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PROCESS EVALUATION AND ECONOMIC ANALYSIS FOR IN SITU COPPER LEACHING AT "LA COLORADA" ORE BODY OF MINERA TAYAHUA, MEXICO

by

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ABSTRACT

With environmental control required in all stages of a mining operation, greater emphasis in the future will be made on improving mining methods and metallurgical processes. With the insurgence of environmental regulations placed on smelting practices, more operations will be adopting hydrometallurgical processing not only to process mineral concentrates but recover the desired values by either dump, heap, vat or in-situ leaching of ore. Techniques and processes must be developed for leaching various metals from ore and host rock without the high cost of mining and subsequent environmental problems.

This thesis presents a case study with the development of techniques, processes and economic assesment of a copper orebody and its posibilities for in situ leaching. The objective is to increase the net present value of the deposit using a low-capital-and-operating-cost mining technique. This methodology provides an oportunity to increase reserves by mining low grade ores and can have a major impact on the economy of the existing operation.

Description and selection of some laboratory testing techniques are presented in this thesis to support the evaluation process for the project. Calculations for capital and operating cost are submitted with a discounted cash flow analysis (DCF-ROR). A sensitivity analysis and a risk analysis are also included which employ a computer program to measure the uncertainty of variables such as copper price, recovery, and operating cost. From these parameters, project risk is evaluated.

Project evaluation results are encouraging, ore samples contain oxide minerals and have copper soluble in acid, testing and field characterization are required to provide more reliable data and enhance the level of confidence.

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CHAPTER 1.

INTRODUCTION

1.1 Overview

Leaching is a hydrometallurgical process for the recovery of metals from ore bearing rocks. Leaching in situ has tremendous potential and seems to be the technology of the future. Copper from secondary sulfide minerals, like chalcocite and covellite, can not be easily dissolved within a short period of time using dilute acid. When an oxide deposit contains some pyrite or sulfide minerals, water alone may be used as the leach solution. In the presence of oxygen, pyrite is acidiced to form sulfate ions which decrease the pH of the water in an autogenous reaction. This reaction enhances the kinetics associated with the dissolution of metallic minerals. The orebody examined in this case study is of an oxide deposit. However, there is a little consistent information related to the leaching of this deposit. Therefore a feasibility study is presented which utilizes risk analysis techniques. The objective is to maximize the net present value of the deposit, with the possibility of using in situ leaching extraction systems in the current operation.

1.1.1 History

Although in situ leaching is considered a modern technique, it has been practiced for centuries. The earliest record was found in a historical volume named the "Huainancius," a book written by Liu-An (177-122 B.C.) who was King of Huainan Kingdom in the West Han Dynasty in China. It stated "Being attacked by iron, the blue vitriol turns into copper". The blue vitriol, or gall-stone, was essentially chalcanthite or copper sulfate ($CuSO_4.5H_2O$) in its natural state. This mineral formed through progressive oxidation of sulfides such as chalcocite, chalcopiryrite, or bornite in contact with a moist atmosphere.

When contacted by surface or underground water, the sulfate dissolved, trickled down through the ground, emerging as a gall-spring. The gall-spring is a natural copper sulfate solution. because it tasted as bitter as gall. The gall-copper process was put into practice in the Tang Dynasty and became rather wide spread in the Soong Dynasty. It was practiced on a considerable scale and with comparatively advanced technology early in the 11th century. The gall-copper process developed in the Soong Dynasty was essentially a prototype of in-situ solution mining using natural water as a leaching agent. Unfortunately, owing to social, political and economic conditions, this technique ceased to develop in subsequent centuries (1). Copper dump leaching at the Rio Tinto Mines in Spain has been reported to have been practiced since the early part of this century. In Hungary, the mines of Schmoellnitz were flooded in 1878, and were pumped out in 1904. Copper was recovered from the mine water and paid for the pumping.

Records indicate that, in England, miners used flooding followed by pumping of a mine approximately two to four times a year for recovering copper from the water, followed by the recovery of iron ochre. Cananea, Mexico, started leaching in place ores in the 1920s from mined out and caved underground stopes to extract residual copper from chalcocite ores using weak sulfuric acid solution containing ferric ions. Ohio Copper Company in Utah started in-situ leaching of mined out areas in 1942. In this case, stope pillars left between mined out veins were fragmented by blasting to obtain a rubblized area measured of 1400x600 feet at the surface. Leach solutions were distributed on the caved zone. Slow percolation was used in the leaching because it was believed this method would promote the oxidizing conditions deemed favorable for continued and rapid dissolution of the copper-sulfide minerals (2).
The San Manuel mine, of the Magma Copper Company is an example of an actual in-situ leaching operation. The project initiated in 1986 and by 1989 had developed into an economic operation. Since then the leaching program has become a viable and cost-effective method to extract copper from the supergene mineralized zone (3).

1.1.2 Present Technology

There are four different applications and associated leaching systems encountered in the field, depending mainly on the location and physical-chemical characteristics of the ore deposit.

a) Dump Leaching
b) Heap Leaching
c) Agitation & Vat Leaching
d) In-Situ Leaching

a) Dump Leaching

In most applications, dump leaching is a convenience for open pit mining operations to dispose of lower grade or waste rock (typically below 0.2% Cu) (4) and stripped material with the intention of recovering the valuable metals contained therein (Figure 1). Dump design is dictated by topography and haulage costs. Copper dumps contain predominantly sulfide mineralization, where the pyrite/copper ratio is an important factor due to acid generation. Leach cycles are usually measured in years.

b) Heap Leaching

Heap leaching is a system in which low grade or mixed oxide-sulfide ores are leached under controlled conditions in relatively small heaps (10 to 50 ft high) on prepared pads or surfaces with positive drainage over a shorter leaching period (days to months). Occasionally the ore is brought to permanent pads for treatment and removal, followed by piling of fresh ore on the same pads for treatment (Figure 2).

Heap leaching is usually selected for moderate to high grade ores. Higher grade ores mined for heap leaching are usually pretreated to optimize copper extraction. Leaching solution is applied to the surface of the heap, usually by spraying or sprinkling.

c) Agitation & Vat Leaching

This system is usually used for tailings or crushed material, where particle size is a critical factor for recovery. Vat leaching is conducted in a container made of concrete, or polymers such as polivinyl chloride (PVC) or high density polyethylene (HDPE), or other material. Such container is filled with the crushed material and acid solution. Pregnant leach solution is recovered and pumped to the plant. Agitation leaching employs the same kind of container with the variation of an agitator or blades that mix the material with the acid solution. The tailings produced by milling pricesses contain some metal values and are characterized by a large portion of fine particle sizes. The permeability of such fine material is so low that the recovery of values is not economic using heap or dump leaching. The use of vat or agitation leaching increases the interaction between the acid solution and the mineral paricles.(Figure 3).

d) In-Situ Leaching

In-situ leaching is an operation in which the ore in an underground mine is broken in-place by some type of blasting technique followed by percolation of leach solution over the fragmented ore-mass. Permeability in the range of 300 to 1000 millidarcys (5) for rock fractures is sufficient to make the ore leachable. Appropriate collection galleries are provided to collect the leach solution for subsequent recovery of metal and recirculation of the barren solution for additional leaching. Since the ore is left in place, such leaching systems are economically attractive (Figure 4) (6). Another situation is the leaching of relatively shallow deposits located less than 300 meters from the surface that are under the water table. Such deposits occasionally need to be fractured in-place and the solution drained to commence the alternative leaching and drying cycles. Metal values from such deposits are extracted by using bore-hole mining-technology, usually consisting of a properly designed pattern of wells suitable for injection of the leaching solution and recovery of metal-bearing liquors from alternating drill holes. The wells are similar to water wells except that the casings are perforated for optimum leaching conditions (7,8,9). Another system variation is characterized by relatively deep-seated deposits more than 300 meters under the water table. Leaching at such depths is enhanced by direct oxidation of sulfide minerals due to increased solubility of oxygen resulting from hydrostatic head. This system variation has the disadvantage of having to control solution loses and ground water pollution under the water table (10).

