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Minerals • Minéraux

SEPARATION OF SULPHIDES FROM MILL TAILINGS PHASE I

MEND Report 2.45.1a

This work was done on behalf of MEND and sponsored by
Lac Minerals
Les Mines Selbaie
Placer Dome Inc., as well as
the Ontario Ministry of Northern Development and Mines and
the Canada Centre for Mineral and Energy Technology (CANMET)
through the CANADA/Northern Ontario Development Agreement (NODA)

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Phase I (MEND)
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FINAL REPORT

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Executive Summary

A review of various mineral processing technologies and their application to the removal of sulphides from mill tailings has been completed for the partners of the MEND consortium. The project included a descriptive review as well as bench scale testing of several processing methods.

Five mill tailings samples were obtained from three operating mines including:

- Placer Dome - Detour Lake Mine
- Lac Minerals - La Mine Doyon
- Les Mines Selbaie - Blast 5180.020 North
- Les Mines Selbaie - Blast 5180.022 Center
- Les Mines Selbaie - Blast 5240.141

The quantity of sample from Les Mines Selbaie - Blast 5180.020 North was insufficient for testing purposes and was therefore rejected. The samples were processed by gravity and flotation separation processes to split the samples into sulphide and non-sulphide fractions. The separation processes tested included:

- Falcon Concentrator
- Knelson Concentrator
- Reichert Tray
- 1/8 Wilfley shaking table
- Carpco LC3000 spiral concentrator
- direct flotation with a variety of pre-processing steps and collectors.

The samples were characterized initially, processed with the above techniques and reanalysed to determine the effectiveness of the sulphide separation. The characterizations included humidity cell tests and acid base accounting, ICP analysis and a mineralogical evaluation.

The results of the study proved that the gravity techniques tested were incapable of separating the sulphides sufficiently to remove the acid generating capability of the tailings. This is likely due to the fact that the sulphide fractions tend to be concentrated in the finest size fractions of the tailings. The separation technique tested that was successful in removing the acid generating potential of the tailings samples was direct flotation.

Sommaire

On a préparé à l'intention des partenaires du consortium du NEDEM (Programme de neutralisation des eaux de drainage dans l'environnement minier) une revue des diverses technologies de traitement des minéraux et de leur application à l'élimination des sulfures des résidus miniers. Le projet comprenait une revue descriptive, ainsi que des essais en laboratoire de plusieurs méthodes classiques de traitement.

On a obtenu cinq échantillons de résidus miniers provenant de trois mines actuellement exploitées, notamment:

- Placer Dome - Mine de Detour Lake
- Lac Minerals - La Mine Doyon
- Les Mines Selbaie - Sautage à 5180.020 Nord.
- Les Mines Selbaie - Sautage à 5180.022 Centre
- Les Mines Selbaie - Sautage à 5240.141

La quantité d'échantillon provenant des Mines Selbaie - Sautage 5180.020 Nord était insuffisante aux fins des tests et par conséquent a été rejetée. Les échantillons ont été traités par des procédés de séparation gravimétrique et de flottation de façon à séparer les échantillons en fractions sulfurées et non sulfurées. Les processus de séparation étudiés comprenaient:

- Le concentrateur Falcon
- Le concentrateur Knelson
- Le plateau de Reichert
- La table à secousses de Wilfley 1/8
- Le concentrateur à spirale Carpo LC3000
- La flottation directe avec divers étapes de prétraitement et collecteurs.

On a caractérisé initialement les échantillons, on les a traités selon les techniques ci-dessus, et on les a réanalysés pour déterminer l'efficacité de la séparation des sulfures. Les caractérisations comprenaient les tests en cellule d'humidité et la détermination acide-base, l'analyse ICP (par plasma inductif) et une évaluation minéralogique.

Les résultats de l'étude ont prouvé que les techniques gravimétriques étudiées ne pouvaient suffisamment bien séparer les sulfures pour éliminer le potentiel de génération d'acides des résidus. Ceci vient probablement du fait que les fractions sulfurées tendent à se concentrer dans les plus petites fractions des résidus. La technique de séparation étudiée qui a réussi à éliminer le potentiel acide des échantillons de résidus était la flottation directe.

1.0 INTRODUCTION

A contract was awarded to Cominco Engineering Services Ltd. on Sept 14, 1993, to study the potential of various conventional mineral processing techniques to reduce the acid generating potential of concentrator tailings. The objectives of the program are summarized as follow:

- Evaluate various mineral processing techniques that could be applied to the separation of sulphides in mill tailings
- Evaluate the relationship between sulphide elimination and acid generation
- Evaluate the operating costs of various separation options

Other issues covered in the Scope of Work include recommendations on further work on the most promising applicable techniques, as well as the presentation of the study findings to the MEND Prevention and Control Committee at a location to be determined.

1.1 Background and Project Scope

The long term environmental impact and financial liabilities of acid mine drainage (AMD) from tailings and waste rock are probably the most serious environmental issues facing the Canadian mining industry today. Improved understanding of the feasibility of separating sulphide tailings into their non-sulphide and sulphide fractions will allow Canadian mine operators greater flexibility with their tailings impoundment options.

Since the beginning of 1988, the Mine Environment Neutral Drainage (MEND) Program, through its Federal, Provincial and Industrial partners, has sponsored research into ways of reducing the environmental impact and financial liabilities of acid mine drainage (AMD) from tailings and waste rock. The establishment of disposal techniques that will enable the operation and closure of acid generating sites in a predictable, affordable, timely and environmentally acceptable manner is one of the main objectives of the MEND program.

The onset of acid generation is due to the bacteria catalyzed oxidation of the sulphides contained in the waste rock or tailings. Most of the acid drainage work to date has looked at the elimination of water and/or oxygen from the tailings. both of which are required to oxidize the sulphides. The present work examines the removal of the sulphides allowing the sulphide portion to be treated separately while the non-sulphide portion may be disposed of by conventional techniques.

If the sulphide fraction may be separated from the non-sulphide fraction, several disposal options could become available. Since the sulphide portion would have a significantly reduced volume, separate underwater disposal in a lined containment area could become realistic. The non-sulphide product, if non-acid generating, could then be disposed of on surface or as part of a cover for existing tailings.

Canada Centre for Mineral and Energy Technology (CANMET) within Energy, Mines and Resources (EMR) Canada, the Ontario Ministry of Northern Development and Mines through the Canada/Northern Ontario Development Agreement (NODA), Lac Minerals of Toronto, Les Mines Selbaie and Placer Dome Inc. have funded the present work as part of the MEND program. The project is to study various conventional mineral processing techniques that could be applied to the separation of sulphides in mill tailings, and the degree to which sulphide elimination impacts on the acid generating tendency of the non-sulphide fraction.

The project as defined in the Statement of Work provided by MEND was to encompass four samples of acid generating mill tailings received from different locations. The project consisted of characterization of those samples, bench scale testing of the most promising samples with various mineral processing techniques applicable to the removal of sulphides from the tailings and recommending the most promising separation technique for further study.

Given the limited time frame involved, it was considered to be outside the scope of this project to optimize the design parameters for each of the process options tested. As a result, reagent cost estimates provided will be high as will many of the design parameters such as flotation times and therefore vessel sizing and costs. It should be noted that the cost estimates provided are order of magnitude estimates only and as such should be considered to be accurate to +30% and -25% only. The project's main objective was to decide which mineral processing options would be applicable to the removal of sulphides from tailings and this has been accomplished.

2.0 PROJECT METHODOLOGY

The project followed a logical progression as set out in CESL's proposal to MEND for the work. The first step was to characterize the samples received according to the following parameters:

> elemental analysis for base metals, total sulphur, total sulphate and CO_2, acid generating capability by acid base accounting (static test) and humidity cells (kinetic test), and mineralogical evaluation for distribution and liberation of sulphides. Following characterization, the tailings were processed by a variety of mineral processing techniques to separate the sulphide fraction of the samples. The types of separation methods evaluated were broken into three groups:

- gravity separation
- flotation
- other

Following processing, the treated tailings were re-evaluated according to the criteria used to characterize the initial samples. Once the feasibility and effectiveness of each technology had been evaluated, costs for implementation and operation of the more promising separation methods were developed. Based upon this data, recommendations for future work and further study have been made.

2.1 Samples Received

The following samples from the MEND participants have been received by CESL in Vancouver.

Participant	Date Rec'd	Comments
Selbaie Sample 1	20/10/93	insufficient sample size requested new sample
Lac Minerals	02/11/93	
Placer Dome	03/11/93	
Selbaie Sample 2	12/11/93	
Selbaie Sample 3	not yet rec'd	replacement for sample 1

Duplicate samples, for the purposes of conducting gravity separation tests were received on approximately the same dates by Lakefield Research at Lakefield, Ontario.

Sample Preparation

The samples were received in thickened slurry form. Each sample was repulped and thoroughly mixed using a mechanical agitator. Representative bulk samples were removed from the agitating pulp using a vacuum line. The bulk samples were air dried, blended and riffled to obtain individual samples for elemental analysis, size analysis, mineralogical analysis, determination of neutralization potential (acid-base accounting) and for kinetic testing.

Sample Identification

The following sample numbers have been assigned to identify the origin of the original samples.

Sample	Company	Origin
1	Placer Dome	Detour Lake Mine CIP tailings
2	Lac Minerals	La Mine Doyon CIP tailings
3.1	Les Mines Selbaie	Concentrator tailings (Blast 5180.020 North)
3.2	Les Mines Selbaie	Concentrator tailings (Blast 5180.022 Center)
3.3	Les Mines Selbaie	Concentrator tailings (Blast 5240.141)

2.2 Elemental Analysis

The elemental analysis performed on each sample included a 32 element ICP, silver, gold, total sulphur total sulphate and CO_2. The analysis was performed by Mineral Environments Laboratories in Vancouver. Results of this analysis are provided in Section 4.0.

2.3 Screen Analysis

Each sample was screened through a Canadian Standard sieve series using the following Tyler Equivalent mesh designations:

Tyler (mesh)	Canadian std. (micron)
65	210
100	149
150	105
200	74
325	44
400	37

The individual size distributions are presented in Section 4.0.

2.4 Acid Base Accounting

Samples received were thoroughly mixed to form a slurry from which representative samples including a head sample were drawn using a vacuum line. All samples were prepared in the same fashion.

Each head sample was air dried then riffled and split to obtain representative samples for screen analysis, analysis of solids and determination of neutralization potential (acid base accounting).

A pulverized fraction of each sample was submitted for a 32 element scan by ICP, analysis of total sulphur, sulphate sulphur, gold, silver, CO_2, and mineral determination.

Each sample was screened through a Canadian Standard sieve series of the following Tyler Equivalent mesh numbers: 65, 100, 150, 200, 325, 400 and shaken on a Ro-Tap for 25 minutes. Each fraction was then weighed with weight percent and cumulative weight percent passing calculated for each sample.

The first part of the Modified Sobek Method of acid base accounting is the determination of paste pH. For this test, a small amount of sample was wetted with distilled water and upon saturation, the pH of the paste was recorded.

Two grams of sample were weighed into an Erlenmeyer flask to which standardized acid was added to obtain a target pH of 1.5 to 2.0. The flask was then placed on an oscillating table and agitated for 24 hours. At the end of the shaking period, the contents of the flask were titrated to pH 8.3 with standardized base. This allowed calculation of the neutralization potential (NP) or acid base accounting (ABA) of the sample which can be balanced against the acid producing potential (AP) derived from the sulphur analysis, to yield the net neutralization potential (net NP).

2.5 Humidity Cells

Samples received were thoroughly mixed to form a slurry from which representative samples including a head sample were drawn using a vacuum line. All samples were prepared in the same fashion.

A humidity cell is a chamber designed to model the geochemical processes of weathering. The air, temperature and moisture can be controlled and the weathered products can be analyzed to determine such parameters as the concentration of metals, the rate of acid generation and the onset of acid mine drainage.

A portion of each test sample (approximately 1kg dry) was charged into a standard humidity cell measuring 20 cm in height, 10 cm in diameter and supported by a perforated disc pad 2.0 cm from the bottom. The cells are equipped with one drain port on the bottom for leachate collection and a port 1.0 cm from the bottom as a humid and dry air inlet. Each cell is covered with a top plate fitted with a plastic adapter through which water is pumped to the cell.

Upon start up, humid air was passed through the cells for one week. The following day, the humid air port was closed, the drain hole opened and the pumping cycle initiated. During the pumping cycle of each cell, 500 ml distilled water was pumped over the sample during a 16 hour period. The leachate collected was then filtered through a 0.45 micron membrane filter and analyzed for pH, conductivity, redox, alkalinity, acidity, sulphate and submitted for analysis by ICP-AES.

After the initial leach day the cells are exposed to three days of dry air then three days of humid air. Over the last day of the cycle, 500 ml of distilled water are pumped over the sample and the leachate is collected and analyzed as above. This seven day leach cycle was repeated until a total of ten leach cycles had been carried out at which time a review of the results was conducted to determine the necessity of continuing the test. At that time it was decided to continue the process and a total of 18 cycles were conducted. The one exception to this was sample 3.3 from Les Mines Selbaie because this sample arrived later than the rest. There were a total of 15 cycles conducted on this sample.

2.6 Mineralogical Evaluation

The mineralogical evaluation was performed by an outside contractor. A mineralogical examination of samples 1, 2, 3.2 and 3.3 were made by Harris Exploration Services of Vancouver. The samples were prepared as smear mount polished thin sections. Detailed descriptions of the individual samples are provided in Section 4.0.

2.7 Mineral Processing Techniques

The types of mineral processing techniques evaluated as potentially useful in the separation of sulphides from the tailings were broken into three categories. These categories included:

- gravity techniques,
- flotation techniques, and
- other.

After a review of these techniques, several were selected to perform bench scale tests on the tailings.

2.7.1 Gravity Separation Techniques

Five gravity separation systems were evaluated for the project. These techniques included:

- a Falcon Concentrator,
- a Knelson Concentrator,
- a Reichert Tray,
- a 1/8 Wilfley Table, and
- a spiral concentrator,

Tests on the Knelson and Falcon concentrators were done by suppliers of the respective equipment who were located in the Vancouver area. The spiral tests, the Reichert Tray tests and the Wilfley Table tests were performed by Lakefield Research in Ontario. Detailed descriptions of these techniques are provided in Section 3.0.

2.7.2 Flotation Techniques

Two types of flotation were initially reviewed but only one was tested on bench scale as the other was deemed inappropriate. These two techniques were:

- direct flotation, and
- reverse flotation.

Flotation tests were combined with a variety of preprocessing steps such as cyanide destruction as well as with several different types of collectors and pH conditions.

Direct flotation tests were performed in the CESL mineral processing laboratory in Vancouver. Reverse flotation systems were not tested in this project but will be reviewed for the reader in the following section of this document.

2.7.3 Other Separation Techniques

Other types of separation techniques evaluated in the preliminary review included:

- magnetic separation, and
- cyclone classification.

These technologies were not tested as a primary means of separation due to physical and mineralogical characteristics of the ore. Magnetic separation was tested as a supplemental separation technique for the Placer Dome sample which contains a significant amount of pyrrhotite. A general review of these technologies is given in the following section of this document.

2.8 Evaluation of Processed Tailings

Once the tailings samples had been processed by the chosen techniques, the heads and tails from selected tests were reevaluated according to the same parameters used to characterize the initial samples. This allowed for the generation of mass balances and calculation of efficiencies of each separation process. In the case of those techniques which did not achieve the expected separation efficiency, additional particle size distribution analysis was undertaken to explain the poor results. Due to time constraints and the delayed arrival of the samples, humidity cell tests on the treated tails could not be performed. Estimates of the acid generating potential of the treated tails were made instead by analysis of elemental distribution in the head and tails samples.

2.9 Costs for Implementation and Operation

Following the evaluation of the various separation techniques, cost estimates were developed for implementation of selected processes at each of the sites involved in the project. The estimated capital costs were developed by obtaining quotations for the major mechanical equipment. A factoring method was used to determine the installed cost based on the mechanical equipment cost. The accuracy level of the capital cost estimate using this method is 25 - 30%. Operating cost estimates for the proposed conceptual flowsheet, are based on extrapolations of reagent consumptions from the test program.

2.10 Recommendations for Further Study

Based upon the analytical results of the study and the economic considerations, areas for further study have been identified. These areas have been chosen as they are thought to hold the greatest potential for reductions in the environmental liability of acid generating mill tailings.

3.0 MINERAL PROCESSING TECHNOLOGY DESCRIPTIONS

The following section of this document provides an overview of each of the mineral processing separation techniques which were identified and subsequently reviewed to assess their usefulness for the removal of sulphides from mill tailings. An explanation of the operating principles of each technology is provided and, where possible, diagrams or pictures are provided.

3.1 Falcon Concentrator

The use of centrifugal force to increase separation efficiency of materials by differential specific gravity has been researched for over a century. The earliest patents were granted on the subject in 1891 and continued into the 1920's. Since that time there have been relatively few new developments in the field.

The Falcon concentrator is a gravity separation technique based on the principle of centrifugal separation. That is, creating a gravitational (centrifugal) force by use of a spinning media. In effect, it is a simulation of gravity but many times stronger with a corresponding increase in separation efficiency. The centrifugal force in the Falcon concentrator is equivalent to approximately 300 G's.

There are two types of Falcon concentrators. The earliest types were batch concentrators while recent developments have led to the development of a continuous feed concentrator. The continuous feed technique is depicted in figure 3.1 and is the one which will be described here.

All continuous Falcon concentrators consist of four primary components:

- a drive frame
- a shroud
- a concentrate launder, and
- a rotor.

The feed is introduced through a central pipe to the impeller zone where it is accelerated toward the wall of the rotor by an impeller which is similar in appearance to the impeller of a pump. The feed is further accelerated by the friction caused by the contact with the migration zone wall of the rotor. The length and slope of this wall is selected so that there is a sufficient component of centrifugal force acting parallel to and up the slope to force solids in the slurry to slide upward. The combination of high gravity and shear forces as well as the relative motion of slurried particles in the centrifuge cause particles with the highest specific gravity to be forced to the bottom of the slurry against the rotor wall in the migration zone. The slurry particles orient themselves in order of increasing specific gravity moving from the inner surface of the slurry towards the rotor wall as they slide upwards to the retention zone.

The area immediately above the migration zone where the wall of the rotor becomes parallel to the axis of rotation is known as the retention zone. As the stratified slurry enters this zone it is pushed upwards by particles coming from below, The upper extent of this zone is characterized by a lip which causes the fraction of the slurry nearest the rotor wall (the denser fraction) to be retained. The remainder of the slurry (the tails) slides over the denser base layer and is removed from the top of the rotor.

In the batch operation of the unit, the rotor is spun until a sufficient fraction of the low density material is removed from the centrifuge bowl and the remainder is the product. In the continuous operation, just below the lip a small portal is installed from where the denser fraction of the slurry (the concentrate) is removed.

If the range of specific gravity of the particles to be retained lies in the middle, the separation may have to be done in two stages, the first to remove the low density fraction from the slurry and the second to remove the product fraction from the denser portion of the slurry.

