RECLAMATION AND CONTAINED DISPOSAL OF PREVIOUSLY PROCESSED MINE TAILINGS

Task 1 Design Team

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EXECUTIVE SUMMARY

A process was designed to recover a pile of 50-year-old mine tailings and transport them to a lined impoundment within two years (excluding permitting approval delays). To defray the costs of the tailings transport, a copper recovery process utilizing batch leaching with sulfuric acid was designed. The process is capable of treating 500 tons of ore per day, with a daily copper recovery of 23.3 tons. The proposed process is capable of recovering 85.5% of the copper present in the tailings pile. The total cost of the project was $5.76 million, while the money generated from recovering the copper was $4.93 million, resulting in a net present value of -$946,000. Although the design did not prove profitable, losses were minimized. Relocating and neutralizing the pile without attempting copper recovery resulted in a net present value of -$2.33 million. A third option which separated the tailings into +325 and -325 mesh fractions was rejected because its net present value was -$2.87 million. Due to the scarcity of available process water and pressures from nearby communities, government, and environmental agencies, the process was designed to recycle all water, thus resulting in a zero-discharge process over the lifetime of the operation. However, after the completion of the mineral recovery process, an approximate volume of 350,000 gallons will be treated to a pH above 7.0 and discharged into the impoundment. An Environmental, Safety, and Health (ES&H) plan was developed to ensure the safety of all personnel involved in the operation of the process. A community relations plan was developed to respond to the concerns of the neighboring community. This plan includes three town meetings, a hotline to answer questions and address complaints about the project, and newsletters sent out periodically to update the community on the progress of the project. The project meets the requirements of the New Mexico Mining Act of 1993 section 507C. Steps to obtain this permit will have to begin at least two years prior to startup. Once the permits are obtained, the process will operate for 260 sixteen-hour days, after which final site remediation procedures will be implemented. This report outlines possible techniques for remediation that depend upon the status of the soil underneath the tailings pile. The proposed process is comparable to existing mine tailings reclamation projects and includes features that represent sound environmental and engineering practices.

INTRODUCTION

Problem Statement

100,000 cubic yards of mine tailings in New Mexico are to be relocated. The tailings must be transported in a slurry ½-mile downhill to a new lined impoundment. The tailings disposal is to take place at a pH between 7.0 and 9.0 to eliminate the possibility of acidic discharge to the local groundwater and soils. Approximately 3.62% w/w Cu and 7.35% w/w Fe are known to be present. Zinc, magnesium, gold, and silver are present in quantities that are too

low to recover economically. Other considerations for the project include site closure, and possible groundwater contamination. A literature survey was conducted to explore the potential for income from mineral recovery and to examine the performance of particular mineral recovery processes.

Site closure involves the preparation of the current tailings site for the construction of a new processing facility that will begin construction in two years. With the possibility of contaminated soil under the current tailings pile, the project required a complete site closure analysis including groundwater and soils contamination monitoring and remediation of the property as necessary.

BACKGROUND

Fear of serious environmental effects has forced the mining industry to remove old tailings piles that are polluting the soils and groundwater. Mine tailings resulting from older and less effective extraction processes contain concentrations of recoverable minerals that are quite high. With the recent advancements in mineral extraction technology, engineers are examining the potential profitability of these tailings. The relocation of these tailings to a lined impoundment will also reduce the risk of groundwater contamination. Often, the cost of moving these piles discourages companies from attempting site remediation. If recovery processes are feasible, the sale of the recovered metals can aid in funding the removal of the old tailings. Reclamation of an old tailings pile can be summarized in five steps: slurrying the tailings, transport of the slurry to a mineral processing facility, mineral extraction (if economical), disposal into a contained impoundment, and process water recycling. Pile reclamation by slurry and compatible mineral extraction techniques present the biggest challenges.

When a tailings pile is left to weather for fifty years, several physical processes occur that hinder reclamation. Packing of the solids may make the adhesion characteristics of the tailings much more pronounced and render the tailings more difficult to handle. A tailings pile with the characteristics of a moist clay is very difficult to reclaim. In addition to solids handling difficulties, surface oxidation which naturally occurs can interfere with the mineral extraction process.

It may be necessary to separate the pile into size fractions that best suit the reclamation process. If the concentration of valuable metals is a strong function of particle size, the size fraction which contains the highest concentration of desirable minerals should be separated and processed for metal recovery. This saves money in the form of reduced capacities on tanks, reduced power requirements for pumps and agitators, and smaller chemical costs. However, additional costs include the screening and de-watering processes required for handling the separate size fractions. To determine the most profitable mineralogical extraction process, an analysis of the tailings is required.

Many mineral extraction techniques are currently available and all offer distinct advantages and disadvantages. These include froth flotation, gravity concentration, leaching, cementation/smelting, and solvent exchange/electrowinning. Each of these is reviewed briefly below.

