cover fill.

By 1994 and 1995, high plasticity clays for underliner bedding fill were being tested for extremely low interface friction strengths at optimum moisture content. The normal confining stresses were applied for simulated high fill loads and allowed to consolidate 0, 12 and 24 hours before testing. The delay in testing to allow consolidation of the sample increased the peak friction strength typically by three to five degrees friction with a reduction in apparent cohesion. Part of the reason for increased friction strengths can be attributed to the dimpling effect, but also may be related to minimizing pore pressures for effective stress conditions.

The first lift of fill in a multiple lift landfill or leach pad operation creates the greatest incremental change in stress on the liner system and underlying subgrade soils. As the fill height increases with the same lift thickness, the incremental change in stress decreases. If the foundation liner system and subgrade soils contain materials with low strength characteristics (i.e., planar synthetic liner surface in contact with fine grained, low permeability, low density, wet of optimum, and/or high plasticity soils),

then a large incremental change in stress may create unstable conditions until the load consolidation has stabilized. Therefore, the early startup controlled fill lift operations are generally the most critical for the geomembrane liner system and slope stability rather than at the ultimate fill loads.

Conclusion

General conclusions concerning the application of laboratory direct shear liner interface strengths to lined fills are as follows:

- The smaller 4-inch square direct shear box was found to be unreliable for testing the interface strength of composite liner systems. The larger 12-inch square direct shear box is recommended for the engineering analyses and design of major fill structures.
- The difference in anchoring the geomembrane liner to the direct shear box for restrained liner interface strengths versus "free-floating" unrestrained liner interface strengths can be significant for the less flexible liners, depending on the geomembrane liner type and thickness.
- The fine grained high plasticity soils placed wet of optimum moisture content for direct shear liner testing are somewhere between total and effective stress conditions to where the test results are not valid. The low permeability and wet of optimum soils require time for load consolidation to dissipate any excess pore water pressures, and may require a very slow strain rate to failure.
- The apparent cohesion should be discarded for all soils tested wet of optimum moisture content to be conservative in design strengths used for stability analyses.
- The less flexible geomembrane liners have a lower direct shear interface friction strength compared to the more flexible geomembrane liners. The apparent difference in test strengths is due to the micro-dimpling or undulating effect of the liner surface under consolidation loads for the more flexible liners in which the shear forces must also shear through a portion of the soil to cause a slide failure along the uneven geomembrane surface.
- The textured interface friction strength of HDPE/MDPE/LDPE geomembrane liners is significantly improved for the more fine-grained and low permeability soils, with less improvement in test strengths for the more clean and coarse-grained soils. A textured HDPE/MDPE liner also approaches the strength of the more flexible smooth CSPE, PVC, and PPE liner sheets.
- The low fill load strengths for the liner caps can be significantly less than high fill load strengths for landfills and heap leach pads due to a dimpling effect or non-planar nature of the geomembrane liner surface under the higher consolidation loads.

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Constitutive relationships for the representation of a heap leach process

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Abstract

Heap leaching of metals is a valuable process which has gained momentum since its first industrial scale application in the early 1960s. More recently, bioleaching of primary metal sulfides has been pursued because of their significant potential economic value. Interest from the mining industry over the last several years has resulted in conceptual models, computational tools, laboratory and pilot scale projects, as well as industrial applications to prove the feasibility of these concepts. Although heap leaching is in principle a simple concept, optimization of the process at an industrial scale has proven challenging. Part of the reason is that optimal design of the (bio) leaching process requires integration of several disciplines, including the following: metallurgy, mineralogy, biochemistry, geology, chemical engineering, civil and geotechnical engineering, mathematics, computer sciences, mechanical engineering, hydraulics and hydrodynamics. The interest in heap leaching has resulted in well-developed concepts and well-understood process constraints; however, scale-up from laboratory experiments under fully controlled conditions to the industrial scale under natural conditions remains challenging.

This paper focuses on the measurement of the constitutive relationships required to characterize fluid flow within an ore mass. These laboratory techniques have been applied to a large variety of ore samples in the precious and base metal industries. The resulting data sets provide, for the first time, a complete characterization of the critical variables which control solution and air flow in the context of a heap leaching operation. These data sets eliminate the need for analytical models developed by soil scientists for significantly different materials. This information should simplify numerical modeling of the process and scale-up by eliminating the need for curve-fitting to estimate the hydrodynamic parameters. Several data sets are used to demonstrate the effect of ore management practices (crushing size, agglomeration techniques and additives, as well as blending) on the constitutive relationships and their

impact on heap leach design and operation. Examples are presented on the use of the laboratory-derived data to optimize process design.

Introduction

Although heap leaching is a simple concept, optimization of the process at an industrial scale is challenging requires integration of many disciplines. A key challenge is scale-up from laboratory conditions to the industrial scale (Scheffel, 2010). In our experience, a major limiting factor for the advance of the heap leaching practice at an industrial scale is the lack of knowledge of the physical and hydraulic (hydrodynamic) properties of the ore which control the movement of solution and air through the ore bed.

Much has been invested in ore leach pad design, mineralogical and metallurgical characterization of ore candidates for a heap leaching process, and their numerical representation (Scheffel, 2006). However, very little has been done to quantify the mechanisms responsible for the movement of solution and air through the heap profile. To put it in the words of one of the pillars of science:

> When you can measure what you are speaking about, and express it in numbers, you know something about it, when you cannot express it in numbers, your knowledge is of a meager and unsatisfactory kind; it may be the beginning of knowledge, but you have scarcely, in your thoughts, advanced to the stage of *Science*, whatever the matter may be.
> (William Thomson − Lord Kelvin)

Heap leaching can be thought of as a special case of reactive transport where solution movement provides for both reagent delivery and removal of the dissolution products (Guzman et al., 2006). The mining industry has refined the formulation of lixiviants necessary to effectively and efficiently dissolve the metals of interest. Unfortunately, it has not typically pursued the understanding of the hydrodynamic mechanisms responsible for the "transport" of solutes. Worldwide experience indicates that limited understanding of these mechanisms has been the reason behind poor performance and even failure of leaching operations.

Consideration of the mechanisms responsible for fluid flow within a heap leach pile lead to the realization that there are two stages of control of the leaching process. The first is available before the ore is placed on the leach pad and is affected by decisions regarding the top size of the ore (run-of-mine versus crushed-ore), the actual size and method of crushing, the choice between agglomeration and as-crushed placement, the selection of agglomeration additives, the choice of blending ratios, the design of the drainage and aeration system (if necessary) and the mode of stacking. It is critical that heap leach practitioners realize these decisions will have a significant impact on the physical and, more importantly, on the metallurgical performance of the process. The second stage of control becomes available once the

ore has been stacked and will depend on the selection of the geometry and method of the irrigation system as well as the irrigation (and aeration) schedule.

This paper focuses on measurement of the constitutive relationships required to characterize fluid flow within an ore mass and use of these measurements to support informed decision-making during the design process. The laboratory techniques discussed within this document have been applied to a large variety of ore samples on the precious and base metal industries. The data are used to demonstrate the effect of ore management practices (crushing size, agglomeration techniques and additives, fine removal, and blending) on the constitutive relationships and their implications for process design.

Methodology

Standard geotechnical characterization of an ore for leach typically includes measurement of the saturated hydraulic conductivity as a function of heap height in an attempt to quantify the potential hydraulic response of the ore mass (Lupo and Dolezal, 2010). However, the hydraulic conductivity of an ore sample is not only a strong function of the degree of compaction, but it also varies non-linearly with the degree of liquid saturation. As such, a good saturated hydraulic conductivity ($\geq 10^{-2}$ cm/s) is a necessary but not a sufficient condition to ensure adequate performance of an ore under percolation leaching at an industrial scale. When dealing with altered ore samples, standard geotechnical testing has on occasion produced overly pessimistic results because of the inherent limitations of the testing procedures, or overly optimistic results which lead to serious problems during the start-up of the operation.

The limitation of the standard techniques becomes critical when dealing with a leaching process which requires natural or forced aeration (leaching/bio-oxidation of sulfides) because standard geotechnical procedures provide no information at all regarding the ability of the ore to support air flow. Moreover, extensive experience with the hydraulic behavior of ore materials shows that the operational degree of saturation resulting from the design solution application rate is a better indicator of the overall performance of the ore than the saturated hydraulic conductivity itself − a piece of information not collected during standard geotechnical testing. Ore samples with an elevated level of fines or a large sand-fraction, those with a tendency to decrepitate, and poorly agglomerated samples, tend to run at higher saturation levels than competent, coarser ores under the same irrigation rates. On a heap, a high level of saturation results in poor solution distribution, solution ponding, wash-outs, slope stability problems and elevated copper inventory. Moreover, in the case of a bioleaching process, a high degree of liquid saturation would be detrimental from the point of view of air permeability and the ability to efficiently deliver oxygen to support the biologically assisted oxidation.

In order to overcome the limitations of the standard geotechnical characterization methods, two specialized testing procedures have been developed to fully characterize the physical and hydraulic

response of an ore under a percolation leaching process. Guzman et al. (2008) present a detailed discussion of the mathematical formulation necessary to represent solution movement through porous materials under variable saturation conditions, while this paper focuses on the measuring techniques and the interpretation of the resulting data. These data sets provide for the first time a complete characterization of the critical variables which control solution and air flow in the context of a heap leaching operation. In the context of modeling of the leach process, these data sets eliminate the need for analytical models developed by soil scientists for significantly different materials (e.g., van Genuchten, 1980; Brooks and Corey, 1964). The use of these data to address data scale-up and simulation of metal extraction has been the subject of a recent publication (Robertson et al., 2013); this topic is addressed in a limited fashion just to highlight its critical importance.

Furthermore, observation at the industrial scale shows that the lack of an agglomeration standard results in significant variability from day to day and from operation to operation. As will be demonstrated from the empirical data presented below, improved agglomeration has a significant impact on hydrodynamic performance of the ore and hence on the metallurgical performance of the process (e.g., Readett and Fox, 2011). With this in mind, the following scale was developed to qualify the agglomeration product; a non-agglomerated sample is assigned a Level 0 (L0), while a sample that has been fully agglomerated is assigned a Level 5 (L5). Most leaching operations are working with agglomerated product below L3 or lower. It is important to note that low levels of agglomeration result in particle size segregation along the heap profile, which in turn promotes solution segregation (channeling), protracted leach cycles and overall poor leaching efficiency.

Recent experience shows, however, that attaining higher levels of agglomeration (L4 or L5) is feasible once the necessary conditions have been identified and operators have been properly trained (Guzman et al., 2006). Experience shows that application of this scale helps explain the discrepancy often encountered amongst metallurgical tests, as well as the differences between lab-scale tests and industrial leaching operations. Based on these observations, our work on this subject lead us to conclude that optimal agglomeration needs to satisfy the following specifications:

1.0 Bind all the dispersed fines to minimize their negative effect on the percolation and solution-retention capacity of the ore.

- 1.1 The percolation capacity of the ore should be at least 100 times larger than the typical application rate (10^{-4} cm/s).
- 1.2 The maximum solution retention capacity during irrigation should result in a liquid degree of saturation smaller than or equal to 75% for an oxide-leaching operation, or 60% for a sulfide leaching operation.
- 1.3 Air conductivity should remain larger than 10^{-3} cm/s.

2.0 The resulting porous structure produces a total porosity larger than 30%, which is partitioned between macro- and micro-porosity of about 50:50 to facilitate bulk solution movement and intimate contact between solution and ore.

3.0 The porous structure is sufficiently resilient to withstand deformation and physical stress resulting from the design heap-height.

4.0 The agglomeration product is sufficiently resilient to chemical decrepitation.

5.0 The agglomeration product and resulting porous structure is able to withstand flooding without major loss of structural integrity.

6.0 Produce an agglomeration product which promotes leaching bed homogeneity, uniform flow distribution and equal opportunity leaching for all the size fractions.

All these specifications become more important in the case of a multi-lift heap. With these requirements in mind, the testing procedures were developed based on nearly 31 years of work in the hydrological sciences and 18 years in the mining industry. The test procedures provide the most complete characterization of the physical and hydrodynamic properties of an ore-for-leach available to the mining industry. The laboratory procedures developed to date include the stacking test (ST) and the hydrodynamic column test (HCT). The following paragraphs provide an overview of these procedures.

Stacking test (ST)

An ST measures the changes in bulk density and hydraulic conductivity as a function of depth. The resulting density and conductivity profiles determine whether the ore as prepared would be able to support the leaching process. The ST is an indicator procedure in that positive results simply indicate that the sample is a candidate for percolation leaching and further testing can be pursued. Negative results, on the other hand, indicate categorically that the ore as prepared would not support percolation leaching. The ST produces data to test specifications 1.1, 2.0, 3.0, and 5.0 and, when using already-leached samples, it provides information on specification 4.0.

An ST is performed by placing an ore sample into a test cell and mechanically increasing the confining load to simulate the effect of a heap weight on the bulk density. The load is increased in a stepwise fashion, allowing for height stabilization during each of the loading steps. The density and permeability of the ore are measured at each step and then the load is increased to simulate additional lithostatic loading. The maximum load for an ST is selected to represent either the maximum lift height in the case of a dynamic heap, or the maximum heap height in the case of a permanent (multi-lift) heap. An inadequate sample produces high density values (leading to total porosity values smaller than 30%) and a steep conductivity profile indicative of a rapidly changing conductivity, while a competent sample produces relatively low density values and a relatively flat conductivity profile. The ST is conducted

under partially saturated conditions, containing only the moisture of agglomeration (or as-drained moisture in some cases), as opposed to fully saturated conditions typically employed in similar studies conducted by geotechnical practitioners. The following results are obtained from an ST:

- *Density profile* – defines the relationship between ore density and heap height. This profile provides a direct measurement of the physical integrity of the ore sample under load and, as such, it quantifies the robustness of the pore structure resulting from the sample-preparation procedure.
- *Hydraulic conductivity profile* – defines the relationship between ore conductivity and heap height. These data represent a direct measurement of the integrity of the porous structure and its resilience under various heap heights, and provide a direct measurement of the effect of physical and chemical decrepitation (for a leached sample) on the ability of the ore to allow solution percolation.
- *The minimum hydraulic conductivity of the sample* – by design, the results from an ST represent the bulk density, conductivity and, at the end of the test, the saturated hydraulic conductivity of unit volume of ore located at the bottom of the heap. Therefore, the saturated hydraulic conductivity value at the end of the load-sequence indicates whether the heap is sufficiently permeable at its ultimate height to allow free drainage of the pregnant leach solution (PLS) to the collection system.
- *The maximum lift and heap height* – data from the density and conductivity profiles provide a direct measurement of the lift and heap height at which the ore sample will allow unimpeded percolation of solution.
- *Preliminary estimates of total-, micro- and macro-porosity* – these preliminary estimates provide a direct indication of the capacity of the sample to support percolation leaching. Ample data from industrial operations indicate that a total porosity of at least 30% is required for proper solution and air percolation. In addition, a 50:50 portioning of the porosity into micro- and macro-components has been determined to provide a good balance between advection and diffusion controlled solution movement.

Hydrodynamic column test (HCT)

The HCT provides the most complete characterization of the physical and hydrodynamic properties of an ore-for-leach at given bulk density. An HCT produces data to test all the requirements for optimal agglomeration defined above. The HCT is performed by placing an ore sample into a column and subjecting the sample to a confining pressure equivalent to desired lift or heap height, as defined from the results of an ST. The diameter of the test cell is selected to minimize potential wall effects on the determination of hydrodynamic parameters of the ore sample. Once the sample has been placed onto the

test cell, the irrigation rate is varied over several orders of magnitude to evaluate the corresponding degree of saturation and the resiliency of the porous structure as the ore becomes increasingly wet. Each flow period is extended until steady state flow conditions are developed, at which point the corresponding degree of saturation is measured. The test is designed to allow simultaneous measurement of the hydraulic and gas conductivity as a function of moisture content (degree of saturation) and pore pressure. The maximum solution application rate is determined by the value at which the surface of the column becomes saturated. Once the ore is flooded, a saturated hydraulic conductivity test is conducted to quantify the maximum flow capacity of the ore. This maximum value typically varies from sample to sample and is strongly controlled by the ore preparation practice. The results generated from the HCT procedure include:

- *Saturated hydraulic conductivity* – defines the maximum percolation capacity of the sample. A saturated hydraulic conductivity (K_s) greater than or equal to 10^{-2} cm/s is a necessary but not sufficient condition to ensure adequate metallurgical performance of an ore under percolation leaching at an industrial scale. The actual criterion for K_s is different for a dynamic heap (initial K_s greater than 100 times the application rate) than for a multi-lift, permanent heap (initial K_s greater than 1,000 times the application rate).
- *Hydraulic conductivity curve* – indicates the degree of liquid saturation resulting from a steady-state solution application rate. This is the most critical parameter in the design of the leaching/bio-oxidation process since the liquid saturation not only defines the ability of the sample to allow solution and air movement, but also determines the mechanical stability of the ore.
- *Moisture retention curve* – provides the degree of saturation (moisture content) of the sample as a function of capillary (pore) pressure and hence defines the physical state of the solution within the sample. This parameter is critical if numerical modeling of the flow process would be pursued but not critical from a practical point of view for design of the leaching process.
- *Air permeability curve* – provides the air flow capacity of the ore as a function of degree of liquid saturation (solution application rate). This is a fundamental design parameter for an air-injection system, which is often overlooked.
- *Drain down curve* – provides an idea of the rate of solution drainage due to the action of gravity. It represents a critical parameter necessary to estimate residual solution (and metal) inventory, required rest time before over-stacking (for a permanent heap) or removal of the leached residue (for a dynamic heap), as well as the potential discharge volume and rate during closure and post-closure conditions.

- *Total-, micro-, macro-, and residual-porosity* – the total porosity indicates the pore space per unit volume of the sample. For a given ore sample, in general, and in an agglomerated ore in particular, the macro-porosity is associated with the pore space between rock particles/agglomerates while the micro-porosity represents the pore space within rock particles/agglomerates themselves. The residual porosity is the fraction of the pore space which will remain saturated even after a prolonged drain down period. Optimal agglomeration process produces a 50:50 micro to macro partition and minimizes the residual porosity.

By design, a sample subjected to an HCT represents a unit volume of ore located at the bottom of the lift – the portion of the heap which experiences the most stringent conditions since it is exposed to the maximum lithostatic load and the maximum degree of liquid saturation along the heap profile. If the hydraulic performance of this portion of the heap/lift is adequate, it is reasonable to conclude that the rest of the heap/lift profile will perform just as well or better.

The results from an HCT provide all the necessary information to understand movement of solution and air under a percolation leaching process. In other words, the HCT generates all the parameters required in the context of numerical representation (computational fluid dynamics) of the leaching process. From a more practical point of view, the HCT answers the key question for the design of a heap – the degree of saturation resulting from a given application rate. Experience shows that for copper oxide leach the maximum liquid saturation should be kept below 85%, and ideally below 75%, to accommodate the natural variability associated with the heap stacking process. Leaching of a sulfide ore typically requires that the degree of saturation remains below 60% to ensure that forced (or natural) aeration can be properly implemented. The operating degree of saturation is the critical design parameter as it controls solution-ore contact, aeration capacity of heap, and mechanical stability of the heap.

In the context of the design of a new leaching operation, hydrodynamic characterization provides a way to quantify the benefits from alternative ore preparation approaches (from mining methods, to crushing, to agglomeration and to stacking) so these results can be used to select the most favorable ore preparation techniques and maximum bulk density for a given ore sample. In addition, this information can be used to select optimal operating conditions, including maximum lift height, maximum heap height, irrigation and aeration schemes and schedule, as well as to determine the operating moisture content, drain-down moisture and maximum air intake, among others.

Data and discussion

This section presents data generated with the hydrodynamic testing techniques described in the previous section. To date, more than 500 samples from around the world from different ore types and minerals (copper, silver, gold, nickel, zinc, and uranium) have been tested. Selected results from these tests are

presented to illustrate the typical results, the type of information that can be obtained and the conclusions that can be derived from these data. The focus of the discussion is on the qualitative interpretation of the data, rather than the specific sample or operation, so the source of the samples is not critical and shall remain confidential. To the extent possible, the following examples include a brief description of the problem addressed by the characterization program, a graphical representation of the data generated by the lab-tests, a few key observations and design implications.

ST measurements to assess the effect of agglomeration

The ore used in this example corresponds to a secondary sulfide initially planned to be leached as a multi-lift run-of-mine (ROM) operation. Inspection of the ore and some small-scale metallurgical tests suggested that efficient leaching may require additional ore preparation. The results from the ST are presented in Figure 1, where the density profile is presented on the left and the conductivity profile is on the right.

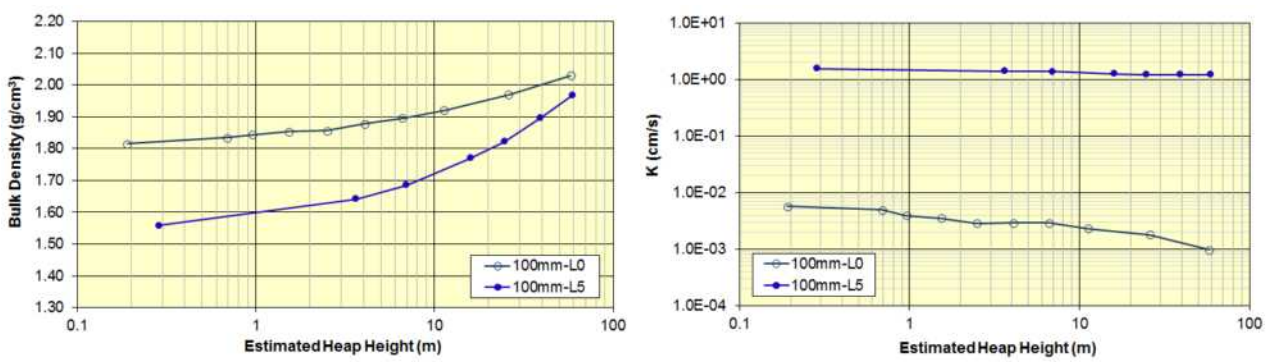

Figure 1: Effect of agglomeration on density and conductivity profiles

Inspection of these plots indicates the following:

- The data from the non-agglomerated sample (100 mm – L0) shows this sample is characterized by "as-placed" density values larger than 1.80 g/cm^3 and hydraulic conductivity values smaller than 6.0×10^{-3} cm/s.
- The density increases while the conductivity decreases in a non-linear fashion as the heap height increases. As the heap height reaches a height of 10 m, the density has increased to about 1.91 g/cm^3, while the conductivity has decreased to about 2.5×10^{-3} cm/s.
- For this particular sample, with a specific gravity (SG) of 2.79, the threshold heap height when the bulk density (1.95 g/cm^3) results in a total porosity less than or equal to 30% is about 20 m. Given that the conductivity at the 20 m height is about 2.0×10^{-3} cm/s, it is concluded that this material will not support heap heights beyond this threshold elevation.

- Interestingly, the hydrodynamic behavior of this coarse sample with 80% of its particles smaller than 100 mm (P_{80}) and a fines fraction (< 74 µm) of about 8% is such that it will not support more than two 10 m lifts.
- Full agglomeration of this sample, on the other hand, decreases the "as-placed" density and reduces the density throughout the range of heap heights tested for this ore. More importantly, full agglomeration increases the conductivity by more than two orders of magnitude (100 fold).
- These results indicate that for this particular sample, the porous structure resulting from full agglomeration is sufficiently robust to support a heap height as high as 60 m.

Design implications

The information derived from the STs in this particular sample shows that a non-agglomerated sample would limit effective leaching of this ore to heap heights under 20 m. Given the relatively low level of fines (~8%), this finding highlights the negative effect of dispersed fines and the ability of agglomeration to neutralize it. Agglomeration of the ore would impose a significant increase in the capital cost and increase operating costs; however, treatment of the ore without agglomeration would likely have resulted in a very difficult start-up and potentially failure – a much more costly alternative.

ST measurements to assess the effect of fines removal

In this example the operator of an oxide leach operation was dealing with an ore with about 20% fines (< 74 µm). The SG of the ore was measured at 2.67. The leaching process was originally designed as a multi-lift, ROM operation which was negatively impacted by the presence of the fines. Two potential options were considered to reduce the negative impact of the fines on the metallurgical performance of the operation: a) fines removal and b) adding an agglomerating drum. STs were conducted on the original particle size distribution (PSD) and two additional samples where 5% and 10% of the fines had been removed. Figure 2 summarizes the density and conductivity profiles generated from the STs.

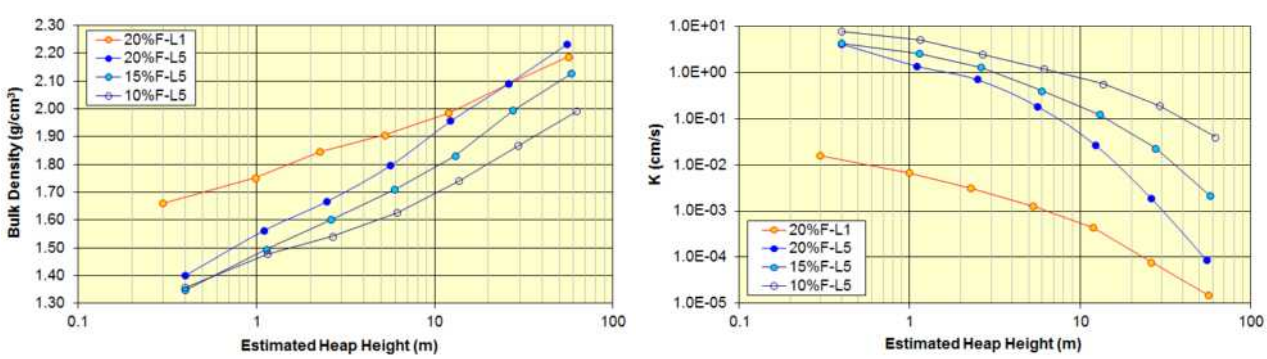

Figure 2: Effect of fines removal on density and conductivity profiles

The results from the STs on these three samples lead to several important observations:

- There is a noticeable improvement on both the density and conductivity profiles of the original sample as the level of agglomeration increases from minimal (20%F-L1 – red line) to full (20%F-L5 – solid blue circles and line).
- Although the reduction in bulk density obtained from full agglomeration diminishes as the heap height increases, the improvement in conductivity subsists over the range of heap heights tested. The minimum conductivity of the L5 agglomerate (at a heap height of 60 m) is almost an order of magnitude larger than that of the L1 agglomerate.
- Removal of 5% (15%F-L5) and 10% (10%F-L5) of the fines combined with full agglomeration shows improvement over full agglomeration of the original PSD (20%F-L5). This improvement is, however, relatively modest when compared to that obtained from the fully agglomerated sample (20%F-L5) with respect to the minimally agglomerated sample (20%F-L1).
- Considering the cost of fines removal (capital and operational), on top of the additional cost necessary to treat the fines in a separate circuit, it was concluded that full agglomeration of the original PSD could be the most viable treatment route. However, given the limited pad area an alternative solution needed to be identified.

Design implications

The results from the STs on this particular sample show that low level agglomeration (or no-agglomeration as originally designed) would not support a multi-lift heap. In fact, even the L1 agglomerate of the original PSD would limit effective leaching of this ore to lifts heights under 3 m – an unsustainable situation. Although full agglomeration significantly improves the hydrodynamic performance of the ore, the maximum lift height that could be leached was still limited to a maximum of about 10 m, which was not compatible with the original design and available pad area. Partial fines removal would impose a significant increase in both capital and operating costs. Moreover, a separate circuit to treat the fines would have added additional economic stress to this low-grade operation. A recommendation was made to identify alternative materials within the ore deposit that could be used to create a more robust sample by blending with this high-fines ore.

ST measurements to assess the effect of top size

This example is based on the sample from a new low-grade, mixed oxide-sulfide project where the design team was interested in determining the optimal top size to be delivered to the heap. The original design called for a top-size of 100 mm. The heap was designed as a multi-lift ROM operation. A key consideration of this analysis is the trade-off between metal liberation versus reduction of hydraulic

conductivity and increase in solution retention capacity resulting from additional crushing. The SG of the ore was determined to be 2.82. The level of fines shows a slight increase from about 8% to 10% as the top-size is reduced. The original sample was screened and the "oversize" fractions crushed to generate the smaller top-size samples. STs were conducted on non-agglomerated (L0) samples. The resulting density and conductivity profiles are presented in Figure 3.