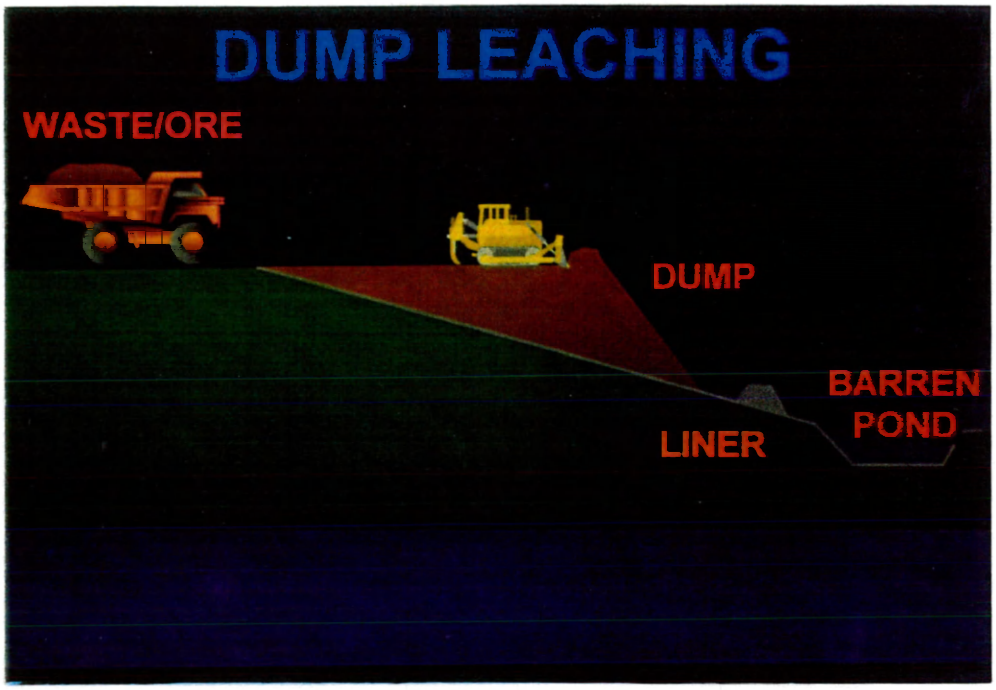

Figure 1. Dump leaching

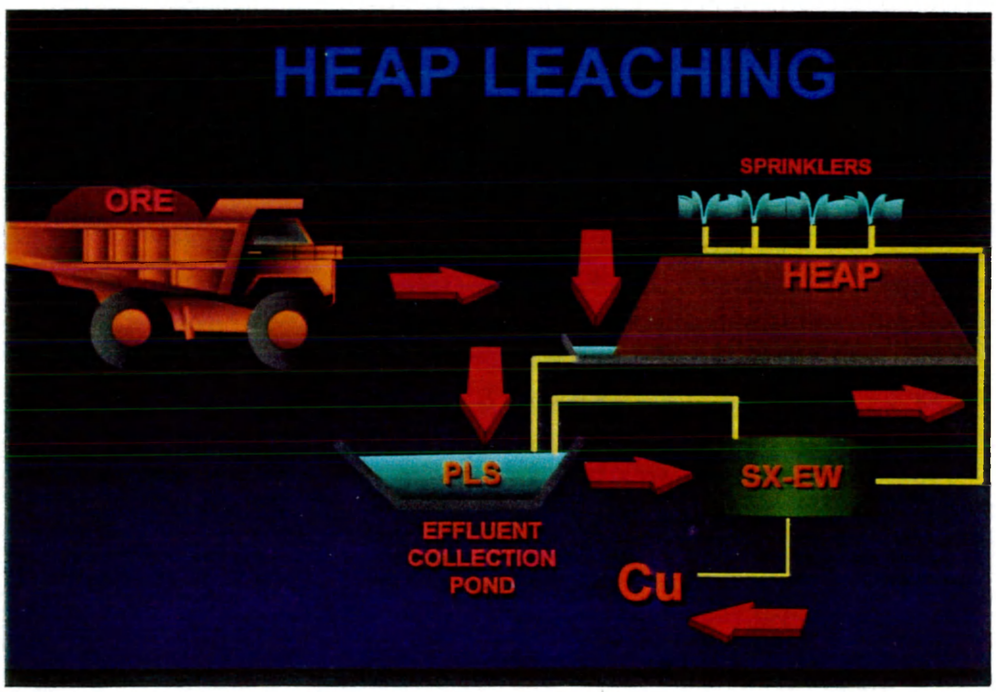

Figure 2. Heap leaching

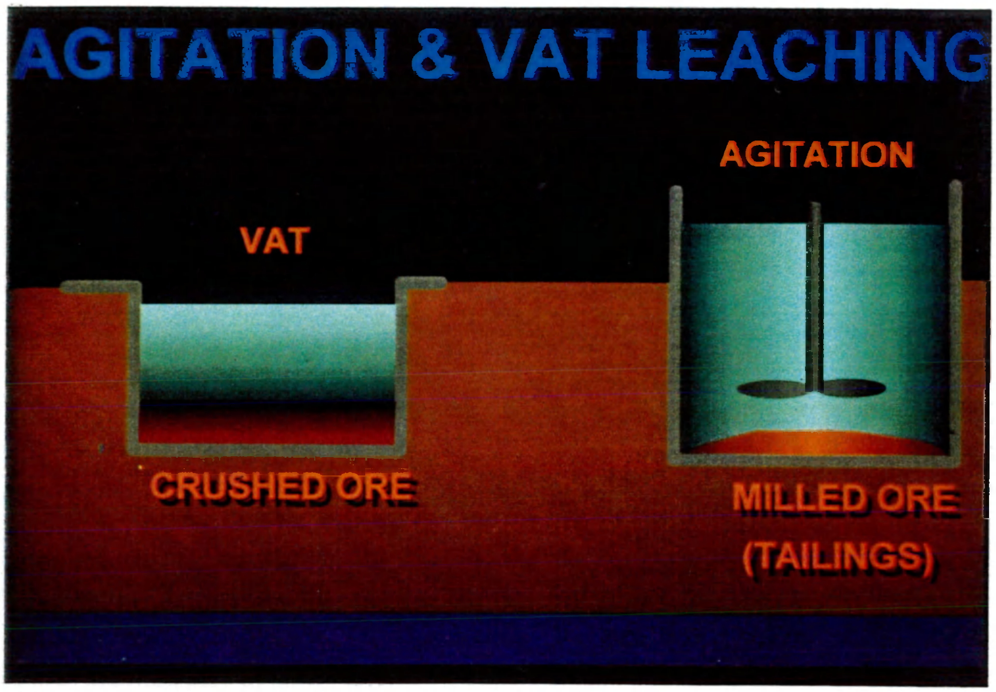

Figure 3. Vat and agitation leaching

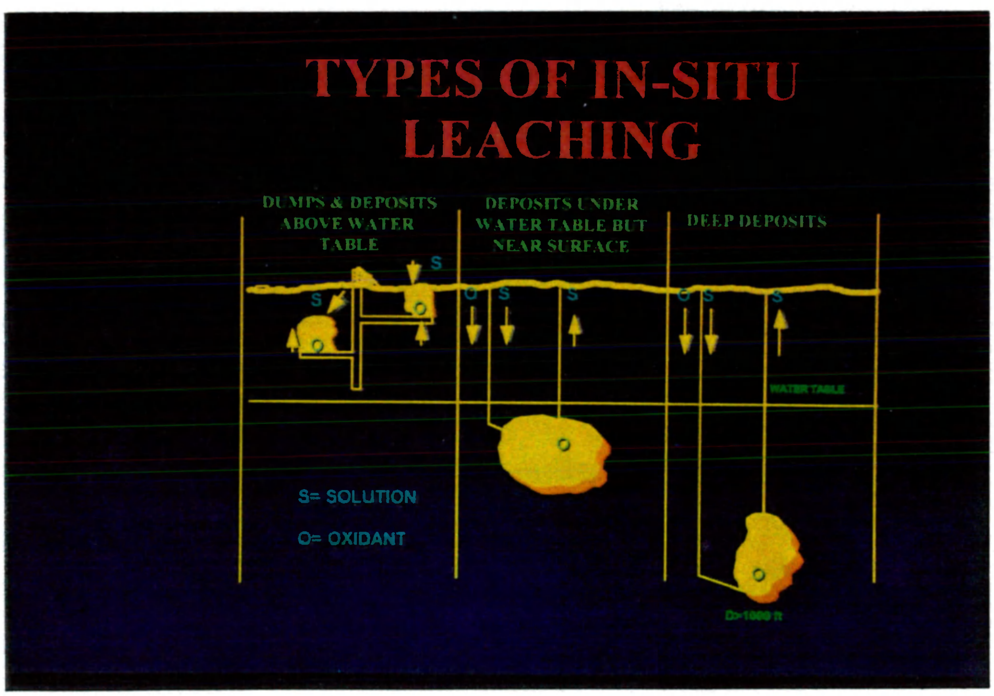

Figure 4. Types of in situ leaching
(after Dr. Milton Wardsworth)-(2)

Some technical innovations important for leaching are:

1.- Improvements in solvent extraction-electrowinning (SX-EW), referring in this thesis as LIX or liquid ion exchange when using SX:

SX-EW is a two-stage process. In the first step, low grade, impure leach solutions containing copper, iron and other base metal ions are contacted with an extractant which selectively removes the copper ions into an organic phase. kerosene containing 15-18% by volume is used as a extracting reagent. The organic phase is immiscible with the aqueous leach solution, so simple phase separation in a mixer/settler unit leaves a copper-poor and acid-rich aqueous solution for recirculation to the dumps. In the second step, the pregnant organic solution is stripped with fresh acid to produce a clean high grade copper solution for electrowinning. Barren organic solution is recirculated to the extraction stage. Over the past 20 years changes in equipment, material construction, and superior reagents have all contributed to reducing capital costs and increasing efficiencies of solvent extraction plants.

2.- Improvements in heap and dump construction.

3.- New materials, liners, monitoring control and reagents.

4.- Advances in in-situ leaching technology:
- Improved techniques for measuring physical properties of ores in-place,
- Improved techniques for drilling and blasting to produce desired fragmentation,
- Improvement in injection and recovery systems,
- Improved understanding of hydrology in broken ore masses,
- Improved design of leaching systems for maximum oxidation,
- Improved studies on leaching of dispersed mineral particles,
- Improved methods of scaling up from laboratory and field tests,
- Improved optimization of parameters for leaching of massive deep-seated deposits,
- Improved environmental controls,
- Improved advances in biotechnology. The sciences of metallurgy and biology employ widely differing techniques and languages. Interdisciplinary approaches are required to bridge this gap,
- Improved genetic engineering for microbes.

1.2 Purpose

The objective of this study is to provide an economically attractive alternative to conventional mining systems, In addition, this thesis presents the basic steps for the evaluation of a in-situ leaching project in terms of increasing reserves and production, and lowering costs.

1.3 Location of Study Area

Salaverna-Terminal District is located in the Northwest portion of the Zacatecas state, 80 miles from the city of Saltillo and 169 miles from the city of Zacatecas, the State capital (Figure 5). The mineralization takes place in irregular ore bodies with a vertical extension greater than 3,600 feet from the surface outcrop. Since the discovery of the ore body in the year of 1530, exploitation of the ore bodies was done by Francisco de Urdiñola. From 1915 to 1972, Minera Peñoles Company started a new period of exploitation for silver, lead and zinc, reaching a production of three million metric tons of oxide ores in that period with an average grade of 12% of lead and 800 gr/ton of silver. In the same period 4,200,000 metric tons of sulfide ores were produced with an average grade of 6.5% of lead, 13% of zinc and 200 gr/ton of silver.

From 1972 to 1978, the exploitation by Minera Providencia del año nuevo Company, was 600,000 metric tons of mineral with an average grade of 2.8% of lead, 6.4% zinc, 154 g/t silver and 1.2% of copper. From 1978 to the present, the operation has been developed by the Minera Tayahua Company with a production rate of 1,200 metric tons per day at an average grade of 1.6% of lead, 4.4% of zinc, 83 g/t of silver and 1.0% of copper.

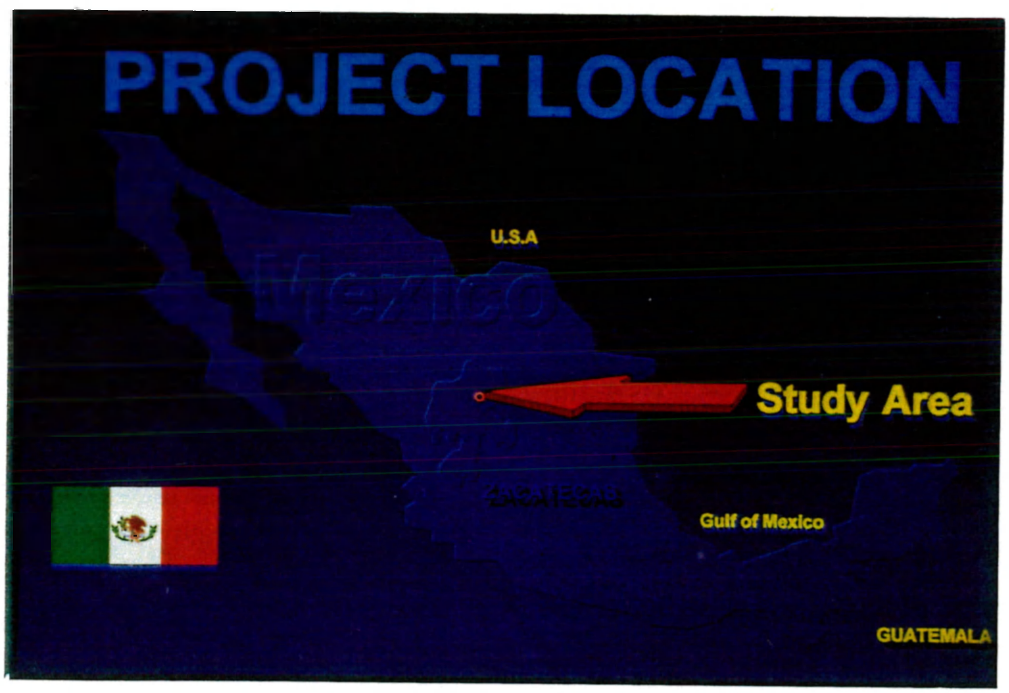

Figure 5. Case study location

1.4 Site Geology

Geology of the area has been described by Burkhardt (1906), Bergeat (1910), Imlay (1938), Barry (1950) and Rogers (1957).The mining district of Salaverna-Terminal, is located on the anticlinal structure of the "Caja" ridge, which include two other ridges known locally as "La Caja" on its Northeast portion and "Concepcion del Oro" to the Southeast. Both form a continuous mountain system, with a length of 28 miles and a width ranging from one to three miles. The axial plane of the anticline, is curved to the West, with a strike direction of North West 45^0 to East West. Rocks are formed by limestones, hornfels, sandstones, shales, and schists. This sequence is intruded by igneous bodies of felsic composition from the Tertiary approximately 40 million years of age, determined by the K-Ar method, on some biotite crystals founded in the igneous bodies by Hart and Buseck in 1961. Mesozoic foundations are the most common, running from the Upper Jurassic to the Upper Cretaceous. The geologic sequence is presented below:

I. Jurassic: -Zuloaga Foundation
-La Caja Foundation

II. Cretaceous: -Taraises Foundation
-Cupido Foundation
-La Peña Foundation
-Cuesta del Cura Foundation
-Indidura Foundation
-Caracol Foundation
-Parras Foundation

On the Nothern side of the anticlinal some ore bodies are of skarn type, basically formed by calcareous silicates, predominating andradite, abundant in quartz. and in less amount of grosularite. The copper mineralization is disseminated, and sometimes replaces the limestone strata of Cuesta del Cura Foundation. It can also be found in thin fractures associated with pyrite with a strike of N45°W and N20°E. The skarn aureole is located bordering an apophysis of the monzonitic intrusion, with a strike of S30°W and 75° dip to the Southeast. The skarn replacement is about 350m high, 60 to 350m in length, and 30m wide. The high grade copper is in a homogeneous fringe of approximately 10m of width, located in the exterior limit of the skarn and is apparently controlled by the contact between the Indidura and Cuesta del Cura Foundations. Supergene copper ore bodies close to the surface (500 m) are oxide in form, with a mineralogy comprised of azurite, malachite, chalcopyrite, cuprite, hematite, and limonite (11).

1.5 Economic Considerations

From an economic viewpoint, the net return from a leaching operation depends upon the quantity and quality of the metal or commodity being extracted from the ore. Overall economics are also influenced by: the capital cost of unit processes, such as cementation, solvent extraction, electrowinning, the cost of utilities, supplies and reagents, and the prevailing prices of metals and byproducts. Since the percent of metal recovery in a in situ leaching environment is very difficult to predict, feasibility studies play a very important role in evaluating these types of projects. The recoveries attained under actual operating conditions may not be as high as those indicated by laboratory or pilot plant tests. For this reason, it is customary in such feasibility studies to carry out cash flow analyses for a wide range of recovery levels (60 to 20 percent). If the cash flow does not appear to be favorable for 50 to 40 percent recovery levels, then it would be preferable to drop the project or undertake additional laboratory and field studies in order to increase the confidence level. On the other hand, if the cash flow looks favorable for 30 to 20 percent recovery levels, then the success of the venture is assured (12). Besides recovery levels, it is mandatory that a cash flow analysis be made for varying metal prices. Fluctuations in metal prices can seriously affect the economic viability of a project. The overall economics is also dictated by the cost of exploration, the capital cost associated with unit operations involved in mining and extraction, and the operating costs for labor, supervision, utilities and supplies. Accordingly, the economic feasibility of a leaching project must be decided in light of all these and several other influencing factors.