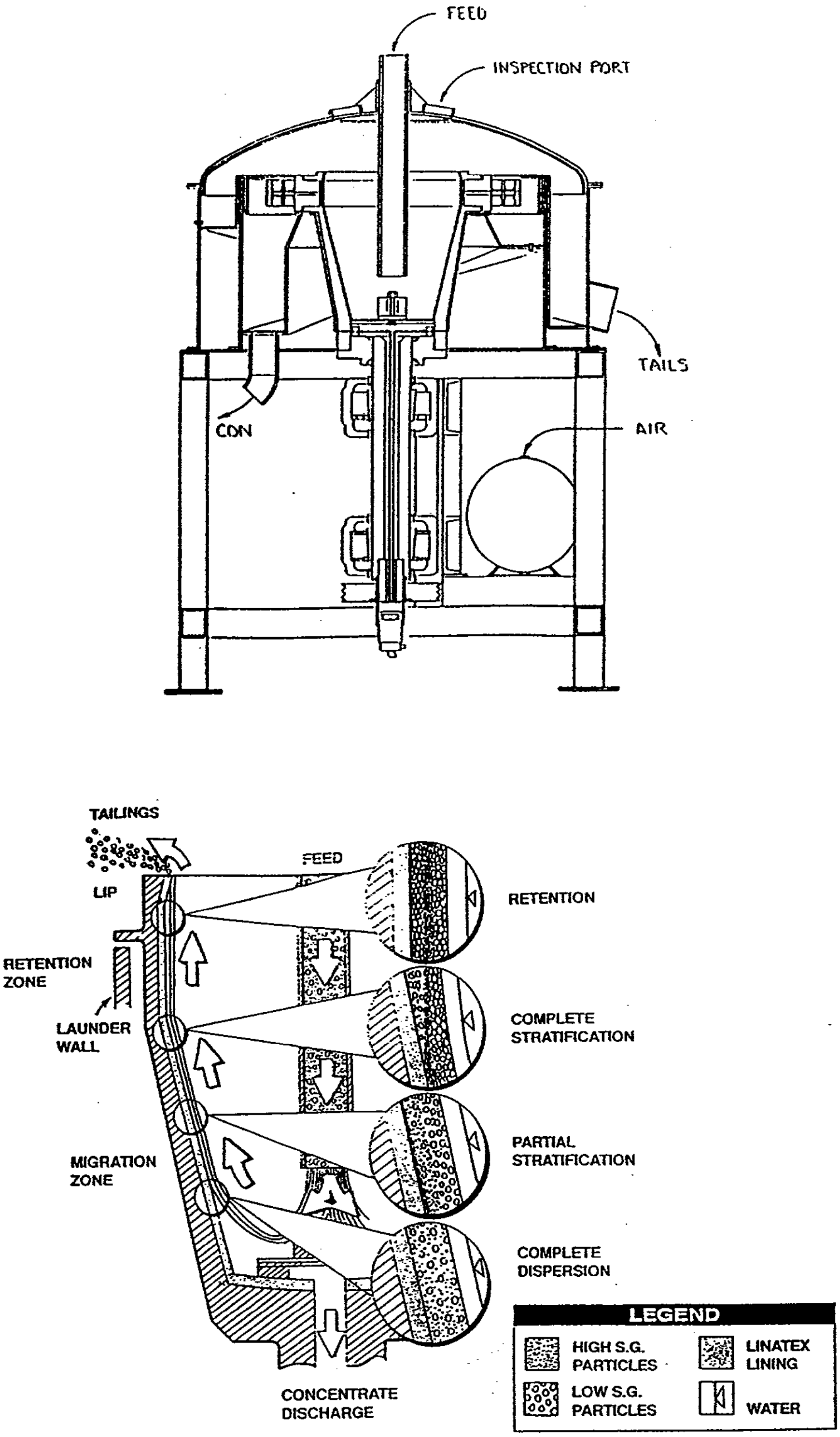


Figure 3.1 - Continuous Feed Falcon Concentrator

3.2 Knelson Concentrator

The Knelson concentrator works on the same principle as the Falcon concentrator. In other words centrifugal force is used to cause the slurried particles to accelerate to the perimeter of a rotating bowl and the relative motion of the particles causes stratification to occur in layers of particle density. A diagram of the Knelson concentrator bowl design is shown in figure 3.2. Whereas the Falcon concentrator creates a 300G environment, the Knelson concentrator separates in the 60G regime.

The gravel slurry is introduced at the centre bottom of a ribbed inner cone that rotates at high rpm. Centrifugal forces cause the initial feed to fill the bottom rib area with solids followed by the next higher slightly larger diameter rib area. Subsequent ribs are sequentially filled a split second later. After that, the following feed moves upward as a thin film over the conical surface of the sand filled ribs until it exits at the top as tailings.

High specific gravity particles in the film of feed slurry passing over the fluidized solids trapped in the ribs of the cone will penetrate the fluidized bed under the enhanced gravity and displace a lower specific gravity particle of the same volume that is trapped in the rib compartment.

Compaction of solids trapped in the spaces between the ribs of the cone is prevented by injecting water through a series of graduated perforations in the cone wall, which fluidizes the trapped solids.

The concentration process continues by "trading" high density particles for less dense particles thereby yielding an enriched concentrate trapped in the bowl.

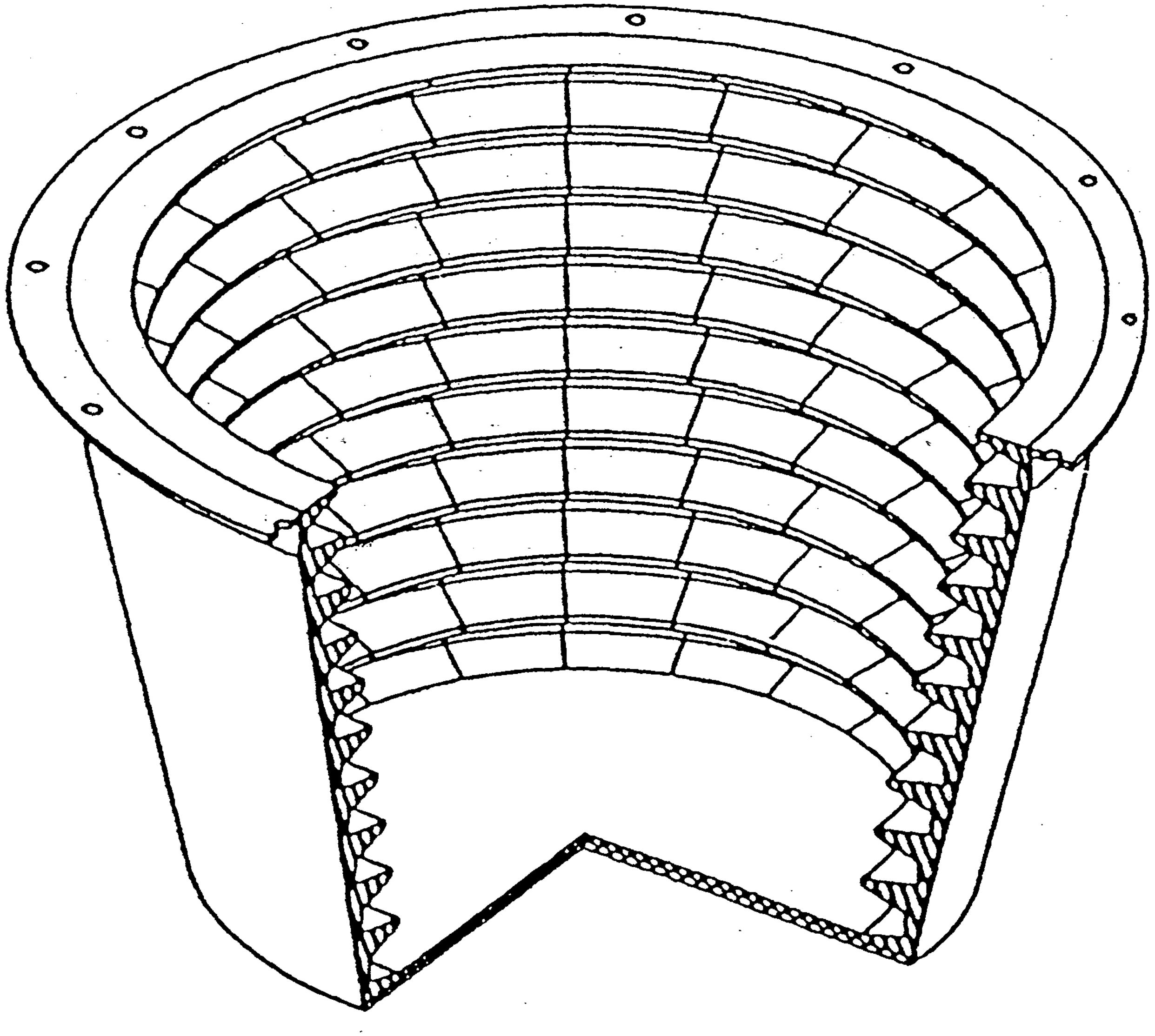


Figure 3.2 - Knelson Concentrator Bowl Design

3.3 Reichert Tray

The Reichert Tray is a bench scale version of the Reichert Cone concentrator. Rather than having a bench scale piece of equipment which requires large quantities of sample to test, a section of the cones has been removed to produce a bench scale testing apparatus which requires much smaller amounts of sample to operate. A schematic representation of a Reichert Tray is shown in figure 3.3. The only difference between the tray and the cone is that the tray sections are replaced by conical pieces which increase its operating capacity.

The Reichert Cone is generally considered a pre-concentrating device. Its main use is for generating a large volume of relatively low grade concentrate. It is somewhat less useful for generating high grade concentrates.

The principle of the Reichert Cone (or tray) is called stream settling. The cones or trays are oriented in sets of two. In the case of cones, the first cone is upright (pointing upwards) while the second in inverted. With trays, the surface area of the tray expands towards the bottom of the tray allowing the feed to spread out while in the second tray the flow area constricts. The slurry is fed to the high end of the first tray. As the slurry flows down the tray and fans out, the depth of flow gets lower and there is preferential settling of higher density particles towards the bottom of the flow. As a result of skin friction, there will be a velocity gradient generated with the velocity of the stream increasing towards the surface of the flow. At the bottom of the first tray, the lower velocity particles will fall through the opening as concentrate and on to the second half of the tray set as feed while the less dense particles will flow over the opening as tails providing feed for the top of the next tray.

The concentrate from the top half of the tray set feeds the bottom half. As the slurry flows towards the bottom of the tray, the flow channel narrows. While there is still preferential settling of denser particle towards the bottom of the flow as the flow moves toward the bottom of the tray its depth increases. At the bottom, a split occurs by use of a weir or trough which retains a bottom fraction of the flow as concentrate and rejects the top of the flow as tails. The concentrate from here is taken off as "concentrate 1" while the tails provide feed to the next tray set along with the tails from the upper tray. The second tray set produces "concentrate 2" and a tails flow which provides feed for the third tray set and so on. In this way the first tray of a set acts for recovery of as much high grade material as possible while the bottom tray in the set acts as a purification step. A series of concentrates are thus produced with decreasing purity.

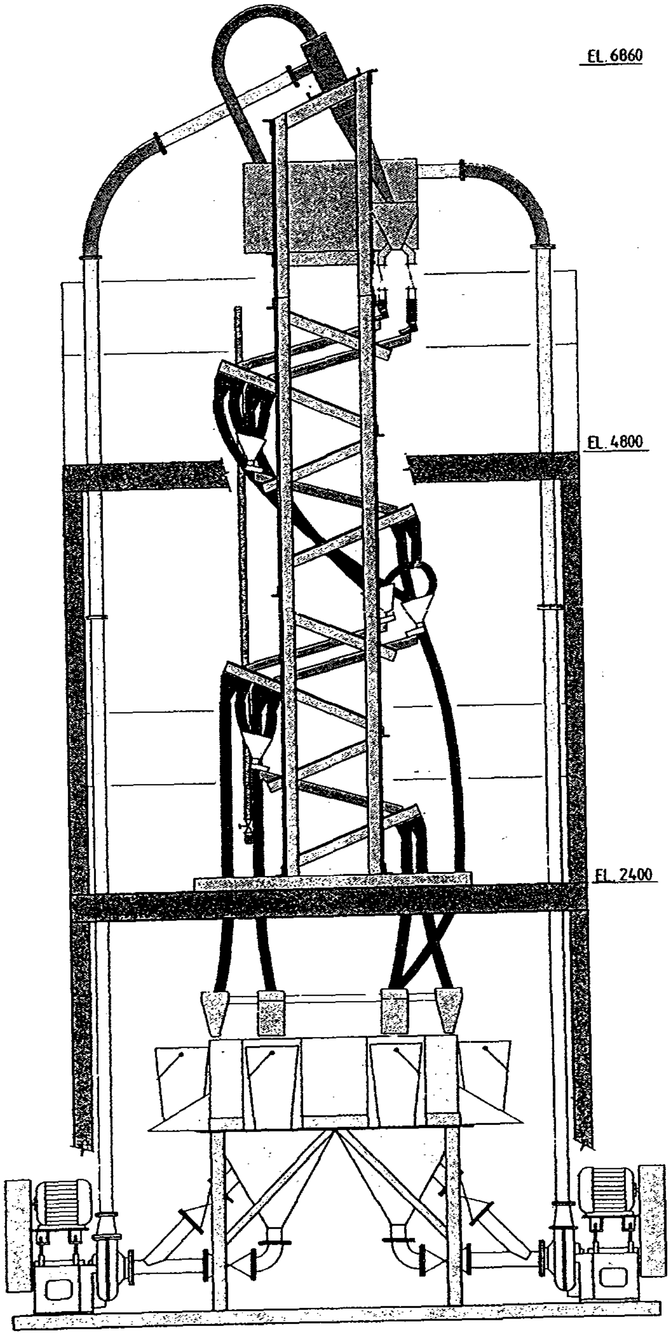


Figure 3.3 - Reichert Tray Schematic

3.4 Spiral Concentrator

The spiral concentrator is also a gravity type separation technique. It consists of an open or closed channel which has been coiled into a helical formation. At the top there is a feed distribution box for even slurry feed and at the bottom there is a collection header to recover the tails from a bank of spirals. There may be several helix coils nested within one another but this primarily a space saving issue. A schematic representation of this technique is presented in figure 3.4

The slurry enters through a feed box at the top of the column and onto the spiral surface. Once on the spiral surface, the slurry particles settle and sort according to size shape and specific gravity. A specific gravity difference of greater than one is required to make separation possible. Particles with the lowest specific gravity are carried with the water towards the outside wall of the spiral. The density of the lower specific gravity stream becomes more dilute as it contains the bulk of the liquid.

The spiral separates the slurry in this fashion as is illustrated above. At intervals along the spiral, there are outlets along the inner wall of the channel which bleed off the denser fraction of the slurry. The less dense material then passes through the spiral and exits as tails.

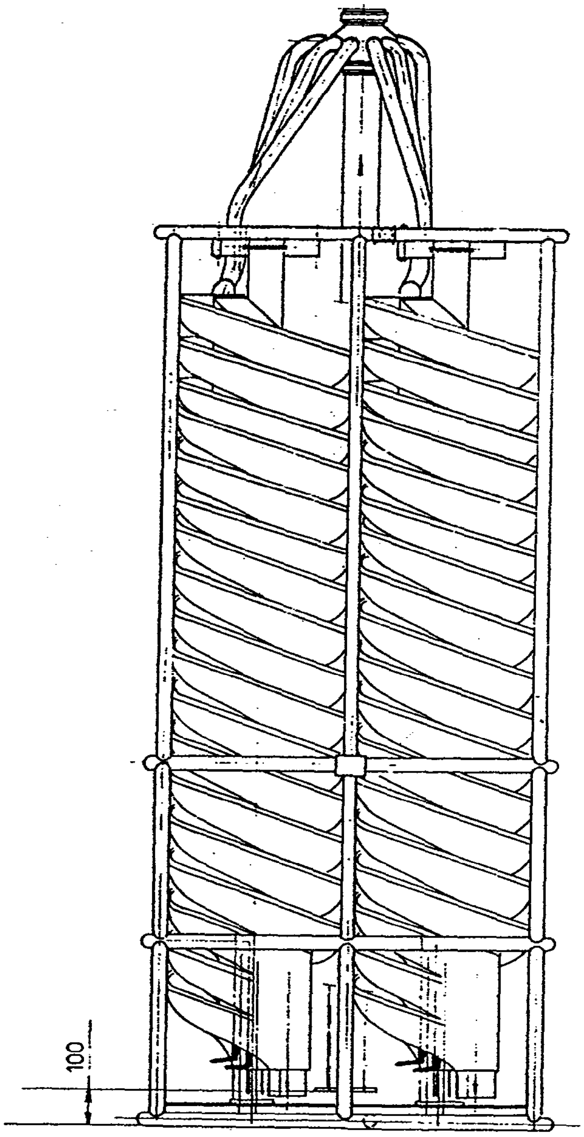


Figure 3.4 - Spiral Concentrator Schematic

3.5 Shaking Table

The shaking table is a gravity separation technique which relies on the differential specific gravity of particles for separation. The table itself is flat with a series of ridges or riffles running longitudinally along it. It is also slightly sloped. Figure 3.5 shows a picture of a bank of shaking tables used for concentration of tin ore.

There are several mineral processing principles at work on a shaking table all at once. These include flowing film concentration, hindered settling, consolidation trickling and asymmetrical acceleration.

The feed for the table is at the high corner so the feed will spread out and down the table in a parabolic motion. The longitudinal baffles or riffles allow an area for the slurry of particles to settle in out of the flow of the feed. Smaller denser particles tend to settle out first at the high end of the table while larger denser particles will settle out at the low end of the table because they are larger and remain exposed to the feed flow longer. The less dense particles do not settle out on the table and are carried through in the tails flow.

The table drive is designed in such a way so that at the end backward stroke (table deck moving longitudinally towards the feed end) the deck and particles on it momentarily come to rest. The deck is accelerated forward until at the end of the forward stroke, the direction is rapidly reversed. The particles trapped behind the riffles on the deck slide forward slightly due to their own momentum while the deck begins its backward stroke. In this way, the trapped particles continuously move towards the discharge end of the table.

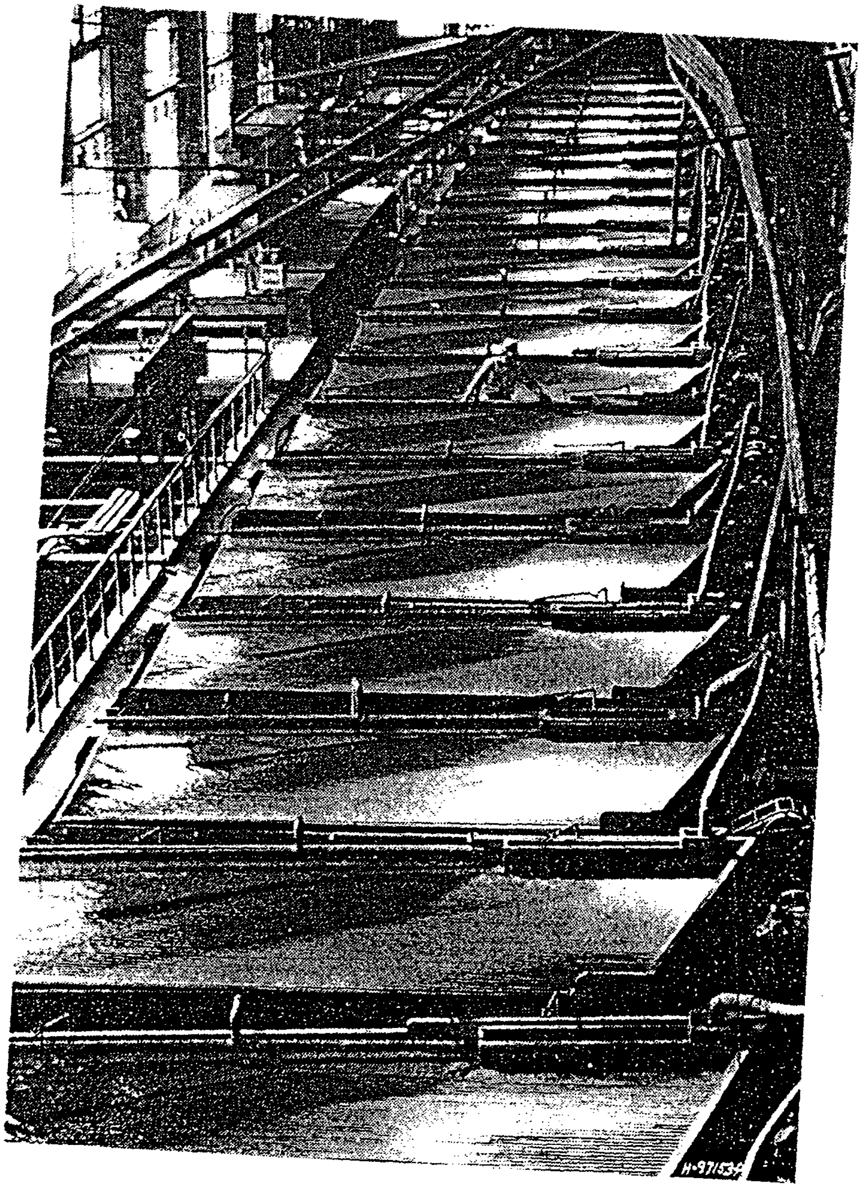


Figure 3.5 - Shaking Tables Concentrating Tin Ore

3.6 Direct Flotation

Flotation is a technique which utilizes the surface chemistry of the ore body rather then the specific gravity of the material. It is therefore better able to remove selective elements by choosing what element to remove rather than removing anything within a definite specific gravity band.

From its beginning in the first decade of this century, flotation has gradually moved to a predominant role in mineral separation. The major reason for this is the trend towards treatment of lower grade and more finely disseminated ores. Also, where gravity, magnetic or electrostatic processes are typically limited to particular minerals or mineral combinations, flotation is able to utilize a wide variety of surface chemistry conditions and reagents for an almost limitless degree of applicability. Figure 3.6 provides a schematic view of a mechanical flotation cell, just one style of flotation equipment.

The basic premise of flotation technology is to add a reagent to the slurry which will preferentially attach itself on one end to the mineral to be removed and at the other end to air bubbles which are introduced to the system. The required mineral in then floated to the top and removed from the system for further processing.

3.7 Reverse Flotation

Reverse flotation works on the same basic principle as direct flotation except that the mineral to be recovered is now in the tails. Typically, it is easier to float the least amount of material. therefore, if the amount of material to be recovered exceeds the amount of rejects or if the surface chemistry of the rejects are more amenable to flotation reagents, then it will be more economical to float the rejects and retain the tails as the concentrated product.