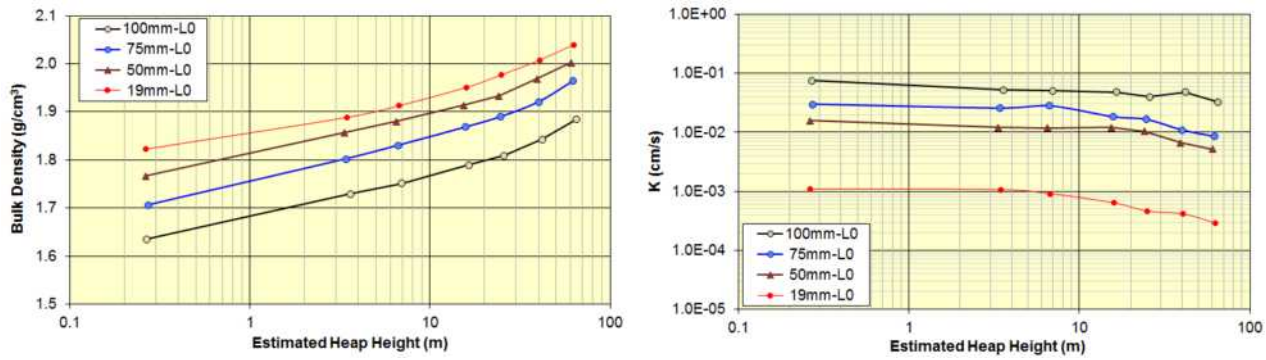

Figure 3: Effect of top-size on density and conductivity profiles

The results from the STs on these samples show the following:

- Reducing the top-size results in higher density and smaller conductivity values throughout the heap profile examined (up to 70 m).
- Given the SG of the ore, reducing the top-size decreases the maximum heap height that would allow efficient hydrodynamic performance (total porosity > 30%). The maximum heap height decreases from over 150 m for the 100 mm sample, to 70 m for the 75 mm sample, to 50 m for the 50 mm sample; and then to about 25 m for the 19 mm sample.
- As the top-size of the sample decreases from 100 mm to 50 mm, the conductivity decreases slowly and then abruptly (about one order of magnitude) as the top size is reduced to 19 mm. The magnitude of the conductivity at the 19 mm top-size is such that it would not support an efficient percolation process and would likely result in high degree of liquid saturation.

Design implications

The results from the STs on this particular sample help to quantify the effect of top-size on the maximum heap height. The data show that crushing to a top size of 50 mm would allow stacking five 10 m lifts, which would satisfy the design criteria. Crushing to a finer top-size would be counterproductive as it would result in too low a percolation capacity. Inspection of the metal distribution by size indicated that nearly 60% of the mineral was contained on the gravel fraction (ϕ > 2 mm), suggesting that additional crushing could improve metal liberation with respect to the original design (P_{100} 100 mm).

ST measurements to assess the effect of decrepitation

An important consideration when designing an acid-based leach process is the potential effect of the acid on the physical integrity of the ore mass. The extent of the chemical decrepitation is a function of the mineralogy of the ore, but also of the acid concentration and the number of cycles to which the ore will be subjected. Therefore, decrepitation will be more significant for a permanent (multi-lift) heap than for a dynamic (single-lift) heap. Decrepitation results in destruction of the porous structure, generation of fines, an increase in bulk density and the reduction of percolation capacity. In order to quantify the potential impact of decrepitation in the context of a mixed oxide-sulfide ore (crushed to a top size of 20 mm), STs were conducted on both the fresh ore and leached residue after a full leach cycle. The results from the STs for these samples are represented by the solid lines and symbols (F 20mm-L3) and dashed lines and open symbols (L 20mm-L3) respectively.

The leached residue was allowed to drain and air dry. The dry material was rolled to break down any residual structure, and then re-agglomerated using acidified raffinate. Both samples were agglomerated to the same level and the same moisture content to facilitate comparison of the ST results. The assumption behind this approach is that any differences in the ST results arise from changes in the competency of the rock fragments and the potential increase in the level of fines.

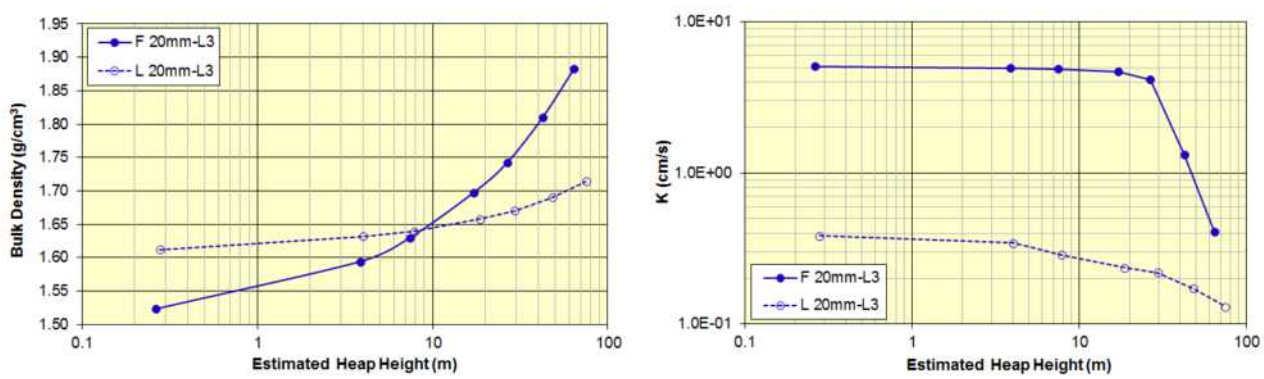

Figure 4: Effect of decrepitation on density and conductivity profiles

The density and conductivity profiles summarized in Figure 4 suggest the following:

- The porous structure resulting from re-agglomerating the leached residue (L 20mm-L3) is more resilient than that generated with the fresh ore (F 20mm-L3); such that its bulk density is lower than that of the head sample. The lower density arises from higher level of fines present in the leached residue and their binding effect, which improves agglomerated structure.
- On the other hand, the higher porous structure resulting from the presence of the fines produces an increase in the micro-porosity fraction (65% for the leached residue compared to 50% for the fresh sample).

- The conductivity profile shows that the leached sample is about one order of magnitude less permeable than the fresh sample. The conductivity curve for the leached sample shows a relatively flat slope even for heap heights over 30 m, while that for the fresh sample shows a sharp decrease as the heap height reaches this value. Notwithstanding this reduction in conductivity, the ST results show that the head sample would be a good candidate for percolation leaching on a heap as high as 80 m.

Design implications

The results from the STs on this particular sample show that although decrepitation is expected to result in compaction and reduction of percolation capacity, the sample is competent enough to potentially support a multi-lift heap design with a maximum height of 80 m.

HCT measurements to assess the effect of bulk density

This example presents the results of HCTs conducted on a mixed oxide-sulfide ore in the context of a multi-lift heap operation. STs on this sample indicated agglomeration at a L3 would allow efficient leaching in a heap as high as 64 m. The purpose of the HCTs was twofold; to confirm the STs results and to determine the operating conditions of the heap for a single 8 m heap, and for the bottom lift once the heap had reached the design height (64 m). The corresponding ore density was selected on the basis of the density profile. The resulting hydraulic and air conductivity curves for both the 8 m and 64 m heap are presented on the left and right frame of Figure 5.

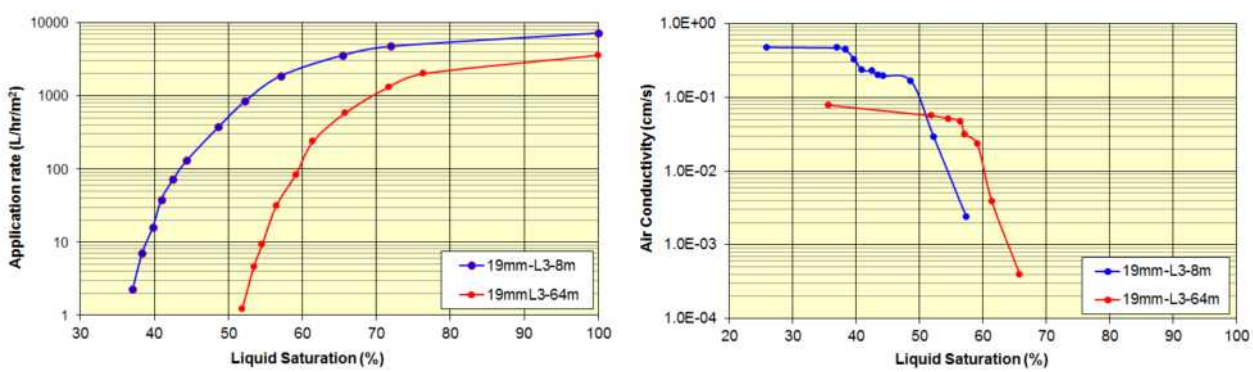

Figure 5: Effect of bulk density on conductivity curves

The data summarized in Figure 5 indicates the following:

- The ability of the ore to support solution and air flow is a strong function of liquid saturation. For the solution flow, the shape of the conductivity curve shows the percolation capacity decreases nearly three orders of magnitude as saturation goes from 100% to its minimal operating values of 38% and 52% for the 8 m and 64 m heaps, respectively.

- This stark difference in percolation capacity from full saturation to lower saturation values lead to the conclusion that a large saturated hydraulic conductivity (*K* at 100% saturation) is a necessary but not a sufficient condition for efficient percolation leaching.
- Air conductivity decreases about two orders of magnitude as liquid saturation goes from it minimal operating value to about 60%. Note that the steep drop in air conductivity occurs much before the pore space is fully flooded (a liquid saturation of 100%). This behavior is responsible for our "rule of thumb" indicating that liquid saturation should not exceed 60% for a leaching process which requires natural or forced aeration.
- According to the conductivity curves, an irrigation rate of 10 L/h/m^2 results in an operating saturation about 38% for the 8 m heap and about 55% for the 60 m heap. The steep shape of the hydraulic conductivity curve shows that this ore would accommodate a large range of irrigation rates without a noticeable change in liquid saturation.
- Although the air conductivity decreases sharply as the liquid saturation increases beyond a threshold value (~49% and ~56% for the 8 m and 64 m samples), the operational range resulting from agglomeration to a L3 is more than sufficient to accommodate forced aeration.

Design implications

The results from the HCTs on this particular sample confirm the results from the STs, which indicated that this ore would be a good candidate for a multi-lift heap leach process when agglomerated to a L3. The hydraulic and air conductivity indicate that an L3 agglomerate would easily accommodate irrigation rates of up to 20 L/h/m^2 to satisfy the reagent demand of the leaching process, with no detriment to air conductivity (forced aeration). In fact, this sample is one of the best-performing samples we have tested. A key task for the design team is to confirm the extent to which this sample represents the ore to be delivered to the heap during the first two years of operation and thereafter.

Summary and conclusions

Ore characterization for an ore-for-leach has been confined to metallurgical and limited hydraulic (geotechnical) testing. Although in most cases this information is adequate, in several instances such limited characterization efforts have resulted in inadequate designs and non-performing leaching operations. This document presents laboratory tools and empirical concepts to assist in the design process of a leaching operation. A comprehensive list of specifications for the leach bed (and optimal agglomeration) is presented. These specifications have been used to guide development of the ST and the HCT so that they provide quantitative data about the extent to which a given sample satisfies these criteria.

Examples of the application of these test procedures on a variety of ore samples are presented to quantify the effect of agglomeration, fines removal, leached-induced decrepitation and densification resulting from self-loading as a function of heap height. Application of these laboratory procedures includes selection of improved design and operational conditions. Extensive experience on characterization of ore-for-leach samples from a variety of minerals indicates the following:

- Ore bulk density controls the hydrodynamic performance of the ore.
- Bulk density varies along the heap profile and strongly depends on sample preparation practices (particle size distribution, crushing approach, agglomeration and additives, and stacking mode).
- Hydrodynamic performance controls operational conditions, metallurgical and geotechnical performance of the ore mass.
- Therefore, it is critical to recognize that operational conditions for a given ore sample depend not only on its geological make-up and its particle size distribution, but also on the ore preparation techniques, the method of stacking, and equally important, the bulk density of the ore.
- Hydrodynamic characterization resulting from the ST and HCT procedures should improve our understanding of percolation mechanisms and help us to improve the design of the leaching process.

We would conclude by saying that our approach has been guided in good part by the words of Lord Kelvin:

> "To measure is to know" and "If you cannot measure it, you cannot improve it."

Acknowledgements

The authors are indebted to the laboratory personnel who have assisted us in executing the test work and provided constructive feedback on the test procedures. Similarly, we are indebted to Mr. Randolph Scheffel for his encouragement and guidance through the last several years and his editorial suggestions on this manuscript.

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Challenges of large heap leach pad design and construction in a cold region of Northern China

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Abstract

The Chang Shang Hao (CSH) Gold Mine is an open pit heap leach mining operation located in Inner Mongolia, a cold and windy region of Northern China. The mine is undergoing a major expansion to increase the production rate from 30,000 to 60,000 tonnes per day. The proposed Phase 2 heap leach pad would have capacity to handle over 100 million tonnes of ore, with an ultimate heap height of 90 m. It will be the largest heap leach facility in China. Numerous challenges were encountered during the detailed design and construction of this large heap leach facility, including complex topography, severe weather conditions, shortage of water and subgrade materials, uncertainty of liner/pipe quality, as well as an aggressive project schedule.

Weather conditions at the mine site change dramatically year round. Temperatures can be as low as −38°C in winter and as high as 37°C in summer. Extreme freezing temperatures and ambient UV radiation demand extraordinary performance from the liner product. Strong winds provide challenges for liner installation. Limited precipitation and tremendous evaporation rates impose extra constraints on water usage and recycling. Freezing temperatures tend to interrupt irrigation and solution recovery in the winter months.

The Phase 2 heap leach pad was originally planned as a stand-alone heap leach facility adjacent to the existing Phase 1 heap. After a site visit, an optimized layout was developed to incorporate the Phase 1 heap into the Phase 2 expansion plan. This revised plan reduces the footprint of the Phase 2 pad by a remarkable 20% and significantly reduces material and construction costs for the expansion project. Unique elements for this heap leach facility also include locally borrowed liner bedding material, a high performance liner system, multi-phased flexible solution collection pipelines, and a filled and covered pregnant solution pond along with buried irrigation tubes on the heap.

Introduction

The CSH Gold Mine is a one of the largest gold mines in China. The low grade gold deposit is located in the Inner Mongolia Autonomous Region of Northern China; approximately 650 km north-west of Beijing and 126 km north-west of the city of Baotou (see Figure 1). It is a conventional open pit, heap leach, gold-mining operation, and commenced commercial production in 2007. The CSH Mine is currently processing 30,000 tonnes of ore per day and producing 133,000 ounces of gold per year.

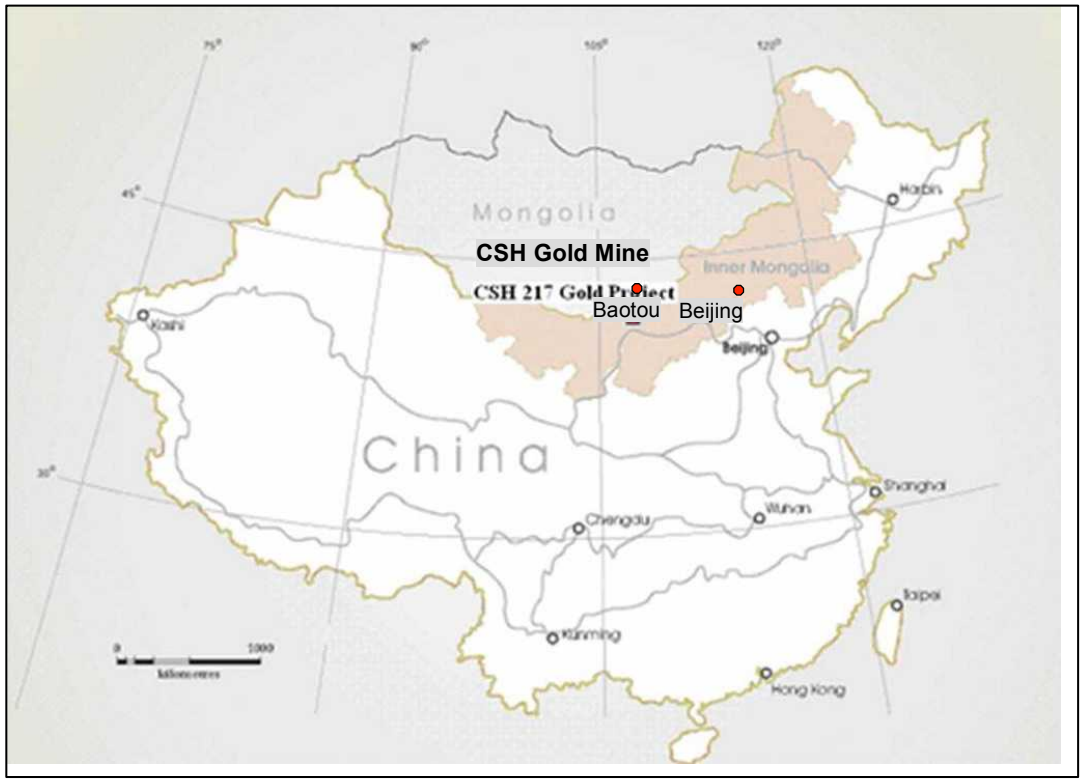

Figure 1: Location of the CSH Mine

The existing heap leach facility (Phase 1) was designed to accommodate 65 million tonnes of crushed ore and will reach design capacity in late 2013. A recently completed feasibility study indicates that the remaining open pit reserves at the CSH Mine stand at over 213 million tonnes of ore containing approximately 4.08 million ounces of gold.

The CSH Mine proposed an expansion project to increase the production rate to 60,000 tonnes per day and produce 260,000 ounces of gold per year starting in 2014. The current mine life of the CSH Mine is expected to be 11 years.

The expansion project includes a new heap leach facility (Phase 2) to handle 100 million tonnes of ore. Combined with the existing heap leach facility (65 million tonnes) and future expansion (Phase 3, potentially over 80 million tonnes), the CSH Mine will become the largest heap leach operation in China.

General site conditions

The CSH Mine is situated on the Inner Mongolian Plateau at an average elevation of 1,550 to 1,750 MASL. The site has a typical continental interior climate in semi-desert conditions. Annual rainfall averages 233.7 mm, which is far exceeded by the estimated evapotranspiration of 2,646.2 mm per year. Most of the rainfall (over 70%) occurs in the brief period between July and September. The summers are hot and the highest temperature is around 37°C. The winters are cold with cold spells down to −38°C. Winter conditions prevail from early October through mid-April, but snowfall is minimal. Strong winds blow at the mine site all year around with typical wind speeds ranging from 30 to 100 km per hour.

The mine site covers an area of gently rolling hills and undulating topography. A veneer of overburden soil covers bedrock. The overburden is comprised of brownish loose silt with a typical thickness ranging from 0 to 2.8 m. The bedrock is dominated by clastic sedimentary formations intercalated with carbonate rich rocks and includes quartz sandstone, greywacke, siltstone and shale with dolomite and limestone. The frost depth is up to 2 m. Groundwater exists in deep aquifers formed in fractured rock formation. The earthquake magnitude for the region is 7 and the design value of peak ground acceleration is 0.1 g.

Layout optimization

The existing heap leach pad (Phase 1 pad) is located at the south side of the deposit, as shown in Figure 2. Ore will be placed on the existing heap leach pad (Phase 1) and will reach full storage capacity in late 2013 at a maximum height of 90 m. The originally proposed second heap leach pad (Phase 2 pad, designed by others) covers a flat area to the east of the existing pad (see Figure 2). The footprint occupies a surface area of 133 ha and the pile will have a capacity of 75 million m^3 or 120 million tonnes at an average ore dry density of 1.6 tonnes per cubic meter. Once the Phase 2 pad reaches its final height of 90 m, the plan is to merge the two pads by placing the balance of ore in the valley between the two pads. The valley capacity is estimated at 40 million m^3 or 64 million tonnes.

The original heap leach pad expansion plan utilized the best (gentlest) topography and provided sufficient capacity for the mine's expansion. However, it occupied two drainage courses (or two catchment areas) and required two separate solution collection channels and two pregnant ponds. This resulted in additional operational complexity in the operation of two pregnant ponds with vertical wet wells. In addition, when the ore is placed in the valley between the two heaps, the Phase 1 pregnant solution pond and processing plant have to be continuously functioning, which would result in a higher operating cost. Furthermore, this plan limits the land use for any future expansion.

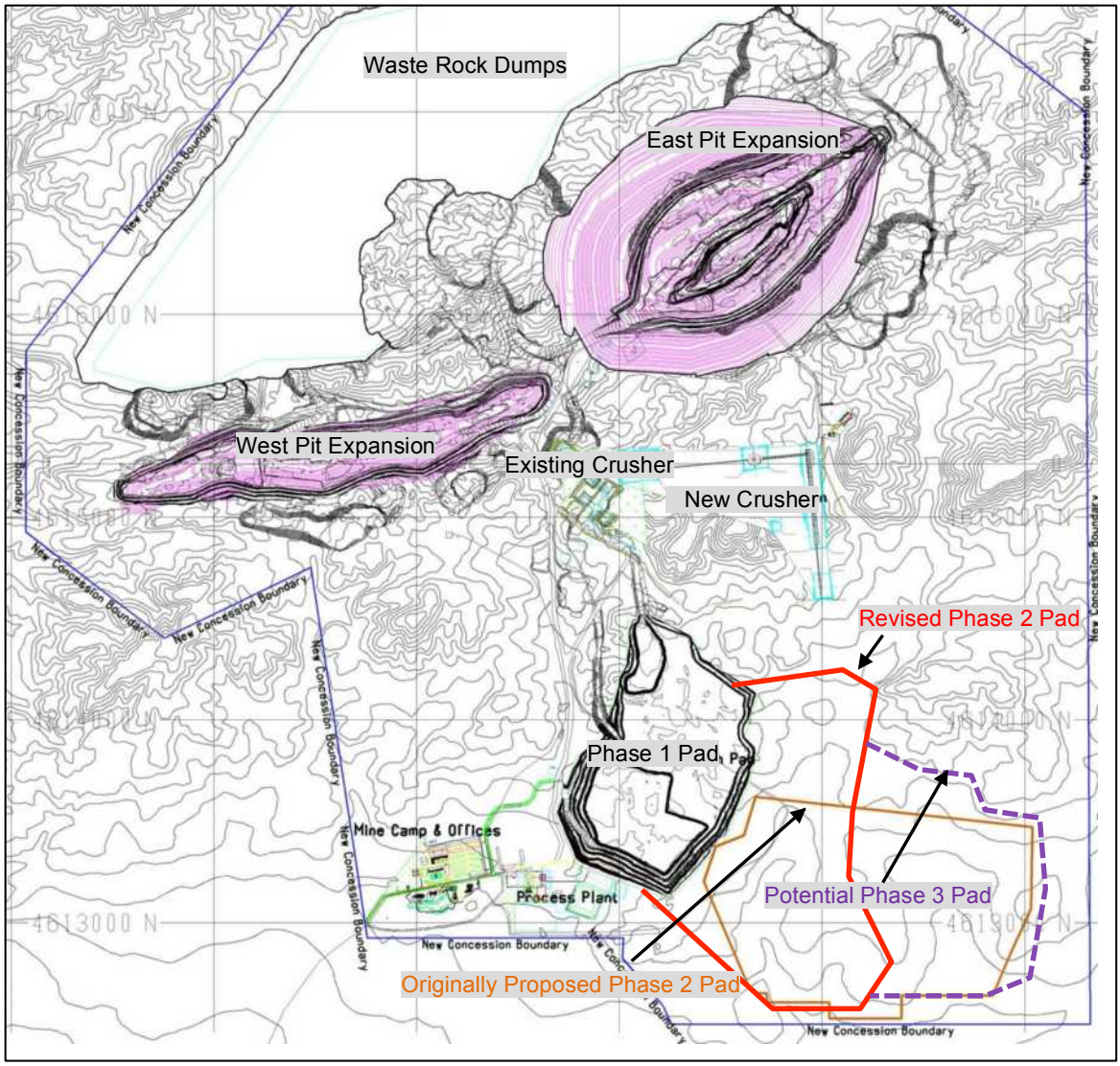

Figure 2: General layout of the CSH Mine expansion project

After the site visit, an optimized layout was developed to sub-divide the original expansion pad into two parts following the natural drainage divides. The revised Phase 2 pad utilizes one natural valley adjacent to the Phase 1 pad and merges the Phase 2 and Phase 1 leach pad areas to minimize slope areas and losses in stackable areas. The new pad footprint was expanded to the north, as shown in Figure 2. This optimization offers a better use of the land and will reduce both initial capital cost and operational costs for the expansion project. The advantages of this optimized plan include:

- maximizing the land use;
- directly incorporating the Phase 1 heap into the Phase 2 heap to limit slope areas;
- reducing the footprint of the Phase 2 pad by 20%, which will significantly cut the material and construction costs;

- requiring construction of one pregnant pond, not two;
- preserves land for future heap leach pad expansion (Phase 3).

The surface area of the revised Phase 2 heap leach pad is approximately 120 ha. The ultimate heap is designed to be 90 m high, similar to the Phase 1 heap. The total storage capacity for the Phase 2 pad is 65.2 million m^3, or 105 million tonnes. The future leap leach pad expansion (Phase 3) occupies an area of 84 ha, with a storage capacity of 80 to 100 million tonnes (see Figure 2).

Challenges in design

Large heap leach operations in China are not common. The general layout of the Phase 2 heap leach pad is illustrated in Figure 3. Challenges faced during the Phase 2 heap leach pad design stages include the selection of the liner bedding materials; design of the high performance liner system, multi-phased pipelines, a filled and covered pregnant solution pond, and strategic heap loading plans.

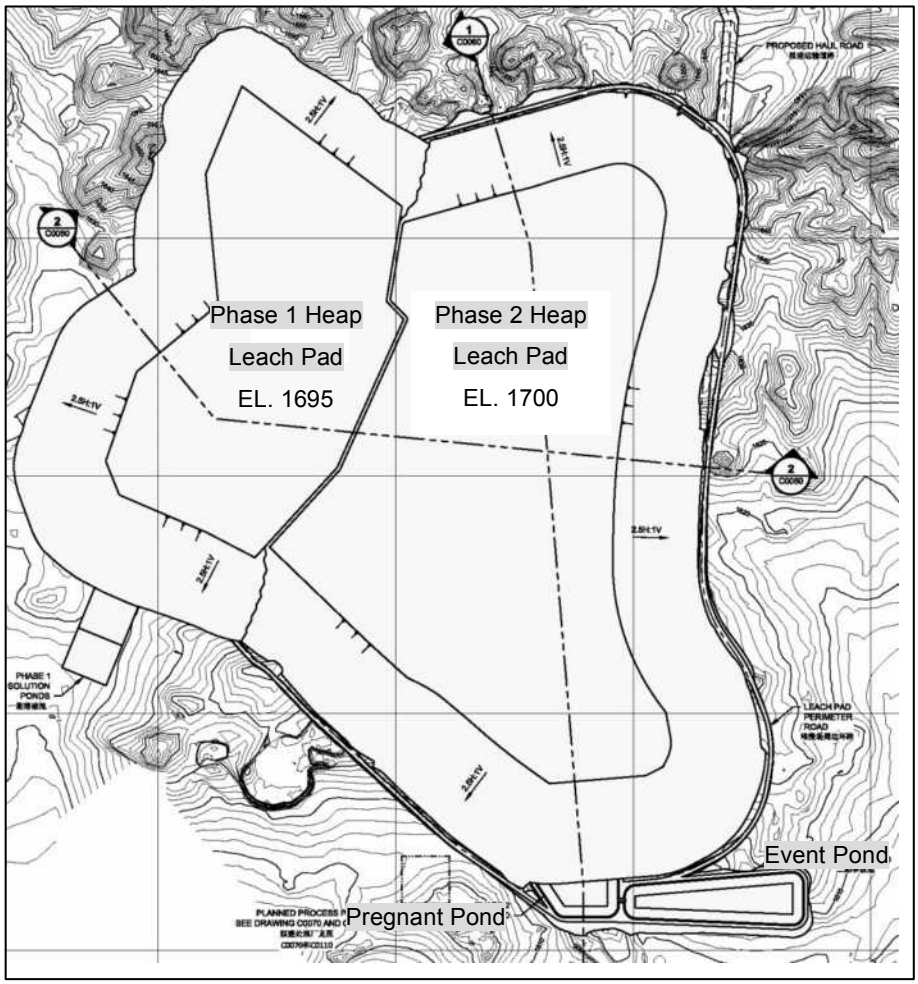

Figure 3: General layout of the Phase 2 heap leach pad

Liner bedding materials

The project is in a semi-desert region with strong winds all year round. The overburden layer is very thin with a maximum thickness of 3 m at the project site. The overburden materials are comprised of loose silt sand and/or sandy silt with a low natural moisture content (less than 3%). It is a challenge to find suitable low permeable material (clayey silts) and sufficient water sources for proper moisture conditioning of the liner bedding material. Geomembrane Clay Liner (GCL) is more expensive. It has lower interface friction angles, and the clay may dry out during construction in this arid climate.

After detailed site reconnaissance, suitable bedding materials (in terms of particle size, permeability, compatibility and strength) were found along some ancient river channels located 5 km east of the mine site. A water supply source was also identified at an upstream reservoir. Therefore, a 300 mm thick, conventional low permeability fine grained compacted soil subgrade layer was designed for the Phase 2 heap leach pad. Low permeability materials used for liner bedding consist of fine-grained soils with minimum amounts of large angular particles. The specified liner bedding materials will be well graded with not less than 25% by weight passing the No. 200 sieve. The upper 100 mm of compacted soil liner will have a maximum particle size of 75 mm. The material should have a permeability of less than 1×10^{-6} cm/s and conform to the gradation limits shown in Table 1.