Flotation is quite sensitive to subtle properties of mine tailings[1]: As mentioned, sufficient weathering of the tailings causes the degree of surface oxidation to render flotation infeasible. In addition, a large concentration of copper oxides interferes with the ability to

efficiently float copper sulfides (chalcopyrite ($CuFeS_2$), chalcocite (Cu_2S), and covellite (CuS). Furthermore, large quantities of slimes (particles less than about 10 μm that do not settle well) present in mine tailings can cause coating of the copper minerals and lower the efficiency of flotation.

Gravity concentration by centrifugation was also considered for concentration of the copper minerals. Several factors can render gravity concentration infeasible. The specific gravity criterion equation is:

$$Q = (D_h - D_f)/(D_l - D_f) \quad (1)$$

where D_h, D_l, and D_f are the specific gravities of the heavy mineral, light mineral, and fluid medium, respectively. Q is a dimensionless quotient used to determine the feasibility of gravity concentration. The quotient for our tailings is 1.4. In general terms, a quotient less than 2.5 is considered a difficult separation.[1] Gravity concentration also depends on particle size, and the large quantity of slimes in these tailings decreases the efficiency of the separation by raising the effective viscosity of the fluid. These slimes can not be effectively removed because the size of the liberated copper minerals in this pile approaches the size of these slimes (i.e., less than 25 μm). A commercial concentrator investigated requires fully liberated mineral particles and is most applicable to particles ranging in size from 50μm to 1.5 mm, which exceeds half of the mass of the tailings pile. Therefore, gravity concentration can not be applied economically to these tailings.

The oxide ores of copper are highly soluble in dilute sulfuric acid. Therefore, leaching is one of the most commonly used extraction processes. However, the copper sulfide minerals are not as readily leached in sulfuric acid and therefore require a leaching aid. Ferric ions act as a leachant for the copper sulfides:

$$2Fe_2(SO_4)_3 + Cu_2S + 2O_2 \rightarrow 2FeSO_4 + 2CuSO_4 \quad (2)$$

The ferric ions can be obtained by oxidizing ferrous ions (using an oxidizing such as hydrogen peroxide) entering
the solution from leaching of copper sulfides or from fresh addition of ferric sulfate. Rapid agitation leaching is typically used for fine particle sizes and mixed distribution of copper oxides and sulfides[2]. This process is especially effective for previously processed mine tailings which contain both oxides and sulfides.

Several processes for purification of the resultant leachate solution were studied. Solvent exchange followed by electrowinning (SX/EW) is a technically superior process but is expensive with a high capital investment and operating cost. Cementation followed by smelting is a much cheaper process and is better suited for projects involving a small amount of tailings and a short project life-span.

Cementation of copper is commonly used after the leaching of mixed tailings[2]. Cementation utilizes scrap steel as a means to plate copper onto a smeltable surface. The cementation of copper only requires one mole of iron to precipitate one mole of copper:

$$Fe^{\circ} + Cu^{2+} \rightarrow Cu^{\circ} + Fe^{2+} \quad (3)$$

However, because of the side reaction caused by the presence of ferrous ions, two to three times the stoichiometric amount is usually required in practice:

$$Fe^{0} + 2Fe^{3+} \rightarrow 3Fe^{2+} \quad (4)$$

The kinetics of copper cementation have been studied extensively[3]. The iron impurities can be

precipitated as ferric hydroxide with lime. The rate controlling step is the diffusion of copper ions onto the surface of steel, and the rate of change of copper concentration can be expressed as:

$$DC_{cu2+} / dt = -kA\ C_{cu2+} \quad (5)$$

where C_{cu2+} is the copper concentration (kg/m$^{3)}$ at time *t*, *k* is the rate constant (m sec^{-1}), and *A* is the exposed surface area (m$^{2)}$ of iron per m^3 bf solution. The cementation rate can therefore be increased by both increasing the temperature and degree of agitation. Also, lowering the presence of ferrous iron and oxygen in solution lowers the effects of the oxidation of the iron surface and thus leads to higher cementation rates. Recoveries of ninety percent and greater before recycling back to the leaching circuit have been reported[2].

TAILINGS CHARACTERIZATION

Although much of the pile was previously analyzed and reported in the problem statement, many of the tailings; qualities vital to the design remained unknown. For this reason, a series of experiments was performed to demonstrate the full-scale design. The ability of the tailings to slurry and de-watering characteristics of the slurry were studied for design of the transport processes. Performance capabilities of metal recovery process were also investigated to determine chemical dosing. These characteristics determined the optimal design option.

SLURRY CHARACTERISTICS

The weight percent of solids in our tailings sample was determined by drying samples in an oven and noting the loss of weight due to the evaporation of water. The oven temperature was kept under 105° C to prevent decomposition of the sulfide minerals to sulphur dioxide. This experiment was performed three times during the design period, and the solids content remained relatively constant at 37%.