CHAPTER 2.

GENERAL BACKGROUND

2.1 Leaching process

In situ leaching appears to offer an attractive alternative to the conventional mining and milling of low-grade metal deposits and requires significantly lower capital investments. In situ leaching at the Santiago Mine is most likely to be underground type of deposit above the water table according to Dr. Milton Wardsworth classification as shown in Figure 4. In situ leaching has taken place as a natural geologic process in the Ninth Level of the Santiago Mine in "La Colorada" ore body (Figure 6). The secondary enrichment in the Indidura foundation (Ksi), is the result of the natural oxidation and copper leaching from ground and surface waters (Figure 7). The leach solution in this natural process is rainfall. The solution is high in oxygen but low in sulfate ions. Leach solution ranges in pH from 2.47 to 4.22.

The sulfate ions that decrease the pH of the water are supplied by the oxidation of pyrite from an autogenous reaction. Natural leaching in this case differs significantly from heap and dump leaching in the rate of solution application and is considerably less than the typical 0.1 to 0.25 gallons per hour per square foot of surface. In the course of natural leaching, cupric sulfate solution moves downward until the oxygen is depleted, at which point the copper in solution precipitates in the reducing environment as shown in Figure 7. In the Eighth Level of the Santiago Mine, minerals precipitated include chalcocite, azurite, malachite, and bornite. As weathering and erosion proceed, the precipitated copper minerals are disolved and reprecipitated numerous times producing a very rich copper deposit. Sometimes when the solution contacts with iron, as in the case of old railroads or pipes (scrap), copper precipitates in metallic form. The presence of microbes has not been confirmed, but the temperature, humidity and acid environment are favorable for microbial life. The ability of certain microbes to enhance the leaching of low-grade metal deposits is a subject of some interest. These microorganisms accelerate the oxidation of metals. In copper sulfide leaching, the microorganisms are always present. In the case of in-situ copper leaching it appears to be simply a matter of providing a favorable environment for their growth and activity.

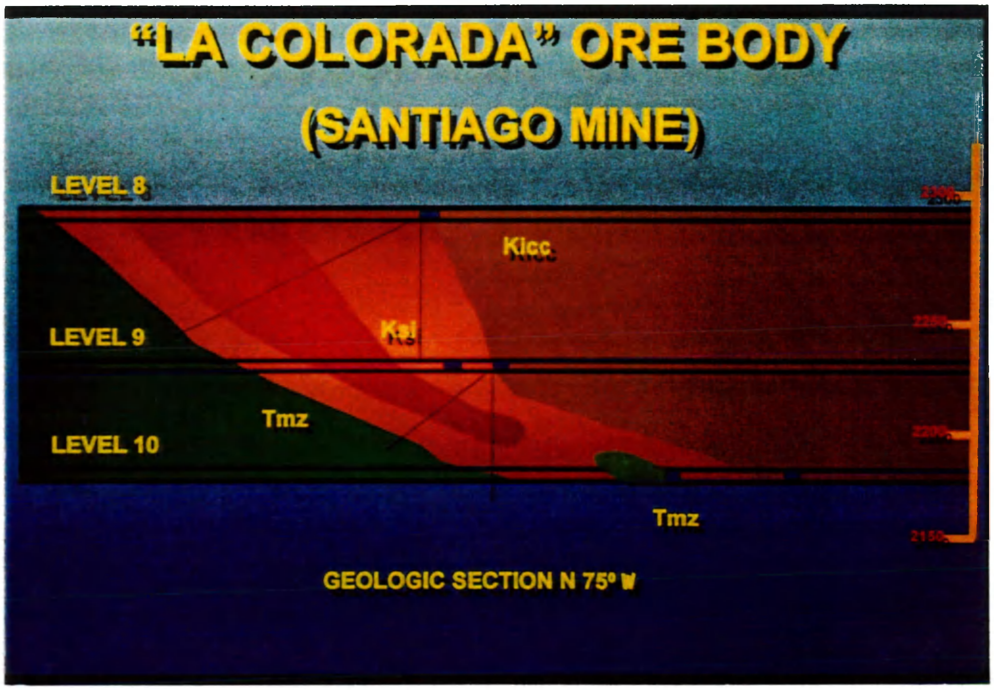

Figure 6. La Colorada" orebody at Santiago Mine.

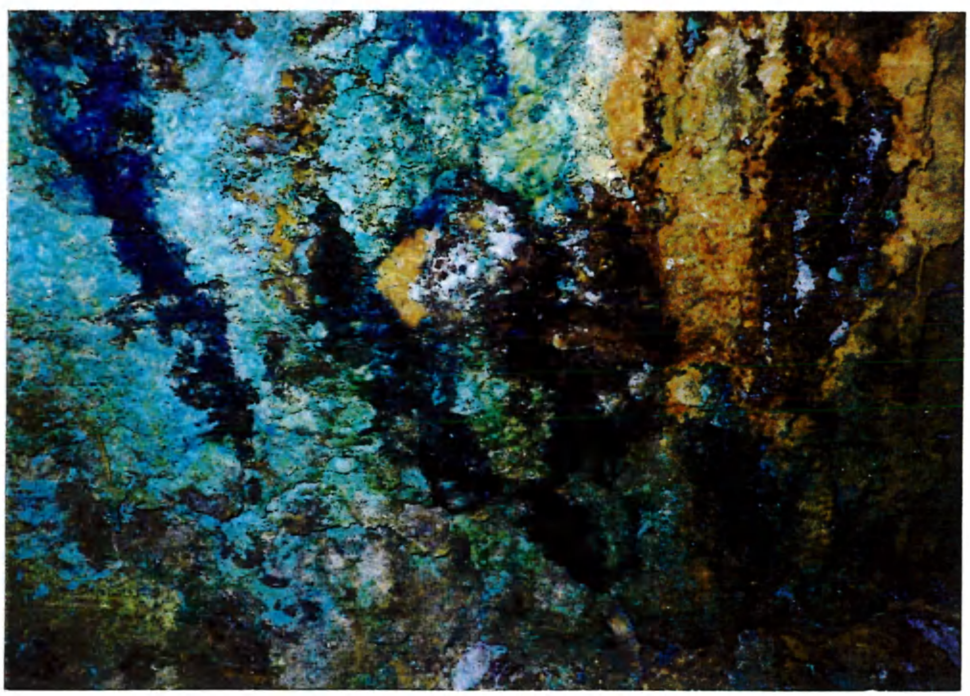

Figure 7. Natural leach solution at Santiago Mine

2.1.1 Laboratory Testing

Various laboratory techniques and methods are currently employed as a means of estimating copper recovery and acid consumption. The most common methods include:

Hot acid soluble copper tests: Used to determine the proportion of copper amenable to recovery by flotation in a mill rather than the amount that could be extracted in a heap leach operation. A small amount of ore is reduced to 100 to 200 mesh and placed in a test tube with a sulfuric acid solution. The sample is agitated while it is heated to 100 degrees Fahrenheit for approximately an hour, solution is then decanted and analyzed for copper. This method will extract most of the copper in the oxide minerals and very little of the copper in sulfide form.

Bottle roll tests: This test generally dissolves all of the copper in oxide minerals, up to to several hundred grams of ore between 100 to 200 mesh are placed in a bottle with a strong sulfuric acid solution, possibly containing several grams per liter of dissolved Fe^{3+}. The bottle is then rolled for 24 hours. The solution is decanted and analyzed.

Column leach tests: Two types of column leach tests are employed in leaching copper:

Closed cycle tests: Closed cycle tests are similar in conception and practice to actual leach operation where LIX-EW is employed to recover copper. Results of these tests provide very reliable data for forecasting actual operating parameters.

Open cycle tests: The leach solution is not recirculated, provided the particle sizing, column height and acid strength duplicate those contemplated in an actual operation. Acid consumption cannot be reliably predicted. There are many factors affecting leach performance, two of the most important are mineralogy and liberation particle size (13).

2.1.2 Laboratory Testing Setup

Based on Tables One and Two, a column size could be 3.25 inches interior diameter and six feet high, with a flow rate of 1.7 ml/minute and an acid concentration of 10 grams per liter of sulfuric acid (Figure 8).

2.1.3 Laboratory Testing Process

Sample Description:

Sample Origin: Solid samples were taken in the Eighth Level of the "La Colorada" orebody at the Santiago Mine and water samples from the Ninth Level of the same orebody.

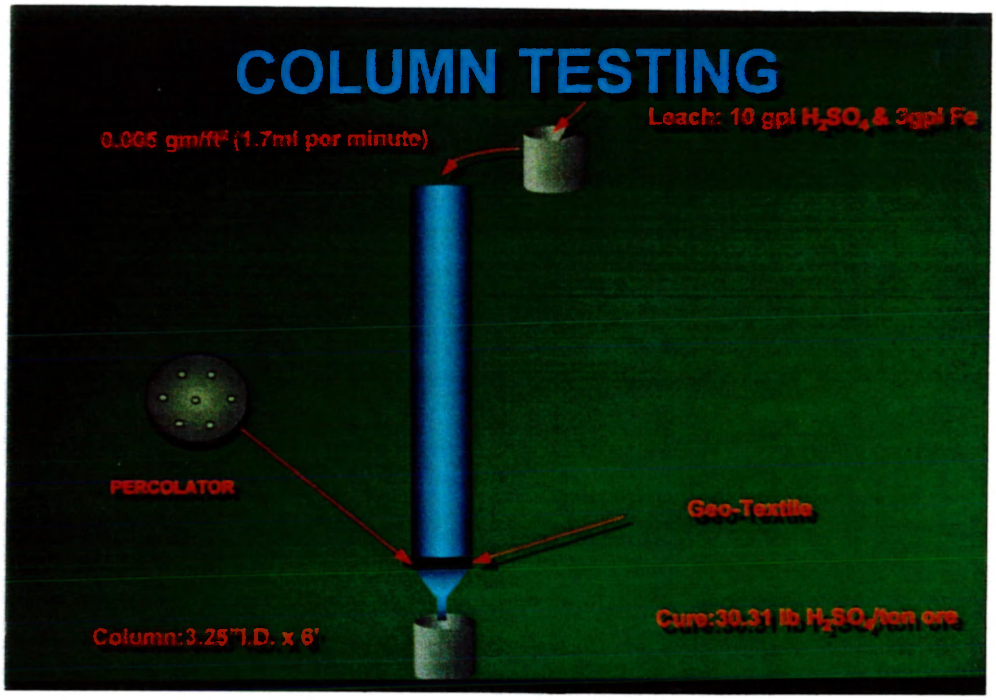

Figure 8. Column testing setup

<u>Mineralogy:</u> Samples one and two contain sulphide and oxide copper minerals in the form of hematite, cuprite, azurite, chalcopyrite, malachite, and limonite.

<u>Method Selection</u>: From a practical point of view, the column leach with closed cycle test is the selected method in order to get the first set of data for estimating the copper recovery and acid consumption. The lixiviant solution is directed to the top of the column. It is important that the lixiviant solution be delivered at a constant flow rate at all times throughout the duration of the test program (Figure 9), for this reason use of a metering pump is recommended. The leach solution traverses the column downward and discharges as pregnant effluent solution at the bottom. The solution is sampled and analyzed on a periodic basis. The volume is measured and the solution is then assayed for elements. The test procedure is ended after a specific number of days have elapsed or when the column of ore no longer contains measurable quantities (10 parts per million) of measurable copper (Figure 10).