Table 1: Gradation of liner bedding materials

Liner bedding materials	Percent passing	
Sieve size	Minimum	Maximum
150 mm	100	100
75 mm	90	100
No. 4 (4.75 mm)	50	100
No. 40 (0.425 mm)	35	85
No. 200 (0.075 mm)	25	70

All placed liner bedding material will be compacted to 95% of the Standard Proctor maximum dry density and moisture conditioned to a maximum of 2% above the optimum moisture content.

Geomembrane liner selection

High density polyethylene (HDPE) geomembrane was selected for use in the project. The liner design considered thickness requirement, tensile behavior, seam behavior, tear resistance, impact resistance, puncture resistance, stress cracking and geomembrane friction. 80 ml (2 mm thick) HDPE geomembranes, smooth and textured, were selected for the Phase 2 heap leach expansion project. The HDPE geomembrane thickness was selected based on experience in similar climates with freezing temperature and high winds. Under strong windy conditions, the selected geomembranes have great

engineering properties from high elasticity and overall mechanical resistance under tensile, tear and puncture modes. The key specifications for smooth geomembranes are listed in Table 2.

Table 2: Smooth geomembrane specifications

Property	Value
Thickness	
Average	2.0 mm
Minimum	1.8 mm
Density (minimum)	0.94 g/cc
Tensile properties (minimum)	
Yield stress	29 N/mm
Break stress	53 N/mm
Yield elongation	12%
Break elongation	700%
Tear resistance (minimum)	249 N
Puncture resistance (minimum)	640 N
Stress crack resistance	400 hours
Carbon black content	2.0-3.0%
Oxidative Induction Time (OIT)	
Standard OIT, minimum	>100 minutes
High pressure OIT, minimum	400 minutes
Oven aging at 85°C	
Standard OIT, minimum	55% retained after 90 days
High Pressure OIT, minimum	80% retained after 90 days

Liner system design

A composite single geomembrane liner system with a compacted low permeability soil subgrade has been chosen for the pad and event pond, while a double liner system with leak detection was designed for the pregnant solution pond. Smooth geomembranes will be used for the central pad, event pond, and the first (lower) liner for the pregnant pond, while textured geomembranes will be used for the sloped pad areas of the pad and the second (upper) liner for the pregnant pond. Geonet is used on the slopes of the pregnant pond and a leak detection granular fill is used between the liner and the floor of the pregnant pond.

Five geomembrane manufacturers, including domestic and foreign, were inspected prior to the liner tenders were issued to pre-qualify the HDPE manufacturers. In-plant quality control is the biggest challenge for Chinese manufacturers even though they have advantages on prices and delivery schedule. In order to ensure high quality, an internationally recognized imported geosynthetic brand was selected for the CSH Mine Phase 2 heap leach facility. The geosynthetics manufacturer was ISO 9000 certified, and the laboratory is certified by GAI/LAP for the tests being performed, and has a third party in-plant independent quality assurance program.

Solution collection piping

The Phase 1 heap leach facility utilized 610 diameter slotted HDPE DR11 pipes for the main solution collection piping. The HDPE pipes used were reliable, but were found to be heavy, less flexible and expensive. It was reported that there were some difficulties during pipework installation on the Phase 1 geomembrane liner surface. The Phase 2 design selected 600 mm diameter corrugated polyethylene tubing (CPT) pipeworks as the main solution collection pipes. In addition to main solution collection pipes, 300 mm and 100 mm diameter perforated CPT pipes were also designed to feed the main solution collection piping system.

The corrugated pipes have been widely used in many heap leach pad projects in the Americas, but have not been commonly used for large heap leach projects in China. The double layered corrugated perforated pipes are lighter, more flexible, and less expensive compared to the Phase 1 pipes. It will be easier for pipework installation as the pipe routes can be adjusted in accordance with the topography in the pad. The pipeworks specifications are listed in Table 3.

Table 3: Corrugated polyethylene pipe specifications

Property	Value
Density	0.941 – 0.955 g/cc
Melt index	< 0.15g/10min
Flexural modulus, 2% secant	760 – 1100 MPa
Tensile strength at yield	145 to 165 kPai
Slow crack growth resistance	100 hrs (min)
Hydrostatic strength classification	11 MPa
Color and UV stabilizer	Black with 2% minimum carbon black
Pipe stiffness (100 and 300 mm diameter)	345 kPa
Pipe Stiffness (600 mm diameter)	235 kPa

The corrugated polyethylene pipes in China are among the low-end quality products and there is no specific industrial standard for heap leach pipeworks. Obtaining quality corrugated pipeworks and fittings in China appeared to be the greatest challenge to the project.

Filled pregnant solution pond

Despite cold temperatures in the winter, the CSH Mine operates the heap leach facility year round. The Phase 1 heap leach operations at the CSH Mine utilized the techniques of buried drip emitters on the heap and a covered pregnant solution pond to maintain solution temperatures in freezing conditions and to minimize evaporation losses. It was the first time that this technique was used in China and the mine

operators are pleased with the operational success to date. This concept was specified for the Phase 2 heap leach facility design. The pregnant pond will be completely filled and covered by gravel (overliner materials) and ore as illustrated in Figure 4.

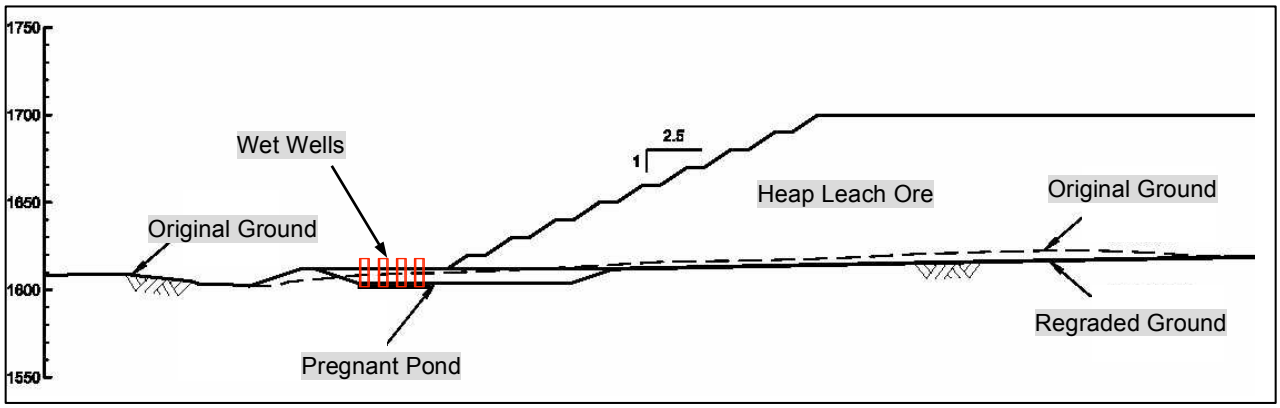

Figure 4: Cross section of pregnant solution pond and heap

The Phase 2 pregnant pond was designed to have a pore volume storage capacity of 56,000 m^3 and the pond has an irregular shape to accommodate and take advantage of the natural topography in this area. The filled and covered pregnant pond will effectively prevent evaporation in the solution pond. Four solution pumps (three operating and one on standby) were designed and each of them has a design pumping rate of 1,000 m^3/hour. A lowered solution pumping platform was designed to minimize the dead storage of the solution pond. It will be the first time that a pregnant solution pond is to be constructed with an irregular shape and a sloping pond floor in China.

Challenges during construction

The Phase 2 heap leach pad construction started in April 2013 and is currently ongoing. Numerous challenges have been encountered during the construction including material/water shortages, severe weather conditions, uncertainty of liner welding/pipe quality, and unyielding construction schedule. Site specific quality assurance and quality control (QA/QC) processes were implemented to ensure the completion of Phase 2 heap leach facility met North American standards of quality.

Earthwork QA/QC

Earthwork of the Phase 2 heap leach pad project includes topsoil stripping, blasting and excavation of rock outcrop, pad rough grading, placement and compaction of structural fills, liner bedding, leak detection fill, sand bedding, drain gravel, and overliner materials. The earthwork contractor has local experience but appears to be inexperienced in construction management or quality control. A field soil laboratory was established specifically for the Phase 2 heap leach earthwork construction. The earthwork

QA/QC has focused very hard on the quality and testing of the 300 mm thick compacted liner bedding material.

Silty clay and clayey silt samples collected from borrow sources are tested in the field laboratory to examine the gradation and compaction properties of liner bedding materials. Suitability of the bedding materials, optimum water contents, and maximum dry density of compacted materials were determined prior to the placement of bedding. Field quality control tests are conducted to ascertain that the fill being placed or in-place materials meet the technical requirements. All laboratory and field testing are performed in accordance with the principles and methods prescribed by the American Society for Testing and Materials (ASTM) standards. The field testing frequency for the liner bedding materials is listed in Table 4. A Chinese-made electronic density gauge (EDG) was initially used for field density testing but was found to be unreliable. The sand cone method has been the most effective method for field density checks at this site.

Table 4: Testing frequency of liner bedding materials

Type of test	Frequency (1 per)
Atterberg limits	5,000 m^3
Moisture content	5,000 m^3
Particle size analyses	5,000 m^3
Moisture-density relationship	10,000 m^3
Field density – sand cone method	5,000 m^3

The site is very dry in the summer months and water is in short supply throughout the moisture conditioning process. Compaction is performed as quickly as possible to preserve the designed moisture content. The surfaced moisture of the compacted liner bedding evaporates quickly. Strong wind often blows away the fine grained materials leaving coarse material on the ground that requires re-compaction. Figure 5 shows a sand storm during construction.

Even though the weather is dry most of the time, severe thunder storms occur during the summer months. Intensive rainfall washed out the bedding layer many times, particularly along the main solution collection channel where flows are concentrated, as shown in Figure 6. In some areas, the bedding materials were re-placed and re-compacted seven times before the deployment of the geomembrane liner. Development of the panel deployment plan to minimize subgrade damage from precipitation has been one of the challenges on site.

Figure 5: Sand storm on site

Figure 6: Destroyed liner bedding due to heavy rainfall

Geomembrane liner QA/QC

The HPDE geomembrane liner contractor is a specialized installer in China, equipped with world-class welding machines and skillful workers. However, they have less experience on heap leach projects. The geomembrane liner QA/QC has focused on visual inspection and a field installation testing program to examine the continuity, integrity and strength of the seams, similar to projects carried out in the Americas. The field program includes:

- Visual inspection to check field seams for squeeze out, foot print, melt and overlap, to check machines for cleanness, temperature and related items, and to mark and repair defects.

- Continuity testing for all field seams and repaired areas, which includes inter-seam pressure or "air testing" and testing using a vacuum box.
- Strength testing on trial welds.

The main challenges to geomembrane liner installation on site are once again the weather conditions. The contractor has to work in the very early mornings and late afternoons to avoid gusty wind and hot temperatures. Strong winds once blew an entire sheet of geomembrane liner up to 5 m high and hundreds of meters away. Numerous sand bags and temporary anchors were applied throughout the liner installation process. Rain and storm water also affected the welding and installation of geomembrane liner. Figure 7 shows the installation of the smooth geomembrane liner at the west area of the Phase 2 heap leach pad.

Figure 7: Smooth geomembrane liner installation

Pipework QA/QC

This is probably the first time that large diameter corrugated polyethylene pipes have been used as a solution collection system for large heap leach operations in China. The pipe material components and stiffness did not meet the specifications in the beginning. The suppliers were forced to modify the manufacturing process to produce suitable pipes. The fittings and connection parts did not meet specifications either and were returned for re-manufacturing.

The pipework QA/QC has focused on third party strength testing of the pipes and connections. It is still a big challenge to prove the quality of the pipework and fittings in China as the testing methods and parameters are different from those in North America. Chinese corrugated polyethylene pipework products are a long way from meeting international standards. Figure 8 shows the 600 mm diameter pipes, and the connection part to the 300 mm diameter pipe is questionable due to the use of different raw material. The domestic pipes were used for the Phase 2 project due to an aggressive schedule. An enhanced third party testing program was applied for the solution collection pipes and connections.

Figure 8: Solution collection pipes

Conclusions

Numerous challenges were encountered during the design and construction of the CSH Mine Phase 2 heap leach facility. The challenges included complex topography, severe weather conditions, shortage of water and subgrade materials, uncertainty of liner/pipe quality, as well as an aggressive project schedule. These challenges have been overcome through optimized layout planning, utilization of local materials, selection and design of a high performance liner system, multi-phased flexible solution collection pipelines, and a filled and covered solution pond. A detailed and unyielding field QA/QC program is prudent to ensure the quality of large heap leach project construction in an inexperienced region.

Acknowledgements

The authors would like to thank Mr. Lisheng Zhang, Mr. Fenglong Wang and Mr. Kevin Wang of Inner Mongolia Pacific Mining Co. Ltd., a subsidiary of China Gold International Resources Corp Ltd., for providing background information, historic data, and valuable insights for the Phase 2 heap leach pad design. This paper only presents a portion of the entire expansion project and the heap leach pad construction is still ongoing as we write. The permission of China Gold International to publish the details of the heap leach pad design and QA/QC is gratefully acknowledged.

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Model analysis of differential settlement for a heap leach constructed on a mine waste storage facility

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Abstract

Existing mining facilities typically have fixed property boundaries, permit limits, or economic considerations that constrain the mining infrastructure to a limited area. Expansion projects at these mines often must fit into these fixed areas and compete with the space requirements of ongoing activities. An alternative for a facility expansion being developed at a copper and gold mine in South America is to locate a new gold heap leach on top of an existing waste storage facility. Because the waste facility was not originally constructed as a heap leach foundation, a concern is that the various types and depth of materials in the waste facility will have different settlement responses. Differential settlement beneath the leach pad could lead to (1) tears in the leach pad liner due to excessive liner strains and (2) poor drainage of pregnant leach solution due to reversal of gradient in the collection pipes. In this paper a series of possible differential settlement scenarios are simulated using the finite difference modeling software, FLAC. The location and extent of the differential settlement is what differentiates between these scenarios. The total differential settlement and the stresses and strains developed in the liner are calculated for each scenario. The modeling demonstrates strategies to mitigate the risk of differential settlement beneath the heap leach.

Introduction

Recent advancements in mining allow for mining resources today that were not possible to mine or process before. We now see mines and facilities located on mountains or in difficult terrains constrained by both property boundaries and topography. Expansion projects at these mines can be challenging due to the space restrictions.

An example is a mine in South America that needs to process a low grade gold ore stockpile through heap leaching. Due to space limitations, and to minimize the environmental disturbance, one of the few

feasible options is to locate the leach pad on an existing waste storage facility (WSF). Since the waste facility was not originally constructed as a compacted foundation, a concern is that variation in types and depths of materials in the waste facility may cause different settlement responses. Differential settlement beneath the leach pad could lead to (1) tears in the leach pad liner due to excessive liner strains and (2) poor drainage of pregnant leach solution due to reversal of gradient in the collection pipes.

The site is surrounded by sensitive environmental and social areas such as off-site rivers and streams that form headwaters to drinking water supplies for downstream communities. Discharges into these rivers and streams are controlled through strict environmental regulations, and the mine's social license to operate requires protecting these water sources. Karst voids, which exist in some areas of the site, are known to sometimes provide links to rivers, streams and groundwater, so potential leakage of the cyanide heap leachate must be eliminated.

As a preliminary risk assessment, the authors evaluated potential differential settlement scenarios as a result of constructing a heap leach facility on the existing waste dump and the associated strains on the geomembrane liner. Our approach for this evaluation was to first develop simplified differential settlement scenarios that could potentially create excessive strains on the liner. We then used a finite-difference numerical model to simulate the differential settlement, under various mitigation strategies, and predict the liner strains. Based on the results from these analyses, the initial assumptions can now be modified to improve the liner design and optimize the costs.

Heap leach pad foundation design

Figure 1 shows the plan view of the WSF on which the heap leach will be constructed. The foundation of the leach pad will include the following components (from bottom to top):

- bedrock;
- up to 50 m of uncompacted and variable waste;
- 5 m compacted waste;
- 0.6 m underliner material;
- Geosynthetic Clay Liner (GCL);
- 80 mil linear low density polyethylene (LLDPE) geosynthetic liner;
- 0.6 m overliner material; and
- 8 m on-off heap.

The purpose of the compacted waste is to provide a more rigid base to the leach pad and minimize, and/ or bridge, settlement. The overliner and underliner are to protect the liner from tears caused by sharp

rocks or vehicle traffic. The geomembrane liner prevents leakage of process solution to the environment. The GCL provides a secondary layer of protection against leakage.

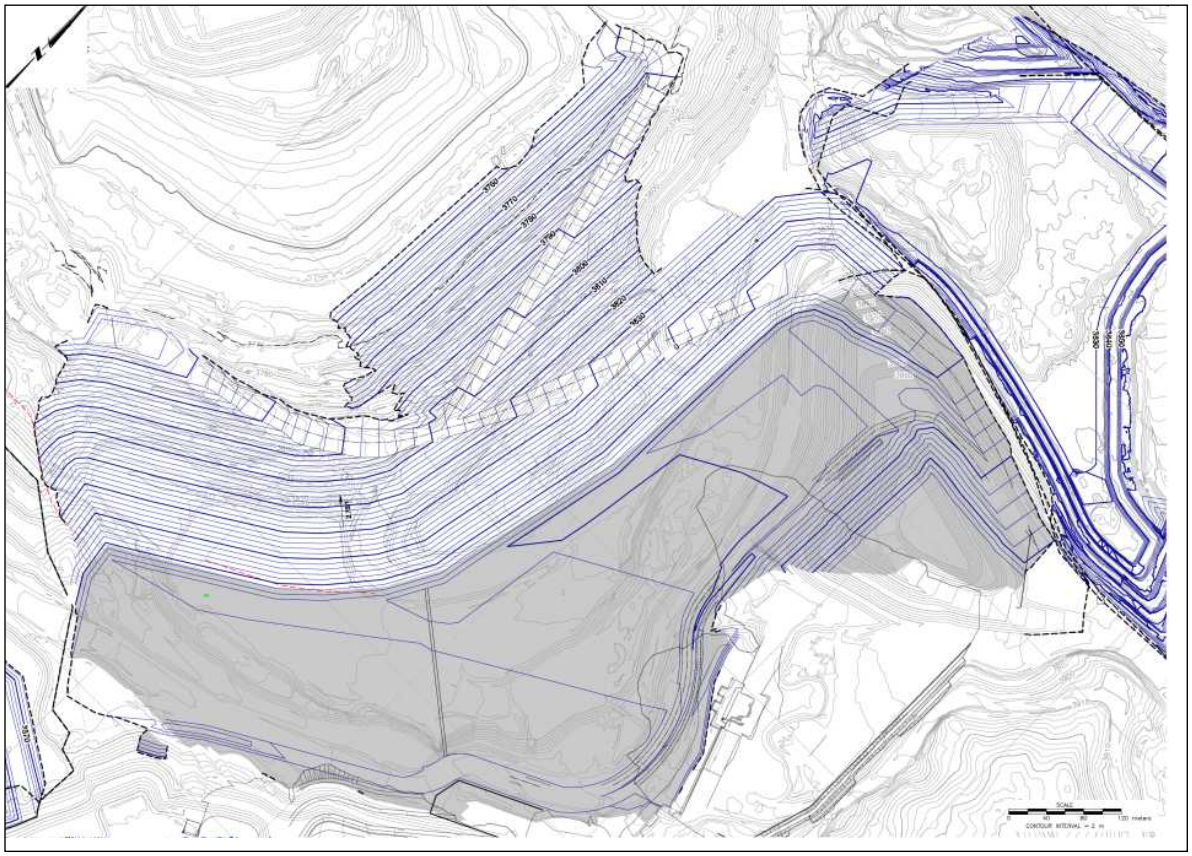

Figure 1: Layout of the Waste Storage Facility (WSF) as foundation of the heap leach pad

Differential settlement scenarios

The impact of differential settlement on the geomembrane liner is sensitive to the length over which the differential settlement occurs. For example, differential settlement occurring abruptly over a short length will induce higher strains in a liner than the same amount of settlement over a longer length. Predicting the length over which differential settlement will occur can be difficult in a waste storage facility where the dumping sequence and variability of the waste properties is not well known. A detailed characterization of the waste storage facility was not practical for our evaluation; therefore we were limited to a series of SPT tests at random locations from which to develop our conceptual understanding of the distribution of material properties in the waste storage facility. Using this data, we developed several simplified scenarios of material distributions under which differential settlement would be most likely to occur. Our intent in developing these scenarios was to consider extreme cases that would give an upper bound on potential differential settlement.

We developed three scenarios. The basic assumption for each scenario was that somewhere within the waste dump there is a 10 m wide prism of soft waste material surrounded by less compressible waste material. The depth of the waste dump in the area of the soft waste material was assumed to be 50 m, which is approximately the maximum thickness of the waste that exists underneath the proposed heap leach pad. Although the exact dimensions of the area over which differential settlement may occur are difficult to predict, these assumed dimensions are more than likely conservative, based on our knowledge of the proportion of soft material in the waste facility. Furthermore, the waste material is unlikely to abruptly change in compressibility as the waste is dumped in 1 to 2 m lifts; and having 10 m of soft material with a depth of 50 m requires that a 10 m wide section of soft material be placed directly over previously placed soft material for at least 25 consecutive lifts, which is highly unlikely.

Figure 2 shows the three possible differential settlement scenarios evaluated in this study. The primary difference between these scenarios is the location and orientation of the soft (compressible) material and the existence of a layer to mitigate differential settlement. This layer was assumed to be a cap of compacted waste directly beneath the pad, or the pad overliner and underliner.

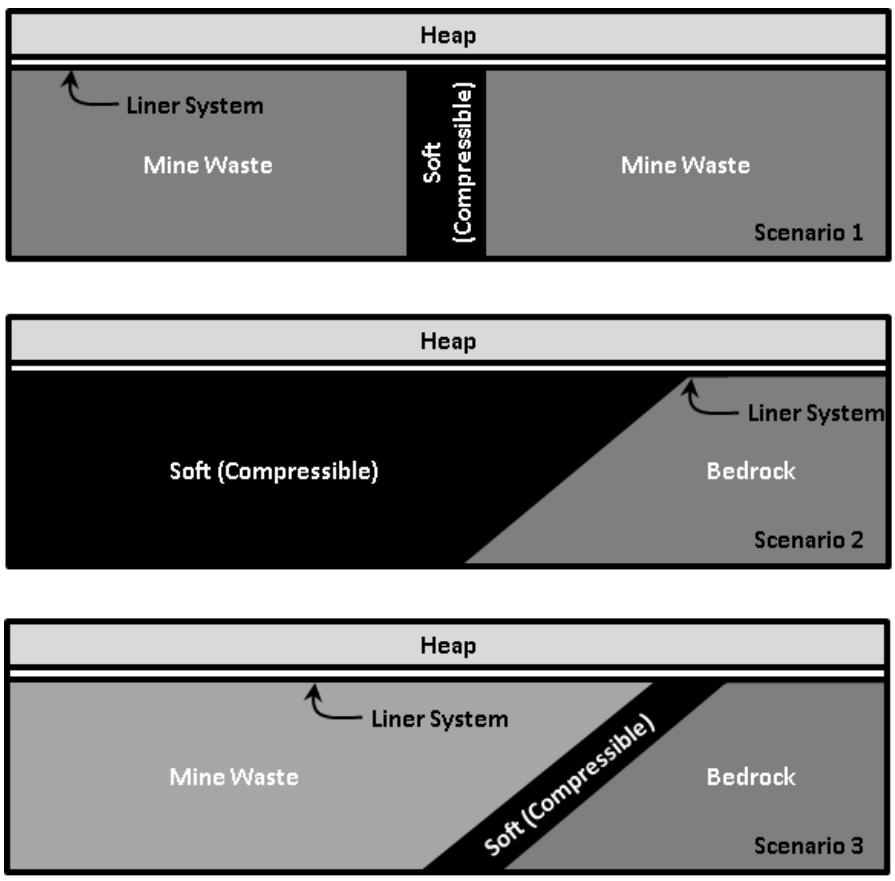

Figure 2: Different scenarios evaluated for differential settlement analysis

Details of these scenarios are discussed below:

(1a) Leach heap located over a 50 m deep and 10 m wide column of highly compressible (soft) material surrounded by less compressible waste material and with no compacted waste cap and no overliner/underliner.

(1b) Leach heap located over a 50 m deep and 10 m wide column of highly compressible (soft) material, surrounded by less compressible waste material and with a 0.6 m thick overliner, a 0.6 m thick underliner and a 5 m thick cap of compacted waste.

(2a) Leach heap located partially over bedrock and partially over highly compressible (soft) material and with no compacted waste cap, and no overliner or underliner. The bedrock-waste interface is sloped so the thickness of the soft waste material varies throughout the leach heap from 0 to 50 m.

(2b) Leach heap located partially over the bedrock and partially over highly compressible (soft) material and with a 5 m thick cap of compacted waste, a 0.6 m thick overliner and a 0.6 m thick underliner. The bedrock-waste interface is sloped so the thickness of the soft waste material varies throughout the leach heap from 0 to 50 m.

(3) Leach heap located over a 10 m wide strip of highly compressible (soft) material bounded by the bedrock on one side and less compressible waste on the other side with no compacted waste cap, and no overliner or underliner.

Numerical modeling of the differential settlement

The differential settlement of the heap was simulated using a FLAC 2D finite difference model. A sensitivity analysis was performed to determine the appropriate mesh size for each scenario. The bottom boundary of the model was fixed in both X (horizontal) and Y (vertical) directions and both sides of the model were fixed in the X direction and free in the Y direction to allow deformation.

For Scenarios 2 and 3, the interface between the bedrock and the compressible material was modeled with consideration for the possibility of sliding of the compressible material relative to the bedrock.

The physical properties and strength parameters employed to model the strong and soft waste, heap, and bedrock material are shown in Table 1.

Table 1: Material properties used for the differential settlement FLAC modeling

Material	ρ (KN/m3)	ρ (kg/m3)	Gmax (shear modulus in MPa)	K (bulk modulus in MPa)	Model	φ	c (kPa)
Bedrock	23	2,345	2344	5,080	Elastic	–	–
Heap leach	15	1,529	120	559	Mohr/Columb	24	9.5
Less compressible waste	20	2,039	279	837	Mohr/Columb	27	9.5
Soft waste (highly compressible)	20	2,000	0.3	0.35	Mohr/Columb	27	9.5
Overliner/underliner and compacted waste	20	2,039	279	837	Mohr/Columb	27	9.5

The parameters for the soft waste material were set by "calibrating" Scenario 1 so that it had 80 cm of settlement after placing the 8 m heap overburden on top of the liner. The 80 cm settlement corresponds to the maximum expected consolidation of a 50 m deep column of the waste material, due to the additional load exerted by placing the heap and a 5 m layer of compacted waste on top of it. This expectation is based on a consolidation calculation in accordance with one-dimensional consolidation theory. The properties used to calculate settlement are estimated from the weakest standard penetration test (SPT) blow count values of the WSF material.

One case of concern for differential settlement is when the soft material is placed next to bedrock: the waste may slide against the bedrock as it settles, causing stress concentrations in the liner located on top of the waste/bedrock interface. This case was modeled in FLAC as Scenarios 2 and 3 to evaluate the potential displacements as well as the strains developed in the liner associated with this case. In Scenario 2, the entire mass of waste material was assumed to consist of soft waste. This scenario is highly unlikely because the properties of the soft material are calibrated based on the weakest material found on the previous SPT exploration of the WSF and is not representative of the entire mass of waste material. In Scenario 3, a 10 m wide strip of soft material is assumed to exist next to the bedrock. The properties of the soft material developed in Scenario 1 were used to represent the compressible material for Scenarios 2 and 3. The boundaries selected for modeling of all three scenarios is shown schematically in Figure 3.