The slurrying characteristics were determined using samples of the mud and tap water. Slurries were created at 3%, 5%, 10%, 15%, and 20% solids to represent the potential range of solids content present in the full-scale design. While the solids concentrations below 15% were easily homogenized by agitation with a spatula, it was evident that it would be uneconomical to pump such a large volume of water. However, at 15% solids, the slurry appeared to be two-phase with a runny liquid on top and a semi-wet cake on the bottom. Pumping costs increase with increasing viscosity and volume. Higher solids contents result in higher viscosities and lower volumes. To determine the optimum solids content, viscosity data over the range of solids contents used for the full-scale design was taken. A summary of the viscosity data is presented in Figure 1.

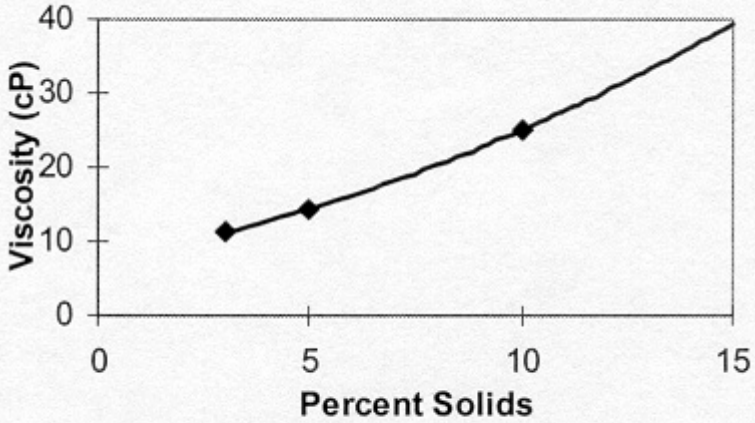

Figure 1. Slurry Viscosities vs. Solids Contents

Settling velocities of the solid particles also affect the ability of the slurry to be efficiently pumped through pipes. Settling velocities were determined using a graduated cylinder and three different solids concentrations. The slurries were placed in a graduated cylinder with marks to show changes in interface position. The time the interface took to move two mm was measured and recorded. Figure 2 demonstrates the settling rates of different solids concentrations.

MINERALS RECOVERY

Mineral Content

To determine the mineral content of the pile, a harsh digestion was performed on the sample which was given to the group at the beginning of the project. To determine the distributions of minerals between the +325 and ~325 mesh fractions of the tailings, a portion of the tailings was separated by passing the tailings as a low solids content slurry through a 325 mesh (45 microns) sieve.

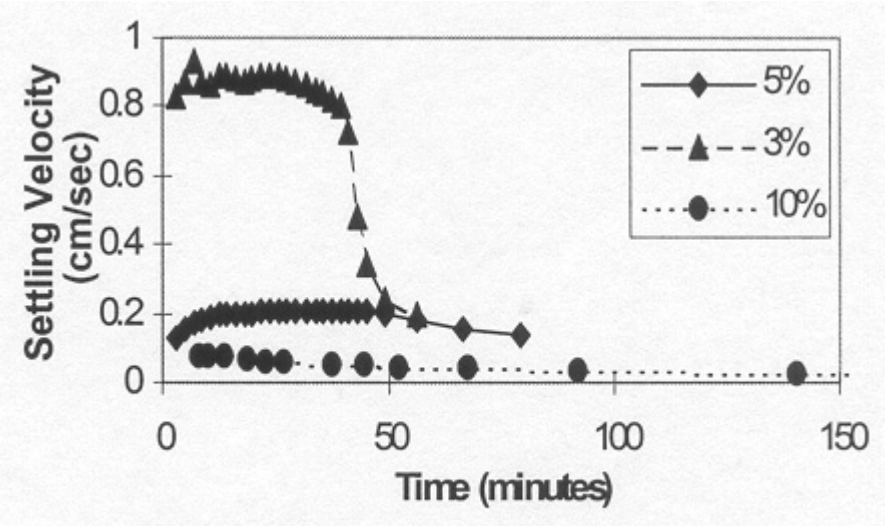

Figure 2. Results of Settling Experiment

The concentrations of copper, iron, sulfur, zinc, and cadmium in the tailings samples were analyzed using the EPA-approved method 200.2 from the EPA/600/494/111 "baseline" analysis procedure. An ICP (inductively-coupled plasma) emission spectrometer provided metals concentrations in the extracted solutions. The procedure was applied to four homogenous one-gram samples of~25 mesh and 'q325 mesh samples. Results from the baseline analysis can be seen in Table 1.