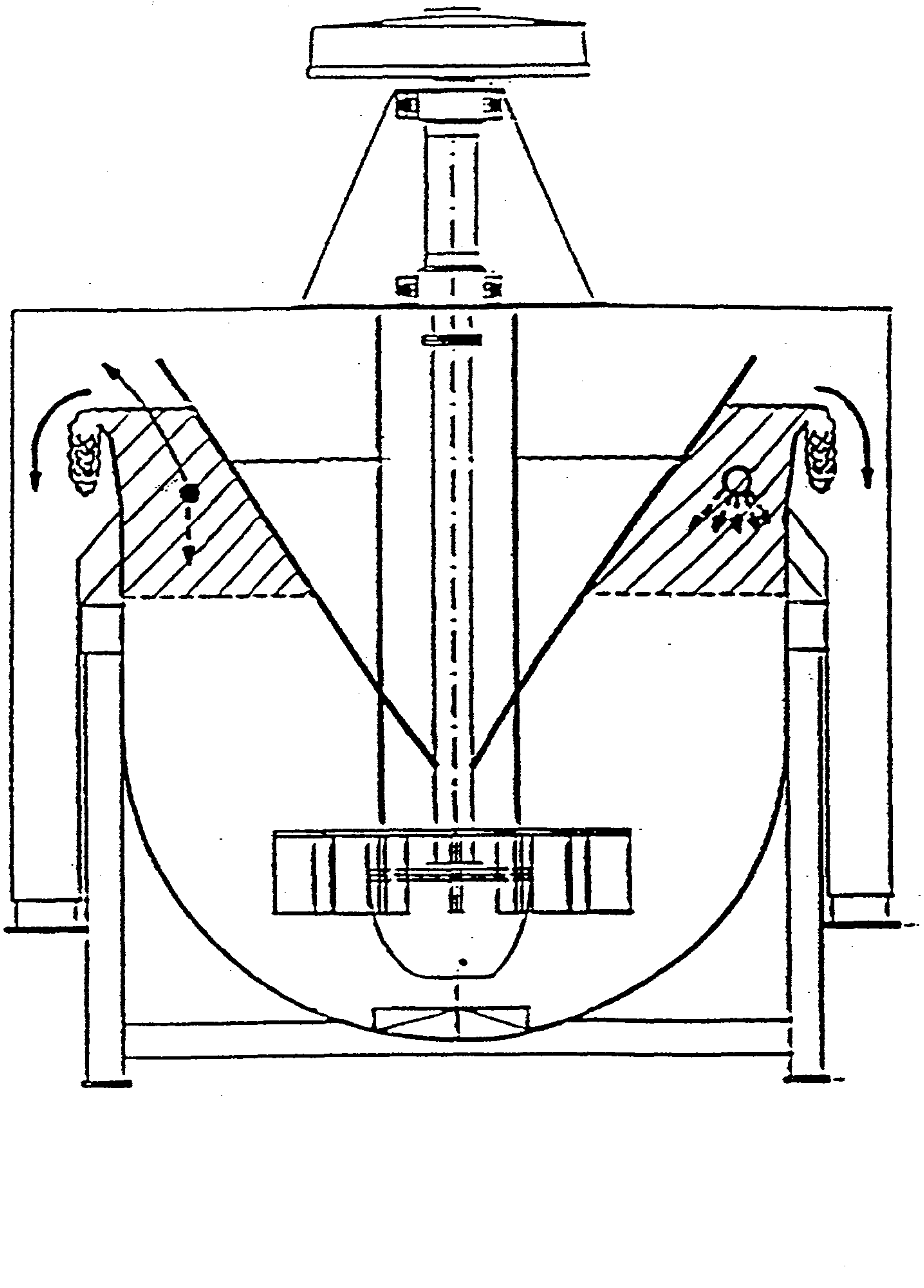


Figure 3.6 - Mechanical Flotation Cell Schematic

3.8 Magnetic Separation

The magnetic separation technique is based upon the magnetic properties of the ore. Magnetic concentration is achieved by simultaneously applying to all particles in an ore a magnetic force which will act only on those particles having magnetic properties while also applying a second force in an opposite direction which will act on all particles of the ore. The most common non-magnetic forces are gravitational, centrifugal and fluid drag.

Magnetic separators can be classified into four categories according to the media used, air (dry separator) or water (wet separator) and the magnetic field strength, standard or high intensity. Figure 3.7 provides schematic representation of several types of magnetic separation techniques.

One of the most common types of magnetic separation systems employs a rotating drum which contains several permanent magnets or electromagnets. These magnets are typically axially oriented and equally spaced with alternating polarity. Sometimes the drum acts as a pulley at the end of a belt or conveyor.

Whether or not a particle is held on the drum and therefore segregated from the mass flow and carried to the magnetic concentrate depends upon the relative magnitudes of the magnetic forces attracting the particle to the drum and the centrifugal forces tending to throw the particle off the drum. In this fashion, magnetic particles may be removed from the bulk stream of ore. The technology is also applicable to the removal of scrap metal such as bolts from ore streams.

The high intensity separation techniques are typically reserved for those particles which have weak magnetic properties. The disadvantage a magnetic separation process is that it is only applicable to streams where the material selected for removal has some inherent magnetic property.

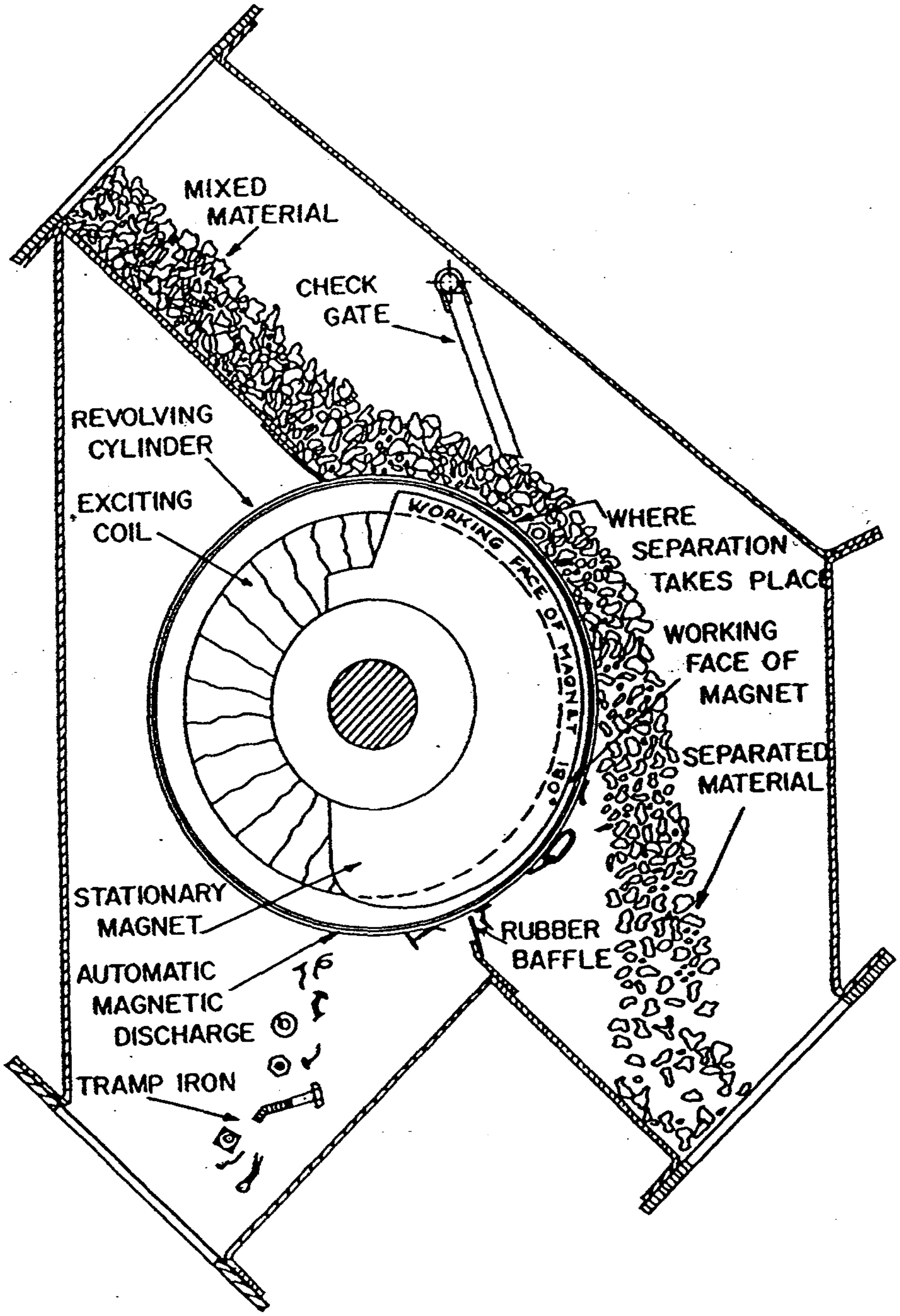


Figure 3.7 - Magnetic Separation Techniques

3.9 Cyclone Classification

A cyclone classifier works on a principle similar to that of the Falcon and Knelson concentrators. Rather than injecting the feed at the bottom of the chamber and having all the material move to the top where it is stratified and removed preferentially, the feed is injected tangentially. The gravitational (G-forces) are not as great in a cyclone and the material with the highest specific gravity falls out the bottom of the cyclone while the less dense material is spun out through the top of the unit. A cyclone classifier is shown schematically in figure 3.8.

As with the Falcon and Knelson Concentrators, the technology can only be used to slit a slurry into two specific gravity fractions.. If the band of specific gravity of the material to be recovered lies in the middle with waste at either end, two stage separation may be required. The split point where particles become part of the heads rather than the tails is defined by the initial tangential velocity and the slope of the sides of the cyclone.

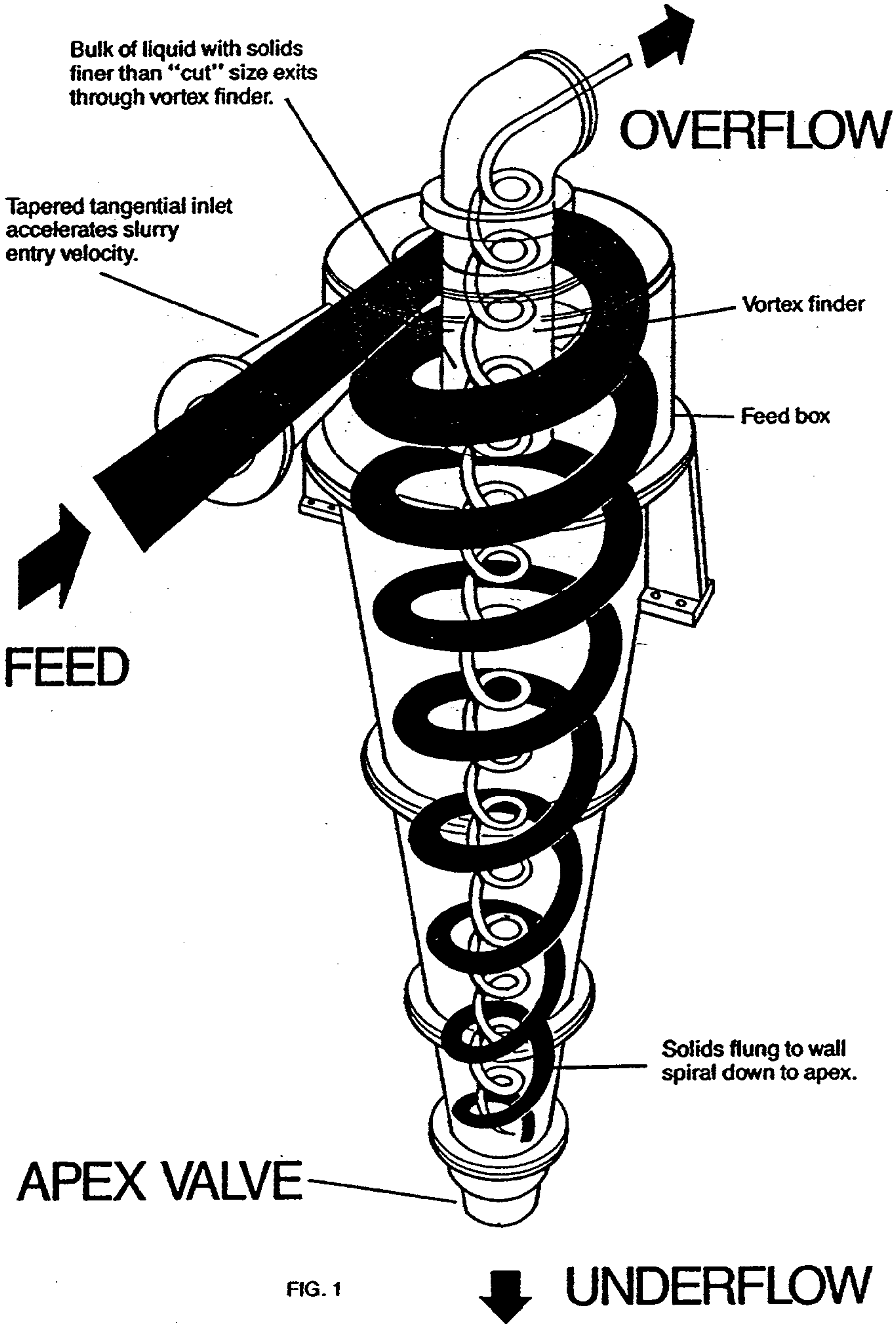


Figure 3.8 - Cyclone Classification Schematic

4.0 DATA PRESENTATION

The following section of this document presents the data which was obtained during bench scale testing of the various mineral processing techniques. The section is broken into sections, each corresponding to a specific separation method. The data is then discussed and evaluated in Section 5.

4.1 Initial Sample Characterization

The samples received by CESL were first characterized through a mineralogical evaluation, elemental analysis, size fraction or screen analysis, acid base accounting (ABA) analysis and humidity cell analysis. The humidity cell and ABA analysis are the subject of a separate section of this report. The following data is that which was obtained as part of the remainder of the initial sample characterization.

Elemental Analysis

Table 4.1 below provides the results of the 32 element ICP scan, the total sulphur, total sulphate, silver, gold, and CO_2 analyses.

Mineralogical Evaluation

A summary of the findings from this evaluation is provided below in Table 4.2.

Sample 1 - Placer Dome, Detour Lake Mine

This product has an overall particle size range of 5-200 microns. It is made up predominantly of silicates. Opaque constituents consist of pyrite and pyrrhotite, plus minor proportions of chalcopyrite and magnetite.

Liberation of sulphides from gangue, and of the different sulphide species from one another, is of a high order (estimated >98% in both cases). The sulphide grains hardly ever exceed 100 microns in size, but very find particles are comparatively rare, and it is estimated that >90% of the total sulphides are in the range of 30-100 microns. The sulphide grains all appear fresh, and should respond well to flotation.

The simple mineralogy and excellent liberation offer the possibility that, despite the low initial concentration of chalcopyrite, separation of a saleable Cu concentrate may be feasible - offering by-product revenue to offset the cost of sulphide removal.

Table 4.1 - Elemental Analysis of Sample Received

Element		Sample 1 (Placer)	Sample 2 (LAC)	Sample 3.2 (Selbaie)	Sample 3.3 (Selbaie)
Ag	ppm	3.1	0.8	25.7	8.0
Al	%	2.67	1.33	0.46	1.06
As	ppm	1	1	164	49
Au	g/ton	0.35	0.2	0.16	0.09
Au	oz/ton	0.01	0.006	0.005	0.003
B	ppm	1	1	224	1
Ba	ppm	81	92	2	8
Be	ppm	0.1	0.1	0.1	0.4
Bi	ppm	29	10	2	3
Ca	%	1.86	1.01	0.80	0.38
Cd	ppm	0.1	0.1	40.8	3.3
Co	ppm	36	13	31	11
Cr	ppm	125	15	120	110
Cu	ppm	1480	424	888	426
CO_2	%	0.7	0.9	1.9	0.7
Fe	%	5.55	4.29	>15.00	4.43
Ga	ppm	39	28	1	14
K	%	0.54	0.21	0.03	0.07
Li	ppm	31	8	4	11
Mg	%	1.96	1.29	0.93	1.18
Mn	ppm	560	325	568	526
Mo	ppm	3	4	1	4
Na	%	0.14	0.05	0.01	0.01
Ni	ppm	55	1	1	1
P	ppm	290	940	30	90
Pb	ppm	241	89	498	211
Sb	ppm	6	1	7	11
Sr	ppm	13	8	44	23
S_T	%	2.34	4.15	23.6	3.47
S (SO_4)	%	0.01	0.04	0.16	0.08
Sn	ppm	1	1	1	1
Te	ppm	-	6.13	-	-
Th	ppm	116	153	14	8
Ti	ppm	1916	208	11	19
V	ppm	107.5	33.1	3.5	3.5
W	ppm	13	4	9	8
Zn	ppm	107	170	8420	1435

Table 4.2 - Summary of Mineralogical Evaluation

Mineral	Les Mines Selbaie - Sample 3.2	Les Mines Selbaie - Sample 3.3	Placer Dome Inc. - Detour Lake Mine	Lac Minerals - La Mine Doyon
Particle size (microns)	5-150	10-150	5-200	5-200
Feldspars	-	74	38% combined	70% combined
Quartz	52	20		
Carbonate	1	0.5	trace	0.5
Sericite	-	trace	-	2.5
Pyrite	46	5	4.5	20
Chalcopyrite	0.1	trace	0.4	0.2
Rutile	-	trace	trace	1.0
Sphalerite	0.6	trace	-	-
Pyrrhotite	-	-	4.0	-
Magnetite	-	-	0.4	trace
Biotite	-	-	6	3.0
Hornblende	-	-	46	-
Epidote	-	-	0.5	0.5
Chlorite	trace	-	-	2.0
Sphene	-	-	-	trace

The silicate gangue is also of simple mineralogy, free of fine-grained intergrowths and notably fresh. It consists predominantly of quartz (and possible untwinned feldspar) and hornblende. Biotite is the principal accessory. The different silicates largely occur as separate (liberated) grains. Carbonate is present only as rare traces.

Sample 2 - Lac Minerals, La Mine Doyon

This product has an overall particle size range of 5-200 microns. The estimated total sulphide content is about 20%, consisting largely of pyrite. Minor chalcopyrite is the only other observable sulphide constituent.

Sulphide grains scarcely ever exceed 100 microns in size. They show virtually complete libration from the silicates (estimated >99%). As in Sample 1 chalcopyrite shows essentially complete liberation from pyrite; however, it is of perceptibly lower abundance than in the previous sample.

The gangue consists predominantly of quartz and feldspars. Biotite, sericite, chlorite and a little epidote and carbonate are minor accessories. Rutile (estimated c.1%) is relatively abundant (as liberated grains).

Sample 3.2 - Les Mines Selbaie, Sample 3.2

This product has an overall particle size range of 5 - 150 microns. It is a sulphide-rich tailings composed essentially of pyrite and quartzose gangue in approximately equal proportions.

Very minor sphalerite and lesser chalcopyrite are the accessory sulphides. The gangue includes a minor component of carbonate.The pyrite is in the form of ragged particles up to 100 microns in size. It shows a strikingly high degree of liberation (essentially 100%) from the gangue- even in the case of the smallest pyrite specks. The pyrite appears mainly to take the form of compact aggregates of minutely fine grain size. It incorporates a minor proportion of a rather brownish-looking variant which may represent secondary (possibly marcasitic) material derived by modification of original pyrrhotite. The accessory sulphides (sphalerite and chalcopyrite) are also mainly liberated, but some 10-15% are estimated to be locked with pyrite (or, rarely gangue) as simple composites or intimate, micron-scale intergrowths.

Sample 3.3 - Les Mines Selbaie, Sample 3.3

This product has an overall particle size range of 10 - 150 microns. It consists predominantly of feldspars, with accessory proportions of quartz. Both occur as liberated grains, with the quartz frequently constituting the upper end of the size distribution. Very minor carbonate is also seen.

The principal sulphide is pyrite, as equant, anhedral-subhedral grains 10-75 microns in size-fully liberated from the silicates. Even the smallest sulphide specks appear to be liberated. The sulphides include a very minor component of chalcopyrite, mostly as tiny liberated specks, but occasionally locked with pyrite grains. One example of chalcopyrite was seen as inclusions in a coarse gangue particle. There are also extremely rare traces of sphalerite, generally locked with pyrite.

In view of the relatively low abundance of sulphides, their simple mineralogy and virtually complete liberation, this tailings should be highly amenable to desulphidation by flotation.