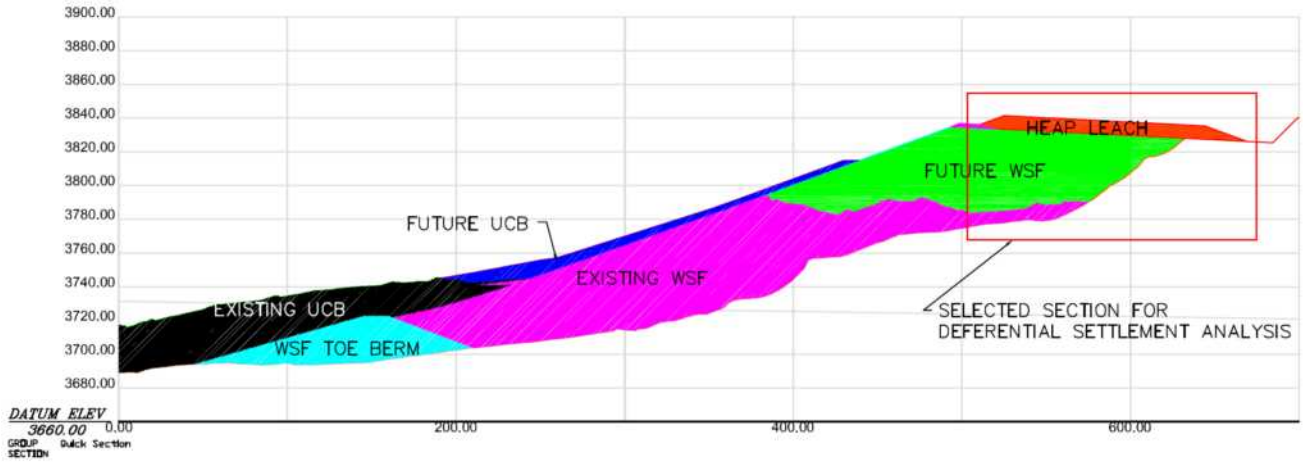

Figure 3: The model boundary selected for the differential settlement analysis

Modeling results

The calculated maximum displacements and strains in the liner for each scenario are shown in Table 2. Results for each scenario are discussed in the following subsections.

Table 2: Summary of the maximum displacements and strains in the liner for different scenarios

Scenario	Max. liner settlement (cm)	Max. liner strain (%)
1a	81.7	11.15
1b	0.47	0.02
2a	5.59	19.48
2b	4.95	7.06
3	8.17	14.33

Scenario 1

In Scenario 1a, the heap leach is modeled with 50 m of waste below the liner and an 8 m heap on top. As shown in Figure 4, a 10 m wide region of soft material with high compressibility is assumed in the middle of the pad in this scenario. The engineering properties of this material were adjusted such that, under the load of heap, the liner settles 80 cm, which is the expected total settlement due to the consolidation of a 50 m deep column of waste material. A sensitivity analysis was performed to determine the appropriate mesh size. In this process, the mesh size for the region of interest (in the middle of the model) was decreased until the results (e.g. liner strain) became independent of the mesh size. Figures 5 through 7 show the results for Scenario 1a (the calibration case). The maximum tensile strain developed in the liner due to the differential settlement for Scenario 1a is 11.15 percent (%).

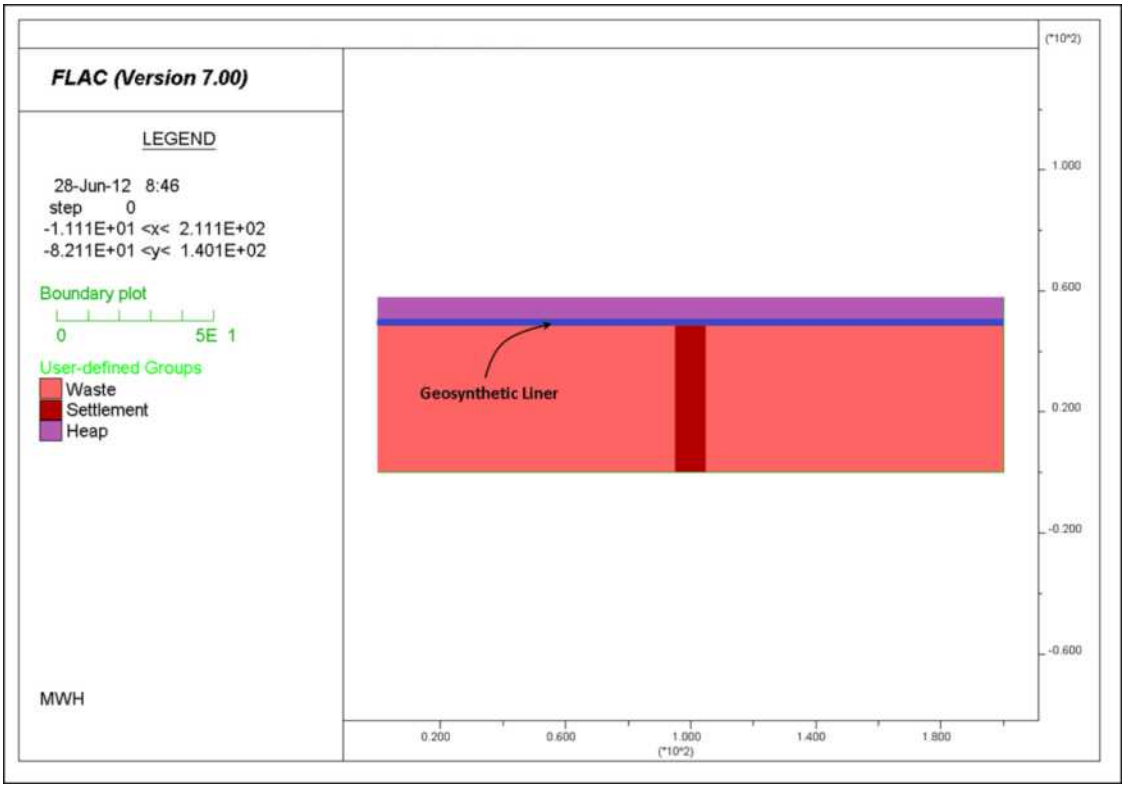

Figure 4: General view of the calibration case model (Scenario 1a) used for the differential settlement analysis

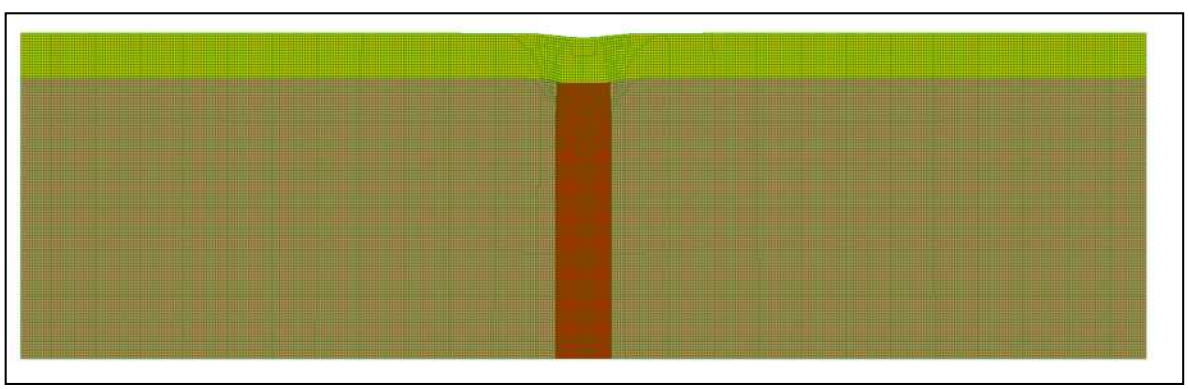

Figure 5: Deformed mesh after placing the 8 m heap for the calibration case (Scenario 1a)

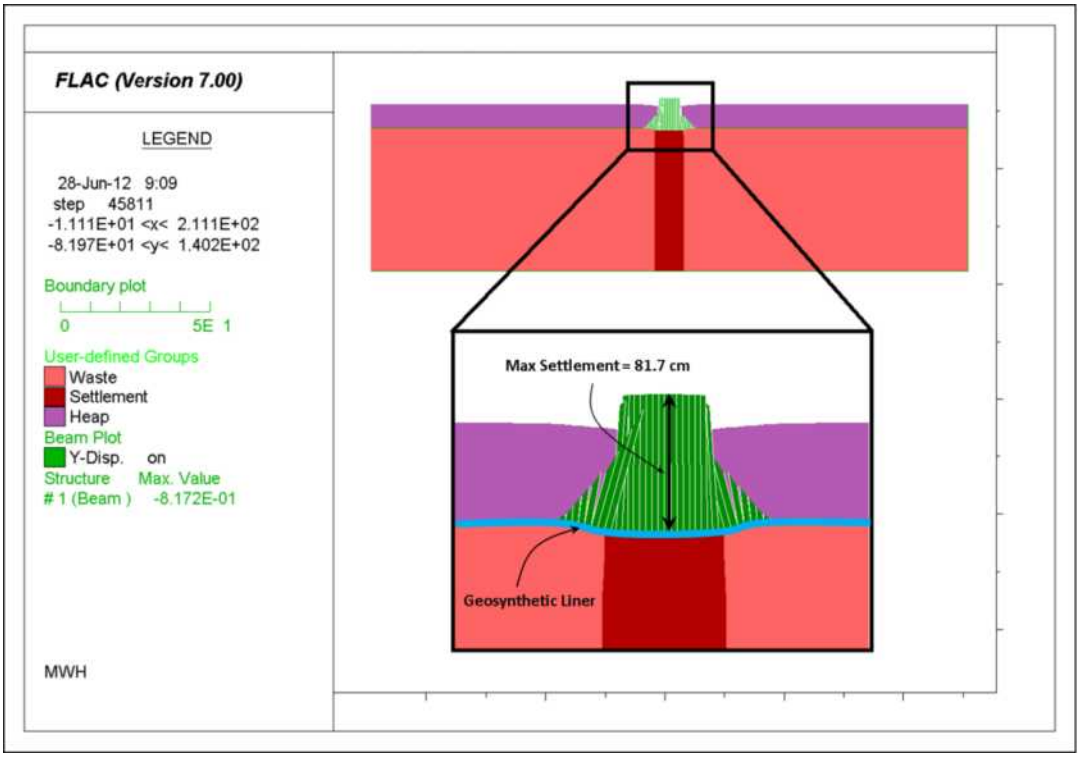

Figure 6: The maximum liner settlement for Scenario 1a

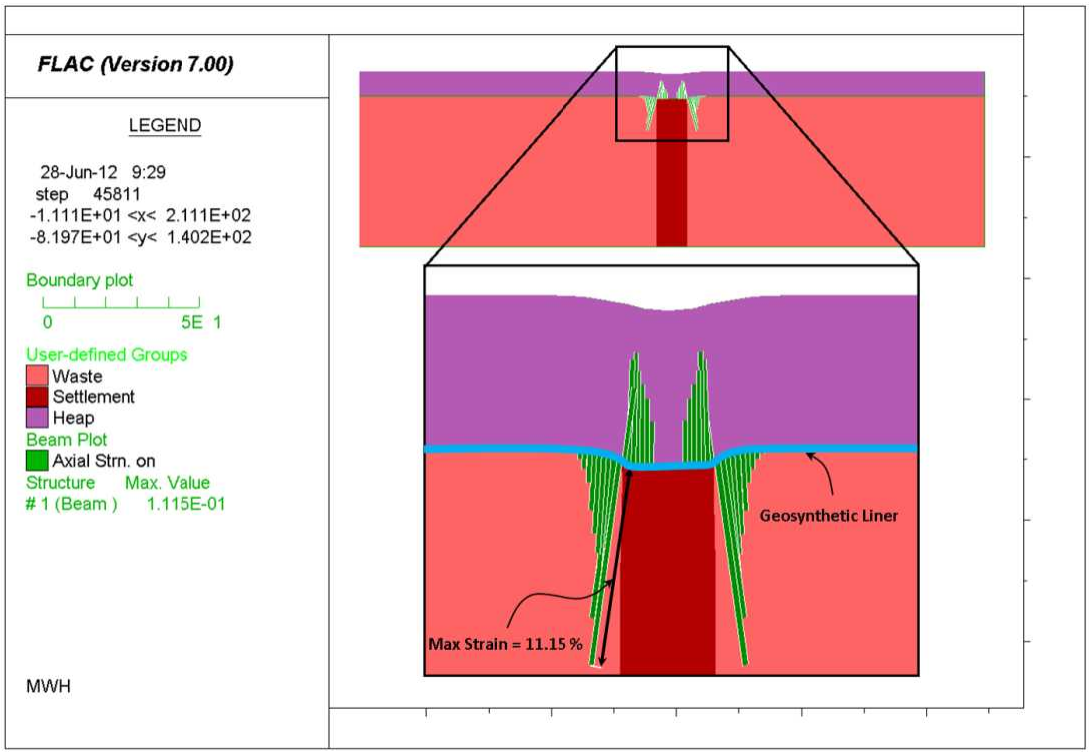

Figure 7: The maximum liner strain for Scenario 1a

The maximum liner tensile strain calculated by the FLAC model is higher than the maximum allowable strain for an LLDPE geomembrane liner for static loads, which the authors estimated to be 7%. However, Scenario 1a does not include the overliner or underliner or the 5 m compacted waste layer. The presence of these materials is expected to reduce the liner strains by providing reinforcement. The effect of inclusion of the overliner, underliner and the compacted waste layer is modeled in Scenario 1b.

Scenario 1b demonstrates that the overliner, underliner and compacted waste will significantly reduce the liner strains by supporting the liner and reducing the liner deformations associated with the differential settlement of a 10-m wide compressible waste column. The maximum liner tensile strain for Scenario 1b was determined to be 0.02%. This result suggests that in Scenario 1b the 5 m compacted waste, 0.6 m underliner and 0.6 m overliner support the liner by spreading the soft waste deformation over a wider area, hence reducing liner strain.

We performed a series of calculations to evaluate the potential for shear failure of the 5 m compacted waste, and the results showed that the stresses induced in the compacted layer by settlement of the underlying waste do not exceed the compacted soil shear strength, which is consistent with the reported FLAC modeling results.

Scenario 2

Scenario 2 simulates the conservative situation when all the waste consists of soft material; whereas, Scenario 3 assumes that only a 10 m band of waste adjacent to the bedrock is made of soft material. The maximum differential settlement for Scenario 2 occurs near the waste-bedrock interface.

Figure 8 provides the general geometry of Scenario 2a. Figure 9 provides the deformed mesh and Figure 10 illustrates the strain distribution of the liner for Scenario 1a. Similar to Scenario 1, once the 5 m compacted waste cap, 0.6 m overliner and 0.6 m underliner layers were added, Scenario 2b, the maximum liner strain was decreased from 19.5% to 7%.

Although this strain may seem marginal in comparison to the acceptable liner strain, it was considered acceptable due to the very conservative assumptions embedded into Scenario 2, including the majority of the heap leach foundation assumed to be compressible.

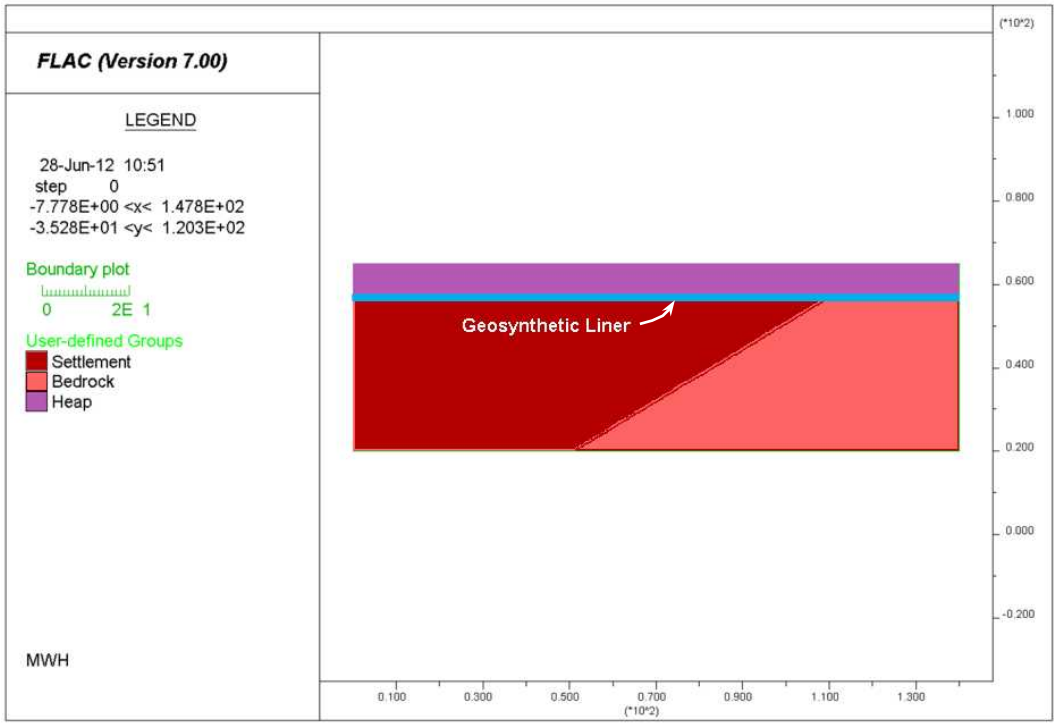

Figure 8: General view of Scenario 2a used for differential settlement analysis

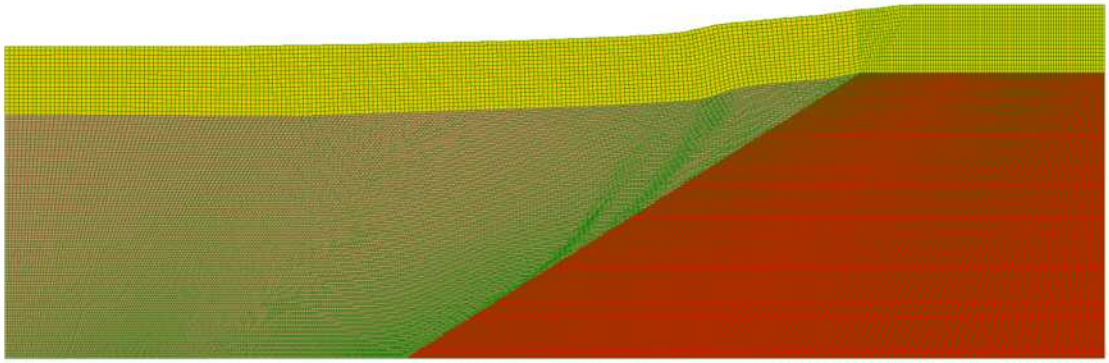

Figure 9: Deformed mesh after placing the 8 m heap for Scenario 2a

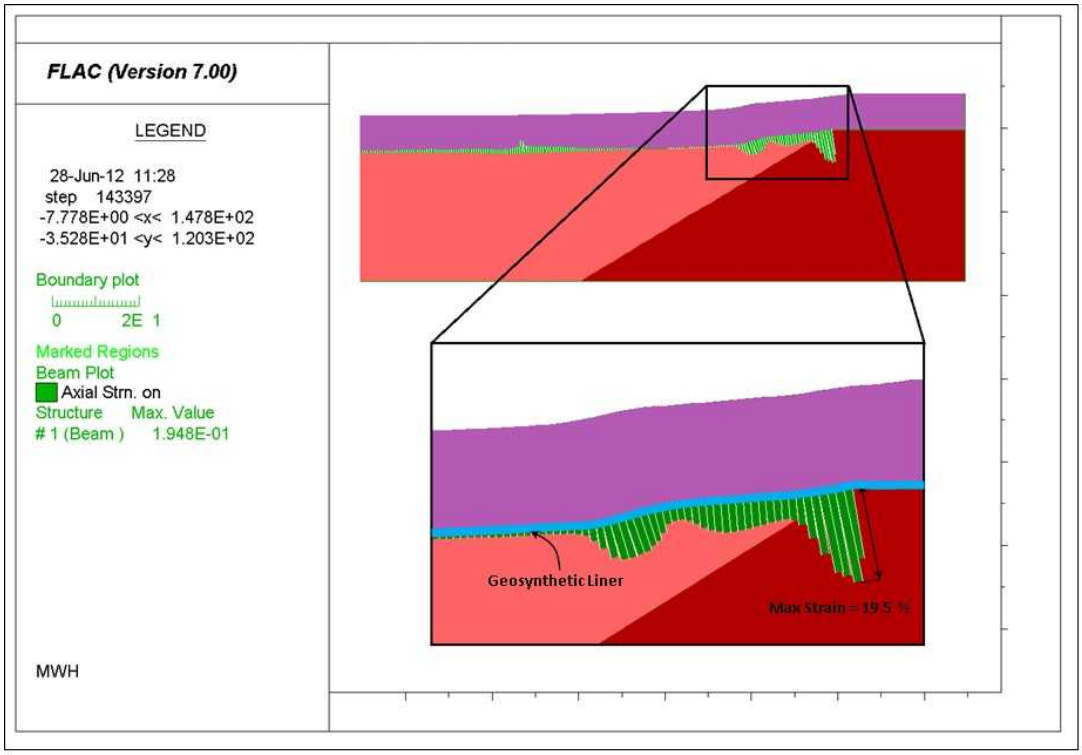

Figure 10: The maximum liner strain for Scenario 2a

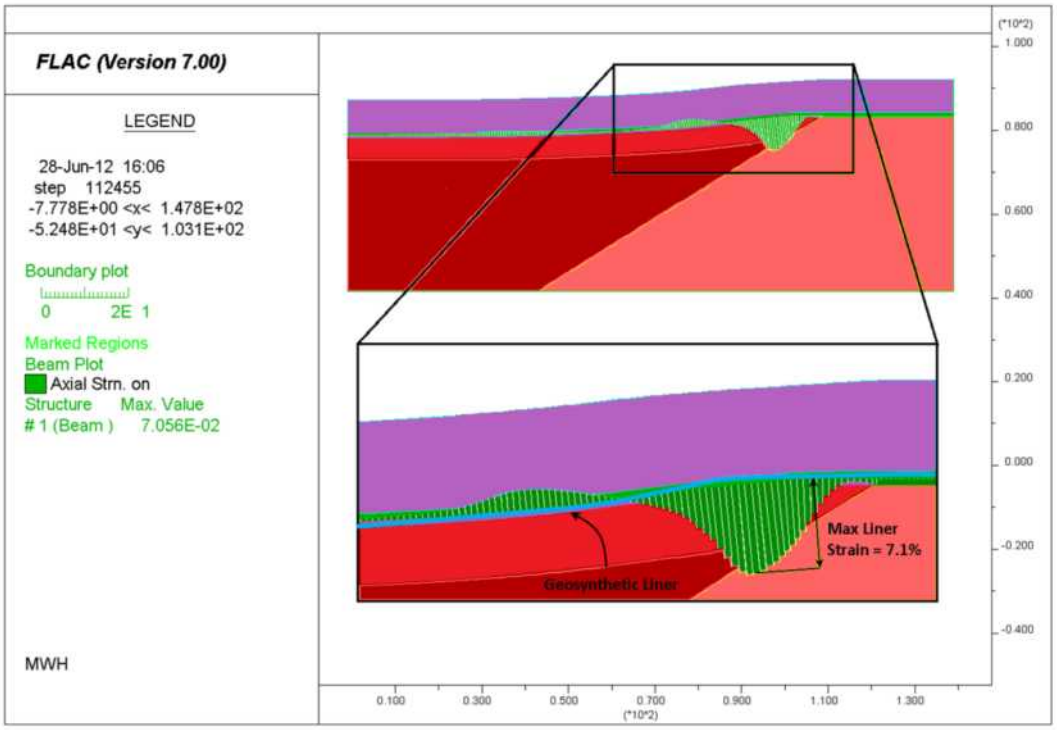

Figure 11: The maximum liner strain for Scenario 2b

Scenario 3

Scenario 3 is similar to Scenario 2, except that instead of assuming that all the WSF consists of soft material, only a 10 m band of WSF adjacent to the bedrock is modeled as soft material. This is shown in Figure 12. The deformed mesh after construction of the 8 m heap and the calculated strains along the liner for Scenario 3 are shown in Figures 13 and 14, respectively.

Scenario 2a showed a higher maximum liner strain than Scenario 3. Therefore, only Scenario 2a was selected for further evaluation using a 5 m thick compacted waste cap, a 0.6m overliner, and a 0.6 m underliner. This evaluation was performed as Scenario 2b and the results shown in Figure 11. It is expected that adding the compacted waste and overliner/underliner layers to Scenario 3 would result in the maximum liner strains lower than that calculated in Scenario 2b (7%) and would be within the acceptable range.

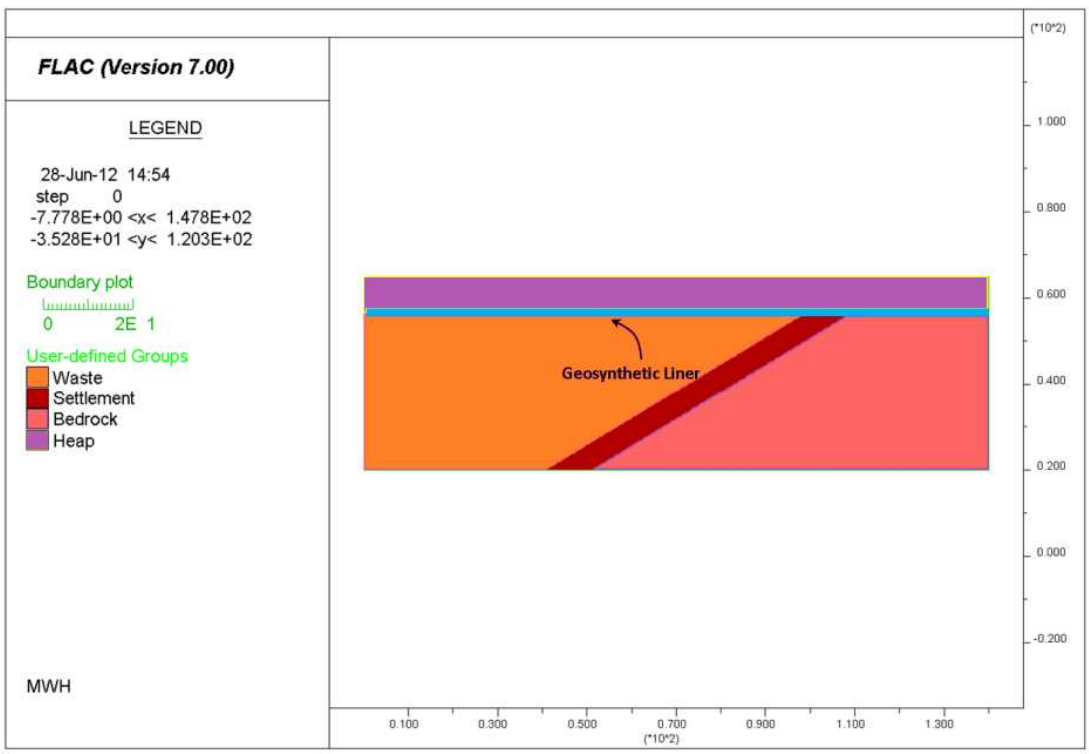

Figure 12: General view of Scenario 3 used for differential settlement analysis

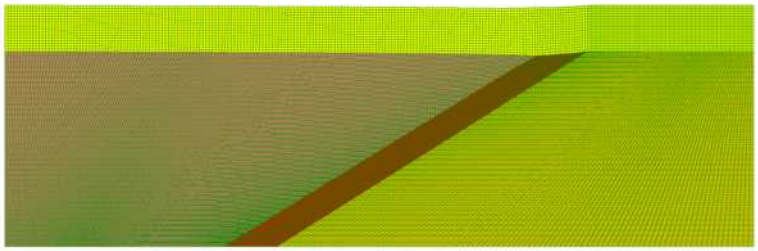

Figure 13: Deformed mesh after pacing the 8 heap for Scenario 3

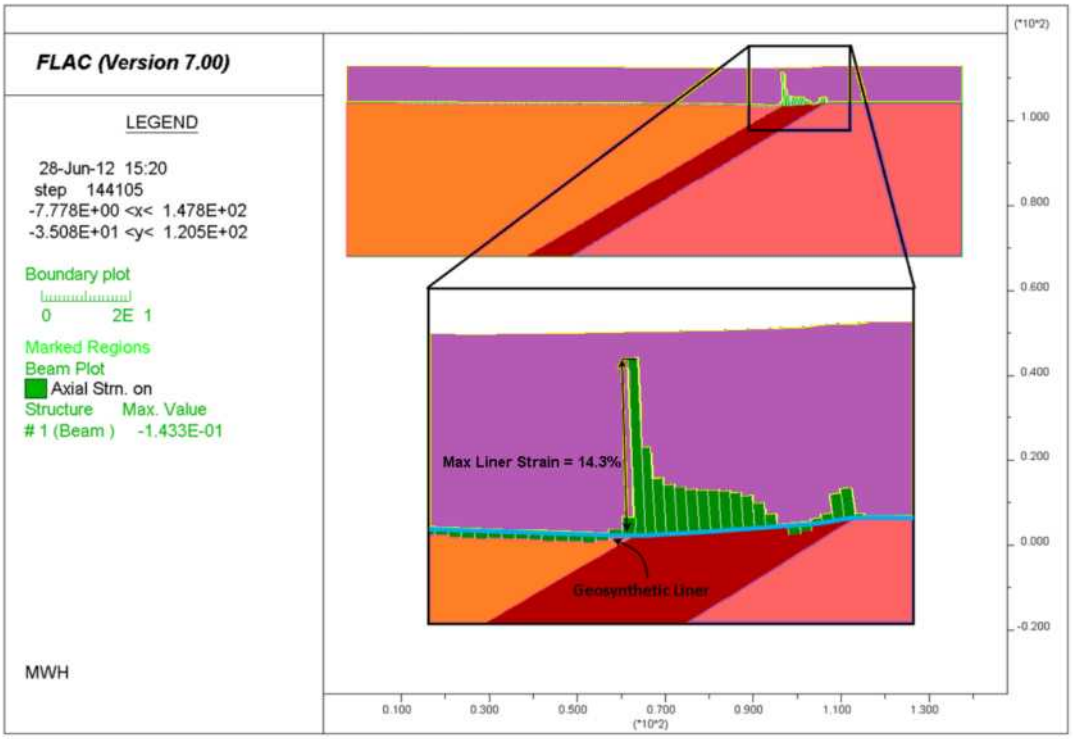

Figure 14: The maximum liner strain for Scenario 3

Conclusions

This paper presents the differential settlement modeling performed to evaluate the integrity of the proposed liner for a heap leach pad at a South American mine subject to differential settlement. First, a 50 m deep and 10 m wide column of soft waste was simulated in the middle of the waste deposit to determine the displacements and strains developed in the liner due to settlement of the soft layer after placement of an 8m high heap. The properties of the soft layer were chosen such that the final settlement in the absence of the underliner, overliner, and compacted waste layer considered in subsequent analyses was 80 cm, where 80 cm is the calculated total consolidation of the existing waste as a result of the additional load exerted by the heap. The maximum liner tensile strain due to this settlement was

calculated as 11.1% without considering the 5 m compacted waste cap, 0.6 m overliner, or 0.6 m underliner. When these components were taken into consideration, the maximum liner tensile strain was reduced to 0.02%.