Table 1. Results from baseline analysis

Sample	%Cu	%Fe	%Cd	%Zn	%S
+325 mesh	2.25	26.00	0.06	0.6	10.63
-325 mesh	5.77	18.29	0.211	1.87	1.29

The concentration of copper in the -325 mesh fraction was 5.77% and the concentration of copper in the +325 fraction was 2.25%. Both concentrations are considered high with respect to concentrations typically handled in modem facilities. However, it is possible that the +325 mesh copper concentration of 2.25% is large due to inaccurate sieve separations. It was determined through SEM (scanning electron microscope) analysis that most of the liberated and leachable copper (both sulfides and oxides) was finer than 45 microns and therefore should have been present in the ,325 mesh. The large copper concentration in the ~325 mesh was most likely due to retention of sub 45 micron particles on the sieve caused by blockage of the apertures.

Flotation

Several mineral extraction processed were analyzed to determine the most feasible process. The first method of recovery examined was flotation. To determine the potential effectiveness of flotation, the degree of liberation of copper and presence of interfering oxides was determined. A compositional mapping and a Z.contrast mapping were performed on both

the -325 mesh fractions and +325 mesh fractions[4]. The results from the analysis showed the extent of oxidation of the tailings in the pile. The ratio of oxides to sulfides was approximately 1:1. Large concentrations of oxides render flotation difficult. Flotation was also ruled out as a possible method of copper recovery because the tailings had already undergone flotation. Even with newly developed flotation reagents, flotation was not likely to be as effective as the simpler process of leaching.

Leaching

Leaching of the copper minerals with sulfuric acid was also tested and determined to be most feasible. The high degree of liberation of mixed copper minerals (observed in SEM photographs) suggested that rapid agitated leaching of the tailings could produce high recoveries of the oxides and sulfides. One-gram samples of the -325 mesh, +325 mesh, and the whole pile were leached with concentrations of sulfuric acid ranging from 2% to 16%. Ferric sulfate was added to the leach solution to aid in leaching of the sulfides. However, it was later determined that the addition of the ferric sulfate was unnecessary due to the large concentration of iron already present in the leach solution from the tailings. Ferric ions were reduced in the leaching process to ferrous ions and therefore required oxidation. Hydrogen peroxide was chosen as an oxidizer both for its low cost and stability when added to the leaching solution. Halfway into the five-hour leaching process, varied concentrations of hydrogen peroxide were spiked into the leaching solution to oxidize ferrous ions back to ferric ions. A stoichiometric amount of hydrogen peroxide and an amount equal to 1.5 times the stoichiometric amount was added to the samples. The stoichiometric amounts of hydrogen peroxide and ferric ions were determined by the reaction:

$$H_2 0_2 + 2F^{+3} \rightarrow H_2 0 + \tfrac{1}{2}\, 0_2 + 2Fe^{+2} \qquad (6)$$

The amounts of hydrogen peroxide spiked were based on an assumption that all of the iron in solution was present as ferrous ions. However, it was noted that many of the leach solution were brown in color and therefore contained large quantities of ferric ions. A green solution indicates a high ferrous ion concentration. Therefore, the amount of hydrogen peroxide added was larger than stoichiometrically required for complete oxidation of the ferrous ions. The samples were leached for five hours at 35°C. The results are presented in Table 2.

Table 2. Leaching Performances

Sample from:	Sample #	% Fe in Leachate	Total Weight of Cu in Sample (µg)	Leached Weight of Cu (µg)	% Recovery
-325 mesh	1	10.28	1684	1660	98.6
"	2	10.44	1659	1631	98.3
"	3	10.49	1724	1702	98.6
"	4	10.48	1705	1682	98.7
Whole pile	5	8.23	872	857	98.3
"	6	8.40	893	886	99.2
"	7	9.50	1021	1011	99.0
"	8	8.86	932	921	98.9

To determine the percent recovery of copper in the leaching process, both the leachate concentration of copper and initial concentration of copper in the sample were determined in the following manner. After the leaching process was complete, the leachate solution was

recovered using a centrifuge and samples of the leachate solution were analyzed by ICP. The remaining solids were then washed to remove any ionic (leached) copper for analysis of the leftover solids. The baseline analysis was performed on the leached solids to determine the amount of copper not leached. The mass balance is evaluated by assuming that the initial amount of copper in the tailings is equal to the amount present in the leachate solution plus the amount remaining in the leftover solids. The minimum sulfuric acid concentration was determined from the leveling of the curve in Figure 3. According to the graph, 14% H_2SO_4 represents the minimum acid concentration for maximum copper recovery.