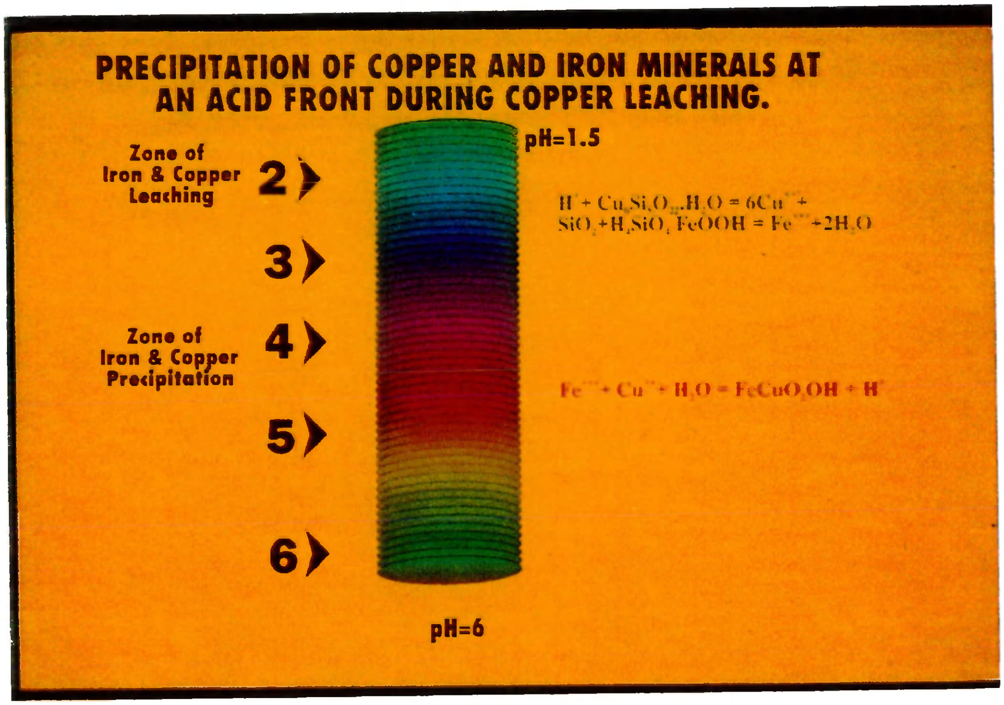

Figure 9. Precipitation of copper and iron during leaching (after Dr. John Dreier)-(13)

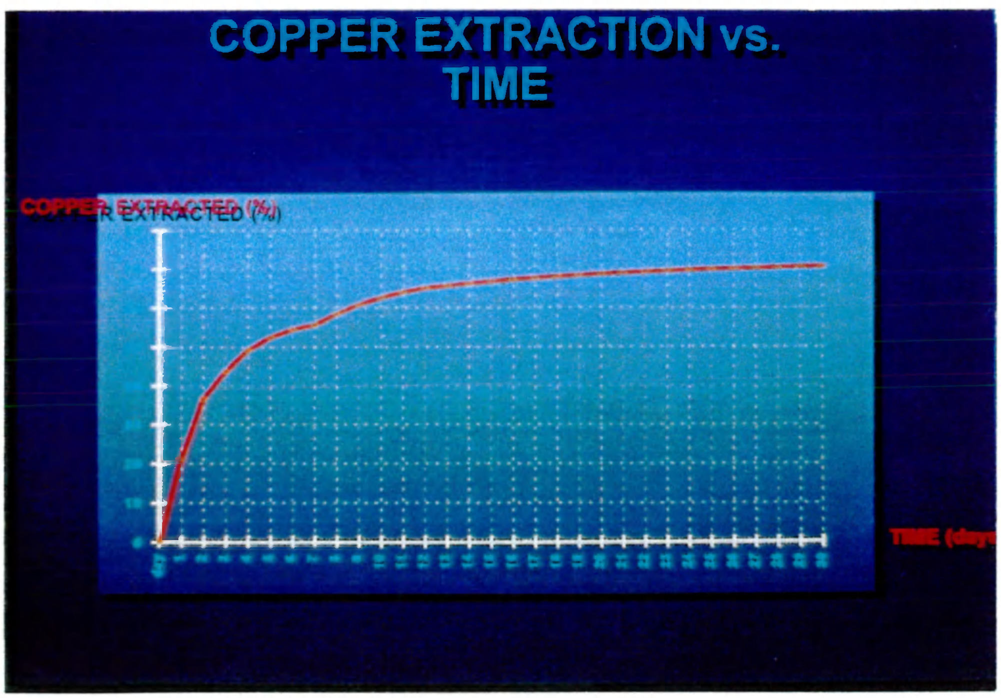

Figure 10. Copper extraction time

The rate at which copper is recovered from a column of ore is dependent on a number of factors including:

Pretreatment: The utilization of a strong sulfuric acid pretreatment for ore has been applied for many years. It has been discovered that the proper application relative to certain ores will minimize sulfuric acid consumption and will significantly improve the project economics. The optimum quantity of sulfuric acid and the concentration at which that quantity is added can only be determined by experimentation.

Ore size distribution: An analysis of different particle sizes is necessary to optimize the amount of copper extracted versus time.

Column diameter and height: The diameter and height of a column are influenced by several parameters. If a small quantity of ore is available, small columns are used to conduct scoping studies. For test material, the particle size is limited to 12 millimeters (1/2 inch). Samples as small as four kilograms (9 lbs) in total weight can be studied in 50 millimeter diameter by 1.2 meter high leach columns.

A "rule-of-thumb" which relates the maximum ore particle size to column diameter has been derived from sampling theory. Column diameter should be three to four times larger than the maximum dimension of the largest particle in the test charge. Table 1 lists the average sample weight required for different leach column diameters and heights, assuming a bulk density of crushed ore of 100 pounds per cubic feet.

Flow rate: The column flow rate is the single most difficult variable to continuosly control. As such, special pumps are built for this purpose. Table 2 translates typical industrial irrigation rates into laboratory solution application for various diameter leach columns.

Acid consumption: The most expensive reagent consumed during copper leaching is sulfuric acid. In a typical laboratory column leaching test, lixiviant and effluent solutions are routinely assayed for ferrous and ferric iron, free sulfuric acid and copper. A running total is kept of acid input and output.

Ancillary reagents: (1)Surfactants such as FC-740 introduced by Minnesota Mining and Manufacturing (3M) are used to reduce the surface tension of leach solutions and many investigations have been conducted using common surface tension modifiers to induce more rapid and thorough lixiviant penetration of copper ore particles. (2) Binding agents such as ammonia which enhance and/or mantain the percolation characteristics of leaching solution and have been utilized for considerable period of time. (3) Chemical modifiers such as alcohols and phenols are used to sequester certain ionic species and are currently being studied in order to determine the influence of solubility modifications upon current copper leaching methodologies (14).

Table 1 ORE WEIGHT REQUIREMENT IN LBS (Bulk Density of 100 lbs/cubic ft)				
Column Diameter (Inches)	Column Height in ft			
	4	6	10	20
2	9	13	22	-
3	20	29	49	-
4	35	52	87	-
6	78	117	196	393
8	140	209	349	698
12	214	471	785	1571
24	1256	1885	3141	6203
36	2827	4241	7068	14137

Table 2 TYPICAL TEST SOLUTION FLOW RATES (ml per minute)				
Column Diameter (Inches)	Irrigation Rate - gpm/sq ft			
	0.0025	0.005	0.075	0.01
2	0.2	0.4	0.6	0.8
3	0.5	0.9	1.4	1.9
4	0.8	1.7	2.5	3.3
6	1.9	3.7	5.6	7.4
8	3.3	6.6	9.9	13.2
12	7.4	14.9	22.3	29.7
24	29.7	59.4	89.1	118.9
36	66.9	133.8	200.7	267.5

Source: K D Engineering Co., Inc., METCON Research Inc. Tucson, Arizona.(14)

Copper mineralogy: Mineralogy is one of the most importants factors for copper dissolution. Common leaching reactions for copper minerals are presented in Figure 11. In the case of silicates, the rate of dissolution is proportional to the solubility of the principal metal oxides in the mineral; this means the more soluble the principal metal oxide is, the faster the mineral will dissolve. The only silicate minerals that are thermodinamically stable in acid leach solutions are quartz and pyrophyllite, all the other silicates are unstable.

Copper sulfide minerals break down much more slowly than copper oxides. The reason for this is that the copper sulfides and iron sulfides are composed of iron and sulfur in their reduced form and in order to break down these minerals the iron and sulfur must be oxidized. Host rock minerals determines acid consumption in the case of limestone, and acid generation as the case of associated pyrite. Typical sulfuric acid leaching solutions are presented in Figure 12, based on data from actual operations and typical copper ores.

2.1.4 Testing Analysis Results

Column testing can generate a large quantity of data. As a consequence, computer spreadsheet and data management are often necessary.

Previous mineralogic reconstruction:

Mineral	Percentage
Cuprite	0.60
Native Copper	0.50
Covelite	0.91
Chalcopyrite	0.47
Azurite and Malachite	5.38

Analysis of samples:

Mineral	Sample#1	Sample#2
Copper (acid soluble),%	2.14	3.77
Calcium, %	12.90	9.80
Total copper, %	2.67	4.26
Silica as Si, %	19.80	21.2
Sulfur, %	4.79	5.07
Gold, oz/ton	0.004	0.006
Silver, oz/ton	0.39	0.63
Cobalt, %	0.004	0.004

Analysis of water samples:

Parameter	Result	Detection limit	Method
Sample#1			
Copper, mg/l (d)	4130.00	0.01	EPA 220.1
pH	4.22	0.01	EPA 150.1
Silver, mg/l (d)	0.04	0.01	EPA 272.1
Zinc, mg/l (d)	1090.00	0.01	EPA 289.1
Sample#2			
Copper, mg/l (d)	1340.00	0.01	EPA 220.1
pH	2.47	0.01	EPA 150.1
Silver, mg/l (d)	0.01	0.01	EPA 272.1
Zinc, mg/l (d)	372.00	0.01	EPA 289.1

(d) = Dissolved.

Permeability range is between 200 and 300 milidarcys.

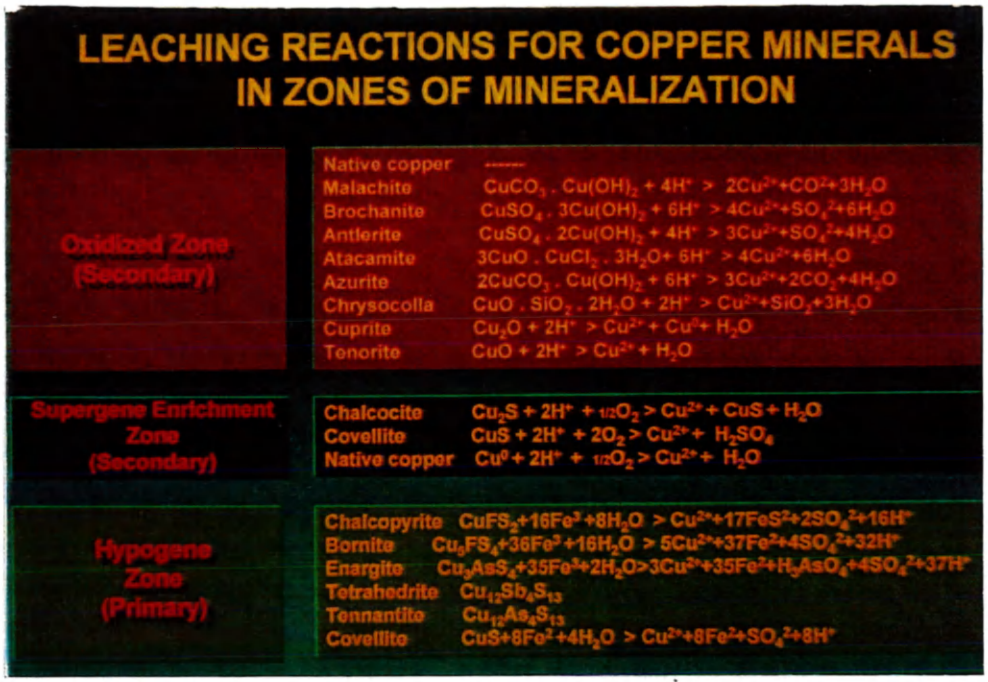

Figure 11. Leaching reactions for copper minerals in zones of mineralization
(after Dr.J.B. Hiskey)-(4)

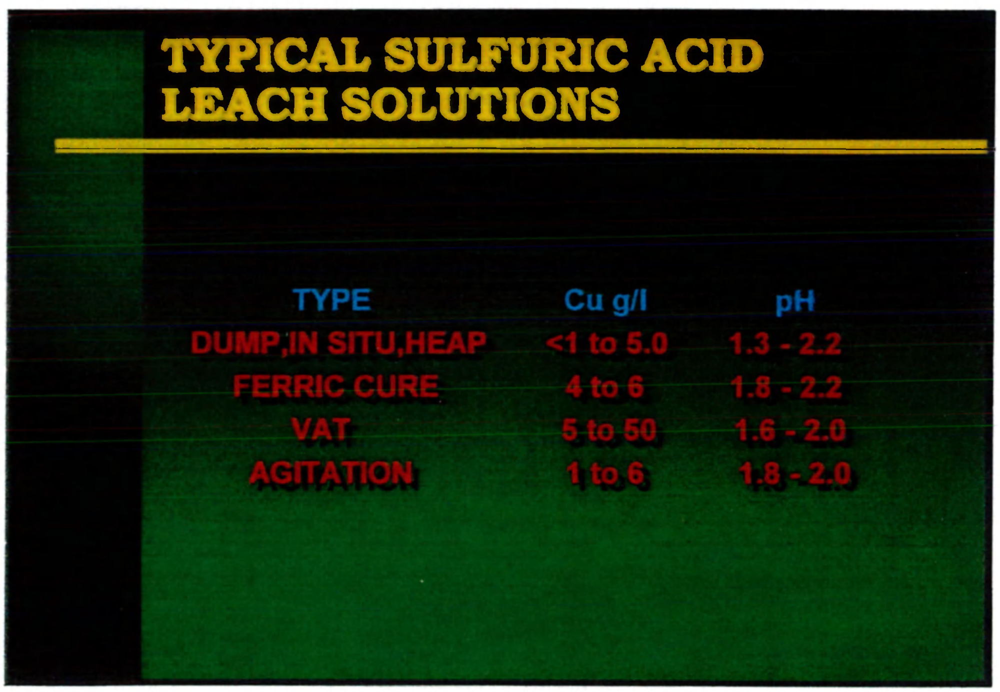

Figure 12. Typical sulfuric acid leaching solutions
(Copper Heap Leach, SME Short Course, Reno,NV, 1993)-(14)

CHAPTER 3.