The second modeling case simulates another potential critical condition of differential settlement. In this case the soft waste material is located adjacent to the bedrock (i.e. adjacent to an incompressible material). The results from the analyses showed that the maximum liner tensile strain can become as high as 19% if the 5 m compacted waste and the overliner and underliner are not included in the simulation. However, by taking the 5 m compacted waste cap, 0.6 m overliner and 0.6 m underliner into consideration, the calculated maximum liner strain was reduced to 7.1%, which is taken as the limit of acceptable tensile strains for an LLDPE liner.

Furthermore, Scenario 2b assumed that the entire WSF consisted entirely of soft waste material, which is a highly conservative assumption as the properties of the soft waste material are based on the weakest material found in the previous SPT exploration of the WSF, which is not representative of the entire WSF. Another conservatism used in this study is assuming a 50 m deep soft layer in all the analyses. Fifty meters is the maximum depth of the waste under the leach heap, but most of the leach heap will be placed over waste with thickness less than 50 m. Additionally, considering the fact that the waste materials are being placed in 1 to 2 m lifts over a long period of time and in a large area, it is highly unlikely that soft material will be placed directly and repeatedly over soft material to build a continuous column of soft waste material that is 10 m wide and 50 m deep.

The results of the modeling showed the important role of the compacted waste cap, overliner, and underliner for reducing liner strains. This emphasizes the importance of the construction of a firm and well compacted foundation for the leach pad.

Based on the FLAC modeling of the settlement and the available data, our judgment, based on conservative design criteria, is that as long as a competent foundation is provided, differential settlement of the underlying waste is not a threat to the integrity of the proposed heap leach pad liner on the WSF.

As the project moves into more detailed design, we will use the differential settlement model presented in this paper to optimize the cap thickness and location, any further waste characterization and the pad design details.

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Recent advances in geosynthetic liner system materials to maximize heap leach pad performance

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Abstract

Historically, geosynthetics have been used to solve a number of environmental challenges in the mining industry. The impacts and costs of environmental challenges can be extraordinary; in addition to traditional environmental protection requirements, mining enterprises have been experiencing increasing pressure to reduce infrastructure cost and improve the timeline and magnitude of project payback. In striving to meet these needs, several common constraints stand in the way, including: (1) limited quantities of select drainage aggregates and water; (2) limited leaching solution concentration due to geomembrane lifespan considerations; (3) leaching solution fouling due to inadequate filtration; (4) delayed metal recovery due to limited drainage rates; and (5) loss of significant quantities of metal product caused by leaching solution leakage through defects in geomembranes resulting from damage during construction.

Seeking solutions for these key challenges has driven a tremendous development effort over the past two years. All of the materials that are traditionally used in both heap leach pads and waste containment systems have been examined for opportunities to shorten heap leach project payback time and improve capture rates in heap leach systems.

The resulting analysis has led to numerous advances in geosynthetic materials that are becoming available to heap leach pad designers, including: (1) drainage geocomposites that provide better drainage performance and lower cost than aggregates in heap leaching conditions; (2) electrically insulating weld techniques for conductive geomembranes that improve the performance of all geoelectric leak detection test methods; (3) refinement of surface texture parameters that govern interface shear performance; (4) development of high performance geomembrane resins that provide increased life span in harsh oxidizing environments; and (5) ultra-high strength reinforced geosynthetic clay liners (GCLs) that can support higher and steeper structures.

These individual material advances naturally led to a holistic system approach for designing heap leach pad lining systems, aggregating to enable design engineers to increase system performance in ways that could not have been previously conceived.

Problem statement and scope of study

Traditional geomembrane lining systems are typically based on standard materials and lowest capital expenditure. Advances are constantly made in geosynthetic materials for the solid waste and transportation industries to improve performance and/or reduce capital expenditures. In examining improvements that can be made for leach pads, the basic functions of earthen materials and geosynthetic materials were reviewed.

Background information about available construction materials

Construction has been done using natural soil materials for centuries in an effort to improve bearing capacity and slope stability, as well as to improve drainage and create dams for water containment. These applications are apparent in prehistoric and early historic building pads, aqueducts, French drains, wells, and earthen dams. In the late 1970s, the first geosynthetics industry conference was held. Since then, vast improvements have been made in the quality and quantity of geosynthetic materials available. For the most part, these materials have been developed to replace earthen materials with alternatives that can be installed more rapidly, will perform better, will be more consistent, and, often, will be available at lower cost than the earthen materials used in more historic designs.

Functions of earthen and geosynthetic materials

Earthen materials and geosynthetic materials fulfill many of the same functions, including separation, reinforcement, filtration, drainage, and containment of a fluid. Generally, the earthen materials used in construction range in particle size from clay, to silt, sand, gravel, and rock. At the lowest permeability end of the spectrum, compacted clay is used for containment. Relatively low strength and low permeability silts are often amended with cements to create soil cements and concrete blocks. Intermediate permeability sands are often used for filtration, drainage, and as a foundation for building structure slabs and footings. Crushed rock is often used as a high strength aggregate road base and, if fine content is removed, can also be used as a drainage material. Similarly, geomembranes offer permeability and permittivity that are several orders of magnitude better for containment than the best clay materials; drainage geonets provide drainage capacity that far exceeds the permeability of most drainage aggregates, and properly selected geotextiles provide much more easily installed filter and separation capacity than lifts of sand and silt.

Although the simplest systems, like pond liners or homogeneous dams, consist of only a containment layer, most systems use multiple layers to provide the various functions and redundancies needed in more critical applications, such as the following examples:

A zone-fill earthen dam is made with different types of soils to create and protect a low permeability clay core and also to drain an outer ballast zone to maintain shear strength and stability.

A composite liner consisting of a geomembrane over a low permeability soil or geosynthetic layer. The composite effect created when these types of materials are in intimate contact can greatly reduce flow through the system, improving containment performance by multiple orders of magnitude as compared to either layer used alone.

Liners that are used to contain liquids that leach from stored solid material include solid waste landfills and mining leach pads. These systems also include a drainage layer above the geomembrane. The drainage layer allows fluid to migrate along slopes to be collected for harvest or treatment.

In systems that contain liquids that carry solids and could plug the drainage materials such as municipal solid waste, coal combustion residuals, and heavily milled ore with fine texture and low cohesion, a filtration layer is typically added above the drainage layer and composite liner to help prolong the useful lifespan for the drainage layer.

In ponds where a liner may hold relatively high hydraulic head, a double geomembrane liner with a leak detection layer between the two geomembranes is very often used. In this case, the primary geomembrane is placed over a drainage layer and a secondary geomembrane layer. The drainage layer and secondary liner serve to help in the detecting leakage through primary liner damage or defects, and greatly reduces leakage into underlying soil and groundwater by minimizing the amount of hydraulic head on top of the secondary liner.

In critical applications such as hazardous waste landfills, a double liner may be constructed with a composite liner as the (lower) secondary liner to further minimize loss of leachate.

This array of liner systems is selected based on cost and availability versus risk and benefit tradeoff analysis on any given project. Generally, this analysis results in a design that includes select native or nearby soils whenever appropriate materials are available, and geosynthetic alternatives to natural soils when select soils are not available in sufficient quantity, are too difficult or costly to segregate and import; or when higher performance is needed than can be provided using available soils.

Available geosynthetic materials

A very wide variety of geosynthetic materials has become available over the past 40 years, beginning with geotextiles used for soil reinforcement and filtration, and geomembranes developed as barrier layers. Today, materials that perform to the generally accepted set of performance requirements have standard specifications and testing protocols that are maintained by the Geosynthetic Research Institute (GRI) ; e.g. GRI GM13 for HDPE (high density polyethylene) geomembranes, GM17 for LLDPE (linear low density polyethylene) geomembranes, GCL3 for geosynthetic clay liners, etc. These standards are self-imposed

by geosynthetics manufacturers through GRI, and provide a baseline for minimum performance that can be counted on from any participating geosynthetics manufacturer.

In addition to these standard materials, specialized materials are available to solve problems that arise due to site-specific issues. These materials are typically developed in collaboration between project owners, specifying engineers, and manufacturers. As a result, there is a wide variety of options available for each of the components used in lining systems. The scope of this paper is to review the functions needed in a leach pad liner system, to determine the best suited existing materials to meet those needs, and to identify any possible need for further material development.

Heap leach pad base liner functions

In the simplest terms, heap leaching is accomplished by heaping ore onto a lined pad that collects and drains solution carrying metal products that are dissolved from the heap after irrigating it with a leaching solution. For discussion purposes, we will consider a liner to be the portion of the leach pad that is placed between graded soil base (below) and the ore being leached (above). In this example, the liner performs only a containment function because ore that happens to have appropriate gradation for drainage can be placed directly on a geomembrane.

For most projects however, it is desirable for a liner to provide better containment performance than a single geomembrane. In many cases, it is necessary for a liner system to serve several more functions in addition to containment, as described below.

Compacted clay liners

Sources of clay

Adequate quantities of appropriate clay soils can be very difficult to find in some locations. Clay deposits must be large enough and consistent enough – large enough to supply the project without costly importing, and consistent enough to quarry without excessive soil segregation efforts.

Need for construction water

Compacting clay soils as a barrier requires that soils be quite wet during the compaction process. The water needed to lubricate clay platelets and pre-swell clay soils to help avoid desiccation cracking can be significant. Managing water is just as important as procuring a supply. Clay soils need to be very wet during this type of construction, but it is impossible to obtain acceptable compaction in soils that are too wet.

Desiccation cracking

The lowest permeability clays tend to be quite expansive, swelling when wet and shrinking when dried. If cohesive forces in the soil are not sufficient to distribute tensile forces during drying/shrinking, desiccation cracking will result and containment performance will be greatly compromised.

Effects of different ore gradations

Excessive fines content

If the ore contains a significant quantity of fine particles, it may either not drain adequately or fine particles may migrate into collection pipes and sumps and ultimately clog the system. In this case, drainage and filtration needs to be incorporated into the liner in order for pregnant leaching solution (PLS) to be collected from the heap. Traditionally, this is done by creating an upper drainage layer from clean fine gravel or sand placed on top of the geomembrane prior to placing ore.

Excessive ore particle size and angularity

If the ore consists of very coarse particles that are angular rather than smooth, the particles can cause yield points that can develop into holes in the geomembrane over time. In this case, a cushion layer needs to be incorporated into the liner design to protect the geomembrane containment layer. This is traditionally done by creating a cushion layer from fine gravel or sand placed on top of the geomembrane prior to placing the ore.

Challenging subgrade soils

Coarse texture and high permeability

As previously mentioned, a composite liner made with a geomembrane over a low permeability subgrade provides much better containment (i.e. less leakage) than either a geomembrane or low permeability soil layer alone. Coarse onsite soils with high permeability cannot provide the performance that is needed for many installations. In this case, additional barrier materials would need to be incorporated into the liner system. In areas with high quality local clay sources and experienced contractors, a compacted clay liner is traditionally used.

Exposed angular clasts

Similar to large angular ore above a geomembrane, relatively large exposed angular clasts in subgrade soils can cause yield points and, eventually, holes in the geomembrane. In this case, the liner system should contain a cushioning element beneath the geomembrane. Traditionally, this was done by either constructing a compacted clay liner or using a lift of sand in locations where barrier performance is not needed from the materials directly beneath the geomembrane.

Geosynthetic alternatives to natural materials

Geomembranes

Standard geomembranes

The GRI publishes standard specifications for geomembrane materials. Those most commonly used in mining applications are for HDPE and LLDPE geomembranes. The GRI standard specifications for these materials are GM13 and GM17, respectively.

Under normal conditions, exposed lifespan (defined as half-life of material properties) for standard GM13 HDPE geomembrane liner is about 40 years if exposed and about 400 years if covered. The elevated temperatures, higher ultraviolet light exposer rates at high elevations, and several other factors can combine to reduce the lifespan for standard material. LLDPE offers better capacity to conform to differential settlement and multiaxial loading, but at the expense of slightly less chemical resistance and lifespan.

Additional geomembrane options currently available

The following additional options are available to improve performance beyond that of a standard geomembrane:

Conductive geomembranes allow a geomembrane layer to be spark tested in accordance with ASTM D7240 during installation to find construction damage before drainage materials or ore are stacked on top of the liner. In addition to post-installation surveys, a new set of installation techniques developed with leading geoelectric leak detection experts now enables GSE Leak Location Liner to improve the accuracy and reliability of all types of geoelectric leak detection surveys. The most notable of these survey methods is the dipole survey (ASTM D7007), which can be used to check for geomembrane damage through overliner materials prior to placing ore on a leach pad.

Coextruded texturing provides texturing with a variety of peak heights, whereas embossed geomembrane texturing provides very uniform texture height. The varying peak heights of coextruded texturing create exceptional shear performance across a very wide range of loading conditions. The most important of these conditions occur (1) following displacement between layers (residual strength); and (2) during construction when the varied peak heights provide shear strength against other materials at low loads – exactly when needed to minimize wrinkle size and keep materials in place as cover soils or ore are placed on top of a liner.

Premium resin formulations such as GSE High Performance HDPE and GSE High Performance LLDPE are available. These materials contain: (1) extremely high grade base resins to prevent stress crack and to accommodate extraordinary amounts of multiaxial elongation, differential settlement; (2) premium antioxidant packages to nearly double the time over which the antioxidant package protects the

base resin and provide longer lifespan in elevated temperatures and high UV exposure locations; and (3) a super-fine grade of carbon black to absorb short wave solar radiation and dissipate it as harmless long wave thermal radiation;

Needs for further development

Leaching pyrite ore types involves exothermic biological reactions that generate temperatures that are quite high. This can greatly increase the rate at which antioxidants are consumed from geomembranes, reducing the geomembrane lifespan. Geomembrane resins that are formulated for high temperatures could greatly increase the types of processes that can be used.

Geosynthetic Clay Liners (GCLs)

Standard Geosynthetic Clay Liners

Standard GCLs are made by placing granular bentonite between geotextiles and needle punching the materials to pull reinforcing fibers together between the geotextiles. High quality sodium bentonite exhibits extremely low permeability, allowing a relatively thin liner to have better containment performance than a liner that is more than 60 cm thick.

Sodium bentonite is also extremely expansive and exhibits very little shear strength when wet, so the loading that can be supported by a GCL is mainly a result of the fiber reinforcement. Standard GCLs exhibit shear strength exceeding 6,500 psf at a 10,800 psf normal load; however the amount of needle punching needed to obtain this shear strength may be associated with wet/dry cycle shrinkage in unballasted GCLs.

Standard bentonite is also subject to cation exchange when exposed to common leaching fluids like sulfuric acid, which commonly results in two orders of magnitude of increased permeability.

Additional Geosynthetic Clay Liner options currently available

- A scrim reinforced nonwoven geotextile can be used to replace a standard nonwoven geotextile carrier fabric in a GCL such as GSE NWL GCL to mitigate the potential shrinkage risk in heavily reinforced GCLs without reinforcement.
- Polymers can be added to sodium bentonite to control cation exchange and maintain barrier performance in contact with leachates that are associated with cation exchange. One such formulation is GSE's CAR (cation/anion resistant) bentonite that is an option in any of GSE's GCLs.

Needs for further development

Because the amount of ore that can be stacked on a leach pad directly depends upon slope stability, additional development of even higher strength GCLs could be beneficial for increasing infrastructure capacity on a per-unit-area basis.

Drainage geocomposites

Standard geocomposites

Standard drainage geocomposites are composed of a drainage geonet core with a geotextile filter fabric laminated to one or both sides. Geocomposites have been used as a reliable replacement for sand and gravel in the collection of solid waste leachates from the top of landfill liners for years, resulting in a wide variety of geocomposites to meet various flow capacity requirements at various compressive loads encountered. In addition to their high drainage capacity, the filter geotextile on the surfaces of a geocomposite help to keep the drainage system clear of particulates that could otherwise clog a natural aggregate drainage layer.

Standard drainage geonets are made with oblong strands that are taller than they are wide (if the plane of flow is in the strand width direction). These strands have a lower moment of inertia vertically than horizontally, resulting in the potential for strands to buckle and roll over under high compressive loads. Geocomposites made with thicker geonet cores (i.e. made with taller strands) are available to provide additional drainage capacity that helps to some extent.

Additional geocomposite options

- Round-strand geonets such as GSE PermaNet are available for applications requiring extremely high compressive strength by using a strand that has the same horizontal and vertical moments of inertia to prevent strand rollover. Testing has shown that these materials maintain flow capacity greater than standard geonets even after 10 or more years at up to 3.8 MPa of overburden pressure.
- TriAxial geonets such as GSE TenFlow are available to provide extremely high flow capacity at slightly lower loads than those that can be accommodated by a round-strand geonet.
- Geocomposites are available with filter fabric on only one side, for use when the other side is to sit directly against a geomembrane. There is a tradeoff between shear strength and flow capacity when either using or not using a bottom geotextile on a geocomposite. Where shear strength of the saturated ore is low and a perimeter buttress is used to provide stability, the higher flow rates in single-sided geocomposites appear to be the more favorable option.

Needs for further development

- Fine particles can migrate out of extremely fine noncohesive soils, either clogging or passing through a filter geotextile. While research has been done to confirm filter compatibility with most common ore types, a new filter geotextile called the GSE GF (graded filter) Geotextile has been developed for draining coal combustion wastes promises to overcome the challenge of meeting both permeability and filtration criteria. Although this will likely not be needed in most heap leaching applications, it may prove beneficial in storing very heavily milled tailings.

Conclusions and recommendations

A leach pad liner system should exhibit the following performance properties:

The system should be robust enough to keep product in the system and limit environmental liability by minimizing leakage into underlying soils and groundwater.

All materials should be able to be consistently manufactured and installed quickly with consistently high quality.

The system should provide high drainage capacity at high compressive loads, and that drainage capacity should be retained on a long-term basis with filtration that is designed to assure that flow capacity into the drainage net will remain high and particles that may migrate toward the filter will be retained to prevent clogging of the drainage layer.

Shear strength for the system should be high enough to increase the steepness of slopes upon which the system can be installed and increase the height of ore that can be processed on top of the system.

In order to provide these performance properties on a long term basis under the range of site conditions encountered, a geosynthetic system will be needed in most cases. In order to optimize leach pad payback and minimize risk, a system comprised of the following materials should be used.

Over the majority of the leach pad surface

Drainage layer

Drainage geocomposites should provide the following performance criteria for most leach pad configurations:

- Adequate flow capacity at the expected compressive loads;
    - In dynamic or on/off leach pads, construction traffic loads will often govern;
    - In static or conventional leach pads, creep thinning due to long term compressive loading will often govern.

- Adequate shear strength for heap stability. In most cases this will be a single-sided geocomposite beneath the center of a saturated ore heap to take advantage of the higher transmissivity in the areas where less shear strength is needed.

GSE developed MineDrain Geocomposite to meet these criteria.

Geomembrane

A geomembrane material should exhibit the following performance criteria:

- A conductive geomembrane that can be surveyed for construction damage in accordance with ASTM D6240 (spark test) following installation, and again in accordance with ASTM D7007 (the dipole method) following placement of the first 50 to 100 cm of drainage materials.
- A premium resin to accommodate multiaxial tensile loading in differential settlement conditions and provide longer lifespan in exposed and elevated temperature settings.

GSE developed High Performance Leak Location Liner to meet these criteria.

Geosynthetic Clay Liner

A GCL material should exhibit the following performance criteria:

- A reinforcing scrim should be used to prevent GCL shrinkage issues.
- A polymer-enhanced bentonite should be used when containing solutions that cause cation exchange in standard sodium bentonite.

GSE developed CAR NWL GCL to meet these criteria.

Additional considerations for a buttress zone along the perimeter of a leach pad

Drainage layer

While the highest strength geocomposites are quite capable of handling typical leach pad loads, the compressive loads that develop at the toe of slope in a leach pad during a seismic event can be extraordinary. Earthen materials should be used in this zone to provide the additional compressive and shear strengths that are needed in this zone.

Geosynthetic Clay Liner

The buttress zone is used to stabilize the entire heap with very high shear strength at the perimeter of the heap. The height and slopes that can be constructed are directly dependent on the internal shear strength of the GCL used here. GSE developed its NWL-60 GCL to provide extremely high shear strength in this type of structure.

Geomembrane

The *peak* shear strength of the *interface* between the geomembrane and the underlying GCL should be slightly lower than the internal shear strength of the GCL material to maintain GCL integrity during seismic events. GSE can adjust texture height of its coextruded textured geomembranes to get the most out of all of the materials in the system.

State-of-the-art heap leach pad configurations

The materials above can be combined into a state-of-the-art leach pad liner system to combine the benefits of all material options and combinations. The following figures illustrate the configurations of these systems. Figure A shows the GSE leach pad liner system for large flat areas inside the heap. Figure B shows the GSE buttress liner system for canyon fills and for the stability zone around the perimeter of relatively flat pads.

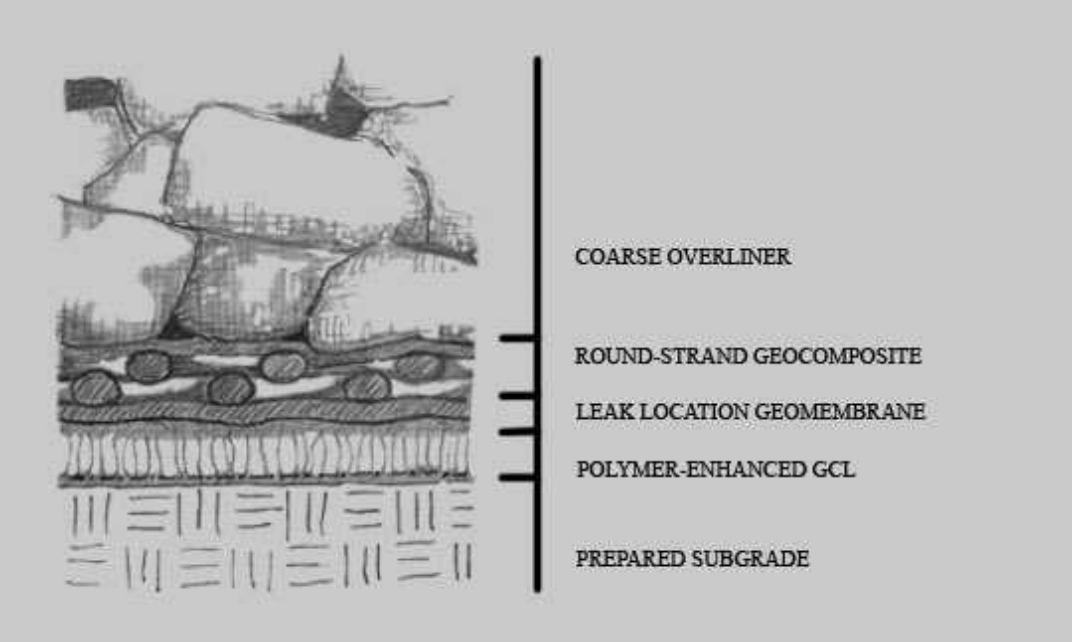

Figure 1: Typical state-of-the-art leach pad liner for flat areas

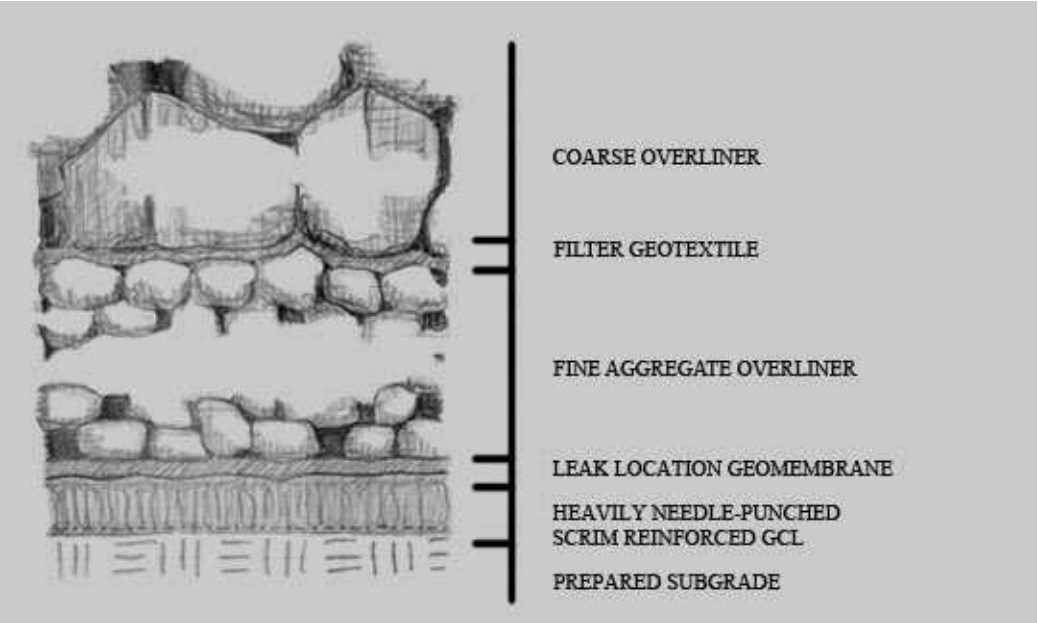

Figure 2: State-of-the-art leach pad liner for stability zones

Hydro-mechanical considerations of high fines content leach ores

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Abstract

Heap leach operations rely on a number of factors for efficient leaching and metal recovery. Of the various factors that can affect leaching, the hydraulic characteristics of the ore are among the most important. Leach ore with favorable hydraulic characteristics (e.g. high hydraulic conductivity) can sustain high percolation rates. These leach ores tend to be relatively coarse with low to moderate fines (particle sizes less than 0.074 mm) content (typically less than 15%). However, as the fines content of the ore increases, the hydraulic performance (e.g. sustained percolation rate) tends to decrease significantly, which may affect the leaching rate, metal recovery rate, and in-heap solution inventory.

While it is clear the fines content directly affects the hydraulic properties of the leach ore, what is less understood is the hydro-mechanical behavior of the ore with increasing fines content. Experience has shown that ores with very high fines content (30% or more) tend to have distinct hydro-mechanical behavior which can have a significant impact on heap performance and stability. Field and laboratory test data from high-fines content leach ore show that these materials tend to be more compressive than low-fines content ore, which can lead to generation of excess pore pressures during ore stacking operations. Under some conditions and loading rates, this can lead to static liquefaction and flow failure of the ore stack. These conditions can occur, even if the ore is agglomerated.

This paper discusses the hydro-mechanical behavior of high fines content leach ore and its implication in the design of heap leach pads. The paper focuses on testing that can be used to characterize the hydro-mechanical behavior of fine-grained ore and how these results can be considered in the design and operation of heap leach pads.

Introduction

Heap leach pads are generally designed based on the results from leach column tests, which help define the relationship between recovery to leach cycle time under various leaching rates and loads. The leach-column test data are then scaled and used as the design basis for sizing the heap leach pad and defining the basic operational parameters. In addition to leach column testing, leach ore is also often tested to

define its hydraulic properties, such as saturated and unsaturated hydraulic conductivity, since these properties have a significant influence on heap performance. For example, leach ore with favorable hydraulic characteristics (e.g., high hydraulic conductivity) can sustain high percolation rates. These leach ores tend to be relatively coarse with low to moderate fines (particle sizes less than 0.074 mm) content (typically less than 15 %). A typical coarse leach ore is shown in Figure 1.

As the fines content of the ore increases, the hydraulic performance (e.g., sustained percolation rate) tends to decrease significantly, thereby affecting the leaching rate, metal recovery rate, and in-heap solution inventory. In high fines-content leach ore, generally considered as ore with fines content exceeding 30% (see Figure 2), not only are the hydraulic characteristics affected, but the hydro-mechanical behavior of the ore becomes important for heap performance. Field observations and laboratory test data from high fines-content leach ore show that these materials tend to be more compressive than low fines-content ore. Under some conditions and loading rates, this can lead to static liquefaction and flow failure of the ore stack. It is important to note that these conditions can develop, even if the leach ore is agglomerated.