Solvent Exchange/Electrowinning

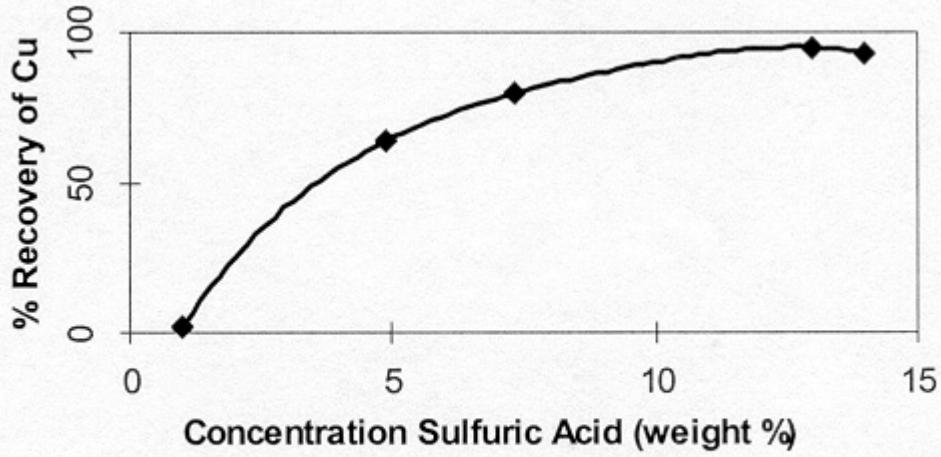

Figure 3. Leaching Performance for Various Sulfuric Acid Concentrations

Solvent exchange with subsequent electrowinning was not tested in the laboratory but was ruled out through consultation with industry professionals. The high capital cost of the equipment and short life-span of the project rendered SX/EW infeasible. The cost of construction of an electrowinning facility with a 12 ton Cu/day capacity on site was estimated to be $600,000.[2] The cost of sending the leachate to an outside SX/EW facility was estimated to be $1.2 million, not including the cost of unrecovered sulfuric acid. The Cu recovery would not be enough to cover this expense.

Cementation

Recoveries of copper by cementation were determined by analyzing the copper concentration prior to cementation and determining the copper concentration remaining in solution after cementation. The leachate solutions were analyzed using ICP analysis. Results of these tests were not available in time for inclusion in the report. Based on the results from these experiments, leaching followed by a cementation process was chosen as the most effective method of recovery.

Because of the significant amount of copper in the +325 mesh fraction of the tailings, a detailed economic analysis is performed to determine whether to treat only the -325 mesh fraction or to treat the whole pile.

ECONOMIC ANALYSIS

Three options were considered for the overall design of the project. Option one involves the slurrying of the initial tailings pile, the pH treatment of this slurry, the transport of the slurry to the impoundment, and the recycling of process water. This option meets the minimum requirement of relocating the tailings pile.

The second option includes leaching the whole pile to recover as much of the copper in the mine tailings as feasible to defray the costs of transporting the tailings to the impoundment. The entire mine tailings pile is processed through a set of agitated leaching tanks to chemically extract copper from the tailings into solution. The leach solution is contacted with scrap steel through a cementation process for copper recovery. The copper-rich scrap metal is then sold to a copper smelter.

The third design option considers the separation of the tailings at a size that enhances copper recovery. A screening process separates a +45 micron fraction of the tailings (low copper content) from a-45 micron fraction of the tailings (high copper content). The screening process may remove unwanted minerals that interfere with the leaching process. However, option three is economical only if the recovered copper profit outweighs the increased cost for screening and de-watering equipment.

An operating time of 16 hours per day was determined to be the most economical based on the tradeoff between labor costs and equipment costs. Therefore, the project was designed to operate for 260 days, leaving sufficient time for site remediation, monitoring, and closure. A cost analysis was performed to determine the optimal number of hours per day of processing.

Option one has a fixed capital investment (FCI) of $1.21 million with a net present value (NPV) of-$2133 million. Option two is most economical based on the amount of copper recovered and the cost of the project. Option two has an FCI of $3.26 million with an NPV of-$946,000. Option three has an FCI of $3.23 million with an NPV of -$2.87 million. These values include start-up and shutdown costs, operating costs, capital costs, and salvage values. Table 4 illustrates the profitability of the three design options. The fixed capital investment (FCI) includes all equipment and startup costs.

Table 4. Cost Analysis for Three Design Options and Optimization of Length of Work Day

Hours per day	8			12			16		
Option	1	2	3	1	2	3	1	2	3
FCI ($ millions)	1.35	3.51	4.67	1.27	3.42	4.11	1.21	3.26	3.23
Operating Cost ($ millions)	1.12	2.39	3.75	1.16	2.44	3.75	1.20	2.50	3.74
Total Cost ($ millions)	2.47	5.90	8.42	2.43	5.86	7.86	2.41	5.76	6.97
Salvage Values ($ millions)	0.195	0.523	0.676	0.184	0.502	0.595	0.175	0.472	0.468
Copper Profit ($ millions)	0.00	4.93	4.09	0.00	4.93	4.09	0.00	4.93	4.09
NPV at 10% ($ millions)	-2.39	-1.09	-4.31	-2.35	-1.05	-3.76	-2.33	-0.946	-2.87

Costs for site remediation are not included for any of the three options. The costs for site remediation will depend upon the amount of contamination found upon completion of the process operation, and therefore cannot be estimated. The possible methods of site remediation are discussed in the site closure plan.