ECONOMIC ANALYSIS.

Economic analysis involves evaluation of the relative merits of investment situations from a profit and cost view point. A cost estimating for capital and operating costs of order of magnitude, discount cash flow rate of return analysis (DCFROR), sensitivity analysis and risk analysis are presented in this economic analysis. Uncertainty is analyzed for variables such as copper price, recovery, grade, and operating costs using statistics with a computer program named @RISK developed by the Palisade Corporation combining Lotus 123 and Excel spreadsheets .

3.1 Cost Model

Capital cost is based on current U.S. dollars; variables are working capital, process materials and installation. Plant equipment cost is for a pilot scale plant. Diamond drilling and induced fracturing are depending upon field characteristics that have to be evaluated by a geologist. Laboratory testing can be performed by an external laboratory or by the resources of the same company, however a criteria for cost and accuracy has to be determined in order to decide which option is prfered. Installation as well as laboratory testing can be performed by an external construction company or by the company resources. Criteria for calculating capital cost is conservative, it means the highest cost in this case is $113,965.00, (See Table 3).

Operating cost is based in U.S. current dollars, using as unit cents per pound of copper produced, for both mine and plant with the major components as reagents, power, labor and maintenance, results from calculations are 16.93 cents for solvent extraction and electrowinning (SX-EW) and 15.44 cents per pound for mine (See Table 3), however, these values can be reduced. Cash flow analysis is for a short term (six years) with the investment set in the first thirty months. Calculations for gross revenue from sales in annual bases and deductions of annual expenses and some basic calculations for taxes are exposed in table 4. Copper composition and pregnant leaching solution (PLS) from the testing analysis results in pages 16 and 17. Flow rate is estimated for the size of the ore body, with SX-EW recovery of a standard plant as well as production rate (See Table 4). Discounted cash flow rate of return analysis (DCFROR) gives a net present value of $157,656.84 and a internal rate of return of 67%, greater than the interest rate of 15% (See Table 5).

3.2 Sensitivity Analysis

Sensitivity analysis is used to see how investment profitability is affected by variation in the parameters that affect overall profitability of the project. Analysis is conducted using normal distribution for PLS grade with a mean of 1.34 grams per liter (gpl) and with a standard deviation of 0.8 for a wide range from 0.54 gpl to 2.14 gpl. For copper price the mean is 90 cents per pound based on the published results of a survey of copper price forecasts by the International Economics Department of the World Bank for the period 1990 to 2000 with a standard deviation of 0.25, in other words a higher price of 115 and lower price of 65 cents per pound. Recovery and operating costs are conducted using a triangular distribution with minimum, most likely, and maximum values (See table 6). Results for this analysis are given in table 7 which gives a bigger net present value of $174,166.32 and an internal rate of return of 69% (Figure 13). Sensitivity analysis for PLS grade gives a minimum of 0.52 grams per liter of copper (See Figure 14). The project can continue to be profitable for a copper price bigger than 65 cents per pound, (See Figure 15).

3.3 Risk Analysis

Risk analysis ascertain estimates as ranges of values associated with probabilities of occurrence. A range of uncertainty is expressed for variables (See Table 6). The computer program @RISK uses Latin Hypercube sampling which accurately recreate the probability distributions specified by the functions (@normal and @triang) in fewer iterations when compared with Monte Carlo sampling. This sampling method, offers great benefits in terms of increased sampling efficiency and faster run times, especially in a personal computer based simulation environment.

Table 3
Capital and operating costs

PROJECT CAPITAL COSTS			
ITEM			U.S., Dlls.
Plant Equipment costs delivered to site			$7,800.00
Bulk materials costs delivered to site			$2,000.00
Installation labor & subcontracts			$3,500.00
Dimond drilling			$35,000.00
Laboratory testing			$5,000.00
Liners			$2,800.00
Drilling and blasting			$12,000.00
Field construction management and supervision			$10,000.00
Engineering and project services			$13,000.00
Pumps and pipes			$4,500.00
Others			$3,500.00
Contingency @15%			$14,865.00
TOTAL CAPITAL COST			**$113,965.00**
PROJECT OPERATING COSTS			
SX-EW			
		UNITS/TON	CENTS/POUND
ITEM	UNIT	OF COPPER	OF COPPER
Sulfuric acid	LBS	610	0.92
Organic diluent	GAL	18.3	1.46
Organic reagent	LBS	7.6	2.42
Anodes	EA	0.05	0.68
Cathode blanks	EA	0.02	0.32
Miscellaneous SX supplies			0.5
Miscellaneous EW supplies			0.9
Total SX-EW supplies			**7.2**
Equipment power	KWH	300	1.2
Lighting	KWH	25	0.1
EW-power rectifiers	KWH	1775	7.1
Miscellaneous	KWH		0.04
Total SX-EW power			**8.44**
SX labor	HRS	1.2	0.3
EW labor	HRS	2	0.5
Total operating labor			**0.8**
Maintenance labor	HRS	0.8	0.26
Maintenance supplies			0.23
Total maintenance			**0.49**
Total SX-EW operating cost			**16.93**
MINE			
			CENTS/POUND
ITEM			OF COPPER
Labor			0.8
Sulfuric acid			1.2
Pumping			10.2
Maintenance			1.5
Others			1.7
Total Mine operating cost			**15.4**
TOTAL OPERATING COST			**32.33**

Table 4
Cash flow calculations

CASH FLOW CALCULATIONS						
YEAR	1994	1995	1996	1997	1998	1999
TONS OF ORE (metric tons)			250,000.00			
Cu GRADE (% acid soluble)			2.14	2.14	2.14	2.14
RECOVERY (%)			0.70	0.70	0.70	0.70
PLS GRADE (GPL)			1.34	1.34	1.34	1.34
FLOW RATE(GPM)			100.00	100.00	100.00	100.00
SX-EW RECOVERY (%)			0.90	0.90	0.90	0.90
CATHODE COPPER PRODUCTION (lbs)			1,449.28	1,449.28	1,449.28	1,449.28
CATHODE COPPER PRODUCTION (lbs per year)			228,986.47	457,972.94	457,972.94	457,972.94
PRICE, CATHODE COPPER ($/lb)			0.90	0.90	0.90	0.90
REVENUE FROM CATHODE COPPER			1,304.35	1,304.35	1,304.35	1,304.35
GROSS REVENUE FROM SALES (annual)			206,087.82	412,175.65	412,175.65	412,175.65
EXPENSES						
UNIT MINE OPERATING COST ($/lb)			0.15	0.15	0.15	0.15
ANNUAL OPERATING COST			35,263.92	70,527.83	70,527.83	70,527.83
SX-EW OPERATING COST ($/lb)			0.17	0.17	0.17	0.17
ANNUAL SX-EW OPERATING COST			38,767.41	77,534.82	77,534.82	77,534.82
FREIGHT ($/lb)			0.05	0.05	0.05	0.05
FREIGHT ANNUAL			11,449.32	22,898.65	22,898.65	22,898.65
TOTAL ANNUAL EXPENSES			(85,480.65)	(170,961.30)	(170,961.30)	(170,961.30)
NET AFTER COSTS			120,607.17	241,214.35	241,214.35	241,214.35
SALES TAX (15%)			30,913.17	61,826.35	61,826.35	61,826.35
CORPORATE TAX(37%)			11,437.87	22,875.75	22,875.75	22,875.75
PROFIT-SHARING FOR EMPLOYEES (13%)			15,678.93	31,357.87	31,357.87	31,357.87
ANNUAL TAX FIXED ASSETS (2%)			1,251.54	2,503.09	2,503.09	2,503.09
NET AFTER TAXES			61,325.65	122,651.30	122,651.30	122,651.30
INVESTMENT CASH FLOW	(40,000.00)	(40,000.00)	(33,965.00)			
NET CASH FLOW	(40,000.00)	(40,000.00)	27,360.65	122,651.30	122,651.30	122,651.30

Table 5
Discounted cash flow, rate of return (DCF-ROR)

YEAR	CASH FLOW	CUM CASH FLOW	CASH FLOW DISC.(15%)	CUM. DISC. CASH FLOW	CASH FLOW (30%)	CASH FLOW (70%)
0	(40,000.00)	(40,000.00)	(40,000.00)	(40,000.00)	(40,000.00)	(40,000.00)
1	(40,000.00)	(80,000.00)	(34,782.61)	(74,782.61)	(30,769.23)	(23,529.41)
2	27,360.65	(52,639.35)	20,688.58	(54,094.03)	16,189.73	9,467.35
3	122,651.30	70,011.95	80,645.22	26,551.19	55,826.72	24,964.64
4	122,651.30	192,663.25	70,126.28	96,677.47	42,943.63	14,685.09
5	122,651.30	315,314.55	60,979.37	157,656.84	33,033.56	8,638.29
		Σ=	157,656.84		77,224.41	(5,774.04)
NPV =	$157,656.84					
IRR =	67%					

Table 6
Sensitivity analysis for cash flow

SENSITIVITY ANALYSIS						
RECOVERY: @TRIANG(0.20,0.60,0.78)						
PLS GRADE: @NORMAL(1.34,0.80)						
SX-EX RECOVERY: @TRIANG(0.85,0.9,0.97)						
COPPER PRICE: @NORMAL(0.9,0.25)						
MINE OP. COST: @TRIANG(0.06,0.154,0.20)						
SX-EW OP. COST: @TRIANG(0.07,0.163,0.27)						
CASH FLOW CALCULATIONS						
YEAR	1994	1995	1996	1997	1998	1999
TONS OF ORE (metric tons)			250,000.00			
Cu GRADE (% acid soluble)			2.14	2.14	2.14	2.14
RECOVERY(%)			0.53	0.53	0.53	0.53
PLS GRADE (GPL)			1.34	1.34	1.34	1.34
FLOW RATE(GPM)			100.00	100.00	100.00	100.00
SX-EW RECOVERY (%)			0.91	0.91	0.91	0.91
CATHODE COPPER PRODUCTION (lbs)			1,460.02	1,460.02	1,460.02	1,460.02
CATHODE COPPER PRODUCTION (lbs per year)			230,682.67	461,365.33	461,365.33	461,365.33
PRICE, CATHODE COPPER ($/lb)			0.90	0.90	0.90	0.90
REVENUE FROM CATHODE COPPER			1,314.02	1,314.02	1,314.02	1,314.02
GROSS REVENUE FROM SALES (annual)			207,614.40	415,228.80	415,228.80	415,228.80
EXPENSES:						
			0.14	0.14	0.14	0.14
ANNUAL OPERATING COST			31,834.21	63,668.42	63,668.42	63,668.42
			0.17	0.17	0.17	0.17
ANNUAL SX-EW OPERATING COST			38,677.79	77,355.59	77,355.59	77,355.59
FREIGHT ($/lb)			0.05	0.05	0.05	0.05
FREIGHT ANNUAL			11,534.13	23,068.27	23,068.27	23,068.27
TOTAL ANNUAL EXPENSES			(82,046.13)	(164,092.27)	(164,092.27)	(164,092.27)
NET AFTER COSTS			125,568.26	251,136.53	251,136.53	251,136.53
SALES TAX(15%)			31,142.16	62,284.32	62,284.32	62,284.32
CORPORATE TAX (37%)			11,522.60	23,045.20	23,045.20	23,045.20
PROFIT-SHARING FOR EMPLOYEES (13%)			16,323.87	32,647.75	32,647.75	32,647.75
ANNUAL TAX FIXED ASSETS (2%)			1,331.59	2,663.19	2,663.19	2,663.19
NET AFTER TAXES			65,248.04	130,496.08	130,496.08	130,496.08
INVESTMENT CASH FLOW	(40,000.00)	(40,000.00)	(33,965.00)	0	0	0
NET CASH FLOW	(40,000.00)	(40,000.00)	31,283.04	130,496.08	130,496.08	130,496.08