Figure 1: Photograph of a typical coarse leach ore

Figure 2: Photograph of a typical high fines-content leach ore

Understanding the hydro-mechanical behavior of high fines-content leach ore is important for both heap leach pad design and operation, as it will influence the overall ore heap stacking height, leaching rate, ore loading rates, and surface water management on the heap. The following sections present discussions on the basic hydro-mechanical behavior of leach ore, testing used to predict ore behavior under load, and how this behavior impacts heap design and operation.

Hydro-mechanical properties of ore

The hydro-mechanical properties of leach ore describe the inter-relationship between ore hydraulic and mechanical behavior under an applied stress. In general terms, as leach ore is loaded (under normal and shear stresses), it will deform which, in turn, affects the hydraulic properties of the ore. In high fines-content ore, this deformation can also result in the generation of excess pore pressures. If the excess pore pressures are high enough, flow failure within the ore heap can occur.

The hydro-mechanical behavior of ore can express itself in many ways. One aspect of the hydro-mechanical response of ore is most easily observed in simple one-dimensional compression tests. One-dimensional compression tests are conducted by placing the ore (fresh or leached) into a rigid-wall test

vessel (see Figure 3). Incremental loads are applied to the ore, resulting in an increase in ore density and decrease in ore porosity as a function of the applied load.

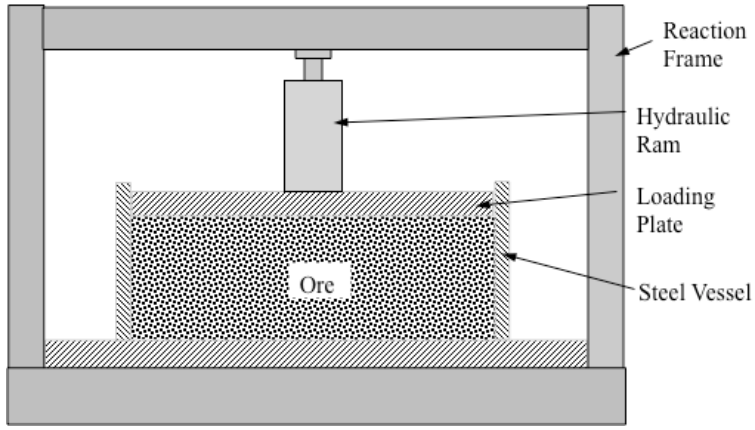

Figure 3: One-dimensional compression test schematic

Typical results from one-dimensional compression tests (from the authors' database) are presented in Figure 4, in terms of ore porosity versus effective ore heap height. As expected, there is a general trend of decreasing ore porosity with applied load (e.g., increased ore height), with a high amount of variability among the test data. The amount of porosity change is directly related to the durability and fines content of the ore sample, as illustrated in Figure 5. In this example, the durable, low fines-content ore sample lost approximately 50% of its original porosity under load, while the compressible high fines-content ore material lost approximately 90% of its original porosity.

The test results presented in Figure 4 illustrate the mechanical response of ore under simple compression. The changes of the hydraulic properties (in this case, saturated hydraulic conductivity) of ore under simple compression are shown in Figure 6. These test results show the ore hydraulic conductivity also decreases with increasing load, which is intuitive given the reduction in porosity (Figure 4). What is not as intuitive is the magnitude of reduction, which can range as much as four orders of magnitude, depending on the ore type and applied load.

The results of the simple one-dimensional compression tests presented in Figures 4 and 6 clearly demonstrate one aspect of the hydro-mechanical behavior of leach ore, namely the reduction in hydraulic conductivity with an applied load corresponding to a decrease in ore porosity. These test results explain the increase in ore saturation with depth that has been observed in many heaps. Based on the results shown in Figure 6, ore near the surface of the heap will generally have a higher hydraulic conductivity than ore buried deep within the heap. Since leach solution application rates are generally derived from column tests, which are typically less than 5 m in length, the application rates reflect the higher permeability of the ore near the surface of the heap. However, as the leach solution moves through the

heap, the ore hydraulic conductivity decreases, resulting in more solution being held within the ore pore space, hence increasing the ore saturation. Another reason for the increase in ore saturation has to do with the decrease in porosity (Figure 4). As ore is compressed and porosity is reduced, the ratio of solution to air within the pore space changes, resulting in an increase in saturation. This concept is illustrated in Figure 7, which shows that an ore originally placed in an unsaturated condition, could become saturated with additional applied load from ore stacking. This is an important concept as it leads to another important aspect of the hydro-mechanical behavior of high fines-content leach ore, namely the ore response under shear.

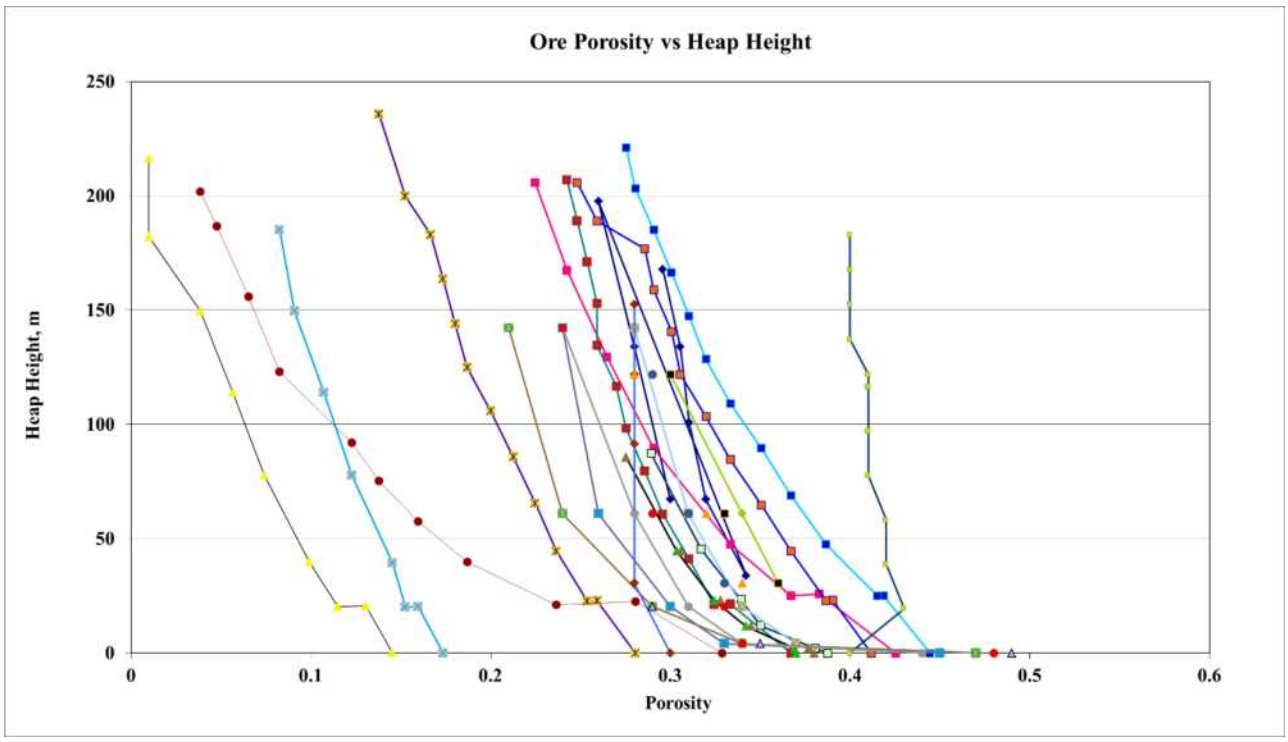

Figure 4: One-dimensional ore compression test results – porosity

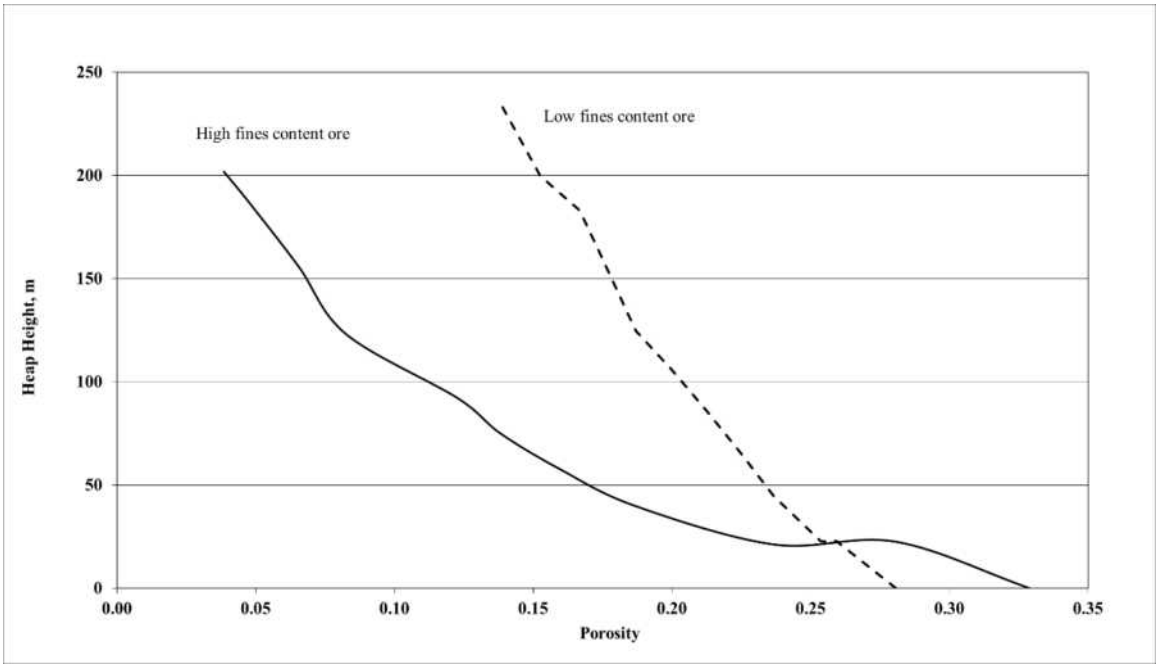

Figure 5: Ore compression versus ore type

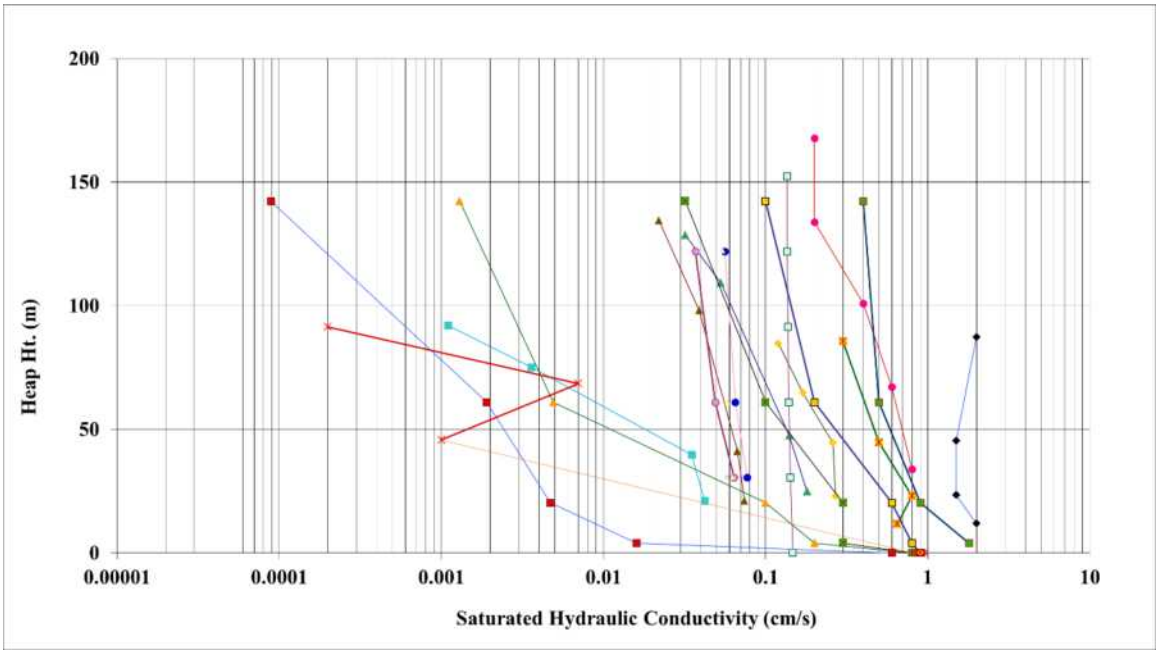

Figure 6: One dimensional compression test results – hydraulic conductivity

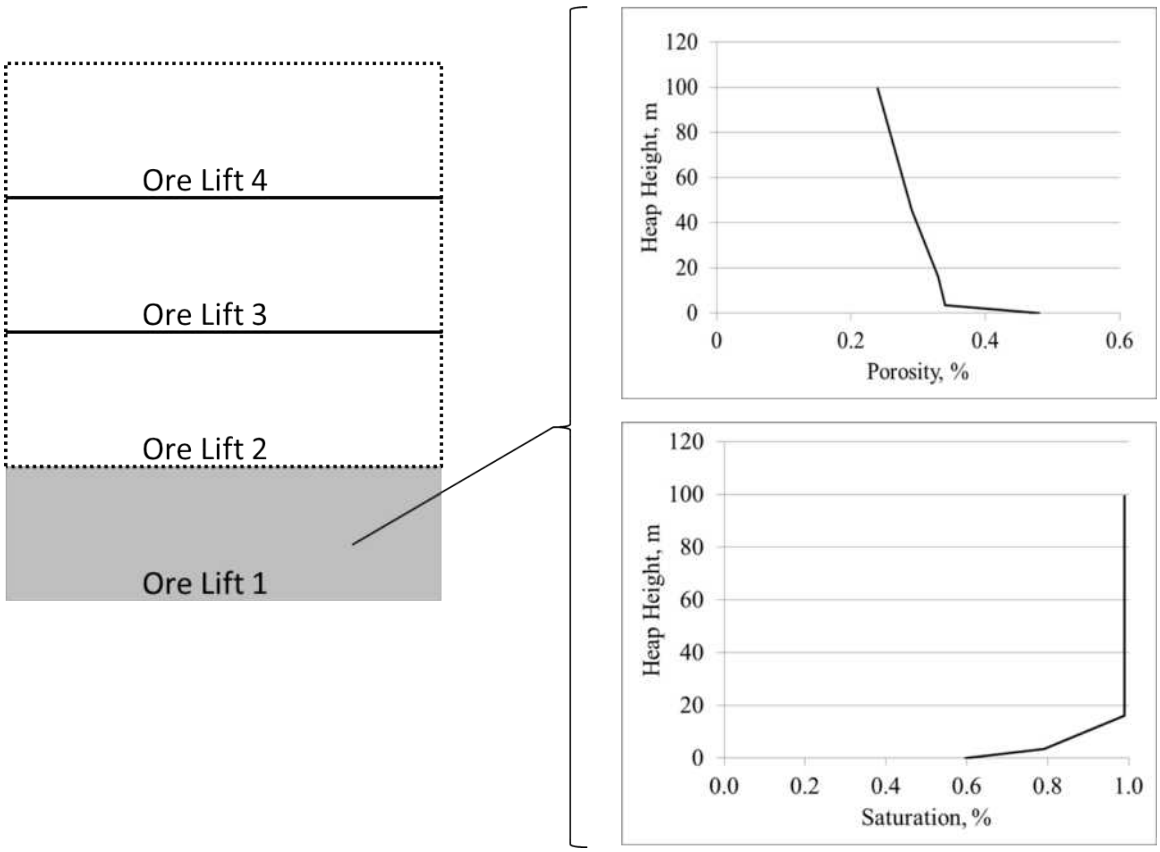

Figure: 7 Ore saturation with load

Shear stresses are normally developed within the ore heap, after the ore is stacked, as a result of self-weight loading and leaching. The shear behavior of ore under load governs the ore heap stacking slope and ore stacking rate, therefore understanding the behavior of the ore under shear is critical to leach pad design and operation.

Before discussing the behavior of ore under an applied shear stress, it is important to briefly discuss the concept of volume change under shear. The test data presented in Figure 4 demonstrated volume (e.g., porosity) reduction with an applied normal stress. However, leach ore may exhibit either a volume reduction or increase with an applied shear stress. Volume or porosity reduction that occurs under shear is termed contraction, while a volume increase under shear is termed dilation. Volume dilation and contraction concepts are generally applied in the field of soil mechanics, however these concepts are equally applicable to leach ore, particularly in high fines-content ore, which can often resemble fine sand and silt (Figure 3).

Early work on the concepts of soil dilation and contraction under shear was conducted by Terzaghi (1929), Casagrande (1936), and Castro (1969). While the details of these topics are beyond the scope and length of this paper, the basic concept is that dense soils tend to dilate under shear, while loose soils tend

to contract under shear. At first, the concept of dilation or contraction may not seem to be too relevant to heap leach pad design and performance. However, when these concepts are applied to ore which is typically placed in a loose state and is leached at high levels of saturation (see previous discussions), the applicability of these concepts becomes clear.

Laboratory testing, primarily consisting of triaxial compression tests under undrained conditions, are typically used to assess the potential dilation/contraction behavior of ore under shear loading. At first, testing the ore under undrained conditions may not seem compatible with the concept of leaching under unsaturated conditions; however, as discussed previously, portions of the heap can become saturated or nearly saturated under normal loading (e.g., ore stacking). Under some conditions, saturated ore may undergo static liquefaction, resulting in failure of the ore heap, as shown in Figure 8. Note that the flow failure shown in this figure occurred in agglomerated ore.

Traditionally, undrained triaxial compression tests are used to simply provide a measure of the ore shear strength, either as values of cohesion with friction angles or as a functional relationship between normal and shear stresses. However, the measure of shear strength does not provide any information on the dilation/contraction behavior of the ore. Luckily, the data from undrained triaxial compression tests can be plotted in stress path space to provide such information.

A discussion on the use and interpretation of plotting data in stress path space is beyond the scope of this paper; however a good general explanation is presented in Lambe and Marr (1979). Yamamuro and Lade (1999) discuss the application of stress paths in terms of static liquefaction, which is relevant to the topic of this paper.

To generate a stress path plot from triaxial compression data, the test data is plotted in terms of mean and shear (sometimes referred to deviatoric) stress. The mean and shear stresses can be calculated as:

- Mean stress = $(S_1' + S_3') / 2$, and
- Shear stress = $(S_1' - S_3') / 2$.

where:

- S_1' = major effective principal stress, and
- S_3' = minor effective principal stress.

Effective stresses must be used as these are undrained tests that have pore pressures that play a significant role in ore behavior under shear.

Figure 9 presents a plot of "typical" stress paths for undrained ore materials undergoing triaxial compression.

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Figure 8: Static liquefaction of ore on a leach pad (agglomerated ore)

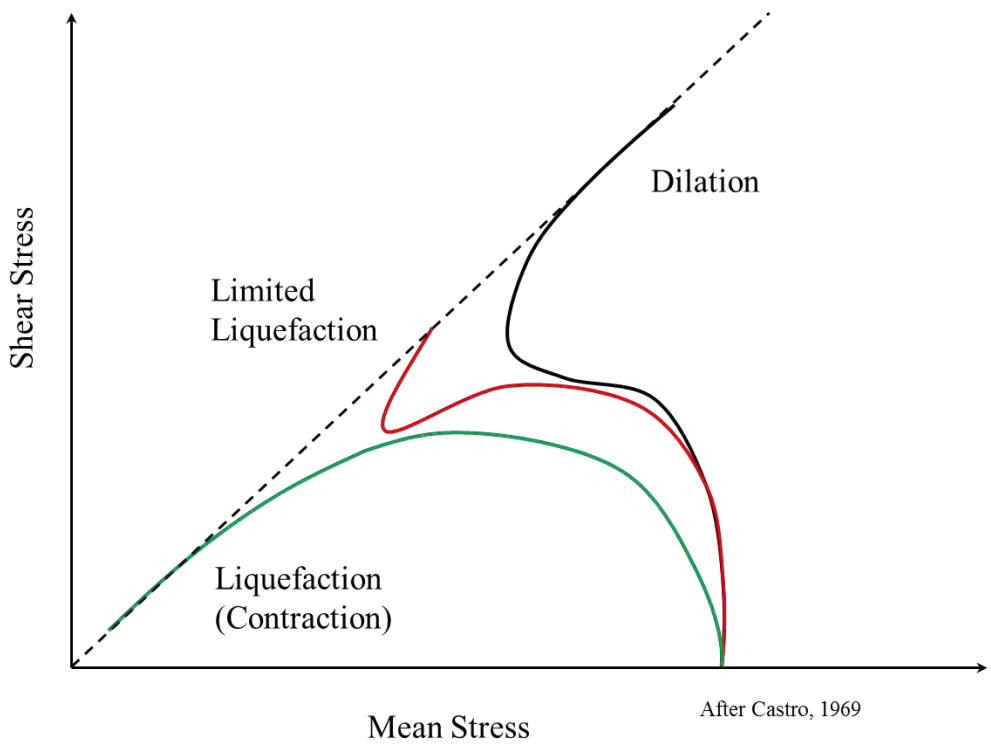

Figure 9: Stress path space plot

As illustrated in Figure 9, the "stress paths" resulting from undrained triaxial compression tests can be categorized in three basic groups: dilation, limited liquefaction, or liquefaction. Dilation is, as described earlier in this paper, volume increase in response to shearing. As the sample shears, the increase in volume prevents generation of high pore pressures, resulting in an increase in a positive trend in the stress path. Liquefaction (full or limited) results from contraction (e.g., volume reduction) of the ore to an extent that the generated pore pressures results in a reduction in shear strength. In the case of limited liquefaction, the volume reduction is reversed by dilation with additional shearing. In full liquefaction, the generated pore pressures are high enough to cause liquefaction. General experience has shown that coarse (e.g., low fines-content) ore tend to have a dilatant behavior under shear, while higher fines-content ores tend to have limited or full liquefaction behavior under shear.

Figure 10 presents results from undrained triaxial compression tests conducted on leach ore with a fines content of over 40%. The test results shown indicate this ore sample has a strong propensity to contractive behavior, resulting in the generation of high pore pressures under shear, which could lead to static liquefaction within the ore heap.

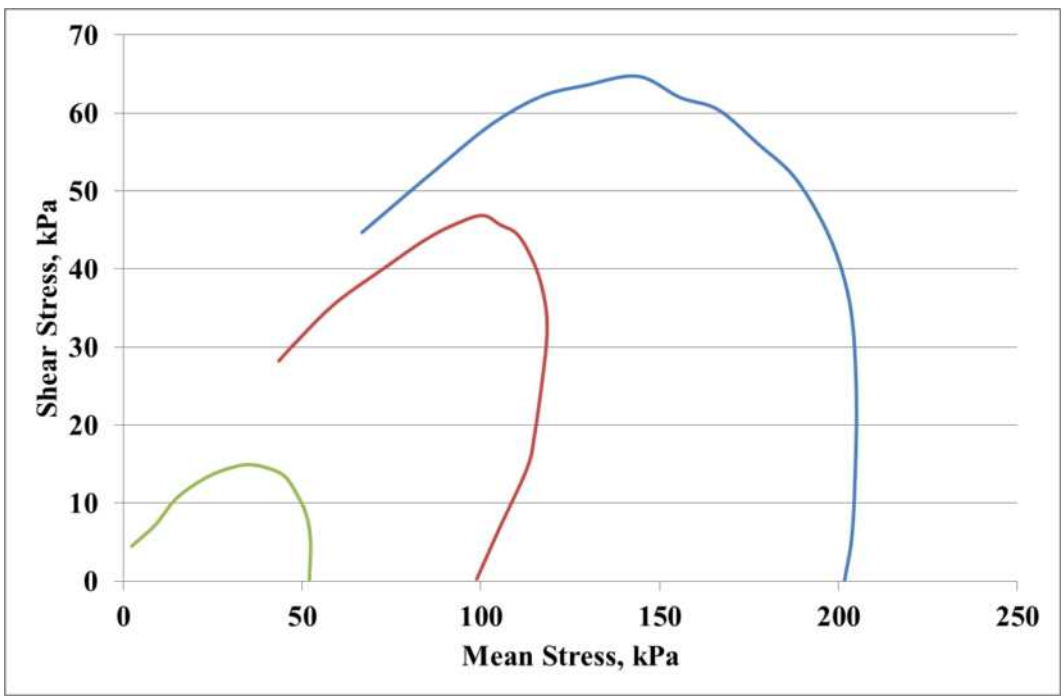

Figure 10: Stress path plot of ore with liquefaction response

The triaxial compression test data from this series of tests indicated the ore has a shear strength defined by a value of cohesion of 1.6 kilopascals (kPa) with an angle of internal friction of 38º. However, if these strength values were used in design, without recognizing the contractive behavior of the ore, there is a possibility the heap would have failed by static liquefaction.

The next section will discuss how to integrate hydro-mechanical considerations for high fines-content ore into heap design.

Hydro-mechanical design considerations

In the previous section, test results from ore with a fines content of over 40% were presented which showed a strong contractive behavior, resulting in liquefaction and loss of shear strength. While the triaxial compression data indicated a shear strength with a cohesion of 1.6 kPa and an angle of internal friction of 38º, these values do not reflect the contractive nature of the ore. In order to integrate the contractive nature of the ore into the heap design, the stress path plot needs to be considered. Figure 11 presents the same plot as Figure 10, but includes additional interpretation. The solid line in Figure 11 represents the failure envelope for the ore sample, with a cohesion of 1.6 kPa and an angle of internal friction of 38º. On each stress path, a horizontal line (dashed) is present at the point the stress path turns horizontal. The tangent point of the stress path to the horizontal line is considered the point of instability (Yamamuro and Lade, 1999). After this point, the stress path continues at decreasing values of stress due to high pore pressures toward the failure envelope.

The dash-dot line in Figure 11 presents a line through the instability points and represents an instability line, beyond which liquefaction develops within the ore. From a leach pad design standpoint, to prevent the possibility of static liquefaction, the state of stress within the ore should be maintained below the instability line. For Figure 11, the instability line could be described with a cohesion value of 0 kPa with an angle of internal friction of 27º, which is considerably lower than the original values of 1.6 kPa and 38º.

Using the instability line for design may require the ore to be stacked in smaller lifts. Additionally, changes in the ore configuration and loading could be made to prevent liquefaction of the ore. For example, increasing the “rest” time between leaching and ore stacking will allow the leached ore to drain, thereby reducing the ore saturation in the heap. Also, it may be possible to stage ore placement and leaching to keep ore with high saturation buttressed by ore with low levels of saturation.

It is also important to recognize that different types could be blended (if available) to reduce the potential for static liquefaction, however, testing should be completed to determine the blend mix.

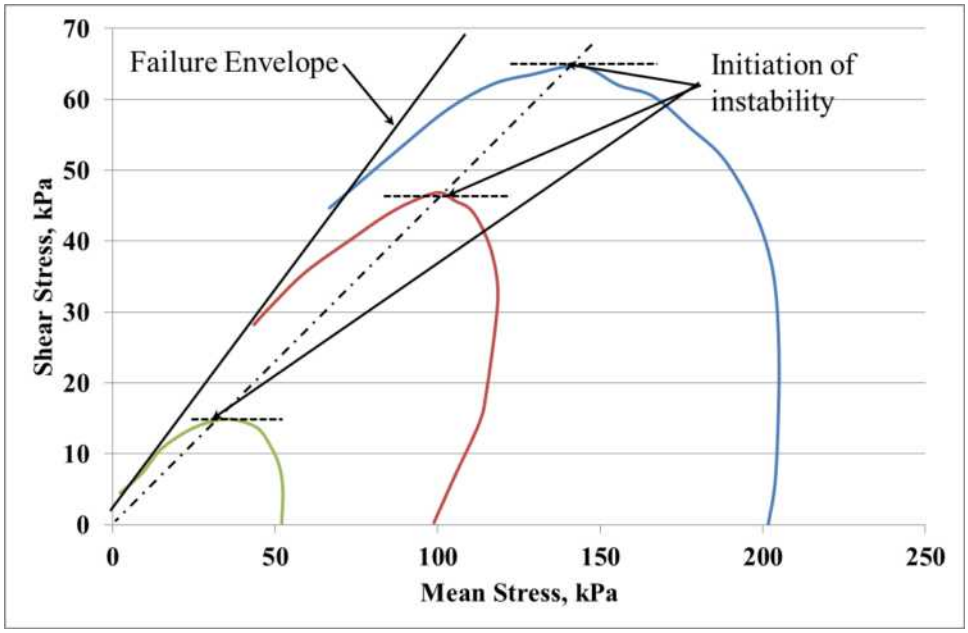

Figure 11: Stress path plot

Conclusion

The objective of this paper is to discuss the importance of understanding the hydro-mechanical behavior of high fines-content ore. As shown in data presented in this paper, the fines content of ore can have a significant influence on the behavior of ore under normal and shear loads and should be considered in leach pad designs.