Screen Analysis

Particle size and total sulphur distributions of the feed samples are shown in Tables 4.3 - 4.6 and Figures 4.1 - 4.4 on the following pages. Precious metal distributions are presented in Tables 4.7 - 4.9 and Figures 4.5 - 4.7.

Table 4.3 - Particle Size and Total Sulphur Distribution of Sample 1, Placer Dome Tails

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	2.3	0.7	0.7	99.3	0.42	0.1	0.1	99.9
147	11.3	3.3	4.0	96.0	0.46	0.7	0.8	99.2
104	34.1	10.1	14.1	85.9	0.67	2.9	3.6	96.4
74	70.6	20.9	35.0	65.0	1.84	16.3	20.0	80.0
44	132.2	39.1	74.0	26.0	2.88	47.8	67.8	32.2
38	32.2	9.5	83.6	16.4	2.81	11.4	79.1	20.9
-38	55.6	16.4	100.0		2.99	20.9	100.0	
Total	338.3	100.0			2.35	100.0		

Figure 4.1 - Particle Size and Total Sulphur Distribution of Sample 1, Placer Dome Tails

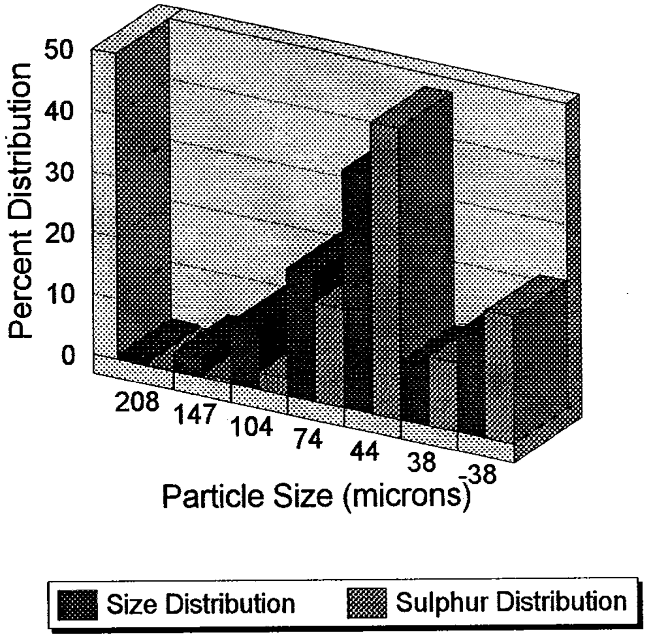


Table 4.4 - Particle Size and Total Sulphur Distribution of Sample 1, LAC Minerals Tails

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.3	0.1	0.1	99.9	0.38	0.0	0.0	100.0
147	6.3	1.9	2.0	98.0	0.38	0.2	0.2	99.8
104	25.7	7.6	9.6	90.4	0.32	0.6	0.8	99.2
74	30.4	9.0	18.6	81.4	0.40	0.9	1.6	98.4
44	78.0	23.1	41.6	58.4	4.69	26.1	27.7	72.3
38	34.4	10.2	51.8	48.2	7.05	17.3	45.0	55.0
-38	162.9	48.2	100.0		4.73	55.0	100.0	
Total	338.0	100.0			4.15	100.0		

Figure 4.1 - Particle Size and Total Sulphur Distribution of Sample 1, LAC Minerals Tails

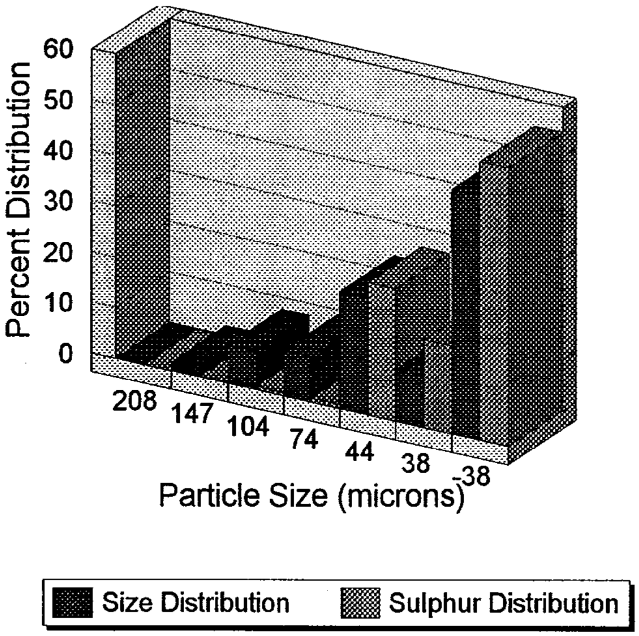


Table 4.5 - Particle Size and Total Sulphur Distribution of Sample 3.2, Les Mines Selbaie

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.4	0.2	0.2	99.8	3.04	0.0	0.0	100.0
147	2.4	1.0	1.1	98.9	3.04	0.1	0.2	99.8
104	7.4	3.0	4.1	95.9	4.18	0.6	0.7	99.3
74	11.0	4.4	8.6	91.4	5.36	1.1	1.9	98.1
44	74.2	30.0	38.5	61.5	15.43	21.6	23.5	76.5
38	32.3	13.0	51.6	48.4	24.15	14.7	38.2	61.8
-38	120.0	48.4	100.0		27.31	61.8	100.0	
Total	247.7	100.0			21.40	100.0		

Figure 4.3 - Particle Size and Total Sulphur Distribution of Sample 3.2, Les Mines Selbaie

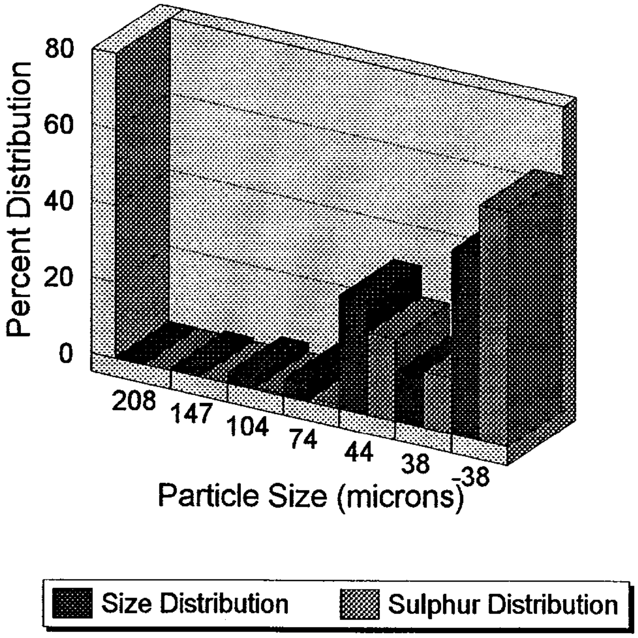


Table 4.6 - Particle Size and Total Sulphur Distribution of Sample 3.3, Les Mines Selbaie

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.3	0.1	0.1	99.9	2.44	0.1	0.1	99.9
147	1.0	0.3	0.5	99.5	2.44	0.2	0.3	99.7
104	4.2	1.5	1.9	98.1	1.14	0.5	0.8	99.2
74	9.2	3.2	5.1	94.9	0.83	0.8	1.6	98.4
44	82.0	28.6	33.8	66.2	2.38	19.6	21.1	78.9
38	53.6	18.7	52.5	47.5	3.55	19.1	40.2	59.8
-38	136.0	47.5	100.0		4.39	59.8	100.0	
Total	286.3	100.0			3.49	100.0		

Figure 4.4 - Particle Size and Total Sulphur Distribution of Sample 3.3, Les Mines Selbaie

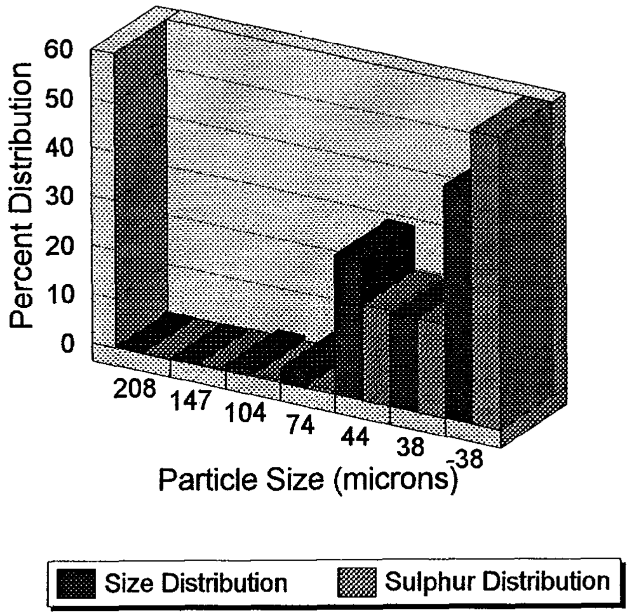


Table 4.7 - Precious Metal Distribution of Sample 1, Placer Dome Tails

			Assays		Distribution	
Mesh Size	**Wt(g)**	**Wt (%)**	**Au (gpt)**		**Au (%)**	
Feed						
150	40.1	16.3	0.85		37.1	
-150 + 325	108.1	43.9	0.38		44.7	
-325	98.3	39.9	0.17		18.2	
Total	246.5	100.0	0.37		100.0	

Figure 4.5 - Precious Metal Distribution of Sample 1, Placer Dome Tails

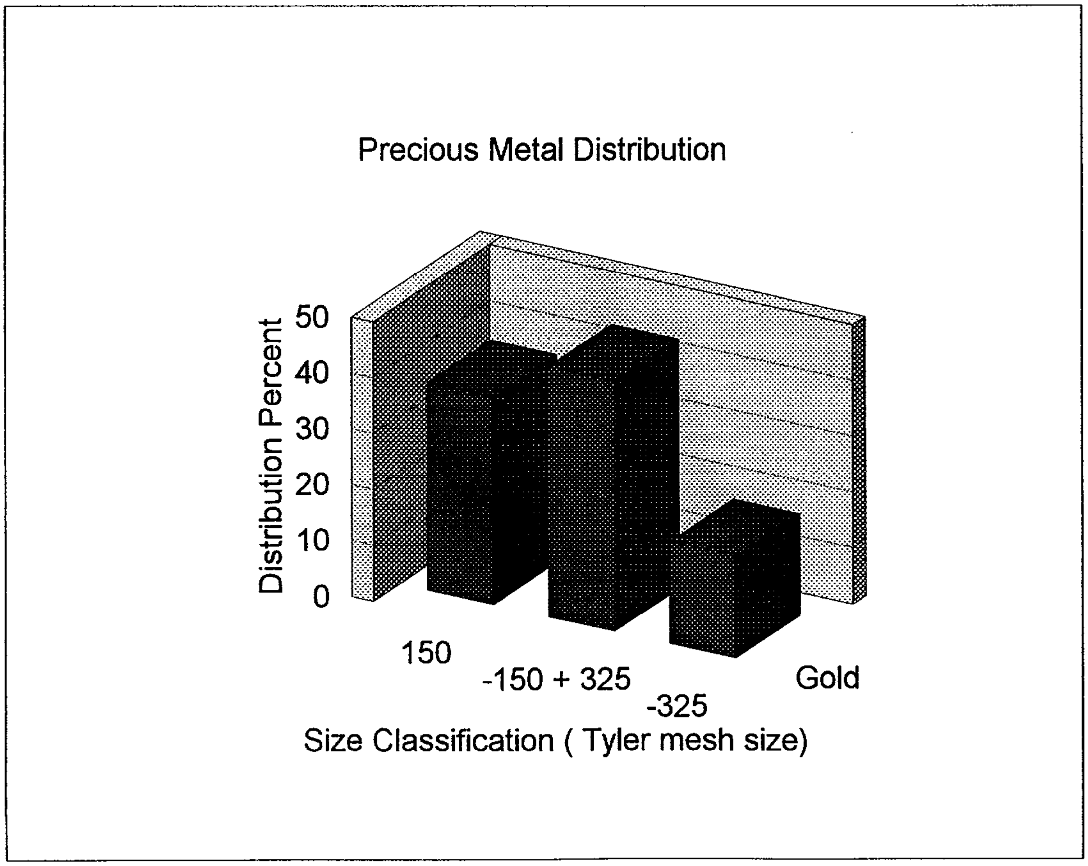


Table 4.8 - Precious Metal Distribution of Sample 2, LAC Minerals Tails

			Assays		Distribution	
Mesh Size	Wt(g)	Wt (%)	Au (gpt)		Au (%)	
Feed						
150	32.3	9.6	0.48		21.0	
-150 + 325	108.4	32.1	0.32		47.0	
-325	197.3	58.4	0.12		32.1	
Total	338.0	100.0	0.22		100.0	

Figure 4.5 - Precious Metal Distribution of Sample 2, LAC Minerals Tails

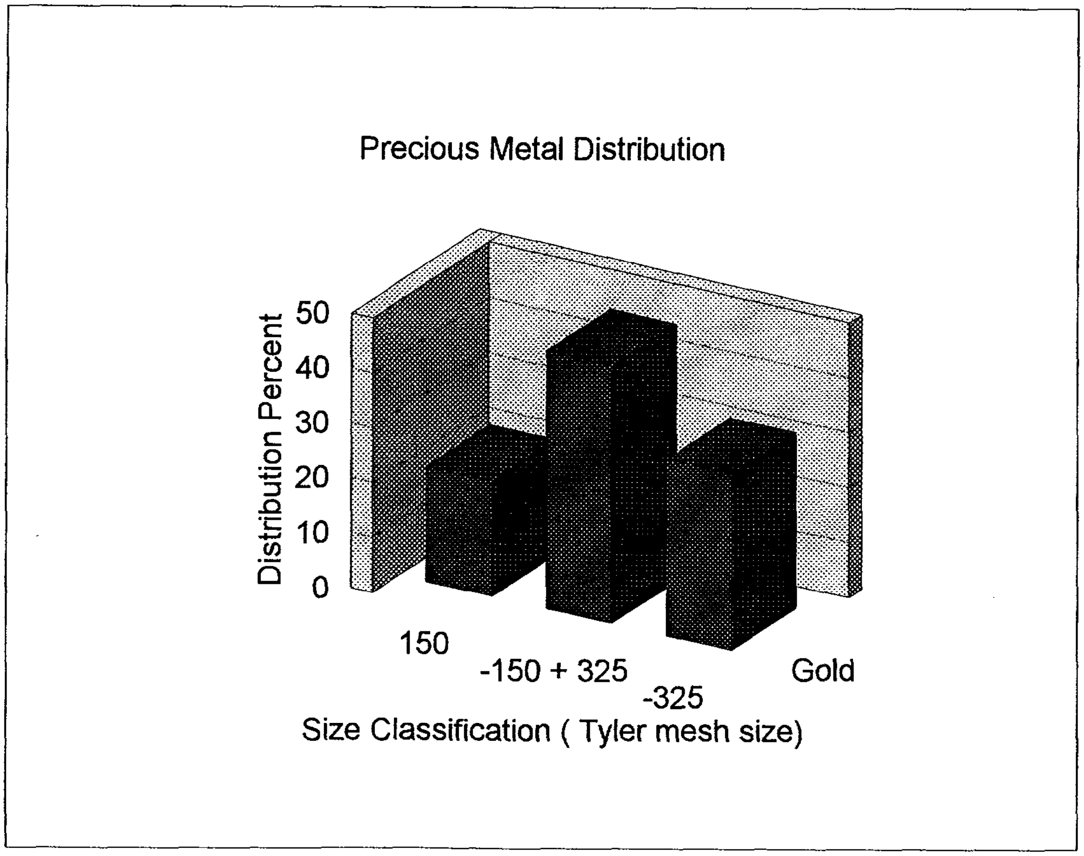
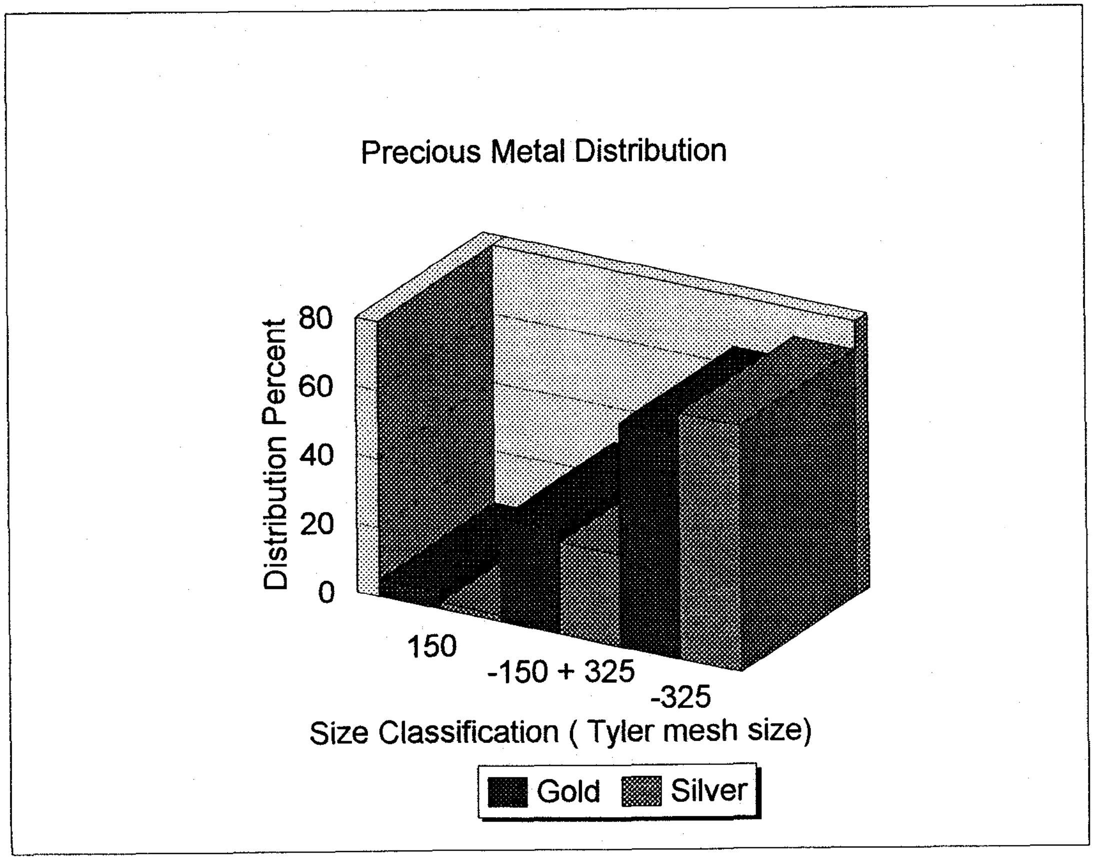


Table 4.9 - Precious Metal Distribution of Sample 3.2, Les Mines Selbaie

			Assays		Distribution	
Mesh Size	**Wt(g)**	**Wt (%)**	**Au (gpt)**	**Ag (gpt)**	**Au (%)**	**Ag (%)**
Feed						
150	10.2	4.1	0.19	10.00	4.9	1.7
-150 + 325	85.2	34.4	0.14	19.20	30.0	26.6
-325	152.3	61.5	0.17	28.90	65.1	71.7
Total	247.7	100.0	0.16	24.79	100.0	100.0

Table 4.7 - Precious Metal Distribution of Sample 3.2, Les Mines Selbaie



4.2 Falcon Concentrator

The Falcon Concentrator tests were conducted by personnel from Falcon Concentrators Inc. using a batch laboratory machine and Falcon's standard testing procedures.
The following samples were tested:

- Placer Dome
- Lac Minerals
- Selbaie (sample 3.2)

Sample 3.3 from Selbaie was not available for testing.