Table 7
Sensitivity analysis for net present value

	CASH FLOW	CUM	CASH FLOW	CUM. DISC.	CASH FLOW	CASH FLOW
YEAR		CASH FLOW	DISC.(15%)	CASH FLOW	(30%)	(70%)
0	(40,000.00)	(40,000.00)	(40,000.00)	(40,000.00)	(40,000.00)	(40,000.00)
1	(40,000.00)	(80,000.00)	(34,782.61)	(74,782.61)	(30,769.23)	(23,529.41)
2	31,283.04	(48,716.96)	23,654.47	(51,128.14)	18,510.67	10,824.58
3	130,496.08	81,779.12	85,803.29	34,675.15	59,397.39	26,561.38
4	130,496.08	212,275.19	74,611.56	109,286.71	45,690.30	15,624.34
5	130,496.08	342,771.27	64,879.61	174,166.32	35,146.39	9,190.79
		Total=	174,166.32		87,975.53	(1,328.31)
NPV=	$174,166.32					
IRR=	69%					

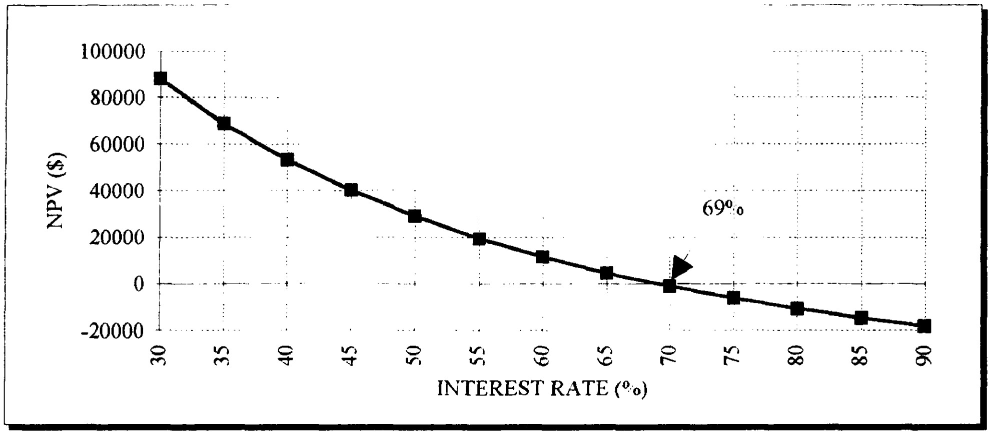

Figure 13
Net present value profile

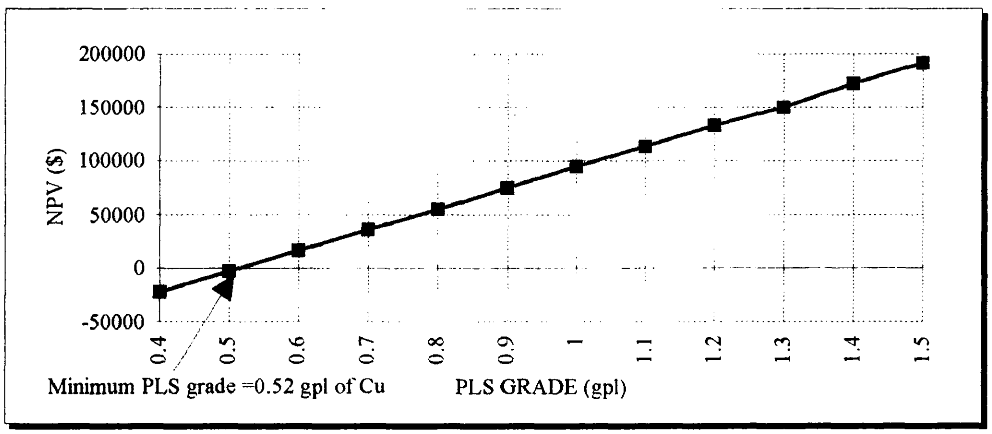

Figure 14
PLS grade sensitivity analysis

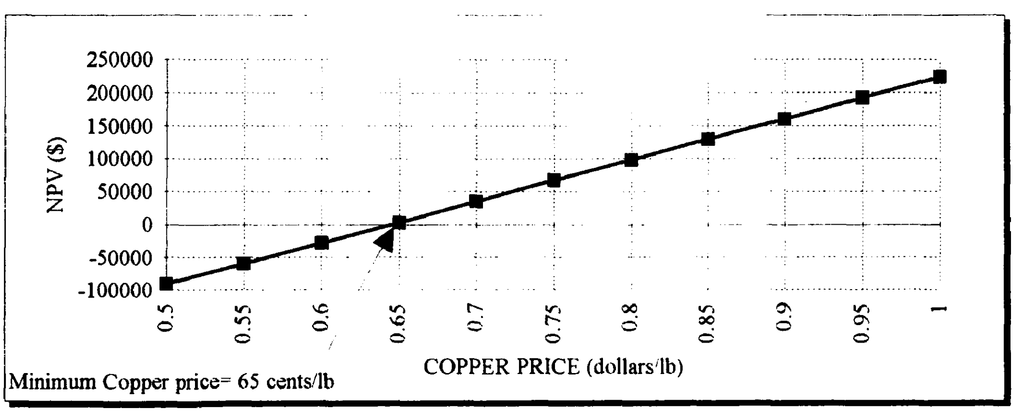

Figure 15
Copper price sensitivity analysis

Figures 16 and 17 show the results in graphic form for the project using 5000 iterations. Statistical results on table 8 shows that the project has a high probability of positive result. Statistics for variables are presented in graphic form from Figure 18 through Figure 23. Even those results are encouraging, the administrative, financial, engineering, and exploration professionals of the company need to establish their own criteria and stategies in finalizing their decisions.

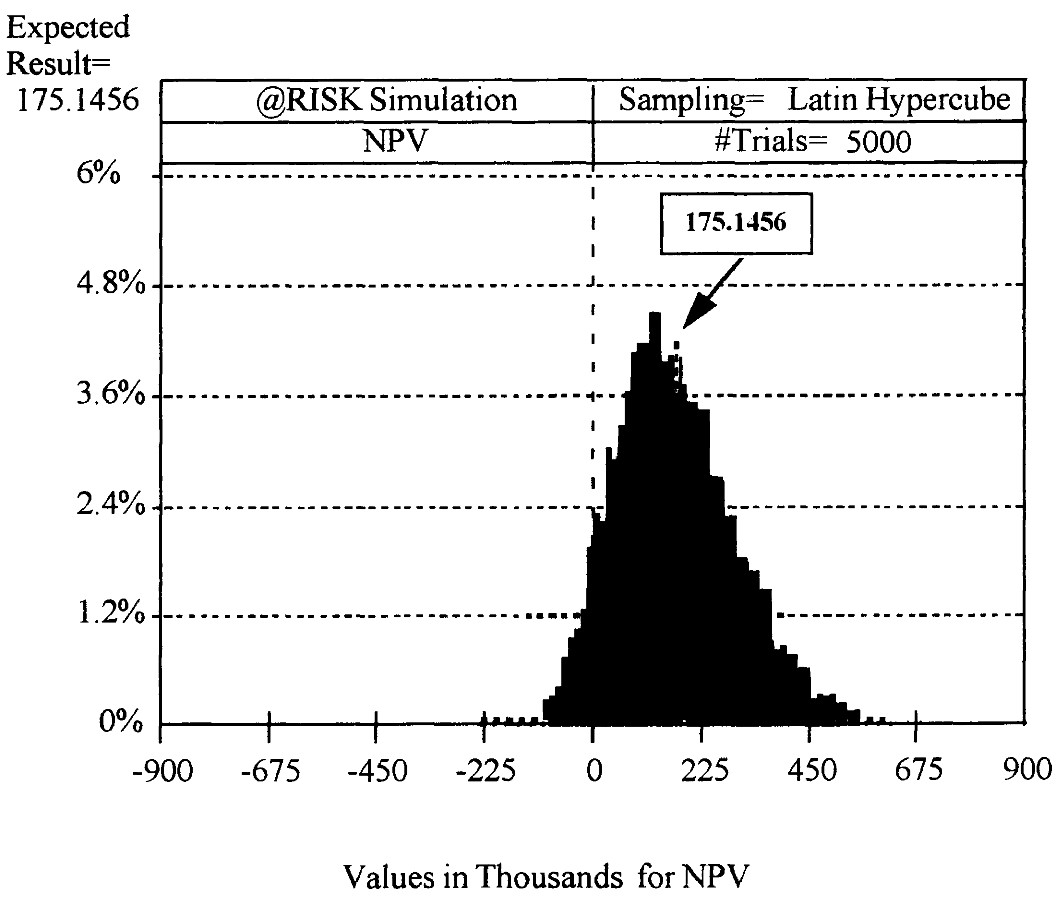

Figure 16. Risk analysis probability distribution

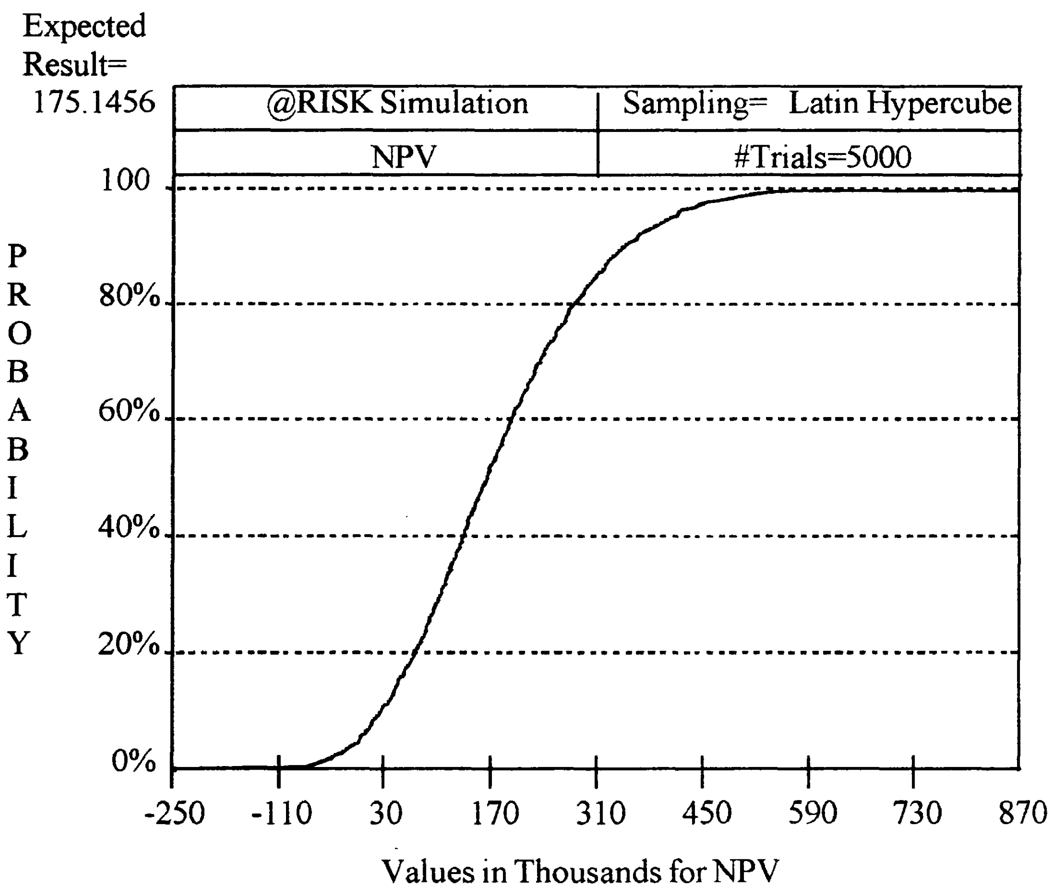

Figure 17. Risk analysis cumulative probability

Table 8
Risk analysis statistics results

EXPECTED/MEAN RESULT=	175,145.60				
MAXIMUM RESULT=	862,586.60				
RANGE OF POSSIBLE RESULT=	1,102,513.00				
MINIMUM RESULT=	(239,926.10)				
PROBABILITY OF POSITIVE RESULT=	93.40%				
PROBABILITY OF NEGATIVE RESULT=	6.60%				
STANDARD DEVIATION=	128080.5				
VARIANCE=	1.64E+10				
SKEWNESS=	0.5505522				
KURTOSIS=	3.516571				
ERRORS CALCULATED=	0				
VALUES FILTERED=	0				
PROBABILITY OF RESULT:(Thousands)					
	>=0 =93.4%			<0.00= 6.60%	
	>=90 =73.4%			< -45= 2.32%	
	>=180=44.7%			< - 90= 0.58%	
	>=270=21.58%			< -135= 0.14%	
	>=360= 8.54%			< -180=0.04%	
	>=450= 2.74%			< -225=0.02%	
	>=540= 0.74%				
CUMULATIVE PROBABILITY:(Thousands)					
	%	VALUE		%	VALUE
	0	<=-239.926		55	<=178.580
	5	<= -11.3591		60	<=194.934
	10	<= 21.2438		65	<=213.231
	15	<= 45.4703		70	<=232.988
	20	<= 66.9022		75	<=253.892
	25	<= 84.9682		80	<=278.137
	30	<= 100.982		85	<=306.917
	35	<= 117.000		90	<=343.915
	40	<= 130.960		95	<=404.539
	45	<= 146.179		100	<=862.587
	50	<= 162.289			