PROCESS EQUIPMENT AND DESIGN

Process Description

Option two (treatment of the whole pile) was determined to be the most economical and feasible design process. A process flow diagram (PFD) for option two is included in Appendix A. The following process description contains references to stream numbers presented in the PFD.

In design option two, the tailings are placed by two front end loaders onto a conveyor and transported to a leaching mill for copper extraction. These tailings were assumed to be 40% solids for the full-scale design. The conveyor places the tailings into seven 10,000 gallon leaching tanks (S2). In each leaching tank, a 27% sulfuric acid leaching solution (S3) enters from a holding tank as the tailings are added to form a solution that is 15% solids and 14% sulfuric acid (pH approximately 2.0). The acid leach holding tank is maintained at approximately 27% acid and is controlled by addition of 98% sulfuric acid (S22) to the holding tank. The solids content of the incoming tailings (likely varying by+/- 5%) is monitored to control the flowrate and concentration of the leaching acid stream (S3). By monitoring the ferrous ion concentration in the leaching tank, the addition of an iron oxidizing agent (hydrogen peroxide) (S20) is also controlled. Each leaching tank has a batch cycle time of five hours, and the tailings are vigorously agitated in each leaching tank.

After the tailings are agitated for five hours, the solutions are pumped out of the leaching tanks (S4) and into a holding tank. The solution leaving the holding tank (S5) is then processed through a set of two parallel 100 hp decanting centrifuges for leachate recovery. The centrifuges

separate the solid tailings from the leached solution and discharge them as a 30% solids stream (S10). The solids proceed to a pH treatment tank where water (S24) and lime are added (S11) to re-slurry the tailings to approximately 15% solids and raise the pH of the acidic solids to between 7.0 and 9.0. The amount of lime needed to neutralize the acidic tailings was estimated from laboratory data. After the tailings are slurried, they are pumped (S 13) one-half mile downhill to the new impoundment site.

At the new site, the tailings are pumped into a thickener to concentrate the solids. A cationic polymeric flocculant is injected (S 15) via an in-line mixer to increase the settling rate of the fine particles that remain in suspension.

The concentration of polymer is held at approximately 12 ppm (based upon microscale tests performed on the tailings sample). The thickening tank uses a large mechanical rake and sludge pump to deliver a 20% solids slurry (S16) to a third centrifuge for further de-watering. The final discharge to the impoundment is 35% solids (S18). The water recovered in the thickener (S23) and centrifuge (S17) is pumped back (S12) to the leaching site for reuse in the pH adjustment process.

The pregnant leach solution recovered from the first two centrifuges is pumped (S6) to a holding tank prior to cementation. The solution pumped (S7) from the holding tank is processed over scrap steel. The liquid solution is then drawn (S8) from the cementation tank for reuse in the leaching process. A portion of the recycle stream is bled (S21) to the pH adjustment process to prevent iron buildup in the acid recycle loop. The amount bled is controlled by monitoring the iron content in the leachant holding tank. The cement copper is then hauled by truck (S9) to a smelter for processing into purified copper.

A detailed equipment list is presented in Appendix B. Operating and other miscellaneous costs are presented in Appendix C. Tanks, piping, and process pumps and motors were selected and priced utilizing cost estimation charts[5]. Screening technology, centrifuges, leaching equipment, slurry blenders, and raw materials were priced through industrial consultants and vendors. Labor costs were calculated according to the following employee arrangement: two front end loader operators will be working full-time on each 8-hour shift; seven operators will monitor the leaching process, cementation process, pH adjustment process, and final discharge to the impoundment; and a night watchman will be employed to oversee the facility after operating hours. An engineer will be present for each shift.

Time Constraints

Figure 4 (next page) is a GANT chart illustrating the time in which the project is completed. All of the time assumptions are based upon good weather, good health for all employees, and good equipment performance. The mining permit may take from six months to eight years to acquire61 For the proposed project, a period of two years was assumed for permit approval.