Results from the Falcon tests are presented in Tables 4.10 - 4.12

Table 4.10 - Falcon Test Results for Sample 1, Placer Dome

Placer Dome

Overall Balance

	Weight		Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Concentrate	3153	30.5	0.44	0.20	4.18	35.8	43.1	52.9
Tailing	7194	69.5	0.34	0.12	1.64	64.2	56.9	47.1
Feed	10347	100.0	0.37	0.14	2.41	100.0	100.0	100.0

Elemental Distributions

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Feed								
150	40.1	16.3	0.85	0.05	0.61	37.1	6.1	4.1
-150 + 325	108.1	43.9	0.38	0.13	2.39	44.7	38.0	43.5
-325	98.3	39.9	0.17	0.20	3.17	18.2	55.9	52.4
Total	246.5	100.0	0.37	0.14	2.41	100.0	100.0	100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Tail								
+150	49.3	19.8	0.76	0.05	0.59	43.8	8.4	7.2
-150 + 325	100.9	40.6	0.34	0.11	1.43	40.1	38.2	35.5
-325	98.3	39.6	0.14	0.16	2.37	16.1	53.4	57.3
Total	248.5	100.0	0.34	0.12	1.64	100.0	100.0	100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Conc								
+150	256	8.1	1.35	0.08	0.61	25.1	3.2	1.2
-150 + 325	1617	51.3	0.45	0.15	4.14	53.0	37.7	50.3
-325	1280	40.6	0.24	0.30	5.05	22.0	59.1	48.6
Total	3153	100.0	0.44	0.20	4.23	100.0	100.0	100.0

Recovery By Size Fraction

+150 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Concentrate	256	15.2	1.35	0.08	0.61	24.2	21.5	15.2
Tailing	1427	84.8	0.76	0.05	0.61	75.8	78.5	84.8
Feed	1683	100.0	0.85	0.05	0.61	100.0	100.0	100.0

-150 + 325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Concentrate	1617	35.6	0.45	0.15	4.14	42.4	42.8	61.8
Tailing	2921	64.4	0.34	0.11	1.42	57.6	57.2	38.2
Feed	4538	100.0	0.38	0.13	2.39	100.0	100.0	100.0

-325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Concentrate	1280	31.0	0.24	0.30	5.05	43.2	45.7	48.9
Tailing	2846	69.0	0.14	0.16	2.38	56.8	54.3	51.1
Feed	4126	100.0	0.17	0.20	3.21	100.0	100.0	100.0

Table 4.11 - Falcon Test Results for Sample 2, LAC

Lac

Overall Balance

	Weight		Assays			Distribution		
Product	**Wt(g)**	**Wt (%)**	**Au (gpt)**	**Te (ppm)**	**S(T) (%)**	**Au (%)**	**Te (%)**	**S(T) (%)**
Concentrate	2430	35.6	0.23		7.99	38.9		70.7
Tailing	4388	64.4	0.20		1.84	61.1		29.3
Feed	6818	100.0	0.21		4.03	100.0		100.0

Elemental Distributions

			Assays			Distribution		
Mesh Size	**Wt(g)**	**Wt (%)**	**Au (gpt)**	**Te (ppm)**	**S(T) (%)**	**Au (%)**	**Te (%)**	**S(T) (%)**
Feed								
150	18	8.9	0.48		0.21	20.4		0.5
-150 + 325	57.4	28.2	0.32		2.66	43.4		18.6
-325	127.9	62.9	0.12		5.18	36.2		80.9
Total	203.3	100.0	0.21		4.03	100.0		100.0

			Assays			Distribution		
Mesh Size	**Wt(g)**	**Wt (%)**	**Au (gpt)**	**Te (ppm)**	**S(T) (%)**	**Au (%)**	**Te (%)**	**S(T) (%)**
Tail								
+150	22	8.8	0.54		0.19	24.0		0.9
-150 + 325	98.5	39.3	0.29		1.71	57.7		36.6
-325	129.9	51.9	0.07		2.21	18.4		62.5
Total	250.4	100.0	0.20		1.84	100.0		100.0

			Assays			Distribution		
Mesh Size	**Wt(g)**	**Wt (%)**	**Au (gpt)**	**Te (ppm)**	**S(T) (%)**	**Au (%)**	**Te (%)**	**S(T) (%)**
Conc								
+150	218	9.0	0.37		0.25	14.8		0.3
-150 + 325	199	8.2	0.58		10.90	20.9		11.2
-325	2013	82.8	0.18		8.54	64.3		88.6
Total	2430	100.0	0.23		7.99	100.0		100.0

Recovery By Size Fraction

+150 m Fraction

			Assays			Distribution		
Product	**Wt(g)**	**Wt (%)**	**Au (gpt)**	**Te (ppm)**	**S(T) (%)**	**Au (%)**	**Te (%)**	**S(T) (%)**
Concentrate	218	36.1	0.37		0.25	28.2		42.2
Tailing	386	63.9	0.54		0.19	71.8		57.8
Feed	604	100.0	0.48		0.21	100.0		100.0

-150 + 325 m Fraction

			Assays			Distribution		
Product	**Wt(g)**	**Wt (%)**	**Au (gpt)**	**Te (ppm)**	**S(T) (%)**	**Au (%)**	**Te (%)**	**S(T) (%)**
Concentrate	199	10.3	0.58		10.90	18.7		42.4
Tailing	1726	89.7	0.29		1.71	81.3		57.6
Feed	1925	100.0	0.32		2.66	100.0		100.0

-325 m Fraction

			Assays			Distribution		
Product	**Wt(g)**	**Wt (%)**	**Au (gpt)**	**Te (ppm)**	**S(T) (%)**	**Au (%)**	**Te (%)**	**S(T) (%)**
Concentrate	2013	46.9	0.18		8.54	69.0		77.4
Tailing	2276	53.1	0.07		2.21	31.0		22.6
Feed	4289	100.0	0.12		5.18	100.0		100.0

Table 4.12 - Falcon Test Results for Sample 3.2, Selbaie

Selbaie (3.2)

Overall Balance

	Weight		Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Concentrate	4726	55.3	0.18	26.36	26.06	61.2	57.8	69.4
Tailing	3813	44.7	0.14	23.83	14.26	38.8	42.2	30.6
Feed	8539	100.0	0.16	25.23	20.79	100.0	100.0	100.0

Elemental Distributions

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Feed								
150	14	5.8	0.19	16.60	3.07	6.8	3.8	0.9
-150 + 325	74.2	30.5	0.14	19.20	11.80	26.4	23.2	17.3
-325	154.9	63.7	0.17	28.90	26.70	66.9	73.0	81.8
Total	243.1	100.0	0.16	25.23	20.79	100.0	100.0	100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Tail								
+150	15.7	6.5	0.15	16.60	5.09	7.0	4.5	2.3
-150 + 325	75	31.2	0.16	21.40	12.10	35.4	28.0	26.4
-325	150	62.3	0.13	25.80	16.30	57.6	67.5	71.2
Total	240.7	100.0	0.14	23.83	14.26	100.0	100.0	100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Conc								
+150	243	5.1	0.23	16.60	1.00	6.6	3.2	0.2
-150 + 325	1418	30.0	0.12	17.36	11.55	20.6	19.8	13.3
-325	3065	64.8	0.20	31.30	34.76	72.7	77.0	86.5
Total	4726	100.0	0.18	26.36	26.06	100.0	100.0	100.0

Recovery By Size Fraction

+150 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Concentrate	243	49.4	0.23	16.60	1.00	60.1	49.4	16.1
Tailing	249	50.6	0.15	16.60	5.09	39.9	50.6	83.9
Feed	492	100.0	0.19	16.60	3.07	100.0	100.0	100.0

-150 + 325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Concentrate	1418	54.4	0.12	17.36	11.55	47.9	49.2	53.3
Tailing	1188	45.6	0.16	21.40	12.10	52.1	50.8	46.7
Feed	2606	100.0	0.14	19.20	11.80	100.0	100.0	100.0

-325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Concentrate	3065	56.3	0.20	31.30	34.76	66.6	61.0	73.3
Tailing	2376	43.7	0.13	25.80	16.30	33.4	39.0	26.7
Feed	5441	100.0	0.17	28.90	26.70	100.0	100.0	100.0

4.3 Knelson Concentrator

The Knelson Concentrator tests were conducted at the facilities of Knelson Gold Concentrators Inc. at Langley, British Columbia. The general testing procedure can be described as follows:

The tailings sample were manually fed into the Knelson Concentrator at 60% solids and were diluted with water during the concentration stage. The mass recovery to concentrate was controlled by varying the feed volume between 500 ml and 2000 ml. The wash water which serves to keep the solids fluidized, was held constant at 2 lpm. The overflow (light fraction) was collected in a pail for the entire duration of the test and then filtered and dryed. The concentrate (heavy fraction) was removed from the concentrate collection rings at the end of the test and was also filtered and dryed. Both products were weighed and assayed.

The mass and metallurgical balances for the Knelson Concentrator tests are summarized in Tables 4.13 - 4.15.

Table 4.13 - Knelson Concentrator Test Results for Sample 1, Placer Dome

Placer Dome

Overall Balance

	Weight		Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Concentrate	89.19	6.4	0.74	0.13	7.20	13.0	5.8	19.2
Tailing	1300	93.6	0.34	0.15	2.08	87.0	94.2	80.8
Feed	1389.19	100.0	0.37	0.14	2.41	100.0	100.0	100.0

Elemental Distributions

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Feed								
150	40.1	16.3	0.85	0.05	0.61	37.5	5.6	4.1
-150 + 325	108.1	43.9	0.37	0.13	2.39	44.1	39.3	43.5
-325	98.3	39.9	0.17	0.20	3.17	18.4	55.0	52.4
Total	246.5	100.0	0.37	0.14	2.41	100.0	100.0	100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Tail								
+150	37.5	16.8	0.83	0.05	0.44	40.7	5.6	3.5
-150 + 325	97.0	43.4	0.34	0.13	1.84	43.1	38.1	38.3
-325	89.0	39.8	0.14	0.21	3.04	16.3	56.3	58.1
Total	223.5	100.0	0.34	0.15	2.08	100.0	100.0	100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Conc								
+150	7.9	8.8	1.40	0.08	5.32	16.7	5.2	6.5
-150 + 325	45.0	50.5	0.75	0.16	9.28	50.6	59.4	65.1
-325	36.3	40.7	0.60	0.11	5.02	32.7	35.4	28.4
Total	89.2	100.0	0.74	0.13	7.20	100.0	100.0	100.0

Recovery By Size Fraction

+150 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Concentrate	7.9	3.5	1.40	0.08	5.32	5.8	5.4	30.4
Tailing	218.1	96.5	0.83	0.05	0.44	94.2	94.6	69.6
Feed	226.0	100.0	0.85	0.05	0.61	100.0	100.0	100.0

-150 + 325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Concentrate	45.0	7.4	0.75	0.16	9.28	14.9	8.8	28.7
Tailing	564.2	92.6	0.34	0.13	1.84	85.1	91.2	71.3
Feed	609.2	100.0	0.37	0.13	2.39	100.0	100.0	100.0

-325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Cu (%)	S(T) (%)	Au (%)	Cu (%)	S(T) (%)
Concentrate	36.3	6.6	0.60	0.11	5.02	23.0	3.8	10.4
Tailing	517.7	93.4	0.14	0.21	3.04	77.0	96.2	89.6
Feed	554.0	100.0	0.17	0.20	3.17	100.0	100.0	100.0

Table 4.14 - Knelson Concentrator Test Results for Sample 2, Lac

Lac

Overall Balance

	Weight		Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Te (ppm)	S(T) (%)	Au (%)	Te (%)	S(T) (%)
Concentrate	80.68	7.5	0.43		11.50	15.5		21.4
Tailing	995	92.5	0.19		3.42	84.5		78.6
Feed	1075.68	100.0	0.21		4.03	100.0		100.0

Elemental Distributions

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Te (ppm)	S(T) (%)	Au (%)	Te (%)	S(T) (%)
Feed								
150	18	8.9	0.48		0.20	20.4		0.4
-150 + 325	57.4	28.2	0.32		2.66	43.4		18.6
-325	127.9	62.9	0.12		5.18	36.2		80.9
Total	203.3	100.0	0.21		4.03	100.0		100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Te (ppm)	S(T) (%)	Au (%)	Te (%)	S(T) (%)
Tail								
+150	21.2	9.1	0.46		0.20	22.0		0.5
-150 + 325	66.1	28.4	0.28		1.58	41.8		13.1
-325	145.2	62.5	0.11		4.73	36.1		86.3
Total	232.5	100.0	0.19		3.42	100.0		100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Te (ppm)	S(T) (%)	Au (%)	Te (%)	S(T) (%)
Conc								
+150	5	5.6	0.88		0.20	11.4		0.1
-150 + 325	21	25.8	0.86		17.33	51.6		38.9
-325	55	68.6	0.23		10.23	36.9		61.0
Total	81	100.0	0.43		11.50	100.0		100.0

Recovery By Size Fraction

+150 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Te (ppm)	S(T) (%)	Au (%)	Te (%)	S(T) (%)
Concentrate	5	4.7	0.88		0.20	8.7		4.7
Tailing	91	95.3	0.46		0.20	91.3		95.3
Feed	95	100.0	0.48		0.20	100.0		100.0

-150 + 325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Te (ppm)	S(T) (%)	Au (%)	Te (%)	S(T) (%)
Concentrate	21	6.9	0.86		17.33	18.5		44.7
Tailing	283	93.1	0.28		1.58	81.5		55.3
Feed	304	100.0	0.32		2.66	100.0		100.0

-325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Te (ppm)	S(T) (%)	Au (%)	Te (%)	S(T) (%)
Concentrate	55	8.2	0.23		10.23	15.8		16.2
Tailing	621	91.8	0.11		4.73	84.2		83.8
Feed	677	100.0	0.12		5.18	100.0		100.0

Table 4.15 - Knelson Concentrator Test Results for Sample 3.2, Selbaie

Selbaie (3.2)

Overall Balance

	Weight		Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Concentrate	97.12	11.8	0.20	41.21	27.43	14.8	19.5	15.6
Tailing	727	88.2	0.16	22.73	19.89	85.2	80.5	84.4
Feed	824.12	100.0	0.16	24.91	20.78	100.0	100.0	100.0

Elemental Distributions

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Feed								
150	10.5	3.9	0.19	10.00	3.07	4.6	1.6	0.6
-150 + 325	90.9	33.6	0.14	19.20	11.80	29.3	25.9	19.1
-325	169.1	62.5	0.17	28.90	26.70	66.1	72.5	80.3
Total	270.5	100.0	0.16	24.91	20.78	100.0	100.0	100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Tail								
+150	11.5	4.2	0.19	10.00	1.46	5.2	1.9	0.3
-150 + 325	85	31.3	0.12	13.60	8.97	24.2	18.7	14.1
-325	175	64.5	0.17	28.00	26.40	70.6	79.4	85.6
Total	271.5	100.0	0.16	22.73	19.89	100.0	100.0	100.0

			Assays			Distribution		
Mesh Size	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Conc								
+150	1	1.2	0.19	10.00	44.52	1.2	0.3	2.0
-150 + 325	49	50.8	0.23	45.04	24.86	58.4	55.5	46.0
-325	47	48.0	0.17	37.95	29.72	40.4	44.2	52.0
Total	97	100.0	0.20	41.21	27.43	100.0	100.0	100.0

Recovery By Size Fraction

+150 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Concentrate	1	3.7	0.19	10.00	44.52	3.7	3.7	54.2
Tailing	31	96.3	0.19	10.00	1.46	96.3	96.3	45.8
Feed	32	100.0	0.19	10.00	3.07	100.0	100.0	100.0

-150 + 325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Concentrate	49	17.8	0.23	45.04	24.86	29.6	41.8	37.5
Tailing	228	82.2	0.12	13.60	8.97	70.4	58.2	62.5
Feed	277	100.0	0.14	19.20	11.80	100.0	100.0	100.0

-325 m Fraction

			Assays			Distribution		
Product	Wt(g)	Wt (%)	Au (gpt)	Ag (gpt)	S(T) (%)	Au (%)	Ag (%)	S(T) (%)
Concentrate	47	9.0	0.17	37.95	29.72	9.0	11.9	10.1
Tailing	469	91.0	0.17	28.00	26.40	91.0	88.1	89.9
Feed	515	100.0	0.17	28.90	26.70	100.0	100.0	100.0

4.4 Reichert Tray

Reichert Tray tests were conducted by Lakefield Research at Lakefield, Ontario. The general test procedure for the tray tests can be summarized as follows:

Slurry as received was pulped to 1420 g/l and passed over a Reichert tray setup consisting of two rougher trays and a cleaner tray. The cleaner tray was used to treat the combined rougher concentrate. The circuit was operated as a closed loop with all products returning to the feed pump during the stabilization period. During this period the trays were adjusted to provide a weight split corresponding to the sulphide content of the sample. Once equilibrium was established, each stream was sampled simultaneously.

Summaries of the mass and metallurgical balances for each sample are presented in Table 4.16. Particle size and total sulphur distributions for the Reichert Tray tails for each sample are presented in Tables 4.17 - 4.20 and Figures 4.8 - 4.11.

Table 4.16 - Mass and Metallurgical Balances for Reichert Tray Tests

Sample 1 (Placer Dome)

Product	Flow Rat kg/h	Wt %	Assays				Distribution			
			Au (gpt)	Cu (%)	S(t) (%)	Fe (%)	Au (%)	Cu (%)	S(t) (%)	Fe (%)
Cleaner Concentrate	405	25.0	0.33	0.10	2.10	8.63	29.3	21.8	26.9	24.7
Cleaner Tail	625	38.6	0.30	0.12	1.92	8.88	41.1	40.3	37.9	39.3
Rougher Tail	589	36.4	0.23	0.12	1.89	8.62	29.7	38.0	35.2	36.0
Feed	1619	100.0	0.28	0.11	1.95	8.72	100.0	100.0	100.0	100.0
Rougher Concentrate (calc)	1030	63.6	0.31	0.11	1.99	8.78	70.3	62.0	64.8	64.0

Sample 2 (Lac Minerals)

Product	Flow Rat kg/h	Wt %	Assays				Distribution			
			Au (gpt)	Te (gpt)	S(t) (%)	Fe (%)	Au (%)	Te (%)	S(t) (%)	Fe (%)
Cleaner Concentrate	307	14.1	0.21	6.00	4.54	5.26	13.1	16.5	15.0	14.7
Cleaner Tail	950	43.7	0.24	4.00	4.31	5.02	46.2	34.1	44.0	43.4
Rougher Tail	916	42.2	0.22	6.00	4.17	5.03	40.8	49.3	41.0	41.9
Feed	2173	100.0	0.23	5.13	4.28	5.06	100.0	100.0	100.0	100.0
Rougher Concentrate (calc)	1257	57.8	0.23	4.49	4.37	5.08	59.2	50.7	59.0	58.1

Sample 3.2 (Selbaie)

Product	Flow Rat kg/h	Wt %	Assays				Distribution			
			Au (gpt)	Ag (gpt)	S(t) (%)	Fe (%)	Au (%)	Ag (%)	S(t) (%)	Fe (%)
Cleaner Concentrate	307	17.9	0.19	25.00	21.20	20.20	18.3	18.7	18.3	18.4
Cleaner Tail	575	33.6	0.18	23.40	20.50	19.40	32.4	32.7	33.1	33.1
Rougher Tail	830	48.5	0.19	24.10	20.80	19.70	49.4	48.6	48.5	48.5
Feed	1712	100.0	0.19	24.03	20.77	19.69	100.0	100.0	100.0	100.0
Rougher Concentrate (calc)	882	51.5	0.18	23.96	20.74	19.68	50.6	51.4	51.5	51.5

Sample 3.3 (Selbaie)

Product	Flow Rat kg/h	Wt %	Assays				Distribution			
			Au (gpt)	Ag (gpt)	S(t) (%)	Fe (%)	Au (%)	Ag (%)	S(t) (%)	Fe (%)
Cleaner Concentrate	463	31.8	0.14	7.60	3.13	4.18	39.4	31.8	33.1	29.7
Cleaner Tail	264	18.1	0.10	7.30	3.03	4.65	16.1	17.4	18.3	18.9
Rougher Tail	731	50.1	0.10	7.70	2.91	4.58	44.5	50.8	48.6	51.4
Feed	1458	100.0	0.11	7.60	3.00	4.47	100.0	100.0	100.0	100.0
Rougher Concentrate (calc)	727	49.9	0.13	7.49	3.09	4.35	55.5	49.2	51.4	48.6

Table 4.17 - Particle Size and Total Sulphur Distribution of Reichert Tray Tails, Sample 1, Placer Dome

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.8	0.3	0.3	99.7	0.32	0.1	0.1	99.9
147	5.1	1.8	2.1	97.9	0.40	0.4	0.5	99.5
104	19.2	6.8	8.9	91.1	0.56	2.2	2.6	97.4
74	76.6	27.2	36.1	63.9	1.33	20.7	23.3	76.7
44	118.7	42.1	78.1	21.9	2.02	48.6	71.9	28.1
38	54.0	19.1	97.3	2.7	2.20	24.1	96.0	4.0
-38	7.7	2.7	100.0		2.56	4.0	100.0	
Total	282.1	100.0			1.75	100.0		

Figure 4.8 - Particle Size and Total Sulphur Distribution of Reichert Tray Tails, Sample 1, Placer Dome

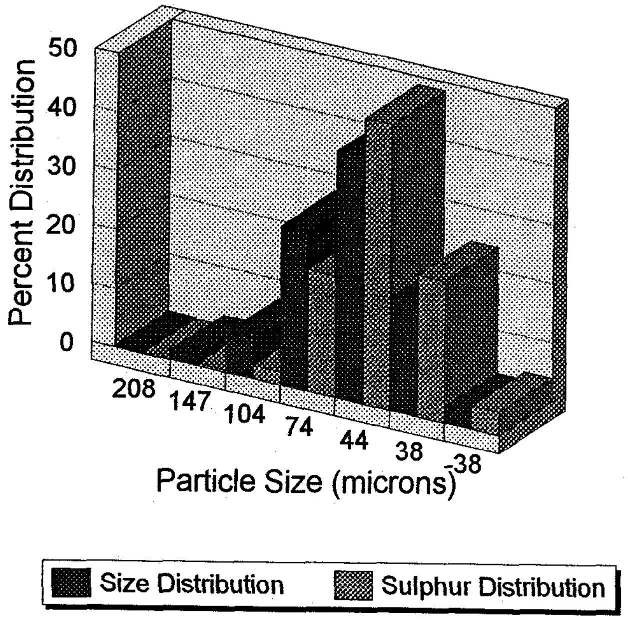


Table 4.18 - Particle Size and Total Sulphur Distribution of Reichert Tray Tails - Sample 2, LAC Minerals