This paper illustrates how stress paths can be used to identify the static liquefaction potential of ore and how this potential can be addressed in design by using the instability line. Once the static liquefaction potential has been recognized, the heap pad design can be revised to reduce the stress state in critical areas of the heap and reduce ore saturation, if possible.

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Heap leaching treatment of clayey ore at Somaïr

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Abstract

Since 2009, Somaïr has been using heap leach to treat ores that are too low in content for a dynamic process plant. Problems of ore permeability appeared after several months of operation. They were attributed to a change in the nature of the ore, which became much more clayey. To combat this problem, the irrigation flow was reduced and the ore particle size increased. These changes proved insufficient for treating ores with high clay content.

To better understand this problem a thorough testwork program was undertaken. All of the dump deposits to be processed were sampled and characterized. The clays identified consisted mostly of kaolinite, as well as illite or illite/smectite interstratified minerals. These are found essentially in the fine particles (<0µm), and also in the coarser particles (>2 mm). These clays are virtually absent from all of the intermediate sizes. Although non-swelling, these clay minerals (except illite/smectite interstratified) do limit the permeability of the ore.

It was decided to produce ore blends using sandstone with clayey ores, in order to limit the clay content to below 12%. This work was determined from numerous column experiments. The problems of heap clogging have almost disappeared. After exploitation, drill core examination of the heap leach residues confirmed the role of the clays. The uranium content of the residue increases with the clay content. The clayey ore is less well washed than the sandstone ore because the solution circulation through the heap to recover the uranium is less regular.

Background

Société des Mines de l'Aïr has been processing uranium ores in Niger since 1968. Up till 2009, annual uranium production varied between 1,000 tonnes (t) and 2,000 t (Durupt and Blanvillain, 2010). Most of

the production was performed by a dynamic treatment process using acid pugging-curing. This process includes five main stages:

- ore crushing and grinding;
- leaching with sulfuric acid with the pugging-curing method;
- filtering the ore on a belt filter;
- extracting the uranium by solvent;
- precipitation of the uranium by soda in a fluidized bed.

During these years, the ore extracted from the opencast mine was classified into six categories (M1 to M6) according to its uranium grade. The ores M5 and M6, and sometimes M4, were processed using the previously-described acid pugging-curing technique. The other ores were stored between the mine and the plant. Parts of these stockpiled ores were exploited between 1979 and 1989 by dump leaching. The uranium recovery yields were quite low and depended very strongly on the permeability of the ore. By 2006, Somaïr had more than 10 million tonnes of these marginal ores. To increase the plant's production, it was therefore decided that a new treatment unit using heap leach would be built, based on the technological progress achieved in this technique over the previous 20 years.

The heap leach process comprises five parts. The ore is first crushed in three stages to a particle size of −10 mm to −20 mm. It is then agglomerated by adding water and concentrated sulfuric acid, and finally placed onto a heap pad, 40 m wide, 400 m long and 6 m high. Each pad is composed of 24 cells. The ore is then leached using an acid irrigation solution. The liquids produced are recovered in ponds and the uranium is extracted using tertiary amine in a solvent-extraction plant. The advancing uranium flow then joins the dynamic treatment stream and undergoes double-precipitation using soda to eliminate zirconium contaminant. The effluents from the solvent workshop are re-acidified and recycled for heap irrigation.

This heap leach process started operation in 2009. Its annual processing capacity was initially designed for 1,000 kt of ore; this was gradually increased to 1,400 kt of ore during the project phase and then to 1,800 kt of ore in 2012. This has enabled Somaïr to produce a total (dynamic + heap leach) of more than 3,000 t of uranium in 2012.

Methodology

The present study concerns the influence of the clay minerals during heap leach operation. It includes a study on characterization of clays present within the various ores to be processed, then a review of the possible processing routes and finally the associated industrial results.

Characterization of clays

Characterization of clay minerals provides a better understanding of the phenomena which may occur when processing ores. It may explain the problems encountered in the acid pugging-curing process or in the heap leach process. For example, within the dynamic process, with a large quantity of clay present in the ores, the filterability of the leach residue on the belt filters is greatly reduced, consumption of acid also increases and the solids transfer is sometimes disrupted due to the sticky nature of these clays. Moreover, in the heap leach process the permeability of the heaps is significantly reduced.

At Somaïr in the 1970s, geologists determined a relationship between the clay content and the concentrations of aluminum and silicon.

Equation 1: $[clay]\ (\%) = 3.215\ [Al_2O_3]\ (\%) - 1.585\ [SiO_2]\ (\%) + 134.72$

For 40 years, this relationship was sufficient for monitoring the acid pugging-curing process. After 2009, with the beginning of heap leach, it became necessary to increase knowledge concerning the clays present in the ores to be processed. As shown by the column tests and certain phases of exploitation in 2010, the permeability of the ore is directly related to the clay grade.

Some 100 samples had been collected from the various deposits. Each sample was analyzed using X-ray diffraction (XRD), then by X-ray fluorescence (XRF). The XRD analysis was used to identify the main clay minerals present and the XRF analysis was used to try to quantify the major elements and extrapolate the grades of these different clays. This data should serve to relate the clay contents to the results of treatment (column tests, permeability measurements, industrial performance, etc.).

Treatment of clayey ores

The performance of the various processing strategies for the clayey ores was evaluated from measurements taken during industrial exploitation.

Before processing, all of the deposits worked before 2009 were sampled with the following procedure:

- sampling the heap by picking 400 t;
- crushing 400 t to a particle size of −300 mm using the plant's primary crusher;
- dividing the sample into two during transfer to secondary crushing;
- second and third crushing to 20 mm of 200 t on a specific installation (throughput 10 t/h);
- sampling of the output from the third crusher to form a representative sample for characterizing the deposits;
- sampling of the output from the third crusher to form 25 representative drums (220 L) for column tests.

New dumps have been regularly worked since 2009 for ore grades that are too low for the acid pugging-curing process. They are sampled using the same procedure when the ore has passed through the plant.

During exploitation, the heaped ore is sampled on the conveyor belt which transfers the ore from the third crusher to the agglomerator. This sampling determines the grade of the uranium and its clay content, as well as the ore moisture. The ore particle size is also determined from this sampling, by sieving at 10 mm, 5 mm, 2 mm, 0.8 mm, 0.4 mm, 0.16 mm and 0.05 mm.

At the end of exploitation, several cells were sampled by coring. 20 to 30 cores of 50 mm diameter were made on each cell. Average samples were compiled for each cell according to depth. Their moisture content and uranium and clay grades were determined. The average samples were re-washed using diluted acid (H2SO4 10 g/l), then re-pulped in water in order to determine their washing yield and leaching yield.

$$\text{Leaching yield} = 1 - \frac{[U]_{washed\ residues}}{[U]_{ore}}$$

$$\text{Washing yield} = \frac{[U]_{ore} - [U]_{non-washed\ residues}}{[U]_{ore} - [U]_{washed\ residues}}$$

$$\text{Total yield} = 1 - \frac{[U]_{non-washed\ residues}}{[U]_{ore}} = \text{Leaching yield} \times \text{Washing yield}$$

To support exploitation, column leach tests (4.5 m high and 0.28 m in diameter) were carried out with the ore sampled from the dumps prior to processing. The ore was agglomerated in a concrete mixer before being placed into the leach columns. These tests served to optimize the exploitation parameters and also to establish the differences in performance between the dumps (i.e., uranium yield, acid consumption, problems of permeability or limited clay content).

Data and discussion

Treatment of clayey ores

Between 2006 and 2009, tests in columns between 4 m and 6 m high on samples taken from the various dumps demonstrated the crucial role of clays. The columns containing the most clayey ores readily clogged. No such problem was encountered with ores with greater sandstone content. The influence of the clay content of the ore was confirmed during exploitation. During the first months of operation (July to November 2009), the ore used was particularly high in sandstone. No problems of permeability were observed. The first problems appeared in December 2009, when this deposit was exhausted and replaced

by a deposit that was much more clayey. Solution pools appeared on the surface of the heaps and some of the heap edges collapsed.

To process ores with low permeability, such as clayey ores, numerous strategies may be considered. For example, at the Mantoverde copper mine in Chile, poor permeability led to the following modifications:

- reduce the irrigation flow from 15 to 7 l/h/m^2;
- increase the particle size of the ore from 10 to 15 mm;
- reduce the slope of the heaps from 5 to 3%;
- shorten the distance between drains from 2 m to 1 m.

The problems of percolation immediately disappeared (Salgado et al., 2011).

At Somaïr, the first strategy tested consisted of reducing the irrigation flows. This led to an increase in the irrigation time for each cell, therefore increasing the irrigated surface. This was performed on waste heaps stacked at several levels. The irrigation time must therefore be less than the time to turn a level into a heap. The initial design was established for a surface flow of 6 l/h/m^2; this was reduced by the operators to 3 l/h/m^2.

A second strategy that was easy to implement consisted of increasing the size of the third crusher's screen in order to modify the particle size distribution of the ore. The particle size increased from 10 mm to 14 mm, then to 17 mm.

The third strategy tested consisted of making ore blends using material from clayey and sandstone dumps. All of the dumps were sampled and characterized before treatment. Some examples of results obtained are shown in Table 1.

Table 1: Composition of five dumps among all the dumps

Dumps	*SiO_2*	*Al_2O_3*	*Fe_2O_3*	*MnO*	*MgO*	*CaO*	*Na_2O*	*K_2O*	*TiO_2*	P_2O_5
A	87.15	5.95	1.50	<0.03	0.28	0.38	<0.05	1.13	0.28	0.05
B	88.35	5.06	1.43	<0.03	0.35	0.47	<0.05	0.85	0.25	0.05
C	81.25	8.33	2.27	<0.03	0.51	0.29	<0.05	1.42	0.55	0.05
D	84.61	6.08	1.76	<0.03	0.38	0.30	<0.05	1.07	0.38	0.10
E	84.44	6.57	1.82	<0.03	0.40	0.36	<0.05	1.22	0.34	0.08

Results

Characterization of clays

The distribution of the uranium and clays, calculated using Equation 1, was studied according to the ore particle size distribution. A sample of crushed ore collected in 2010 from the industrial heap was sieved.

As shown in Figure 1, the uranium and the clay are found essentially in the fine particles (<50 µm) and the grade of uranium and clays are closely related. These two observations simply demonstrate that uranium is associated with clays, which has long been known. This figure primarily allows the particle size to be related to the nature of the ore. Below 50µm, the ore is almost exclusively with clays. Between 50 µm and 2 mm, it is exclusively with sandstone. Beyond 2 mm, the content in clay and uranium are close to the average ore grades.

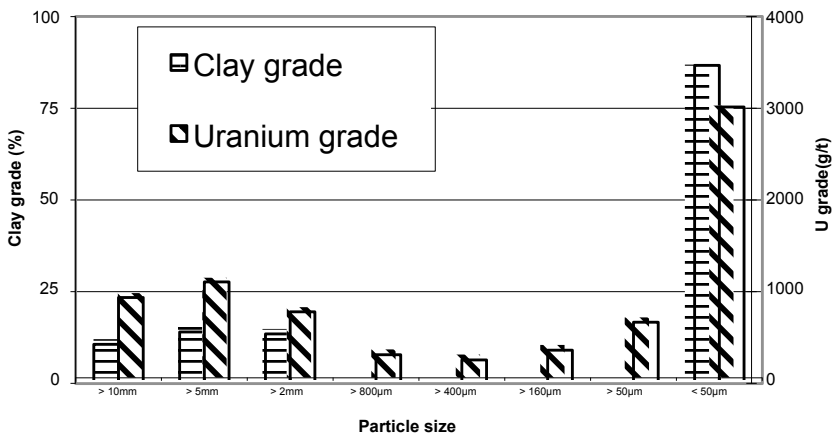

Figure 1: Repartition of uranium and clays in relation to the particle size

The empirical relationship (Equation 1) that estimates the clay content using Al and Si concentrations provides a simple and quick estimate of the total quantity of clays present in the ore. Nevertheless, it has several disadvantages. It remains quite approximate and only provides overall clay content, without differentiating the nature of the different types of clays that compose the ore.

The XRD analysis conducted during this study identified the main clay minerals present within Somaïr ores. Kaolinite was predominant, accompanied by illite, often with illite/smectite interstratified minerals in low quantities.

Kaolinite is a clay mineral of type 1/1 and with the general formula, $Al_2Si_2O_5(OH)_4$. Sheets of kaolinite are formed of two layers:

- an octahedral layer, which contains aluminum;
- a tetrahedral layer, which contains silica.

Illite is a clay mineral of type 2/1 and with the general formula $(K, H_3O)(Al, Mg, Fe)_2(Si, Al)_4O_{10}[(OH)_2,H_2O]$. Sheets of illite are formed of three layers:

- an octahedral layer;
- two tetrahedral layers.

These two clay minerals do not swell. With kaolinite, the space between the sheets is closed with hydrogen bonds. In the case of the illite, the space between the sheets is occupied by K^+ ions with a low degree of hydration, which limits their swelling. Illite/smectite interstratified, on the other hand, is a clay mineral that swells in the presence of water.

The hydraulic conductivity of kaolinite is close to 10^{-8} m/s, while that of the illite is 10^{-11} m/s for apparent densities of 1.6 (Pusch, 2006). Hydraulic conductivity of illite/smectite interstrafied is probably much less than this, thus a small quantity of this clay can considerably reduce the hydraulic conductivity of an ore. These differences in hydraulic conductivity between the clay minerals, and therefore ore permeability, are very high, therefore we cannot rule out the fact that illite/smectite interstratified (present in trace quantities in the ore) might be the origin of the problems encountered.

Treatment of clayey ores

The first operation, which consisted of reducing the irrigation flow from 6 to 3 l/h/m^2, was very positive, but insufficient for the very clayey ores (clay grade greater than 12%). This drop in flow increases the leaching time for the cells.

The second option, which consisted of increasing the ore particle size from −10 to −17 mm, brought significant gains in permeability, but these changes did not enable the treatment of clayey ores (clay grade greater than 12%). They were also accompanied by a drop in the uranium recovery yield. Samples taken from the industrial unit and correlation of the recovery yields of uranium versus particle size distribution enabled the determination of these yield losses. Increasing ore particle size from 10 mm to 20 mm reduced the yield by 0.9%, and with a particle size of 40 mm, these losses were between 2 and 4%.

Finally, the most effective option to avoid problems of permeability consisted of blending ores using the clay and sandstone dumps.

Numerous column tests provided information on the behavior of each deposit (uranium yield, acid consumption, problems of permeability) and led to the establishment of a clay content limit to avoid permeability problems. This limit was fixed at 12%, based on the results displayed in Figure 2. This limit was determined according to Equation 1 from the Al_2O_3 and SiO_2 analyses.

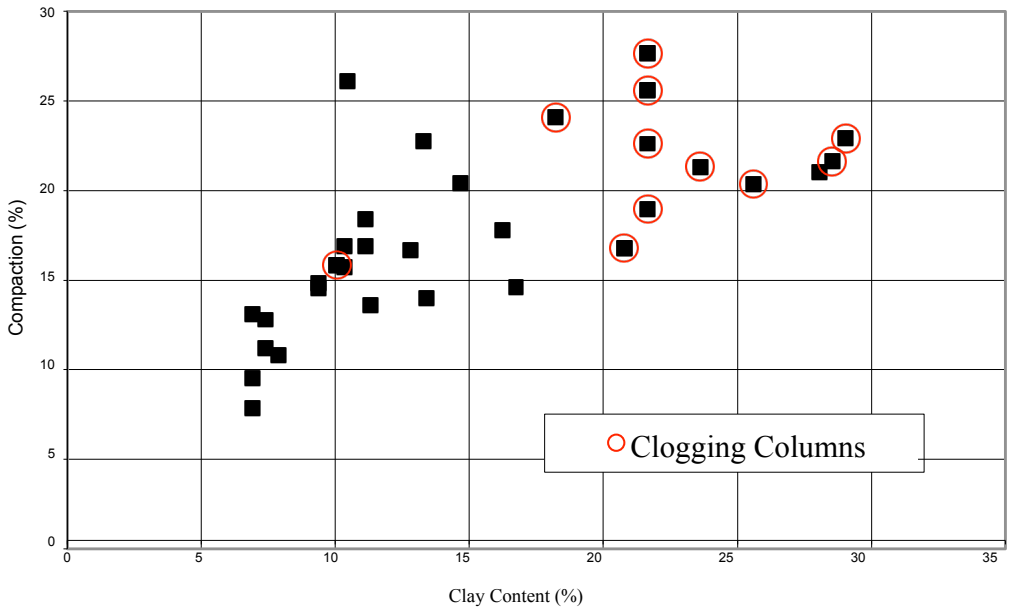

Figure 2: Column compaction in relation to ore clay content

Thus, the ore blending was defined by the clay grade (<12%) and uranium. Since 2011, these recommendations to control the clay grade have been applied by the operators, and problems of permeability have almost disappeared. Only a few pools occasionally remain on the surface, and collapses of heap walls have been eliminated. It is probable that the drop in the irrigation flow and the increase in particle size have also been beneficial, but the bulk of the improvement is due to controlling the clay ore content using ore blending.

With good permeability, the maximum uranium concentration for each cell in the PLS occurs in less than 20 days of cell irrigation. When the clay content increases, the uranium takes longer to come out. It takes longer to reach this maximum value (25 to 50 days) because the permeability of the heaps is poorer.

The analysis of the heap cores after exploitation clearly shows that the highest uranium grades in the residues correspond to the most clayey cells (Figure 3).

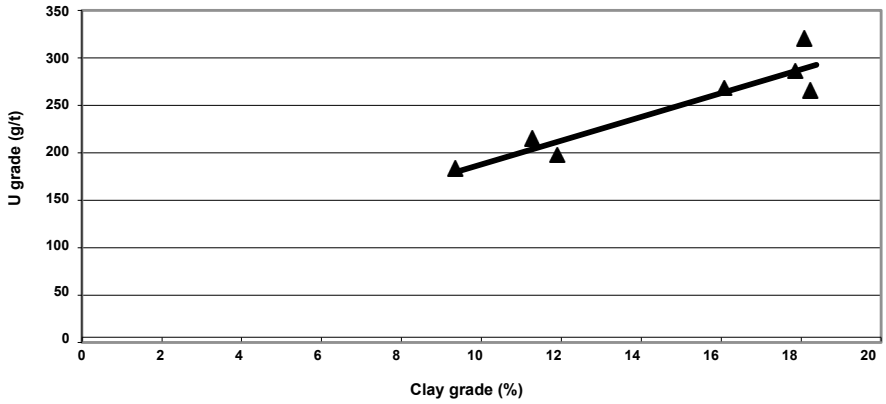

Figure 3: U grade in the residues in relation to ore clay content

The clays consume more acid. The quantities of acid added during agglomeration have not been augmented according to the nature of the ore. It is logical that the clayey ores will have leached less. At the same time, the irrigation flow was reduced in the cells containing clayey ore due to poor permeability. The total irrigation ratio (L/S) was also lower. The ore was less well washed.

Figure 4 shows the differences in behavior between an ore with particularly high sandstone grade (cell A13) and a more clayey ore (cell D7). For the sandstone ore, the grades with and without re-washing of the residues are closer when compared to the clayey ores, which translates as greater washing yields. In both cases, the curves with and without re-washing move closer with the depth of the heap (level 1 corresponds to the top of the heap and level 3 to the bottom of the heap). The circulation of solutions through the heap improves with depth.

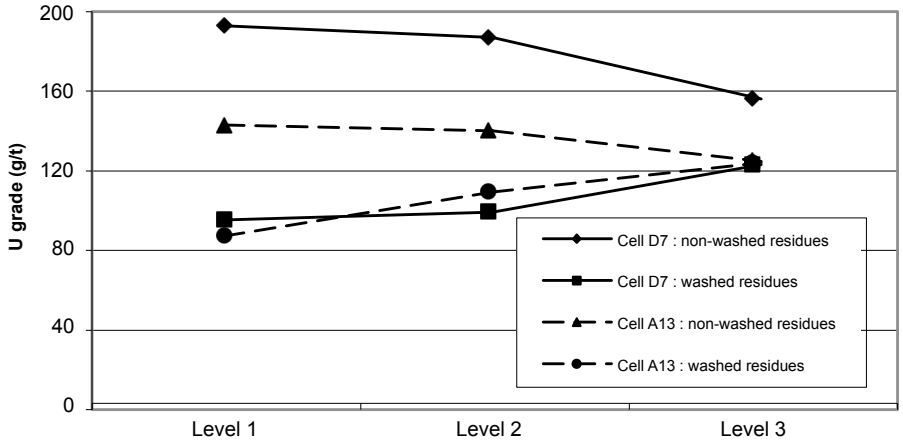

Figure 4: Residues analysis from cells A13 and D7

The processing route, which consists of blending of sandstone and clayey minerals, will nevertheless reach its limit within a few years, when all of the sandstone deposits are exhausted.

Other options that are currently being studied include:

- Addition of a binder may allow stronger agglomerates to be produced. Several binders have already been selected at the laboratory scale. They are in the process of validation at the pilot scale. Essentially, the cost of the binder will have to be compared with the other processing methods and uranium losses caused by poor permeability.
- Reduction of heap heights can also increase the permeability of the heaps. However, to sufficiently improve this parameter, the heap must be reduced by at least half. With the current "waste heap" functioning, this implies doubling the surface of the leach areas and thereby increasing the construction costs.
- Elimination of fines after crushing can also considerably improve the permeability of the ore. This option would require a thorough review to determine the ore cut-off point for the heap leach and acid pugging-curing processes. Currently this division is done according to the uranium ore grade. A future approach could use particle size, where crushed ores (coarser particles) would be treated by heap leach and the fines treated by an acid pugging-curing process.

Conclusions

The problems of permeability encountered at the Somaïr heap leach operation are directly related to the clay grade. They manifest themselves primarily by the appearance of pools on the surface of heaps and the collapse of the heap edges. The consequences are as follows:

- Delays in the uranium production because the solutions take longer to circulate through the heaps and the irrigation flows must be reduced.
- A drop in uranium recovery yield because the clay consumes more acid and the ore is less thoroughly washed by the irrigation solutions.

To resolve these permeability problems, the irrigation flows were reduced from 6 to 3 $l/h/m^2$ and the particle size was increased from −10 mm to −17 mm. However, the most effective measure consisted of limiting the clay content of the ore to 12% by ore blending. To define this limit, all of the dumps were characterized and column tested. The ores consist of different clay minerals (mainly kaolinite and illite, but also illite/smectite in small quantities) and the 12% limit is based on the total quantity of clays. Additional tests are warranted to determine the role of each of the specific clay minerals.

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Linear low density polyethylene geomembranes for cold climate mining applications

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Abstract

Linear low density polyethylene (LLDPE) geomembranes offer superior engineering properties and installation advantages over high density polyethylene (HDPE) geomembranes in mining sector applications customarily characterized by large overburdens and locally subsiding and/or acid-dissolving subgrades. Used in conjunction with natural low-permeability soil under-liners, they constitute the most secure and economical hydraulic barriers for the protection of underlying water tables from industrial activities.

Introduction

Mineral processing from mining activities is an industrial sector that is especially environmentally sensitive on account of the large volumes and nature of the mineral elements and compounds, as well as the processing agents used for metallurgical extraction such as hydrometallurgy (heap leaching). Mining pollutant sources are numerous and span most of the traditional mining practices, from liberalization to metal extraction, to waste disposal. Grinding for instance, which is usually performed in the presence of water, generates tailings and mine waters, as well as downstream separation processes such as distillation, magnetic separation, electrometallurgy (electrowinning), gravity separation, flotation and selective dissolution. Mine waters may therefore contain residual mineral compounds and hydrometallurgical reagents, as well as oxidants such as propylene glycol, aliphatic alcohol, and cresylic acid (from flotation processes), lime, calcite, soda ash, sodium hydroxide, ammonia, sulfuric nitric, hydrochloric acids (e.g. modifiers), copper sulfate, lead nitrate and kerosene (e.g. activators and depressants), metal hydroxides (e.g. flocculants), ferric and ferrous sulfides, aluminum sulfide and ferric chloride (e.g. coagulants), sulfuric acid and sodium cyanide (from hydrometallurgy), hydrogen peroxide, sodium hypochlorite and potassium permanganate (e.g. oxidants). Therefore in general, mine waters will often require environmentally secured large storage and treatment retention ponds prior to their final discharge,

involving the use of engineered containment solutions using reliable hydraulic barriers in order to prevent the contamination of underlying water table from contaminant percolation. Although natural soil hydraulic barriers such as compacted cohesive soil layers (clayey soils) have historically been extensively used, state-of-the-art composite hydraulic barriers involving the use of synthetic liners (geomembranes) in intimate contact with natural low-permeability soils are now preferably endorsed by most regulators as they offer enhanced environmental protection.

Polyethylene geomembranes

Geomembranes are impervious sheets of polymeric material primarily used to reduce the hydraulic conductivity of in situ soils. They represent a branch of engineering materials called geosynthetics which are specifically designed to enhance the technical properties of natural soils which have been deemed deficient from an engineering point of view. Geomembranes are thus widely used to provide additional imperviousness to natural soils, so as to protect water tables against contamination from industrial activities such as mining.

Geomembranes come in a variety of polymers. Polyethylene is the most common because of its inherent mechanical and endurance properties, including chemical resistance to a wide range of mining contaminants and natural oxidation processes, which causes its renowned longevity. Since their inception in the early 1980s polyethylene geomembranes have become the material of choice of every industrialized nation when it comes to mining containment applications. Their rapid growth and industry wide acceptance through worldwide governmental endorsements and regulations are a result of their many advantages when compared to more traditional waterproofing materials. As opposed to previous unlined facilities and associated dismal containment properties, modern engineered mining waste enclosures with polyethylene geomembranes are now the safest and most economical containment method.

Polyethylene geomembrane generalities

Out of the many types of hydraulic barriers available today, polyethylene geomembranes are usually the first choice for mine designers as they are both highly impervious and chemically resistant to a very wide range of contaminants. Polyethylene geomembranes are hence rot-proof and designed to withstand long term use in both buried and exposed conditions. Furthermore, as opposed to natural soils which require specific water contents to achieve proper hydraulic barrier status, polyethylene geomembranes are inherently impervious regardless of their environment, and are unharmed by freeze-thaw cycles and extreme freezing temperatures. They also constitute great space savers because of their essentially bi-dimensional structure, achieving higher levels of imperviousness in less than a few millimeters than meters thick of natural clay.

Polyethylene geomembranes also offer excellent engineering properties due to their high elasticity and overall outstanding mechanical resistance under tensile, tear and puncture modes, making them especially reliable when used in seismic zones. And since they are manufactured under controlled conditions, all material properties of polyethylene geomembranes, including waterproofing, are also much more isotropic than natural soils. Moreover, they can be custom manufactured to meet specific friction requirements, thereby insuring heap leaching pad and tailings dam stability. The use of polyethylene geomembranes also offers the added benefit that every phase of their manufacturing and installation processes can fully be quantified and documented, enabling fast and accurate conformance verification to project technical requirements for both quality assurance and permitting.

LLDPE generalities

Although HDPE geomembranes usually constitute the material of choice when it comes to environmental protection structures, LLDPE is a polymeric product with many advantageous engineering properties that are beneficial within the mining sector, especially in cold climate applications. For instance, LLDPE geomembranes are now often preferred for heap leaching pads due to their enhanced elastic properties, which better accommodate typical mining subgrades and overburdens.

LLDPE differs from its HDPE counterpart in many ways, e.g. elongation at break, tear resistance and puncture resistance, but it makes up for this with its unequaled deformability from its low modulus (Geosynthetic Research Institute GRI-GM17, 2012, and GRI-GM13, 2012). As its name implies it is a substantially linear polymer, although braced with a significant amount of short side branches produced by a copolymerization process of ethylene and alpha-olefins, namely hexene. The comonomer side-branches keep the linear polymer chains from packing too closely together, thereby resulting in overall lower density. LLDPE should not be confused with Low Density Polyethylene (LDPE) with characteristically long chain branching and narrower, yet higher molecular weight distribution, yielding lower tensile, impact and puncture resistances as well as limited flexibility. LDPE is usually not considered as a viable contender for geomembrane intent. Originally introduced in the mid-seventies as a rotomolding resin, it now has well over 50% of the world's lower density type polyethylene market. The geosynthetic industry was not left behind, as it began to introduce LLDPE geomembranes as early as the early nineties, culminating in its consecration by the adoption of the first industry standard, the GRI-GM17 (Geosynthetic Research Institute, 2012) in 2000.