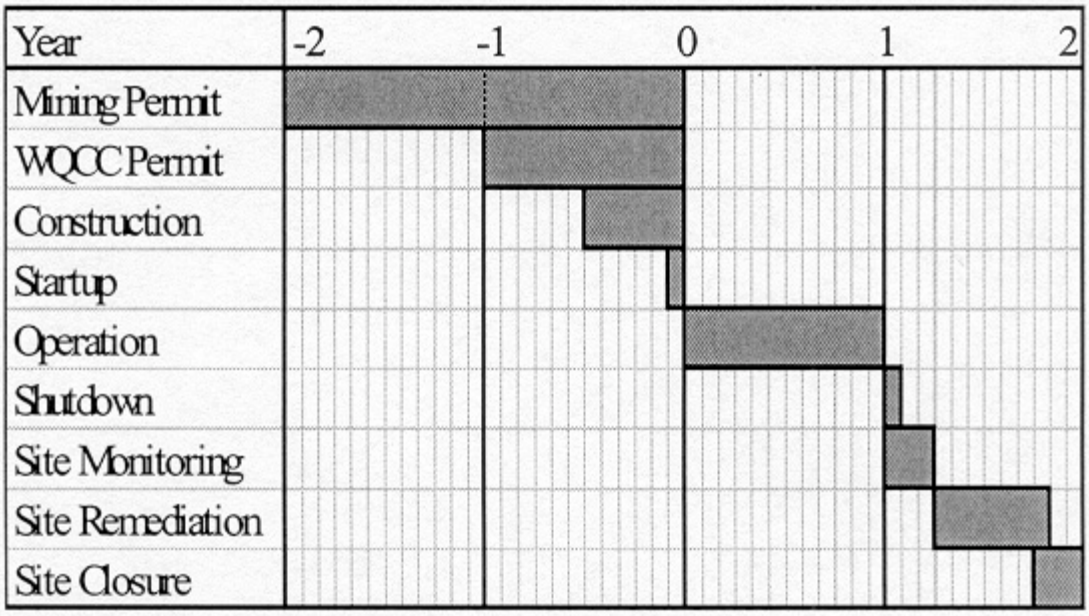

Figure 4. Gant Chart for Tailings Recovery Project

Federal Waste Regulations

Because we are proposing a new process to an existing mine, a permit is required under the New Mexico Mining Act of 1993 section 507C (NMMA 507C)[6]. This permit encompasses nearly all aspects of the project, including employee safety, public safety, wildlife protection, public relations, water usage, impoundments, transportation, site stabilization and surface configuration, erosion control, and revegetation.

Employee and public safety are covered in the ES&H plan. Public relations issues are included in the Community Right-to-Know plan. In accordance with the NMMA 507C permit a detailed outline of the process and of all the streams would be submitted to the New Mexico Environment Department. A strict hydrologic balance will be outlined which will cover the recycle streams and the optimal water usage for this process. Because the lined impoundment has already been constructed, it will be checked to insure that it meets all of the requirements specified by this permit. Transportation of the recovered copper is available to the smelter to which the cemented copper will be sent. The old tailings site should be stabilized such that it will not become a threat to the ecosystem after site closure. After removal of the pile, the surface configuration should be assessed, and the surface should be reconfigured to minimize the potential of contamination by water moving through the site and transporting contaminants to unprotected soil. Revegetation is not considered because plans exist to build a new shop complex on the current tailings site.

In addition to the permit above, two permits are required from the New Mexico Water Quality Control Commission (WQCC) under sections 3103 and 3104XX. In accordance with WQCC 3103, an analysis of the existing ground water will be performed and compared to the ground water standards of New Mexico. During and after operation of the process, the ground water near the old tailings site will be analyzed. After project completion, the ground water analysis will again be performed at the site of the lined pond. From the given analysis on the pile, the ground water standards that would pertain to this process are: arsenic (As) 0.1 mg/1,

cadmium (Cd) 0.01 mg/1, lead (Pb) 0.05 mg/1, total mercury (Hg) 0.002 mg/1, selenium (Se) 0.05 mg/1, silver (Ag) 0.05 mg/1, copper (Cu) 1.0 mg/1, iron (Fe) 1.0 mg/1, manganese (Mn) 0.2 mg/1, zinc (Zn) 10.0 mg/1, and pH 6-9. Ifa leak occurs in the process streams, and the ground water contamination is forced above these standards, a plan for remediation of the ground water would be implemented. The construction of a zeolite wall and a pump-and-treat method are suitable remedies for contamination.

WQCC permit 3104 is concerned only with the contamination levels of the discharge streams of a process. Because the waste water is recycled, the only stream that will be discharged is the stream which is deposited into the lined impoundment. Therefore, a detailed description of this discharge stream will be submitted, as well as the processes used to reduce the contamination levels. Because this single discharge stream is being discharged into a lined impoundment, the acceptable levels of contamination are higher than if the discharge was directly onto the ground.

SITE CLOSURE

In accordance with the closure guidance for mining sites given by the New Mexico Environment Department, the site closure consists of two stages[9]. The first stage of the closure plan is for the site where the tailings pile is currently located. Soil testing begins by taking either split spoon samples or core samples as soon as the pile has been removed from the current site. These samples are taken at the same time that monitoring wells are drilled.