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.1	0.0	0.0	100.0	0.23	0.0	0.0	100.0
147	1.9	0.7	0.8	99.2	0.23	0.0	0.0	100.0
104	11.1	4.3	5.1	94.9	0.43	0.5	0.5	99.5
74	18.5	7.1	12.2	87.8	0.74	1.3	1.9	98.1
44	72.7	28.0	40.2	59.8	3.81	27.3	29.1	70.9
38	38.1	14.7	54.9	45.1	5.42	20.3	49.5	50.5
-38	116.9	45.1	100.0		4.39	50.5	100.0	
Total	259.3	100.0			3.92	100.0		

Figure 4.9 - Particle Size and Total Sulphur Distribution of Reichert Tray Tails - Sample 2, LAC Minerals

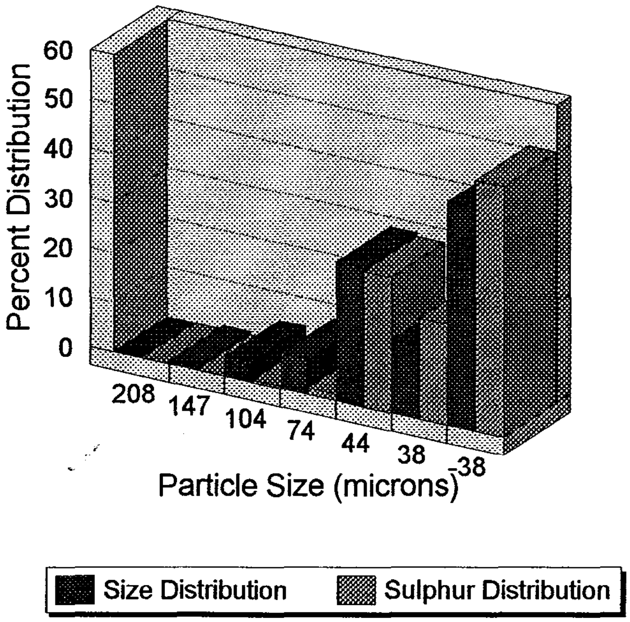


Table 4.19 - Particle Size and Total Sulphur Distribution of Reichert Tray Tails Sample 3.2, Les Mines Selbaie

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.9	0.3	0.3	99.7	1.74	0.0	0.0	100.0
147	4.7	1.4	1.7	98.3	1.99	0.1	0.1	99.9
104	15.0	4.5	6.1	93.9	0.26	0.1	0.2	99.8
74	22.7	6.8	12.9	87.1	6.02	1.8	2.0	98.0
44	130.6	39.0	51.9	48.1	28.82	48.5	50.5	49.5
38	60.3	18.0	69.8	30.2	18.26	14.2	64.7	35.3
-38	101.1	30.2	100.0		27.11	35.3	100.0	
Total	335.3	100.0			23.14	100.0		

Figure 4.19 - Particle Size and Total Sulphur Distribution of Reichert Tray Tails Sample 3.2, Les Mines Selbaie

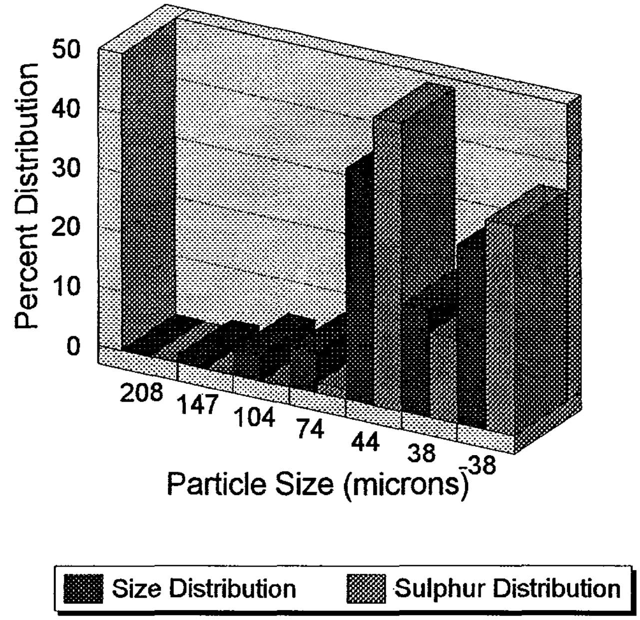
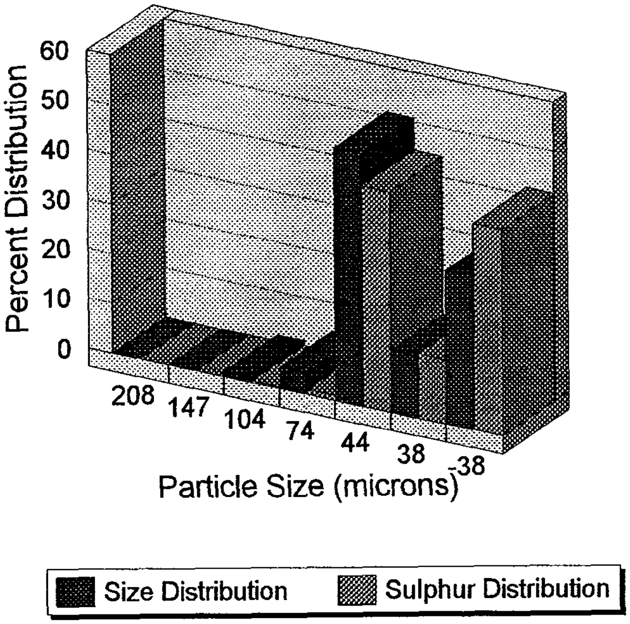


Table 4.20 - Particle Size and Total Sulphur Distribution of Reichert Tray Tails - Sample 3.3, Les Mines Selbaie

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.1	0.0	0.0	100.0	0.56	0.0	0.0	100.0
147	0.4	0.1	0.2	99.8	0.56	0.0	0.0	100.0
104	3.8	1.3	1.4	98.6	0.66	0.3	0.3	99.7
74	13.3	4.4	5.8	94.2	0.66	1.0	1.4	98.6
44	153.5	50.8	56.6	43.4	2.45	44.2	45.6	54.4
38	37.6	12.4	69.1	30.9	2.92	12.9	58.5	41.5
-38	93.4	30.9	100.0		3.78	41.5	100.0	
Total	302.1	100.0			2.82	100.0		

Figure 4.11 - Particle Size and Total Sulphur Distribution of Reichert Tray Tails - Sample 3.3, Les Mines Selbaie



4.5 Spiral Concentrator

The spiral concentration tests were conducted at Lakefield Research using a Carpco LC3000 Spiral. Slurry, adjusted to a pulp consistency of 1300 g/l was fed to the spiral using a closed loop pumping system. The cutters for concentrate and middlings were adjusted to produce the desired weight split prior to sampling. The results of the spiral tests are reported in Table 4.21. Particle size and total sulphur distributions are detailed in Tables 4.22 - 4.25 and are graphically displayed in Figure 4.12 - 4.15.

Table 4.21 - Mass and Metallurgical Balances for Spiral Tests

Sample 1 (Placer Dome)

			Assays				Distribution			
Product	Flow Rat kg/h	Wt %	Au (gpt)	Cu (%)	S(t) (%)	Fe (%)	Au (%)	Cu (%)	S(t) (%)	Fe (%)
Concentrate	109	8.1	0.55	0.19	4.17	11.10	17.1	13.7	17.7	10.5
Middlings	265	19.8	0.33	0.09	1.73	7.87	25.0	16.0	17.8	18.2
Tails	965	72.1	0.21	0.11	1.72	8.48	57.9	70.3	64.5	71.3
Feed	1339	100.0	0.26	0.11	1.92	8.57	100.0	100.0	100.0	100.0
Concentrate + Midds	374	27.9	0.39	0.12	2.44	8.81	42.1	29.7	35.5	28.7

Sample 2 (Lac Minerals)

			Assays				Distribution			
Product	Flow Rat kg/h	Wt %	Au (gpt)	Te (gpt)	S(t) (%)	Fe (%)	Au (%)	Te (%)	S(t) (%)	Fe (%)
Concentrate	82	6.8	0.46	8.00	5.50	6.70	15.5	5.8	9.7	9.1
Middlings	148	12.3	0.21	6.00	3.65	4.82	12.7	7.9	11.6	11.8
Tails	973	80.9	0.18	10.00	3.76	4.89	71.8	86.3	78.7	79.0
Feed	1203	100.0	0.20	9.37	3.87	5.00	100.0	100.0	100.0	100.0
Concentrate + Midds	230	19.1	0.30	6.71	4.31	5.49	28.2	13.7	21.3	21.0

Sample 3.2 (Selbaie)

			Assays				Distribution			
Product	Flow Rat kg/h	Wt %	Au (gpt)	Ag (gpt)	S(t) (%)	Fe (%)	Au (%)	Ag (%)	S(t) (%)	Fe (%)
Concentrate	108	9.1	0.30	30.60	27.00	25.30	15.0	12.4	12.5	12.3
Middlings	205	17.2	0.17	20.40	16.30	15.60	16.1	15.6	14.3	14.4
Tails	876	73.7	0.17	22.00	19.50	18.60	68.9	72.0	73.2	73.3
Feed	1189	100.0	0.18	22.51	19.63	18.69	100.0	100.0	100.0	100.0
Concentrate + Midds	313	26.3	0.21	23.92	19.99	18.95	31.1	28.0	26.8	26.7

Sample 3.3 (Selbaie)

			Assays				Distribution			
Product	Flow Rat kg/h	Wt %	Au (gpt)	Ag (gpt)	S(t) (%)	Fe (%)	Au (%)	Ag (%)	S(t) (%)	Fe (%)
Concentrate	94	4.6	0.71	18.80	8.56	9.09	20.6	9.7	10.3	7.8
Middlings	237	11.6	0.14	8.30	3.44	4.88	10.3	10.8	10.5	10.6
Tails	1720	83.9	0.13	8.40	3.58	5.16	69.1	79.5	79.2	81.5
Feed	2051	100.0	0.16	8.87	3.79	5.31	100.0	100.0	100.0	100.0
Concentrate + Midds	331	16.1	0.30	11.28	4.89	6.08	30.9	20.5	20.8	18.5

Table 4.22 - Particle Size and Total Sulphur Distribution of Spiral Tails Sample 1, Placer Dome

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.3	0.1	0.1	99.9	0.31	0.0	0.0	100.0
147	4.1	1.5	1.6	98.4	0.31	0.3	0.3	99.7
104	19.0	7.0	8.6	91.4	0.42	1.8	2.1	97.9
74	57.7	21.2	29.8	70.2	0.83	10.9	13.1	86.9
44	112.7	41.5	71.3	28.7	1.75	45.0	58.1	41.9
38	55.9	20.6	91.9	8.1	2.25	28.7	86.7	13.3
-38	22.0	8.1	100.0		2.64	13.3	100.0	
Total	271.7	100.0			1.61	100.0		

Figure 4.12 - Particle Size and Total Sulphur Distribution of Spiral Tails Sample 1, Placer Dome

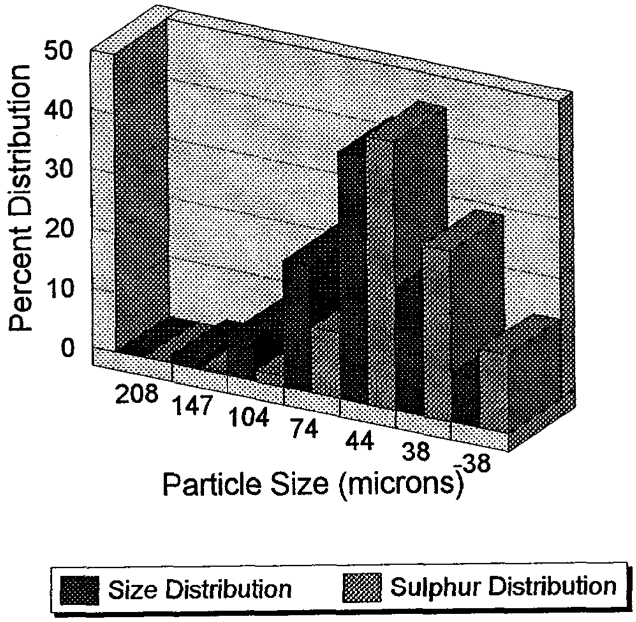


Table 4.23 - Particle Size and Total Sulphur Distribution of Spiral Tails Sample 2, Lac Minerals

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.1	0.0	0.0	100.0	0.32	0.0	0.0	100.0
147	1.7	0.6	0.6	99.4	0.32	0.0	0.1	99.9
104	12.2	4.1	4.7	95.3	0.37	0.4	0.5	99.5
74	20.4	6.8	11.5	88.5	0.54	1.0	1.5	98.5
44	72.3	24.1	35.6	64.4	2.69	17.8	19.3	80.7
38	30.3	10.1	45.7	54.3	5.23	14.5	33.9	66.1
-38	162.8	54.3	100.0		4.43	66.1	100.0	
Total	299.8	100.0			3.64	100.0		

Figure 4.13 - Particle Size and Total Sulphur Distribution of Spiral Tails Sample 2, Lac Minerals

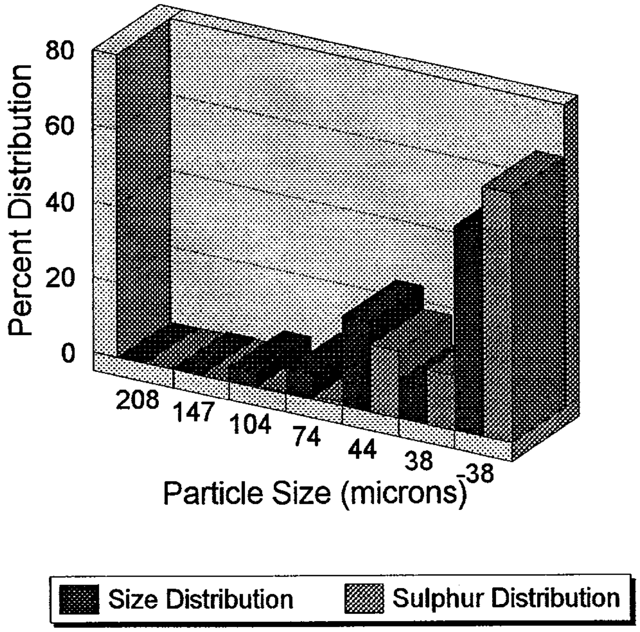


Table 4.24 - Particle Size and Total Sulphur Distribution of Spiral Tails - Sample 3.2, Les Mines Selbaie

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.1	0.0	0.0	100.0	2.24	0.0	0.0	100.0
147	1.3	0.4	0.4	99.6	2.24	0.0	0.0	100.0
104	10.0	3.1	3.5	96.5	4.71	0.7	0.8	99.2
74	19.3	6.0	9.5	90.5	5.76	1.8	2.5	97.5
44	106.1	32.8	42.4	57.6	13.76	23.1	25.6	74.4
38	59.7	18.5	60.8	39.2	22.83	21.5	47.1	52.9
-38	126.5	39.2	100.0		26.46	52.9	100.0	
Total	323.0	100.0			19.60	100.0		

Figure 4.14 - Particle Size and Total Sulphur Distribution of Spiral Tails - Sample 3.2, Les Mines Selbaie

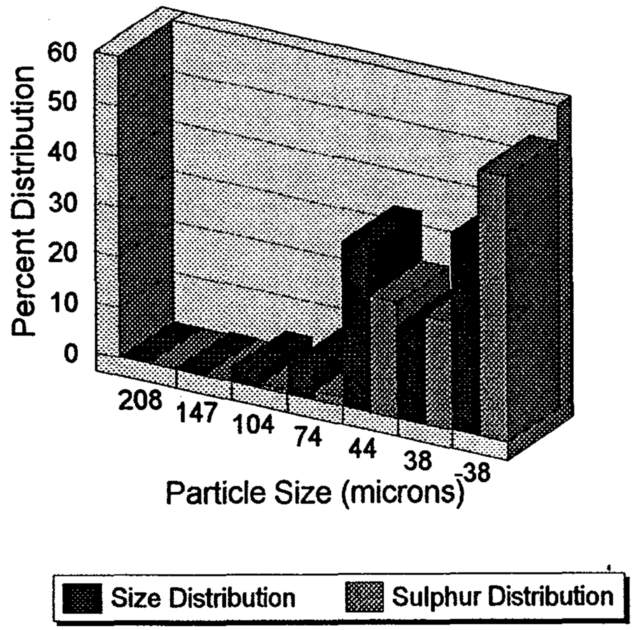
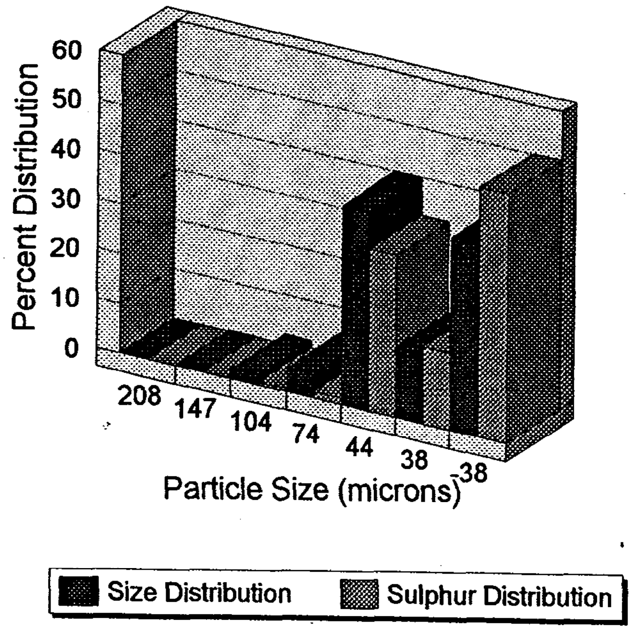


Table 4.25 - Particle Size and Total Sulphur Distribution of Spiral Tails - Sample 3.3, Les Mines Selbaie

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.1	0.0	0.0	100.0	1.35	0.0	0.0	100.0
147	0.7	0.2	0.3	99.7	1.35	0.1	0.1	99.9
104	5.8	2.0	2.2	97.8	0.93	0.6	0.7	99.3
74	14.1	4.8	7.0	93.0	0.85	1.2	1.9	98.1
44	118.8	40.1	47.1	52.9	2.66	32.5	34.4	65.6
38	41.7	14.1	61.2	38.8	3.62	15.5	49.9	50.1
-38	115.1	38.8	100.0		4.24	50.1	100.0	
Total	296.3	100.0			3.29	100.0		

Figure 4.15 - Particle Size and Total Sulphur Distribution of Spiral Tails - Sample 3.3, Les Mines Selbaie



4.6 Shaking Table

Table tests were conducted by Lakefield Research using over a 1/8 scale Wilfley shaking table. Slurry, adjusted to 1277 g/l pulp consistency, was fed to the table using a variable speed peristaltic pump. The table was adjusted to provide a well defined sulphide band and then sampled over a period of 3 - 5 minutes. During the sampling period, all products streams were collected in drums, filtered weighed and submitted for assay. The measured flow rated and calculated metallurgical balances are presented in Table 4.26. Tables 4.27 - 4.30 and Figures 4.16 - 4.19 show the particle size and total sulphur distributions of the table tails.