LLDPE elasticity

Polyethylene is a semi-crystalline thermoplastic polymer composed of both crystalline and amorphous molecular phases. Semi-crystallinity is a desirable property for most plastics because it combines the

mechanical strength of crystalline polymers with the flexibility of amorphous ones, such as flexible polyvinyl chloride (PVC). Semi-crystalline polymers can be tough, with an ability to bend without breaking. Polymer crystallinity can be determined with Differential Scanning Calorimetry (DSC) by quantifying the heat associated with the melting temperature of the polymer. As LLDPE contains less crystalline molecular phase (≈ 10%) than HDPE (≈ 90%), its density and melting point are lower while its elasticity is enhanced (Islam et al., 2011), making it a much more ductile material highlighted by superior multi-axial deformation of 90% and above (American Society for Testing and Materials [ASTM] D 5617, 2010). Elasticity is often a prime factor in heap leach pads and tailings dam design to prevent the polyethylene geomembranes from extending beyond their mechanical yield point and into permanent deformation from the customary sizeable overburden pressures, coupled with uneven or local subsiding subgrades and/or coarse and angular tailings.

LLDPE and stress cracking

Stress cracking is a visually brittle failure that occurs at a constant stress lower than the yield stress of the material itself. It is a phenomenon to which all HDPE geomembranes are susceptible. Commercially available HDPE geomembranes show a range of stress cracking resistances that vary by a factor of 10, while in all other respects these HDPEs have very similar mechanical properties. As in many materials, a gain in one performance characteristic is often countered by a loss in another; the high crystallinity that gives HDPE their excellent mechanical strength characteristics are also responsible for their susceptibility to stress crack.

Clearly, without stress there can be no stress cracking. Thus, all liners should be designed and installed to perform solely as a barrier and not to contain stress. However this is not practical in a construction environment, especially under cold climate conditions. Stress cracking often originates from inadequate welding procedures, such as thermal shock and excessive pre-welding surface preparation. Under those conditions, stress cracking may still occur in spite of the materials meeting industry standard stress cracking resistance (ASTM D 5397, 2012).

Figure 1: Stress-cracking of HDPE from improper welding procedures

Thermal shock occurs when a thermal gradient causes different parts of an object to expand by different amounts. This differential expansion can be understood in terms of stress. At some point, this stress can exceed the strength of the material, causing a minute initial crack to form. If nothing stops this crack from spreading through the material, it will cause the object's structure to fail. Thermal shock is most likely to occur when welding polyethylene geomembranes in cold temperatures. As the cold geomembrane is exposed to rapid heat transfer, its crystalline phase percentage may actually increase from molecular rearrangement, causing fragility. Therefore, every attempt should be made to select a polyethylene geomembrane made from a resin that has sufficient stress cracking resistance to tolerate the damage induced during deployment, overheating during welding, and installation stresses that unavoidably occur.

Due to its high amorphous phase percentage, LLDPE does not crack under stress, even under the worst installation conditions and cold environments (as evidenced by the industry-standard GRI-GM17 which does not even require testing to that effect). Since LLDPE has a lower density and melting point than HDPE, LLDPE geomembranes also require less heat transfer for their welded assembly. This enables more expedient assemblies at low ambient temperatures without stress cracking concerns.

LLDPE chemical and UV resistances

HDPE is known to have excellent chemical resistance to a wide range of contaminants, especially to sulfuric acid and sodium cyanide which are typically used in heap leaching processes. Although LLDPE has a lower density than HDPE, which from the onset should reduce its chemical resistance to polymer oxidation and cross-linking, LLDPE geomembrane densities (0.939 g/cc) are by industry decree very

close to their HDPE counterpart (0.940 g/cc). Thus, LLDPE geomembranes may still be used in lieu of HDPE geomembranes in most mining applications without detrimental accelerated chemical ageing, as markedly exemplified by several LLDPE geomembrane heap leaching designs notably in South America.

While the chemical resistances of LLDPE and HDPE geomembranes are closely matched, their resistances to UV exposure from solar irradiation are not. Although most mining applications call for geomembranes to be used under buried conditions, other ancillary containment structures such as barren and pregnant solution ponds, acidic mine drainage holding ponds and waste water reservoirs usually call for exposed conditions. High energy UV solar radiation is detrimental to polymers as it elicits molecular chain scission and ensuing photo-degradation, hence the use of formulated additives such as carbon black and light stabilizers.

But photo-degradation is also linked to polymer density and as such LLDPE is somewhat disadvantaged. A study by the Geosynthetic Research Institute (Geosynthetic Research Institute, 2012) establishes the half-life prediction of LLDPE at 33 years for a GRI-GM17 specified 1, 0 mm geomembrane under an annual solar irradiation of 160 kLy (160 kcal/cm^2), corresponding to a Southern California/West Texas exposure, where the half-life is defined as the time period required for the material to have lost 50% of its original properties. However, photo-degradation is also inversely proportional to material thickness (Rowe et al., 2010). Considering that 1.5 mm thickness geomembranes and beyond are emblematic to the mining industry, half-lives of 50 years are usually anticipated.

White LLDPE geomembranes

Both HDPE and LLDPE geomembranes exhibit very high coefficients of thermal expansion, which among other issues may hinder their site installation, even in cold climates as deployed sheet temperature rapidly rises with solar irradiation. When light strikes the surface of a non-metallic material it bounces off in all directions due to multiple reflections by the microscopic irregularities of its surface, constituting diffuse reflection. The color of opaque objects is determined by which wavelengths of light they scatter strongly (with the light that is not scattered being absorbed). If objects scatter all wavelengths with roughly equal strength, they appear white. If they absorb all wavelengths, they appear black. Black polyethylene geomembranes hence heat up from the absorbed solar energy, yielding thermal expansion in the form of so-called expansion waves.

The use of white LLDPE geomembranes offers many benefits in that regard, as its core temperature gradient is reduced by up to 50%. Expansion waves are reduced accordingly, facilitating their site assembly and maximizing intimate contact with the underlying substrate, a key to enhanced impermeability.

Figure 2: Expansion waves in black polyethylene geomembranes (from Rowe, 2011)

Figure 3: White polyethylene geomembrane with reduced thermal expansion waves

White LLDPE geomembranes are manufactured from a multi-extrusion process with a thin coalesced surface layer (≈ 0,2 mm) of specially formulated UV resistant white polyethylene resin of titanium dioxide and hindered amines, while the geomembrane's underlying core remains black, which does not affect any material properties of the original geomembrane. Hindered amines are chemical compounds containing an amine functional group surrounded by a crowded steric environment. They are

extremely efficient stabilizers against light-induced degradation of polymers. A great advantage of hindered amine light stabilizers is that no specific layer thickness or concentration needs to be reached to guarantee excellent results.

Reduction of thermal expansion also reduces habitual geomembrane toe of slope trampolining and facilitates backfilling operations by minimizing vehicular damages. Backfilling operations may therefore be performed at any time of the day. By the same token white geomembranes enable closer adherence to minimum designed cover layer thicknesses. White geomembranes are also proven to reduce subgrade desiccation as well as permafrost surface thaw due to their lower heat sink capacity. Due to their white surface they also enable easier visual inspection for defects by exposing the black underlying material. Additionally, on account of their lower inner core temperatures, white geomembranes also enable the deceleration of their anti-oxidants depletion rates, thereby prolonging their half-lives. Depletion rates of anti-oxidants are proportional to their in-use temperature, as can be seen in Figure 4. White geomembranes are also beneficial for evaporation mining processes as they redirect reflected solar irradiation from the perimeter slopes towards the upper surface of the contained brines, thereby enhancing the vaporization process.

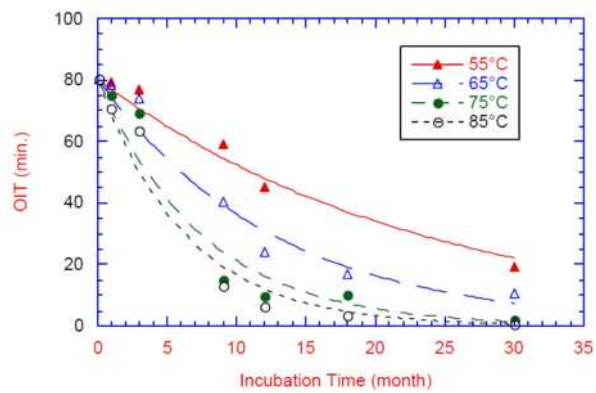

Figure 4: Anti-oxidant depletion rates at different temperatures (from Geosynthetic Research Institute Report #16, 1995)

Rough textured LLDPE geomembranes

Mining applications often require special design considerations for the stability of unusually large fill structures brought about by the interface shear strength (e.g. friction angle) of the geomembrane in contact with the underlying and overlying soils (or tailings), as translational (lateral movement) wedge slip failures generally occur along those interfaces. Therefore, it is customary to properly evaluate the shear strength of these interfaces with approved industry methods such as ASTM D5321 (ASTM D 5321, 2012), for both peak and residual friction angles (especially called for by seismic considerations as mining deposits usually coincide with orogenesis zones).

In addition to conventional textured polyethylene geomembranes with characteristic peak friction angles of 20° to 25°, LLDPE geomembranes also come in a wide range of enhanced friction property products with high asperity heights (0.7 mm in lieu of industry-standard 0.25 mm) with corresponding peak friction angles ranging as high as 35°, accommodating most non-cohesive and cohesive soil nature and conditions (Blond and Élie, 2006). As for valley-fill heap leach pads, enhanced friction LLDPE geomembranes are usually considered in order to prevent stability issues on steep slopes.

Electrically conductive LLDPE geomembranes

Numerous documented tailing dams failures have been linked to seepage and ensuing dissolution and erosion across their basal, slope and toe berm's natural soil hydraulic barriers. Therefore it is very advantageous to use electrically conductive polyethylene geomembranes that enable complete geo-electrical leak surveys both during and after construction with all industry-approved methods: puddle method, water lance method, water covered method, and soil covered method (ASTM D 7007, 2009, D 7002, 2010, D 7240, 2011, D 6747, 2012, D 7852, 2013). The greatest advantage in using electrically conductive geomembranes as opposed to other electrically conductive mediums such as electrically conductive subgrades, electrically conductive geotextiles and geosynthetic clay liners (GCL), is that intimate contact between the geomembranes and the underlying mediums is not necessary since the geomembrane itself is actually carrying the electrode on its underside by way of a thin co-extruded electrically conductive carbon black layer (≈ 0,1 mm) which does not affect the geomembrane's original material properties.

Carbon black is a material produced from the incomplete combustion of heavy petroleum. The main parameters influencing the final conductivity of carbon black are its structure and aggregate chaining, particle size, purity and loading level.

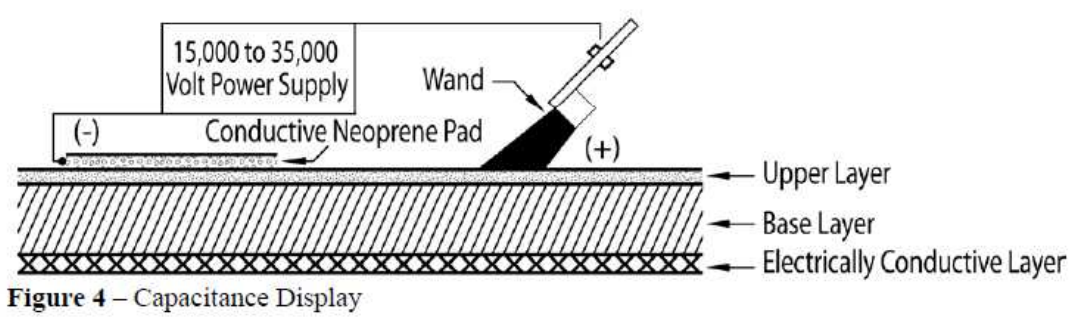

Figure 5: Geo-electrical leak detection schematic (from ASTM D 7852, 2013)

Figure 6: Geoelectrical leak detection apparatus

Additional considerations

Because of their lower melting point and heat transfer demand, site assemblies of LLDPE geomembranes are accelerated by having their welding equipment speed increased by up to 50% (Solmax, 2012). Due to this, and to their stress-crack resistance, LLDPE geomembranes are often considered to be a fail-proof material for construction due to their wider window of welding parameters. This is especially beneficial in cold weather seaming conditions. As opposed to HDPE geomembranes, the increased flexibility of LLDPE geomembranes enables easier handling and fitting around acute rigid appurtenances, and allows shop pre-fabrication and accordion folding on thinner gauges without permanent damage to the material.

LLDPE geomembranes may also be easily assembled to HDPE geomembranes using industry-standard welding protocols. Under those circumstances, LLDPE's mechanical properties should govern acceptable weld testing passing values, in similar fashion to welding two different thicknesses of the same polymer whereby the thinner one's mechanical properties govern acceptable weld testing passing values (Geosynthetic Research Institute, 2011).

Conclusion

LLDPE geomembranes are characterized by their elevated material elasticity, which makes them a material of choice in mining applications such as heap leaching and tailings dump, where site conditions are often associated with large overburdens of angular fills and locally subsiding and/or acid dissolving subgrades. LLDPE geomembranes may in addition be laden with additional specialty features such as light reflecting properties, enhanced interface friction and electrically conductive attributes, enabling improved engineering designs and more expedient and higher quality site installations.

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Heap leach pipe design, specification, and installation

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Abstract

Heap leaching is an industrial mining method used to extract precious metals, copper, nickel, uranium zinc and lead compounds from ore. Heap leaching has been used for centuries to extract metals from their ores. Gold, silver, copper, nickel, uranium, zinc, and lead are all extracted using some variation of this method. The oldest continuously active example is the extraction of copper at Rio Tinto, Spain, which has been in operation since 1752. There are a number of variants of the process, and the design of the heap leaching pads, that are dependent on the ore being processed and the topography of the mine site.

The pipe used to collect the leachate from the pile is a significant component in determining the performance of the process. Different heap leach pipe designs place quite different demands on the collection pipe in terms of loads, temperatures, and chemical environments. The proper specification of these pipes will have an impact on the system efficiency and the pipe durability.

Introduction

Heap leaching entails placing crushed ore, either crushed and graded, or run-of-mine material, on an impermeable barrier upon which a collection system is placed, usually consisting of 100 mm to 150 mm diameter perforated pipe, placed in a graded aggregate drainage course. The ore is then placed above this collection system. A leaching solution, selected based on the metal being extracted, is then trickled through the heap leach pile to dissolve the desired metal, and the resulting leachate is then collected by the collection system and transported to further processing. Gold and silver are extracted using sodium cyanide or potassium cyanide solutions with a pH of about 11. Copper, nickel, and uranium are extracted using dilute sulfuric acid.

Table 1: Metals heap leached

Metal	Leachate
Copper	Sulfuric Acid – H_2SO_4
Gold	Sodium Cyanide – NaCN Potassium Cyanide – KCN
Nickel	Sulfuric Acid – H_2SO_4
Silver	Sodium Cyanide – NaCN Potassium Cyanide – KCN
Uranium	Sulfuric Acid – H_2SO_4

Basically, there are four types of heap leach pads; "flat" pads, dump leach pads, valley fills and on/off pads. Each type places different loads on the collection system pipe.

Conventional or flat pads are placed on relatively smooth terrain, with lifts of from 5 m to 20 m. The effective loads on the pipe, if placed in a drainage aggregated layer, are mitigated by the arching of the gravel and the ore above it. This vertical arching factor (VAF) can result in a reduction of the theoretical column load on the pipe of as much as 80%. It has also been found that in cases of multiple lifts placed after the leaching of the first lift, there is little or no change in the pipe shape or deflection.

Figure 1: Typical collection system for conventional heap leach pile

Dump leach pads can include flat or rolling terrain, and the ore is typically run-of-mine. Lift thicknesses are often greater, as thick as 50 m. Loads on the leachate collection system can be somewhat greater and more irregular in the run of pipe.

Valley fills consist of filling a valley behind some form of impoundment dam. These valleys may be gently sloped, or can be quite steep. Loading on the pipe can be very high, especially at the bottom of the valley, with little arching occurring, and in some cases loads created are greater than the theoretical column load over the pipe. The collection pipe at the bottom of the valley is often much larger in diameter, with 600 mm being common. The total fill heights in these designs can be 300 m or greater. Installation of the pipe to minimize the effects of the increased load is critical to the long term performance of the collection pipe. Placing the pipe in a trench and backfilling carefully around and over it can mitigate some of these loads.

Figure 2: Valley fill

On/off pads are relatively flat pads built over a sturdy liner and overliner system. Normally, a single lift of ore, from 4 m to 20 m thick, is placed and leached. When that ore is spent at the end of the leaching process it is removed and the pad is reloaded with new ore. The highest loads on the collection system are not from the ore, but from the equipment used to place the ore, such as trucks and conveyor systems.

Figure 3: On/off pad

In all cases the ore should be as uniformly permeable as possible or practical. The leachate solution will find the easiest pathway through the heap leach pile, often by-passing sections of the pile. Also, as much as possible, the ore should have a low percentage of fines, defined as material passing number 100 sieve. Table 1 presents the effect of fines on permeability based on Cedergren (1989).

Table 2: Influence of percent of minus 100 fraction on permeability

Percentage passing number 100 sieve	Permeability, m/day
0	25 to 100
2	3 to 30
4	0.6 to 15
6	0.15 to 6
7	0.06 to 1

Fines can also migrate with the flow, agglomerating and plugging drainage pathways.

Low permeability can also create an increase in the phreatic level in the pile, which can create pipe instability, increased fines migration potential, and system failure. In Figure 4, the ore, with >12% passing number 100 sieve, had a permeability of <12 cm per day. The mine operator, in order to increase flow from his collection system, increased the phreatic surface until the pile became unstable and failed.

Figure 4: Heap leach pile failure: >12% passing number 100 sieve

The process of heap leaching can, and usually does, result in a breakdown of the ore into finer pieces, which can lead to heavier unit weights and self-loading consolidation, and increasing the percentage of fine material and decreasing permeability during operation of the process. A study conducted on a South American copper ore at Ohio University demonstrated that the leachate alone caused a 9% vertical consolidation of the ore. Whether this is typical of all ores is unknown, but the consolidation could be somewhat greater or much less. There are very limited test reports available on this subject, but it might be recommended that similar testing be done prior to any heap leaching process is started.

Gold and silver

Crushed ore is placed in the heap leach pad and irrigated with sodium cyanide or potassium cyanide. The leachate is collected by the pipe system at the bottom of the pile and drained to a holding pond or tank. After removing the metal from the solution, the cyanide solution is typically reused in the heap leaching process.

Copper

Roughly 40% of all copper comes from heap leaching operations. Heap leaching of copper was done commercially in Spain in 1752. All of the four heap leach pile types listed in this paper are used at various mines throughout the world. Some forms of copper ore react well with sulfuric acid and are easily leached. Sulfide and pyritic ores, however, require some form of biologic leaching. These heap leach piles

perform best if air is added to the pile. These biological reactions are exothermic and can produce temperatures in the heap leach pile in excess of 40°C. This creates additional issues for the leachate collection systems, including loss of antioxidants and softening of the resins. The aeration pipe is installed in the ore, which is effected by the leaching solution, and the pipe will deflect as much as the ore consolidates vertically.

Nickel

Heap leaching of nickel is also done utilizing sulfuric acid. The quantity of acid required is significantly greater than needed for extracting copper. The final product from heap leaching can include nickel hydroxide precipitates or metal hydroxide precipitates that are then refined by conventional smelting to produce metallic nickel.

Uranium

Heap leaching of uranium ores is practiced in Australia and Africa, and is similar to copper heap leaching. The final product is yellowcake. The compositions were variable and depended upon the leachate. Among the compounds identified in yellowcakes include: uranyl hydroxide, uranyl sulfate, sodium para-uranate, and uranyl peroxide, along with various uranium oxides. Yellowcake typically contains 70 to 90% triuranium octoxide (U_3O_8) by weight. Other oxides such as uranium dioxide (UO_2) and uranium trioxide (UO_3) may be present. Yellowcake is used in the preparation of uranium fuel for nuclear reactors.

Heap leaching pipe design

In heap leaching applications, the environment is considered quite aggressive chemically, thermally, and structurally. To get the best possible performance from the pipe leachate collection systems, the pipe specifications must assure a product that can withstand those extreme conditions. Where needed, installation requirements must be modified to assist the pipe structurally. The differences from application to application (i.e., leachate collection versus aeration pipe) must be understood and designs must accommodate those differences.

Pipe supplied to a project must be tested and certified by an accredited independent testing laboratory as meeting the specification requirements. There are two current ASTM specifications that specifically address these mining applications, ASTM Designation: F2986 − 12 Standard Specification for Corrugated Polyethylene Pipe and Fittings for Mine Leachate Applications and ASTM Designation: F2987 – 12 Standard Specification for Corrugated Polyethylene Pipe and Fittings for Mine Heap Leach Aeration Applications. Compared to previous standards used by the mining industry that were primarily intended for land drainage applications, these two standards require:

1. high quality virgin resins;
2. higher anti-oxidant levels (Oxidation Induction Time – OIT);
3. tighter UV protection levels;
4. higher stress crack resistance;
5. options for leak resistant joints; and
6. special perforation designs for aeration pipe.

Because of the very high compressive and bending strains possible in some of these applications, a virgin resin with high stress crack resistance in needed; not so much because of the very high stresses, which are primarily compressive, but because of the potentially high long-term strains. A lesser quality resin, or a recycled resin, will have much greater potential for cracking due to resin quality, and, in the case of recycled material, contamination creating stress raisers in the pipe wall.

The higher anti-oxidant requirement is needed to address the potential elevated temperatures in many applications and the effect of oxidizing acids. Again, recycled resins will not provide this protection, potentially leading to premature cracking.

The tighter UV protection requirements are designed for protection in many of the mine environments that are at elevations above 3,000 m in arid environments. Without appropriate UV protection, embrittlement can occur.

The pipe joint requirements included in both standards provide an option for highly leak resistant joints. These are necessary for the aeration application, where joint leakage might not permit air to reach all of the pile. In the collection standard, it provides an option for utilizing the pipe for leachate transmission from the heap leach pipe to the holding pond or tank.

The standard drainage pipe configuration of perforations works well for leachate collection pipe. The aeration pipe perforations must be designed to deliver air uniformly throughout the pipe. Those perforations must not permit leachate to enter the aeration system and must be protected from plugging by the ore. This is covered in the standard. Design of the aeration systems involves sizing the pipe properly to deliver air throughout the system, and sizing the perforations using the orifice formulae to maintain uniform distribution through the heap leach pile. The drawing from ASTM F2987 shows the perforation locations and the protective cover. The number and diameter of the perforations will vary throughout the pipe run, with the fewest perforations nearest the edges of the pipe and the largest open area in the center of the pile.

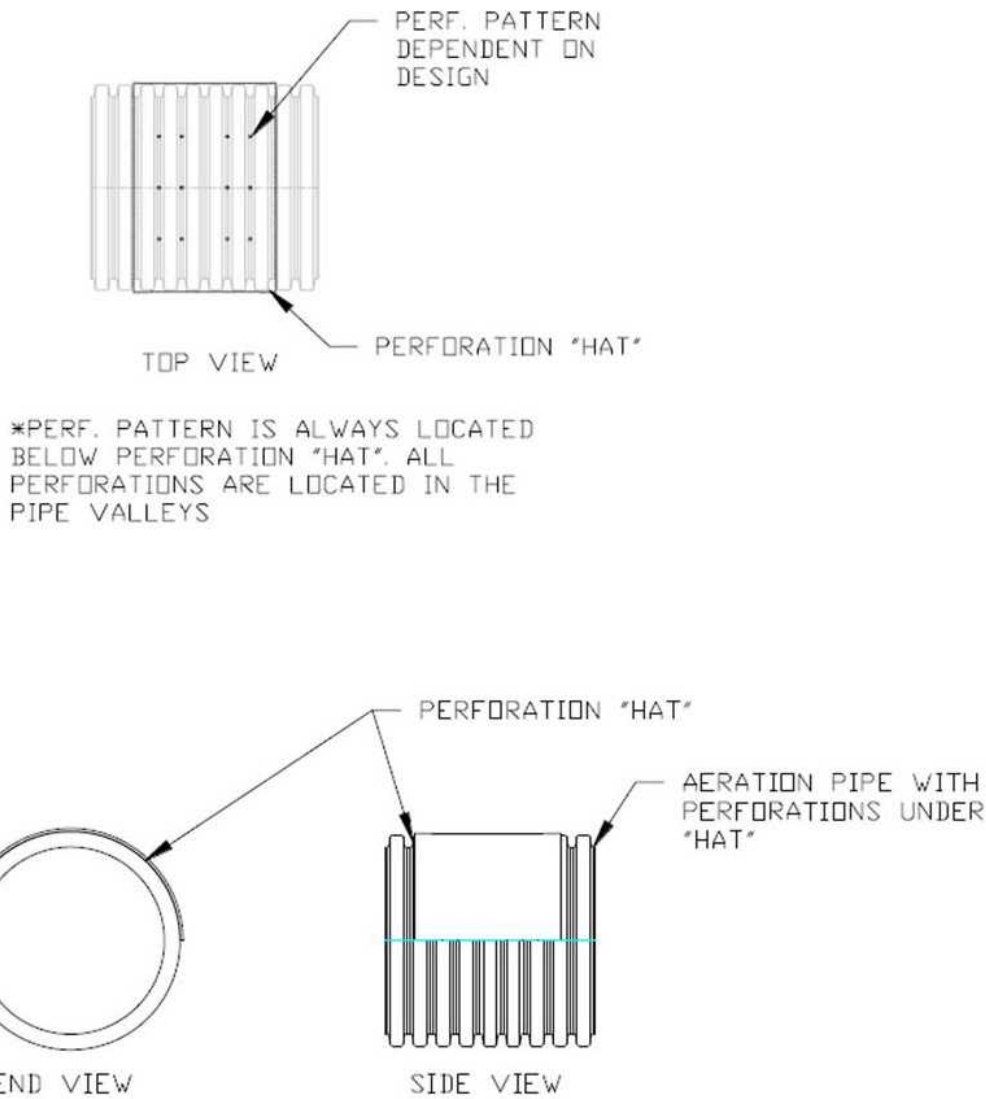

Figure 5: Perforation detail from ASTM F2987

Cost

The construction cost of a heap leach pad varies a great deal, but the contribution of each element generally falls in the same range in terms of percentage, as shown in Table 3.

Table 3: Construction cost of leach pads

Construction element	COST, $/M²
Preliminary earthworks – removal of topsoil, building of edge berms and collection ditches. Cost assumes minimal alterations to topography. Sometimes it is necessary to do extensive site preparation, at a cost of several dollars per square meter	$1.00
150 to 300 mm of compacted clay-rich soil, engineered to a permeability of 10–6 cm/sec.	$1.00 – $3.00
Limited leak detection, usually embedded small-diameter perforated pipes placed near the lower edge of the heap and in areas of solution concentration. These "daylight" to collection sumps at the front of the heap. Leakage is usually permitted up to a certain small limit provided the area is not extremely environmentally sensitive	$0.50
Plastic liner, usually 0.75–1.00 mm (30–40 mil) thick PVC, or 1.50 to 2.00 mm thick HDPE or LLDPE. The liner is delivered in rolls up to 2,000 m^2 each, and field-welded to form the total liner. The initial installation for a "small" heap leach may cover 100,000 m^2; large installations may install 500,000 m^2 each year. An HDPE liner of 2.00 mm thickness has sufficient strength and puncture resistance to support a heap up to 150 m high	$3.00 – $5.50
Geotextile cover may be placed above the plastic to prevent damage of the plastic by rocks in the drainage layer. The use of the geotextile is an economic tradeoff with the crush size of the gravel	$1.50
Drain pipes, usually 75–100 mm perforated flexible tubing, are placed on 6 m centers above the plastic. Where solution does not drain directly out the front of the heap, large collector pipes may also be embedded in the drainage layer	$0.50
Gravel cover, up to 1,000 mm thick, is placed to protect the pipes and the liner, and to provide a permeable base below the heap. Cost may be low if the gravel can be produced from the ore	$0.50 – 5.00
Total installed pad cost	**$8.00 – $17.00**

The pipe costs are an insignificant portion of the total construction cost, ranging from 3% to 6%. Adding aeration pipe more than doubles the pipe cost, but it still is a small portion of the total pad construction cost. Proper specification and inspection should not significantly change the total costs, yet will contribute positively toward the performance of the system.

Conclusions

Polyethylene pipe systems can perform quite well in heap leach applications if they are properly designed and installed.

Inspection of the pipe installation practices is highly recommended.

Reference pipe specifications should be tailored specifically to mining applications. ASTM specifications should be referenced for the following reasons:

1. Substantially improved durability in mining environments.
2. Improved performance of collection systems.
3. Improved aeration system performance.
4. Reduced loss of leachate, and the metals in it.
5. Improved structural performance.

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Author Index

Heap leaching is a complex process that requires the integration of many aspects to establish an economically viable and technically sound project, both during operation and closure. Many other conferences focus on specific scientific and engineering aspects of heap leaching, or have heap leaching as a subtopic. This conference brings it all together so that a wide range of professionals can share their expertise and experience in the development, operation and closure of heap leach projects.

The objective of this conference is to provide a forum for scientists and engineers to share technologies, practices and advances for establishing successful heap leach projects for a range of minerals in a variety of different climates.

The themes of this conference are:

- Ore characterization/metallurgical test work
- Design and construction
- Operations and metal recovery
- Closure and post-closure
- Technological developments

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