If the top soil of this site is contaminated, up to 6 inches of top soil are removed and processed through the recovery unit. The remaining soil is tilled with lime to adjust the pH. If the contamination is within the vadose zone depth, a geo-synthetic liner is placed over this site. A geo-synthetic liner is used instead of a clay liner because of the dry climate in New Mexico and the chance of dessication and cracking of a clay liner. The geo-synthetic liner is generally more expensive than the clay liner, but does not require a top soil cover. The soil type beneath the tailings pile is Plack-Lonti-Pits[10]. This type of soil is characterized by nearly level to very steep contours, good drainage, shallow or deep soils, and pits located on broad terraces and hills. With a percolation rate in the range of 0.06 to 6.0 inches per hour, an average monthly rainfall of 0.75 inches[11], and an average soil pH of 8.84,12 the rate of migration of the heavy minerals into the ground water is significantly reduced.

Even with the decreased chance for ground water contamination, monitoring wells will be drilled for the ongoing analysis of the groundwater. A monitoring well will be implemented every 200,000 square feet. These wells cost $100 per foot of well depth. If the ground water is determined to be contaminated, a zeolite wall will be built at the south end of the site, because the ground water flows from north to south. The zeolite wall consists of a trench that would be fifty feet deep and filled with zeolite. The zeolite captures the metals contaminating the ground water and therefore acts as a filter[13, 14]. In addition to metal contamination testing, the ground water needs to be sampled for pH levels, sulfate levels, and dissolved solids. If this type of contamination is present, more monitoring wells will be drilled, and they will all be used in a pump-and-treat method for ground water remediation. Monitoring wells will also be placed downstream of the lined impoundment to ensure that the ground water is not contaminated by tailings that may have penetrated the lining of the impoundment. Costs of these methods of remediation have not been examined in detail, as the plan for them is only in place should extensive contamination be found when the process operation has ceased.

The second stage of the site closure plan consists of dismantling the recovery operation. In accordance with hazardous chemical disposal requirements, the process streams are neutralized and discharged into the lined impoundment. WQCC 3104 covers the requirements for the discharge stream pumped into the lined impoundment. After removing the tailings from the pile and discharging the waste into the pond, a top soil would be placed on top of the pond to serve as a cover.

The National Pollution Discharge Elimination System (NPDES) permit allows the discharge of a process stream into a stream, river, or creek. Although our recovery process discharges into a lined pond, this NPDES permit will be required in the case of upset conditions due to a leak.

PERSONAL RELATIONS

Community Right to Know

According to Federal Law, a community right-to-know plan requires reporting to the local authorities the hazardous chemicals used. One town meeting is required to inform the community of the project. However, following the minimum requirements of that plan will not be sufficient. Because the operation is within ¼mile of the surrounding community, it would be beneficial to have the community well-informed. In order to make the meetings available for everyone, a series of three town meetings held over one week. In addition to increased availability, this multi-meeting format has other advantages over a single meeting format. Multiple meetings allow informed citizens to come back second or third times with more poignant questions regarding the project. These meetings will consist of an oral presentation with a question and answer session to follow.

Process tours and complaints handling will also help maintain good community relations. Any member of the community wishing to tour the plant will be trained in basic chemical process safety. A phone number for complaints will also be available. This demonstrates the company's wish to address community's concerns.

ES&H PLAN

The purpose of an ES&H plan is to ensure that employees are well-informed of the dangers that may arise from normal operation of the process, as well as potential hazards. Although there are no specific regulations on the use of sulfuric acid in a process, the company should make special efforts to warn employees of the dangers of acid. Safety showers and eye-wash stations should be provided in at least three locations: two near the seven leaching tanks, and one near the pH adjustment tank. In addition, warning signs will be placed throughout the site for restricted access and to inform workers and the public of the danger due to heavy machinery and acid usage. There would also be a sign upon the entrance of the site with the phone number of the on-site engineer to contact in case of an emergency.

As construction nears completion, all employees will gather for several meetings during which an engineer will explain the process and point out the potential hazards mentioned above. The regulations regarding arsenic (29CFR1910-1018), lead (29CFR1910-1025), and cadmium (29CFR1910-1027) will be adhered to by the process. These regulations will be summarized during the meeting. The on-site engineer will be responsible for reporting all accidents, and should be reported to by all employees should an accident occur. Safety equipment that will be

provided by the company will include rubber boots and gloves, eye goggles, and lab coats. These items should minimize the harmful effects of the acid in case of a spill.

CONCLUSIONS AND RECOMMENDATIONS

This report summarizes an economically and environmentally sound solution to the reclamation of the mine tailings pile located just outside of Silver City, NM. The recovery of minerals was treated only as necessary and as would be profitable to the company. Should new technology arise that would permit the recovery of small amounts of the more precious minerals such as gold and silver, they should also be considered as potential value for the process. The technology that is being developed now should be continuously monitored to ensure that the best solution to the process is obtained.

The contamination of creeks is an issue of great concern with the public because of other mine tailings pollution problems in the area. To reduce the negative effects of the tailings pile on the environment and to ensure good relations with the community, the project should begin immediately. In addition to the benefits for the environment and community, the design provides a means for minimizing losses for the company through copper recovery.

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