Table 4.26 - Mass and Metallurgical Balances for Table Tests

Sample 1 (Placer Dome)

			Assays				Distribution			
Product	Flow Rat kg/h	Wt %	Au (gpt)	Cu (%)	S(t) (%)	Fe (%)	Au (%)	Cu (%)	S(t) (%)	Fe (%)
Concentrate	6	5.6	0.56	0.39	8.64	16.90	11.2	18.7	24.2	10.6
Middlings	27	25.1	0.35	0.12	2.62	9.75	31.6	26.1	33.2	27.8
Tails	75	69.3	0.23	0.09	1.22	7.82	57.2	55.2	42.6	61.5
Feed	108	100.0	0.28	0.12	1.98	8.81	100.0	100.0	100.0	100.0
Concentrate + Midds	33	30.7	0.39	0.17	3.71	11.04	42.8	44.8	57.4	38.5

Sample 2 (Lac Minerals)

			Assays				Distribution			
Product	Flow Rat kg/h	Wt %	Au (gpt)	Te (gpt)	S(t) (%)	Fe (%)	Au (%)	Te (%)	S(t) (%)	Fe (%)
Concentrate	8	7.1	0.65	6.00	13.70	13.20	24.0	7.6	25.2	18.6
Middlings	20	17.9	0.32	4.00	2.80	3.77	29.5	12.7	12.9	13.2
Tails	84	75.0	0.12	6.00	3.20	4.62	46.5	79.7	61.9	68.2
Feed	112	100.0	0.19	5.64	3.88	5.08	100.0	100.0	100.0	100.0
Concentrate + Midds	28	25.0	0.41	4.57	5.91	6.46	53.5	20.3	38.1	31.8

Sample 3.2 (Selbaie)

			Assays				Distribution			
Product	Flow Rat kg/h	Wt %	Au (gpt)	Ag (gpt)	S(t) (%)	Fe (%)	Au (%)	Ag (%)	S(t) (%)	Fe (%)
Concentrate	8	3.0	0.37	37.50	40.50	35.00	8.9	7.0	9.7	8.6
Middlings	205	80.6	0.11	13.90	10.40	10.20	71.4	70.2	67.0	68.0
Tails	42	16.4	0.15	22.20	17.80	17.30	19.7	22.8	23.3	23.4
Feed	254	100.0	0.12	15.96	12.51	12.10	100.0	100.0	100.0	100.0
Concentrate + Midds	213	83.6	0.12	14.74	11.47	11.09	80.3	77.2	76.7	76.6

Sample 3.3 (Selbaie)

			Assays				Distribution			
Product	Flow Rat kg/h	Wt %	Au (gpt)	Ag (gpt)	S(t) (%)	Fe (%)	Au (%)	Ag (%)	S(t) (%)	Fe (%)
Concentrate	14	4.7	0.47	30.50	16.80	16.30	18.2	16.0	21.2	14.7
Middlings	26	8.8	0.15	6.90	2.52	3.57	10.8	6.7	5.9	6.0
Tails	257	86.5	0.10	8.00	3.15	4.81	71.0	77.2	72.9	79.4
Feed	297	100.0	0.12	8.96	3.74	5.24	100.0	100.0	100.0	100.0
Concentrate + Midds	40	13.5	0.26	15.16	7.52	8.03	29.0	22.8	27.1	20.6

Table 4.27 - Particle Size and Total Sulphur Distribution of Table Tails Sample 1, Placer Dome

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.8	0.3	0.3	99.7	0.26	0.1	0.1	99.9
147	6.9	2.4	2.7	97.3	0.33	0.7	0.7	99.3
104	22.5	8.0	10.7	89.3	0.34	2.3	3.1	96.9
74	40.0	14.2	24.9	75.1	0.54	6.5	9.6	90.4
44	101.8	36.1	61.0	39.0	1.13	34.6	44.2	55.8
38	45.5	16.1	77.2	22.8	1.51	20.7	64.9	35.1
-38	64.4	22.8	100.0		1.81	35.1	100.0	
Total	281.9	100.0			1.18	100.0		

Figure 4.16 - Particle Size and Total Sulphur Distribution of Table Tails Sample 1, Placer Dome

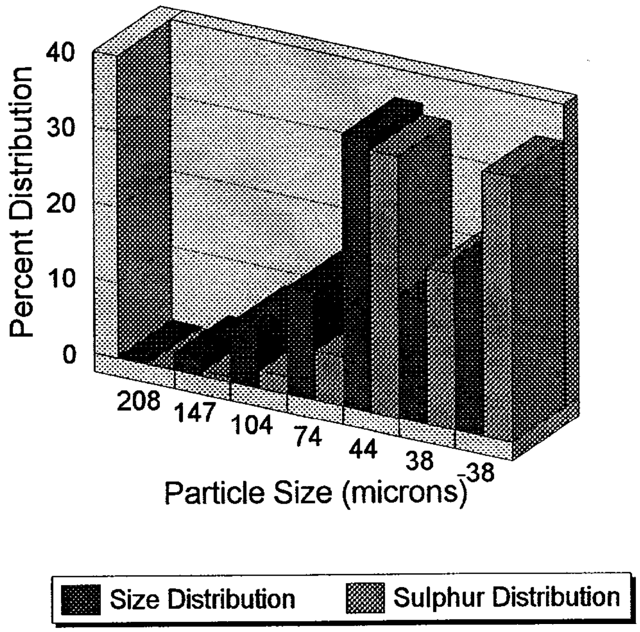


Table 4.28 - Particle Size and Total Sulphur Distribution of Table Tails Sample 2, Lac Minerals

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.1	0.0	0.0	100.0	2.24	0.0	0.0	100.0
147	0.8	0.3	0.3	99.7	2.24	0.2	0.2	99.8
104	7.0	2.3	2.6	97.4	1.64	1.2	1.4	98.6
74	13.6	4.5	7.1	92.9	1.67	2.4	3.8	96.2
44	54.6	18.0	25.0	75.0	1.35	7.7	11.4	88.6
38	33.7	11.1	36.1	63.9	3.27	11.5	22.9	77.1
-38	194.3	63.9	100.0		3.81	77.1	100.0	
Total	304.1	100.0			3.16	100.0		

Figure 4.17 - Particle Size and Total Sulphur Distribution of Table Tails Sample 2, Lac Minerals

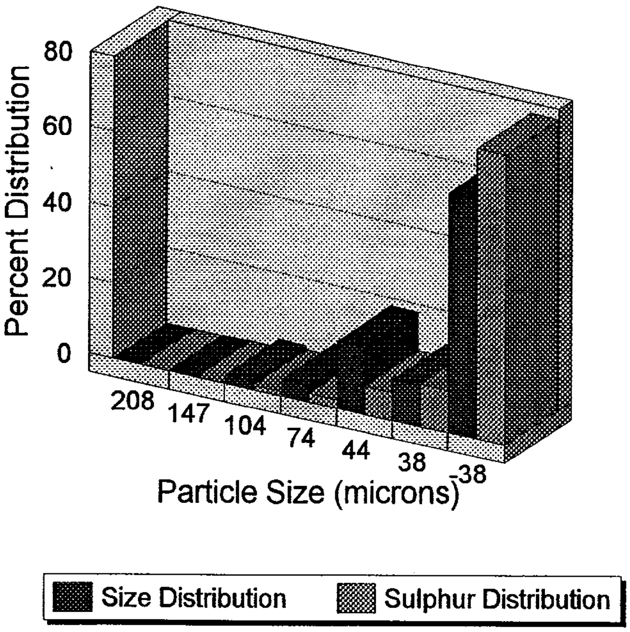


Table 4.29 - Particle Size and Total Sulphur Distribution of Table Tails Sample 3.2, Les Mines Selbaie

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.1	0.0	0.0	100.0	8.70	0.0	0.0	100.0
147	2.3	0.7	0.7	99.3	8.70	0.3	0.4	99.6
104	11.4	3.4	4.1	95.9	9.88	1.9	2.2	97.8
74	15.3	4.6	8.7	91.3	6.37	1.6	3.9	96.1
44	90.2	27.0	35.7	64.3	8.88	13.5	17.3	82.7
38	56.1	16.8	52.5	47.5	18.20	17.2	34.5	65.5
-38	159.0	47.5	100.0		24.53	65.5	100.0	
Total	334.4	100.0			17.80	100.0		

Figure 4.18 - Particle Size and Total Sulphur Distribution of Table Tails Sample 3.2, Les Mines Selbaie

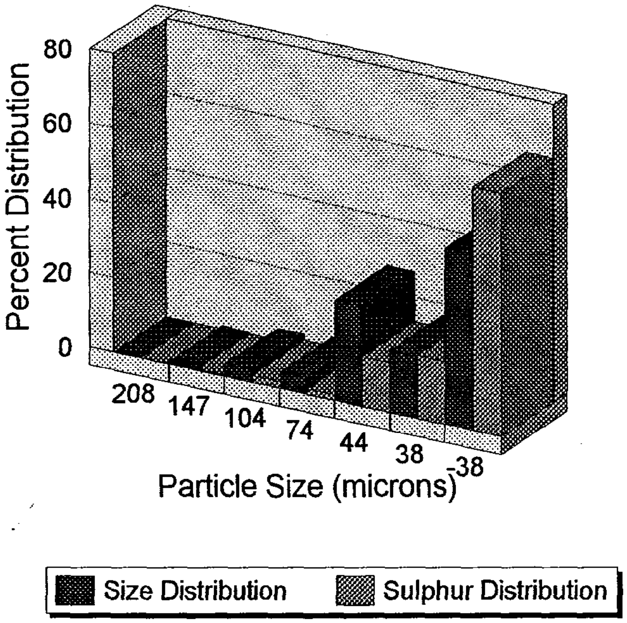
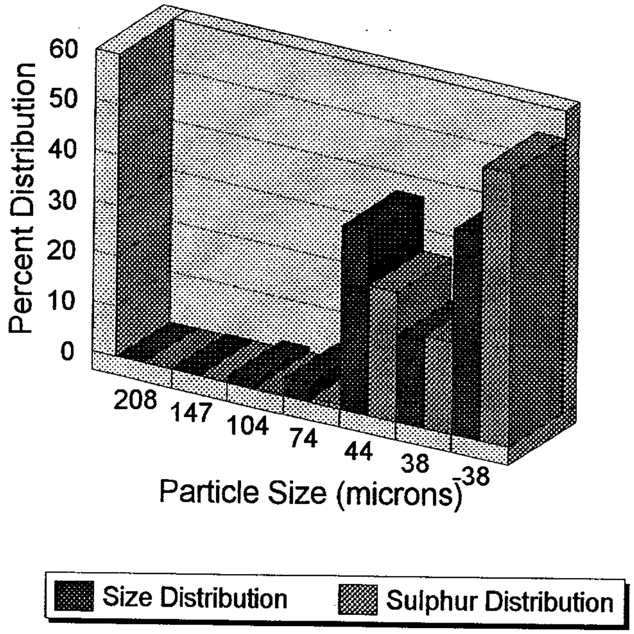


Table 4.30 - Particle Size and Total Sulphur Distribution of Table Tails Sample 3.3, Les Mines Selbaie

Size Fraction (microns)	Weight (g)	Weight (%)	Cum Wt (%)	Cum Wt % Passing	% S(t)	Dist. % S(t)	Cum Dist % S(t)	Cum % Passing
208	0.0	0.0	0.0	100.0	0.00	0.0	0.0	100.0
147	0.7	0.2	0.2	99.8	0.70	0.1	0.1	99.9
104	4.4	1.5	1.8	98.2	0.78	0.4	0.5	99.5
74	10.7	3.7	5.4	94.6	0.75	1.0	1.4	98.6
44	106.0	36.4	41.9	58.1	2.05	25.7	27.1	72.9
38	49.9	17.2	59.0	41.0	3.11	18.4	45.5	54.5
-38	119.2	41.0	100.0		3.86	54.5	100.0	
Total	290.9	100.0			2.90	100.0		

Figure 4.19 - Particle Size and Total Sulphur Distribution of Table Tails Sample 3.3, Les Mines Selbaie



4.7 Direction Flotation

Direct flotation experiments were preformed on all samples. In each case the flotation test was combined with one or several pre-processing steps. These pre-processing steps are defined below.

Placer - Detour Lake Mine

The tests which were done on the Placer Dome sample include the following:

- Trial 1 - CN destruction prior to flotation,
- Trial 2 - CN Destruction at higher pH,
- Trial 3 - Removal of Pyrrhotite by magnetic separation; add $CuSO_4$ to CN destruction step,
- Trial 4 - Removal of Pyrrhotite by magnetic separation; no CN destruction,
- Trial 5 - Removal of Pyrrhotite by magnetic separation; different Cu collector, and
- Trial 6 - Removal of Pyrrhotite by magnetic separation from rougher concentrate.

Lac Minerals - La Mine Doyon

The tests performed on the Lac Minerals sample include the following:

- Trial 1 - CN destruction prior to flotation,
- Trial 2 - Test with 5100 and 404,
- Trial 3 - Test with 5100 and 404,
- Trial 4 - No CN destruction; use Texeco SP-160 as collector,
- Trial 5 - CN destruction prior to flotation; use Texeco SP-160 as collector, and
- Trial 6 - CN destruction prior to flotation; float using PAX only.

Les Mines Selbaie - Sample 3.2

The tests performed on the Selbaie sample 3.2 include the following:

- Trial 1 - Test with $CuSO_4$ and PAX,
- Trial 2 - Lower pH with H_2SO_4 and addition of supplemental collector (M-91),
- Trial 3 - Low pH; add M-91 only,
- Trial 4 - Higher pH using mixture of 404 and PAX with pre-aeration,

- Trial 5 - Higher pH using mixture of M-91 and PAX,
- Trial 6 - Higher pH using mixture of Texeco SP-160,
- Trial 7 - Condition with sodium sulphite,
- Trial 8 - Use Texeco SP-160 as collector with longer conditioning time, and
- Trial 9 - Use Texeco SP-160; pre-aerate for 20 minutes.

Les Mines Selbaie - Sample 3.3

The tests performed on the Selbaie sample 3.3 include the following:

- Trial 1 - Higher pH using mixture of 404 and PAX with pre-aeration,
- Trial 2 - Mixture of PAX and M-91,
- Trial 3 - Use Texeco SP-160 as collector,
- Trial 4 - Use Texeco SP-160 as collector; lower addition rate, and
- Trial 5 - Use PAX as collector.

Summaries of the individual tests by feed sample are presented in Tables 4.31 - 4.34. Individual flotation data sheets including metallurgical balances are included in Appendix A.

Table 4.31 - Summary of Flotation Tests - Placer Dome

		Test Number					
Product		1	2	3	4	5	6
Magnetics							
Assay	Au (gpt)			0.49	0.34	0.43	1.27
	Cu (%)			0.18	0.2	0.18	2.09
	S(t) (%)			18.4	19.44	15.69	32.81
Distribution	Wt. (%)			6.81	5.5	7.36	0.4
	Au (%)			9.69	13.74	6.36	1.27
	Cu (%)			8.71	7.24	8.38	5.63
	S(t) (%)			52.05	47.83	53.23	5.72
Copper Concentrate							
Assay	Au (gpt)	1.21	1.05	1.56	1.96	1.56	2.61
	Cu (%)	3.35	5.09	8.27	15.4	15.4	14.57
	S(t) (%)	26	21.1	13.9	25.86	22.04	16.49
Distribution	Wt. (%)	3.58	2.17	1.21	0.61	0.62	0.66
	Au (%)	12.98	6.81	5.48	8.97	1.93	4.25
	Cu (%)	79.14	73.02	70.26	62.32	59.92	64.13
	S(t) (%)	40.37	19.42	6.98	7.11	6.25	4.7
Sulphide Concentrate							
Assay	Au (gpt)	0.96	0.51	0.78	1.15	1.03	1.09
	Cu (%)	0.62	0.47	0.36	0.27	0.29	0.39
	S(t) (%)	21.6	25.1	22.6	27.91	27.15	35.29
Distribution	Wt. (%)	1.57	3.89	3.35	2.82	2.83	4.19
	Au (%)	4.35	5.94	7.58	23.99	5.87	11.27
	Cu (%)	6.45	12.05	8.56	5.01	5.19	10.94
	S(t) (%)	14.76	41.46	31.41	35.17	35.39	64.03
Tailings							
Assay	Au (gpt)	0.29	0.31	0.3	0.07	0.48	0.35
	Cu (%)	0.02	0.02	0.02	0.01	0.02	0.01
	S(t) (%)	1.09	0.98	0.26	0.08	0.06	0.59
Distribution	Wt. (%)	94.85	93.95	88.64	89.62	88.4	94.69
	Au (%)	82.49	87.25	77.25	45.96	85.42	81.59
	Cu (%)	14.41	14.93	12.46	5.9	11.18	6.33
	S(t) (%)	44.88	39.12	9.57	3.21	2.44	24.18

Table 4.32 - Summary of Flotation Tests - Lac

		Test Number					
Product		1	2	3	4	5	6
Sulphide Concentrate							
Assay	Au (gpt)	0.49	0.46	0.45	0.26	0.52	0.70
	Te (gpt)	14.00	17.00	15.00	11.00	12.34	11.86
	S(t) (%)	28.42	27.78	25.11	4.05	23.95	38.83
Distribution	Wt. (%)	14.08	15.45	15.82	17.02	15.51	9.22
	Au (%)	27.36	28.27	28.51	15.01	26.85	21.57
	Te (%)	21.26	26.16	24.47	25.77	25.00	13.09
	S(t) (%)	98.66	93.15	93.46	16.68	93.81	85.86
Tailings							
Assay	Au (gpt)	0.22	0.20	0.21	0.30	0.26	0.26
	Te (gpt)	8.50	8.00	8.70	6.50	6.80	8.00
	S(t) (%)	0.15	0.34	0.33	4.15	0.29	0.65
Distribution	Wt. (%)	85.92	84.55	84.18	82.98	84.49	90.78
	Au (%)	72.64	71.73	71.49	84.99	73.15	78.43
	Te (%)	78.74	73.84	75.53	74.23	75.00	86.91
	S(t) (%)	3.12	6.85	6.54	83.32	6.19	14.14

Table 4.33 - Summary of Flotation Tests - Selbaie Sample 3.2

		Test Number								
Product		1	2	3	4	5	6	7	8	9
Sulphide Concentrate										
Assay	Au (gpt)	0.35	0.29	0.19	0.30	0.42	0.26	0.21	0.40	0.37
	Ag(gpt)	79.09	51.00	50.10	41.59	44.44	45.50	46.65	47.21	47.07
	S(t) (%)	29.58	46.87	46.44	41.95	43.50	39.39	41.82	46.27	46.02
Distribution	Wt. (%)	10.81	46.50	48.62	50.43	48.77	52.44	50.38	46.38	44.70
	Au (%)	24.92	83.15	77.90	71.59	65.51	81.04	75.19	60.34	62.93
	Ag(%)	32.62	93.07	94.05	89.37	89.94	90.14	89.61	91.84	91.51
	S(t) (%)	14.52	98.31	98.30	97.31	97.71	98.35	96.74	96.27	96.90
Tailings										
Assay	Au (gpt)	0.13	0.05	0.05	0.12	0.21	0.07	0.07	0.23	0.17
	Ag(gpt)	19.80	3.30	3.00	5.03	4.73	5.49	5.49	3.63	3.53
	S(t) (%)	21.10	0.70	0.76	1.18	0.97	0.73	1.43	1.55	1.19
Distribution	Wt. (%)	89.19	53.50	51.38	49.57	51.23	47.56	49.62	53.62	55.30
	Au (%)	75.08	16.85	22.10	28.41	34.49	18.96	24.81	39.66	37.07
	Ag(%)	67.38	6.93	5.95	10.63	10.06	9.86	10.39	8.16	8.49
	S(t) (%)	85.48	1.69	1.70	2.69	2.29	1.65	3.26	3.73	3.10

Table 4.34 - Summary of Flotation Tests - Selbaie Sample 3.3

		Test Number				
Product		1	2	3	4	5
Sulphide Concentrate						
Assay	Au (gpt)	0.50	0.68	0.33	0.45	0.51
	Ag(gpt)	51.45	40.11	33.62	48.24	52.65
	S(t) (%)	29.96	20.47	16.69	25.79	28.13
Distribution	Wt. (%)	10.62	15.78	19.29	11.12	11.21
	Au (%)	33.39	38.43	32.01	22.53	23.79
	Ag(%)	68.05	73.35	98.41	82.28	90.11
	S(t) (%)	87.89	92.31	90.67	81.95	90.10
Tailings						
Assay	Au (gpt)	0.12	0.21	0.17	0.20	0.21
	Ag(gpt)	2.87	2.73	0.13	1.30	0.73
	S(t) (%)	0.49	0.32	0.41	0.71	0.39
Distribution	Wt. (%)	89.38	84.22	80.71	88.88	88.79
	Au (%)	66.61	61.57	67.99	77.47	76.21
	Ag(%)	31.95	26.65	1.59	17.72	9.89
	S(t) (%)	12.11	7.69	9.33	18.05	9.90

4.8 Acid Generating Capability

Evaluation Approach

All data was reviewed with respect to the project objectives, methodology and results. Mass balance and acid/neutralization potential calculations were made to gain a clearer analysis of the results of the processing techniques employed and graphical presentations of this analysis are presented in Section 5.0 as appropriate. Tight project scheduling and delays in sample delivery did not allow CESL sufficient time to carry out kinetic tests on the products of the processing techniques which showed the greatest promise for success. The above analytical approach was necessary to enable predictions to be made with regard to the environmental performance of the separation products. The kinetic tests performed on the head samples were also evaluated to allow further insight into the potential weathering characteristics of the products of the processing procedures.