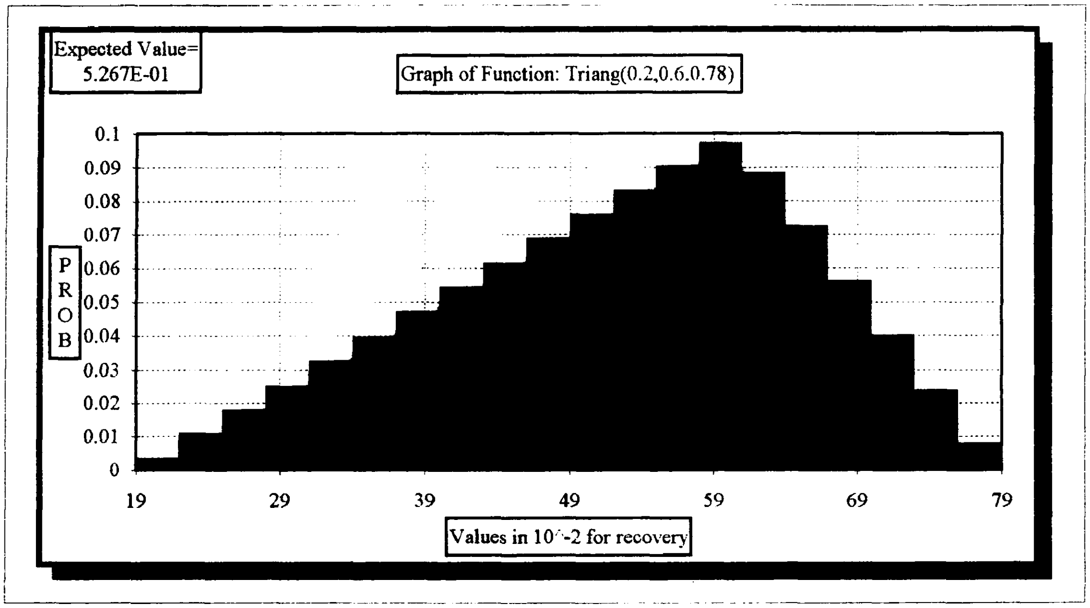

Figure 18
Risk analysis results for recovery

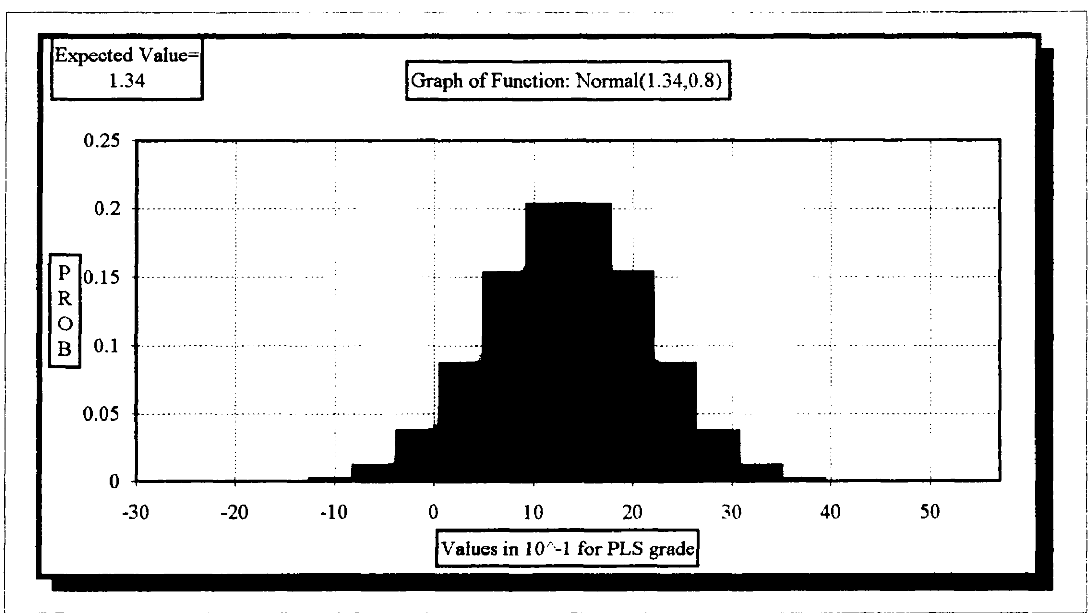

Figure 19
Risk analysis results for PLS grade

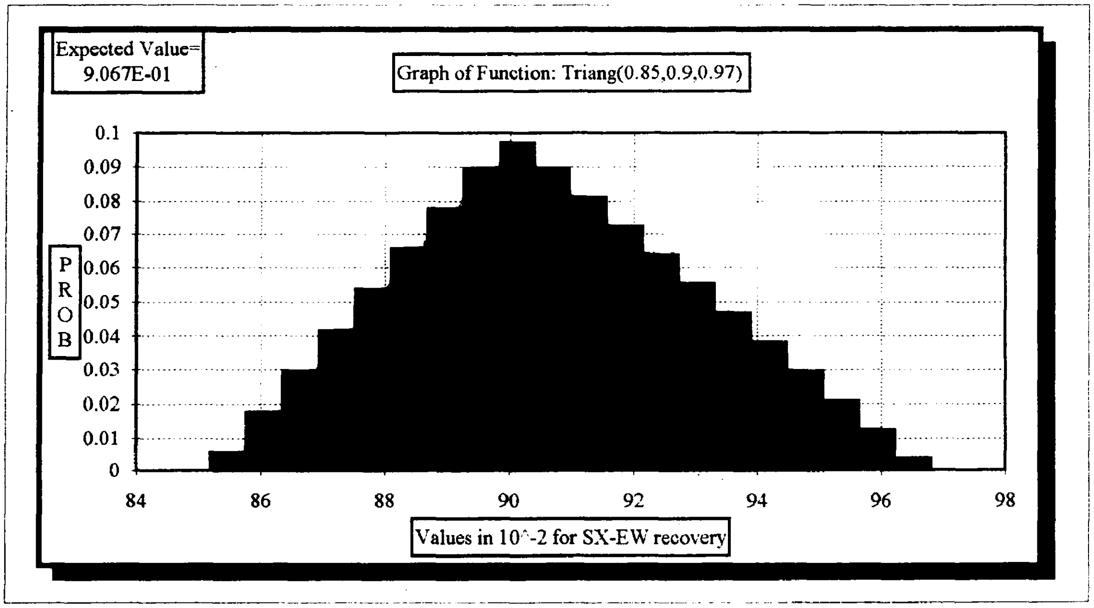

Figure 20
Risk analysis results for SX-EW recovery

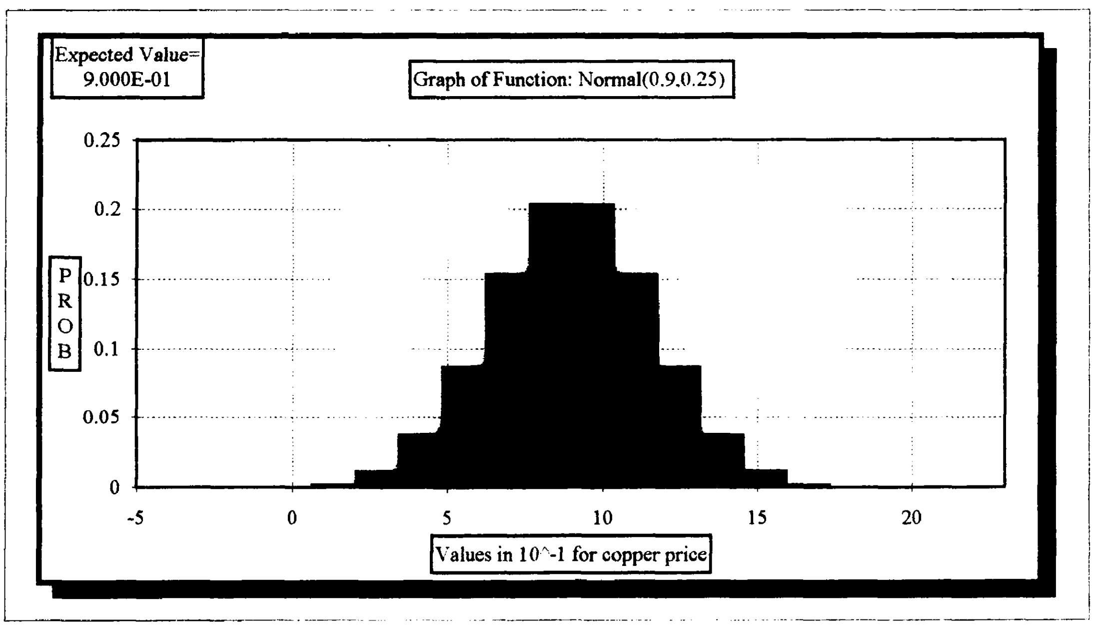

Figure 21
Risk analysis results for copper price

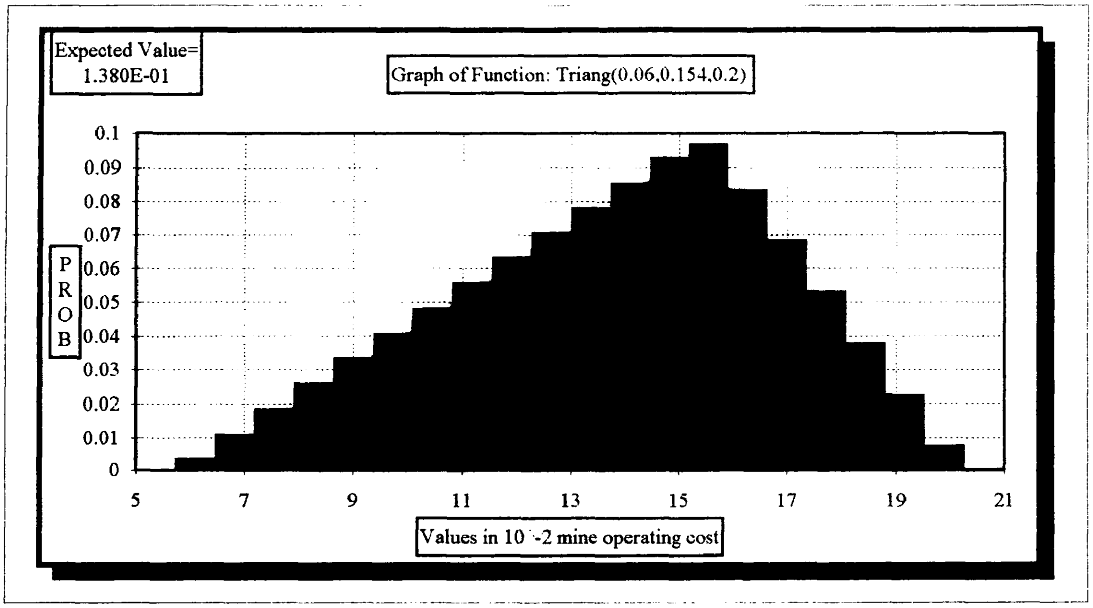

Figure 22
Risk analysis results mine operating cost

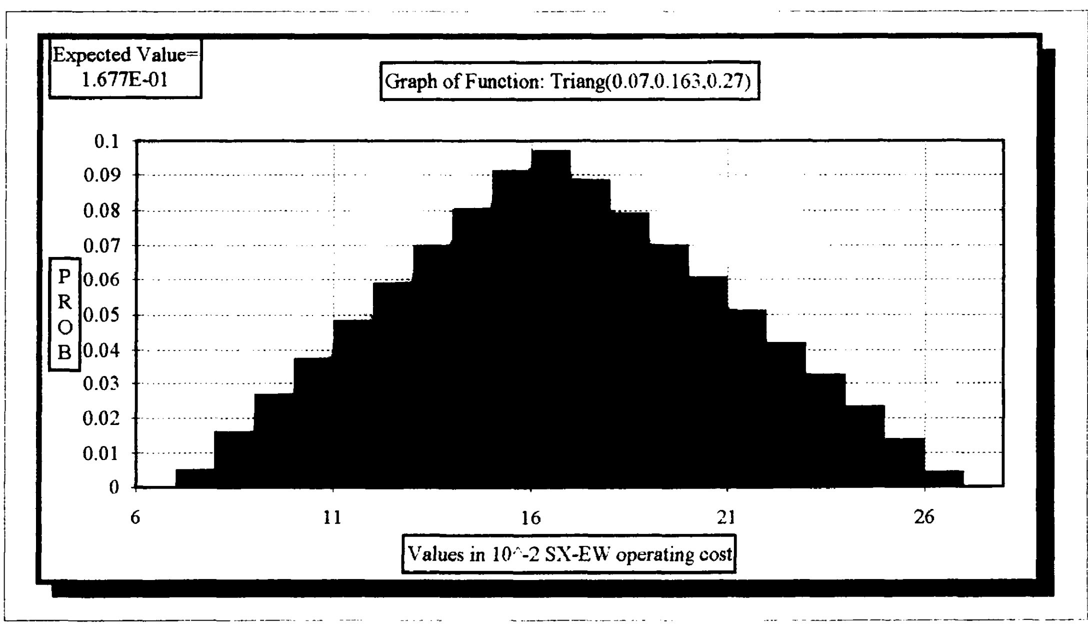

Figure 23
Risk analysis results for SX-EW operating cost

CHAPTER 4.

CONCLUSION AND RECOMMENDATIONS

Leaching is found to be potentially an effective and economical method of recovering copper from low-grade ore deposits and can probably be successfully applied at the Santiago Mine. All leaching processes are facilitated by bacterial action. Determining the presence or lack of microorganisms will be helpful but not indispensable for the operation. Utilizing native microorganisms present in mineral deposits takes less time to adapt to leaching conditions and can be stimulated with appropriate nutrients.

Implementation of a Leaching System in Minera Tayahua could contribute to improved mining by:

*Maximum extraction of ore bodies that are small, deep and/or low-grade which would be uneconomical to mine using conventional methods.
*Lower production costs, because energy needs are reduced, ore extraction and crushing are eliminated, and labor requirements are lessened.
*Better worker health and safety, because there is no haulage equipment operating on site, and mine employees are not exposed to underground mining hazards.
*Less environmental disruption, because there is no need to mine.

In general, for an in-situ operation advantages and disadvantages are as follow:

Advantages:

1). Is more environmentally attractive, since it creates less surface disturbance.
2). Requires less capital and lower operating costs. Energy requirements in leaching are much lower than conventional methods to obtain the same quantity of metal.
3). Recovers metals economically from mineralized deposits that could not be extracted by more conventional mining, milling and smelting processes.
4). Increases a mine's ore reserves. Low-grade or inaccessible ores zones, caved ore, dumps and tailings may become ores, thereby leading to more effective mineral utilization and conservation.
5). Leach solutions obtained usually lend themselves to a variety of metal recovery processes. The pure metal or metal compounds obtained may be of greater value to a mine owner than the sulfide or oxide products normally obtained through conventional milling processes.
6). Can often be used in conjunction with a conventional mining or milling processes to boost metal recoveries and increase ore reserves.

Disadvantages:

1). Both physical and chemical restrictions may limit the usefulness of leaching. The effectiveness of contacting ore with solutions and recovery of leach solutions from the system without appreciable loss are two important physical factors. Dissolution rates, metal precipitation during the process, and solution regeneration chemistry are major factors.
2). Testing an in situ mining project short of actual field operation may prove to be difficult.
3). Ground-water contamination may result from in situ mining operations.
4). Detailed information on the physical, chemical, and bacteriological factors involved in situ extraction of metals and minerals have not yet been established commercially.

Recommendations:

A detailed geological and hidrological study of the orebody is necessary in order to prevent solution losses or ground water contamination. Testing is the basis of the predevelopment stage, it is mandatory to run several laboratory tests with variation in particle size, flow rate, reagents, and acid concentration in order to make reliable predictions concerning acid consumption, rate of recovery and the percent of extraction. Since leaching is present in natural form in the mine, an in situ pilot test is also suggested at the "La Colorada" orebody (Figure 24).

If copper is to be recovered by iron cementation, ferric iron in pregnant leach solution must first be reduced to the ferrous state by scrap iron before precipitation of elemental copper. This reaction results in the production of ferrous sulfate without the regeneration of sulfuric acid. The cementation process consumes more sulfuric acid than utilization of the solvent extraction-electrowinning process on the same ore. Therefore, recommend the last method be employed. In addition, a better quality copper can be produced.

A pilot scale test can be structured in five stages after testing results:

First Stage: Leaching without blasting, measurement of parameters.
Second Stage: Leaching subsecuent to blasting, leach and obtain data.
Thirth Stage: Compare results, economics, model development.
Fourth Stage: Construction of a pilot scale SX-EW plant (underground).
Fifth Stage: Evaluation for a full scale process following Figure 25.

Evaluation Process: Special attention should be directed towards the mineral composition during the early stages of the project. Recognizing the minerals and understanding of their chemical characteristics are the basis through which project decisions are made and expenses are minimized. The next step is the laboratory testing, which has to give a level of confidence in the recovery rates, acid consumption and pregnant leaching solution grade.

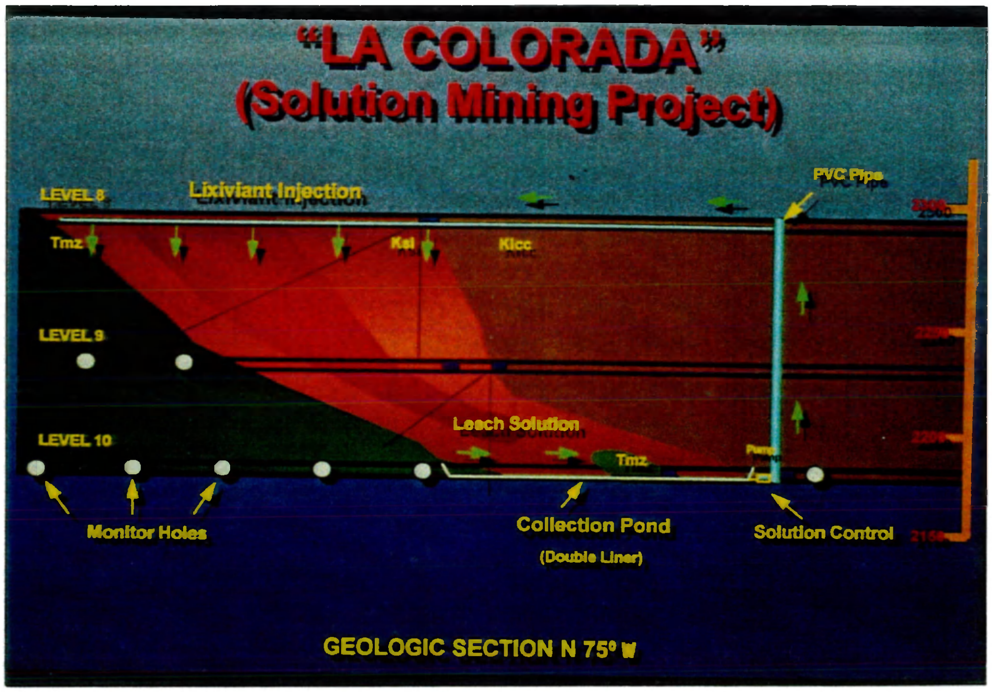

Figure 24. In situ leaching design for "La Colorada" orebody at Santiago Mine

After collecting information on the parameters, a feasibility study has to be performed. Project decisions will depend on the amount of risk the company is willing to accept. The less risk the company is willing to bear, however, the longer it will take and more expensive it may be to make the decision. In more aggressive organizations decision making is based on relatively limited information. A more conservative organization may continue to evaluate the same project by conducting a large detailed engineering design. It is important realize that laboratory test results may differ from the field, however, it gives a base to evaluate the project. One approach for decision making is to invest only in quality opportunities that allow the investment to at the least break even under a number of worst-case scenarios.

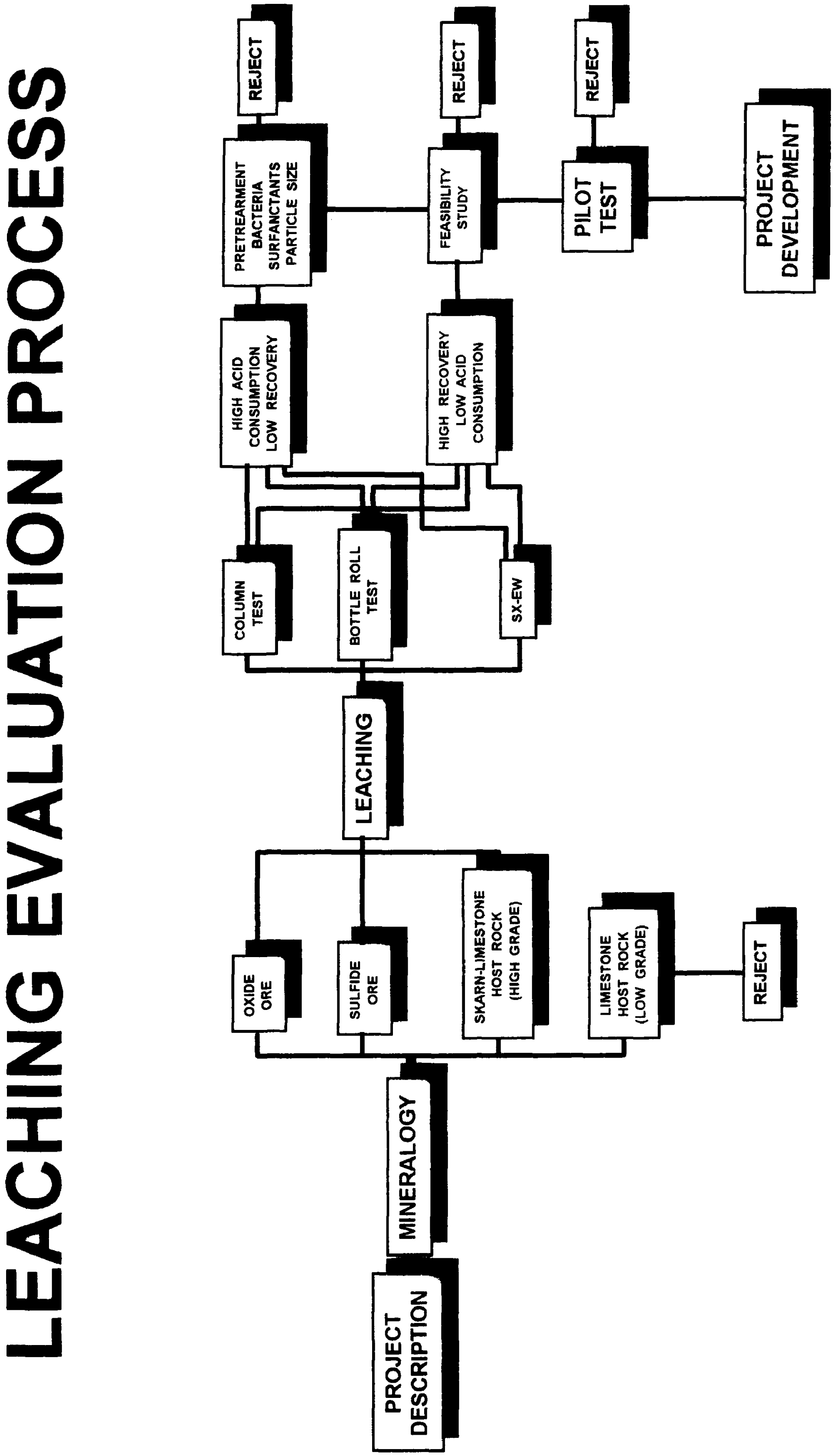

Figure 25. Leaching evaluation process

Suggested design criteria for in situ leaching is presented below:

General design criteria:

<u>Design criteria requirements:</u>

1. Economic:
    - -Ore Reserves
    - -Mine Life
    - -Copper price
    - -Operating and Capital costs
    - -Cutoff grade
    - -Extraction rate
    - -Recovery: Mine and SX-EW plant.

2. Regulatory:
    - -Permitting
    - -Environmental

3. Operational:
    - -Layout
    - -Infrastructure
    - -Solution application rate
    - -Particle size: Fragmentation

<u>Constraints:</u>

1. Mineralogical:
    - -Mineral solubility in acid
    - -Host rock
    - -Unwanted minerals precipitation

2. Geological:
    - -Structure
    - -Faulting
    - -Ground water
    - -Foundations

3. Chemical
    - -Acid consumption
    - -Reaction balance control: Eh-pH
    - -Reagents: surfactants, modifiers, binders.

4. Physical:
   - -Location
   - -Geometry
   - -Dimensions
   - -Particle size

5. Metallurgical:

   -SX: Flow rate, solution composition,settler specific flow, mixer residence time, mixer configuration, depth of organic and aqueous, mixer and settler configuration, materials of construction, instrumentation and control, crud removal system.

   -EW: Cathode production rate, cathode area and current density and efficiency, cell size, cell configuration, tank house piping, rectifier sizing, cathode harvest area

6. Biological:
   - -Bacteria recognition and activation
   - -Bacteria addition

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