Static Test Results - Acid Base Accounting

Table 4.35 (a-d) provides a summary of the data from the acid base accounting tests on the head samples and products from the various separation tests. The mass fraction of the head sample recovered, weight percent of sulphur in the products, percent of the original sulphur present, net neutralization potential (Net NP) and neutralization potential to acid potential ratio (NP/AP) data are presented.

The acid base accounting tests that were preformed constitute one method of assessing the neutralization potential of the samples. Alternatively, a theoretical neutralization potential may be calculated from the elemental analysis for carbon dioxide. Table 4.36 provides a summary of neutralization potentials derived from the CO_2 analysis and by the acid base accounting There are some substantial differences in these two sets of numbers particularly for the Placer Dome, Detour Lake sample. Discussions of these data are provided in Section 5.0,

Metals Distribution in Heads and Products

Tables 4.37 to 4.40 provide data on the distribution of selected metals of concern in the various products as determined by ICP. It is noted that several discrepancies in the analyses are apparent, particularly with regard to As, Mo, Ni, and Sb. The mass balance calculations indicate significantly higher quantities reporting to the products when compared to the head samples. These discrepancies are due to the level of accuracy of ICP tests as well as interferences associated with the analysis. The distribution of other elements should therefore be evaluated with some caution.

Kinetic Test Results - Humidity Cells

Table 4.41 through 4.44 summarize the data from the humidity cell tests. Provided for review are some of the more environmentally relevant parameters including pH, conductivity, cumulative sulphate and acidity, arsenic, nickel, zinc, lead, copper and cadmium. The data provides a review of the analysis of leaching products throughout the 19 week test with one analysis per week. The exception to this is the Selbaie sample 3.3 which has only a 15 week test period.

Table 4.35(a) - Acid Base Accounting Data
Placer Dome

	Weight	S %	S Distrib	Net NP	NP/AP
Head	100.0	2.34	100.0	-40.6	.04
Float 1	94.9	1.09	44.9	-6.6	0.8
Float 3	88.6	0.26	9.6	21.5	3.6
Float 4	89.6	0.08	3.2	27.7	12.1
Falcon	69.5	1.60	47.1	-19.5	0.6
Table	69.3	1.20	44.9	-11.1	0.7

Table 4.35(b) - Acid Base Accounting Data
Lac Minerals

	Weight	S %	S Distrib	Net NP	NP/AP
Head	100.0	4.15	100.0	-101.9	0.2
Float 1	85.9	0.15	3.1	21.4	5.6
Float 3	84.6	0.34	6.7	14.4	2.4
Table	75.1	3.20	61.9	-73.1	0.3
Falcon	64.4	1.84	29.3	-32.3	0.4
Knelson	88.2	3.42	49.8	-82.0	0.2

Table 4.35(c) - Acid Base Accounting Data
Selbaie Sample 3.2

	Weight	S %	S Distrib	Net NP	NP/AP
Head	100.0	23.6	100.0	-685.5	0.1
Float 2	53.5	0.7	1.7	51.1	3.3
Float 4	49.6	1.2	2.7	33.9	1.9
Float 7	49.6	1.4	3.3	27.7	1.9
Falcon	44.7	12.4	26.6	-337.3	0.1
Table	71.3	17.8	65.8	-514.0	0.1

Table 4.35(d) - Acid Base Accounting Data
Selbaie Sample 3.3

	Weight	S %	S Distrib	Net NP	NP/AP
Head	100.0	3.5	100.0	-91.0	0.1
Float 1	89.4	0.5	12.1	-0.4	1.0
Float 2	84.2	0.3	7.7	4.6	1.5
Tray	50.1	2.9	48.6	-77.5	0.1

Table 4.36 - Comparison of CO_2 and ABA Neutralization Potential

	CO_2 (%)	NP by CO_2	NP by ABA	% NP as CO_3
Placer Dome - Detour Lake	0.7	15.9	32.2	49
Lac Minerals - Mine Doyon	0.9	20.5	26.5	77
Selbaie 3.2	1.9	43.2	47.0	92
Selbaie 3.3	0.7	15.9	15.0	100

PLACER DOME, DETOUR LAKE

Table 4.37(a) - ICP/AES Analysis of Tails

Element		Sample FS1-1 Tail	Sample FS1-3 Tail	Sample FS1-4 Tail	Sample #1 Falcon Tail	Sample #1 Table Tail
As	ppm	1	1	1	1	1
Bi	ppm	14	14	13	21	20
Cd	ppm	0.1	0.1	0.1	0.1	0.1
Co	ppm	25	9	9	22	23
Cu	ppm	202	178	170	1166	1051
Fe	%	3.33	2.87	3.05	4.73	4.55
Mn	ppm	415	450	485	531	540
Mo	ppm	2	2	3	2	2
Ni	ppm	36	28	28	61	68
Pb	ppm	184	186	180	295	323
Sb	ppm	13	17	17	17	17
Zn	ppm	47	48	52	111	152
Cr	ppm	85	90	98	120	151

Table 4.37(b) - Elemental Distribution in Tails

Element		Sample FS1-1 Tail	Sample FS1-3 Tail	Sample FS1-4 Tail	Sample #1 Falcon Tail	Sample #1 Table Tail
		94.9	88.6	89.6	69.5	69.3
As	ppm	95	89	90	70	69
Bi	ppm	46	43	40	50	48
Cd	ppm	95	89	90	70	69
Co	ppm	66	22	22	42	44
Cu	ppm	13	11	10	55	49
Fe	%	57	46	49	59	57
Mn	ppm	70	71	78	66	67
Mo	ppm	63	59	90	46	46
Ni	ppm	62	45	46	77	86
Pb	ppm	72	68	67	85	93
Sb	ppm	206	251	254	197	196
Zn	ppm	42	40	44	72	98
Cr	ppm	65	64	70	67	84

LAC MINERALS, LA MINE DOYON

Table 4.38(a) - ICP/AES Analysis of Tails

Element		Sample FS2-1 Tail	Sample FS2-2 Tail	Sample #2 Falcon Tail	Sample #2 Knelson Tail (2)	Sample #2 Table Tail
As	ppm	1	1	1	1	4
Bi	ppm	6	5	7	6	7
Cd	ppm	0.1	0.1	0.1	0.1	0.1
Co	ppm	2	3	6	10	9
Cu	ppm	105	96	254	381	386
Fe	%	1.39	1.5	2.57	3.66	3.52
Mn	ppm	352	358	338	318	364
Mo	ppm	4	3	3	3	5
Ni	ppm	10	10	12	16	19
Pb	ppm	78	78	96	85	64
Sb	ppm	12	13	11	9	10
Zn	ppm	40	44	135	147	79
Cr	ppm	16	17	15	14	18

Table 4.38(b) - Elemental Distribution in Tails

	Sample FS2-1 Tail	Sample FS2-2 Tail	Sample #2 Falcon Tail	Sample #2 Knelson Tail (2)	Sample #2 Table Tail
Weight	85.9	84.6	64.4	88.2	75.1
As ppm	86	85	64	88	300
Bi ppm	52	42	45	83	53
Cd ppm	86	85	64	88	75
Co ppm	13	20	30	68	52
Cu ppm	21	19	39	79	68
Fe %	28	30	39	75	62
Mn ppm	93	93	67	86	84
Mo ppm	86	63	48	66	94
Ni ppm	859	846	773	1411	1427
Pb ppm	75	74	69	84	54
Sb ppm	1031	1100	708	794	751
Zn ppm	20	22	51	76	35
Cr ppm	92	96	64	82	90

LES MINES SELBAIE, SAMPLE 3.2

Table 4.39(a) - ICP/AES Analysis of Tails

Element		Sample FS3.2-2 Tail	Sample FS3.2-4 Tail	Sample FS3.2-7 Tail	Sample #3.2 Table Tail	Sample #3.2 Falcon Tail
As	ppm	16	26	34	147	111
Bi	ppm	2	3	3	1	5
Cd	ppm	0.1	2	4.6	29.7	45
Co	ppm	3	4	4	22	16
Cu	ppm	101	172	169	836	1178
Fe	%	2.32	2.66	2.89	14.73	10.44
Mn	ppm	904	899	913	609	635
Mo	ppm	3	3	2	1	3
Ni	ppm	12	14	13	46	36
Pb	ppm	130	157	134	399	475
Sb	ppm	9	11	9	4	13
Zn	ppm	716	1303	1899	6929	8981
Cr	ppm	72	93	81	107	157

Table 4.39(b) - Elemental Distribution in Tails

Element		Sample FS3.2-2 Tail	Sample FS3.2-4 Tail	Sample FS3.2-7 Tail	Sample #3.2 Table Tail	Sample #3.2 Falcon Tail
		53.5	49.6	49.6	71.3	44.7
As	ppm	5	8	10	64	30
Bi	ppm	54	74	74	36	112
Cd	ppm	0	2	6	52	49
Co	ppm	5	6	6	51	23
Cu	ppm	6	10	9	67	59
Fe	%	8	9	10	70	31
Mn	ppm	85	79	80	76	50
Mo	ppm	161	149	99	71	134
Ni	ppm	642	694	645	3280	1609
Pb	ppm	14	16	13	57	43
Sb	ppm	69	78	64	41	83
Zn	ppm	5	8	11	59	48
Cr	ppm	32	38	33	64	58

LES MINES SELBAIE, SAMPLE 3.3

Table 4.40(a) - ICP/AES Analysis of Tails

Element		Sample FS3.3-1 Tail	Sample FS3.3-2 Tail	Sample #3.3 Tray Tail
As	ppm	9	12	50
Bi	ppm	3	3	4
Cd	ppm	0.1	0.1	1.1
Co	ppm	3	3	10
Cu	ppm	165	152	464
Fe	%	2.15	2.1	4.06
Mn	ppm	552	556	530
Mo	ppm	2	3	2
Ni	ppm	11	9	74
Pb	ppm	106	96	200
Sb	ppm	10	9	9
Zn	ppm	694	750	1405
Cr	ppm	88	84	98

Table 4.40(b) - Elemental Distribution in Tails

Element		Sample FS3.3-1 Tail	Sample FS3.3-2 Tail	Sample #3.2 Table Tail
		89.4	84.2	50.1
As	ppm	16	21	51
Bi	ppm	89	84	67
Cd	ppm	3	3	17
Co	ppm	24	23	46
Cu	ppm	35	30	55
Fe	%	43	40	46
Mn	ppm	94	89	50
Mo	ppm	45	63	25
Ni	ppm	983	758	3707
Pb	ppm	45	38	47
Sb	ppm	81	69	41
Zn	ppm	43	44	49
Cr	ppm	72	64	45

Table 4.41 - Humidity Cell Data
Placer Dome, Detour Lake

Cycle	pH	Conductivity	Cumulative Sulphate	Cumulative Acidity	Arsenic ug/l	Nickle ug/l	Zinc ug/l	Lead ug/l	Copper ug/l	Cadmium ug/l
1	8.15	819	74.5	0.2	40	5	1	14	2	6
2	7.95	706	140.5	0.9	100	5	1	32	2	4
3	7.57	524	228.8	1.8	170	10	22	56	4	7
4	7.85	771	369.4	2.4	40	5	1	20	2	7
5	7.81	802	538.4	3.2	130	5	5	24	14	5
6	7.77	681	676.3	3.8	130	5	31	8	2	1
7	7.8	572	782.1	4.5	140	5	16	14	2	1
8	7.63	572	897.2	5.1	10	5	1	24	2	1
9	7.74	676	1018.6	5.7	70	10	15	52	2	5
10	7.73	711	1171.4	6.5	70	10	1	20	10	10
11	7.66	640	1317.5	7.4	130	10	31	60	14	9
12	7.45	570	1439.3	8.2	20	5	1	38	4	3
13	7.81	713	1583.1	8.6	60	5	1	16	2	7
14	7.7	819	1757.7	9.3	10	5	1	2	2	6
15	7.69	685	1892.5	9.7	10	5	5	24	2	2
16	7.97	722	2034.7	10.3	20	5	4	20	4	5
17	8.01	869	2179.4	10.8	60	5	1	2	8	8
18	7.69	503	2284.3	11.9	100	5	36	14	4	1
19	7.98	525	2347.3	12.6						
20										

Table 4.42 - Humidity Cell Data
Lac Minerals, La Mine Doyon

Cycle	pH	Conductivity	Cumulativ Sulphate	Cumulative Acidity	Arsenic ug/l	Nickle ug/l	Zinc ug/l	Lead ug/l	Copper ug/l	Cadmium ug/l
1	10.25	1393	92.3	0	160	60	2	36		3
2	7.83	1008	232.3	0.5	150	10	3	36	926	1
3	7.76	245	263.8	1.1	90	5	2	18	2	1
4	7.48	929	389.6	1.7	50	5	1	2	4	1
5	7.15	738	495.6	2.6	20	5	1	2	2	1
6	6.38	694	642.9	3.9	40	5	17	2	2	1
7	7.57	498	744.7	4.6	170	5		66	2	22
8	7.39	401	821.1	5.5	10	5	1	2	2	1
9	7.47	626	934.2	6	10	5	3	2	2	1
10	7.62	717	1092.3	6.9	80	5	1	34	10	6
11	7.42	481	1200.1	8.4	130	10	7	54	10	6
12	7.3	505	1314.4	9.3	20	10	1	16	2	1
13	7.57	454	1408.9	9.7	10	5	1	6	2	7
14	7.62	819	1606.3	10.5	10	5	1	2	2	3
15	7.35	605	1751	11.4	10	5	6	2	2	4
16	7.5	678	1916	12.2	40	5	1260	20	4	13
17	7.57	642	2066.6	13.1	40	5	9	56	6	1
18	7.25	703	2212.9	14.5	10	5	6	40	6	8
19	7.63	565	2323.3	15.6						
20										

Table 4.43 - Humidity Cell Data
Les Mines Selbaie, Sample 3.2

Cycle	pH	Conductivity	Cumulative Sulphate	Cumulativ Acidity	Arsenic ug/l	Nickle mg/l	Zinc ug/l	Lead mg/l	Copper mg/l	Cadmium mg/l
1	8.05	864	111.1	0.2	10	10	0.066	0.028	0.156	0.01
2	6.89	1155	302.5	12.5	350	55	17.7	0.2	0.014	0.081
3	6.51	1962	728	72.4	700	85	82.7	0.45	0.02	0.281
4	6.64	2860	1406.6	153.8	1080	40	149	0.734	0.03	0.55
5	6.77	3730	2552.4	251.3	1240	5	166.6	0.774	0.026	0.699
6	5.03	2350	3280.9	314.4	850	20	104.7	0.572	0.07	0.425
7	6.45	2180	3922.6	357.8	830	5	67	0.544	0.018	0.283
8	6.17	2240	4544.2	406.7	770	5	83.9	0.532	0.03	0.343
9	6.29	2630	5189.4	475.8	850	5	116.8	0.584	0.046	0.578
10	6.27	3160	5976	556.8	1030	5	149.5	0.732	0.094	0.581
11	6.39	5820	8056.4	701.2	1750	5	253.9	1.136	0.124	1.693
12	5.97	3580	9150.2	798	1200	5	165.5	0.852	0.102	1.101
13	6.45	3710	10191.7	890.6	1130	5	177.6	0.74	0.09	1.154
14	5.9	4720	11820	1014.6	1350	5	225.7	0.854	0.212	1.621
15	6.03	3420	12916.8	1121.2	1010	5	157.4	0.704	0.152	1.094
16	5.62	3970	14352.7	1259.8	1390	5	245	0.95	0.668	1.667
17	5	5210	15883.6	1421.6	1500	5	369.2	1.146	1.228	2.645
18	4.3	5500	17634.5	1696.9	1560	195	388.6	1.143	5.174	2.95
19	3.72	5990	19556.7	2144.5						
20										

Table 4.44 - Humidity Cell Data
Les Mines Selbaie, Sample 3.3

Cycle	pH	Conductivity	Cumulative Sulphate	Cumulative Acidity	Arsenic ug/l	Nickle ug/l	Zinc ug/l	Lead ug/l	Copper ug/l	Cadmium ug/l
1	8.07	426	53.4	0.2	10	5	150	10	22	6
2	7.66	271	81.3	0.9	20	5	453	2	26	1
3	7.9	410	133	1.4	110	5	225	2	36	1
4	7.5	92	142.7	1.8	90	10	847	14	2	1
5	7.28	93	154.1	2.5	10	10	183	2	2	1
6	7.27	466	228.1	2.9	10	5	867	10	2	3
7	7.32	2420	690.8	11.3	310	35	11326	252	46	91
8	7.63	889	896	13.5	150	15	2497	96	12	33
9	7.87	564	1010.6	14.2	50	5	475	50	4	11
10	7.7	1841	1368.6	16.6	250	5	3908	190	8	66
11	7.43	247	1414.2	17	10	5	453	2	2	8
12	7.05	438	1498	17.9	90	5	725	52	2	15
13	7.66	1101	1723.2	19.1	210	5	2790	128	10	37
14	7.59	2210	2270.5	21.7	480	5	4892	326	20	102
15	8.09	729	2410.3	22.6						
16										
17										
18										
19										
20										

5.0 DATA DISCUSSION AND EVALUATION

The following section of this document will present explanations of the data obtained and assess the effectiveness of the various mineral processing technologies which were tested. For those techniques tested that were not effective in separating the sulphide fraction of the tailings, discussion will be provided to explain the inefficiency.

5.1 Sample Characterization

Results from the elemental and mineralogical examinations (Tables 4.1 and 4.2) indicate that the tailings sample from Placer Dome's Detour Lake Mine, is made up primarily of silicates with a small quantity of sulphides present as approximately equal proportions of pyrite and pyrrhotite and a minor amounts of chalcopyrite (0.15 % Cu) and gold (0.3 gpt Au). The contained level of total sulphur (2.34 % $S_{(T)}$) was the lowest of the four samples tested. The silicates are composed of Quartz/Feldspars and Hornblende. The mineralogy is simple with liberation of sulphides from the gangue estimated to be greater than 98 %. The overall particle size range, observed during the mineralogical examination, is 5 - 200 microns. From Figure 4.1 it can been seen that the sulphides tend to be concentrated in the finer size fractions. Whereas the d_{80} of the overall sample is estimated to be 95 microns, the d_{80} size of the sulphides is 74 microns. The distribution of gold shown by Figure 4.7, is concentrated in the coarser size fractions with less than 20% of the gold reporting to the -44 micron (325 mesh) size fraction. This implies that the residual gold values are not associated with the sulphides and are likely present as inclusions with the silicate gangue.

The sample of tailings from Lac's La Mine Doyon consists predominantly of quartz and feldspars with pyrite as the principle sulphide constituent. The total sulphur content was measured at 4.15 %S(t). The sample also contains a minor amount of gold (0.2 gpt Au), tellurium (6 gpt Te) and titanium (0.2 % Ti) present as liberated grains of rutile. The carbonate content was estimated to be about 0.5 %.

The mineralogy, as observed with the Detour Lake sample is very simple and shows virtually complete liberation of the sulphides from the gangue as well as the sulphides from each other. The overall particle size range was observed to fall between 5 - 200 microns. From Table-4.4 and Figure-4.2 it can be seen that the sample contains a high proportion of fines with almost 50 % by weight being finer than 38 microns. The distribution of the sulphides is even finer with 55 % of the sulphur contained in the -38 micron size fraction. The gold is more evenly distributed amongst the various size fractions as shown by Figure 4.5. As observed with the Detour Lake sample, it appears as though the residual gold has little association with the pyrite and is likely present as inclusions in the silicates.

Two samples were received from Les Mines Selbaie representing ore from different sections of the mine. The first sample (ID 3.2) was collected from a high sulphide ore zone whereas the second sample (ID 3.3) originated from a low sulphide ore zone. The two samples represent the worst case and best case scenarios for the ore types processed through the concentrator.

For both ore types, a very fine grind is required to achieve liberation. The overall range of particle size, as seen by Figures 4.3 and 4.4, is from 5 - 150 microns with approximately 80 % being finer than 60 microns. The sulphides tend to be concentrated in the fine size fractions, similar to the other samples tested. The 80 % passing size of the sulphides for the Selbaie samples 45 - 48 microns. The major difference between the two samples is the sulphur content. Sample 3.2 is an approximately equal mixture of pyrite and quartzose gangue with a total sulphur content of greater than 20% S(t) whereas sample 3.3 consists predominantly of feldspars and quartz with a small amount of pyrite (3.5% S(t)). Gold and silver are present in both samples in quantities proportional to the sulphide content. Sample 3.2 contained 0.16 gpt Au and 25.7 gpt Ag compared with 0.09 gpt Au and 8.0 gpt Ag contained in Sample 3.3. Both samples contained a minor amount of carbonate (0.5% - 1.0%) which could delay the onset of acid rock drainage.

The common characteristics shared by each of the samples can be summarized as follows:

- acid production potential exceeds neutralization potential and increases with increasing sulphide content.
- The sulphide minerals tend to be concentrated in the fine size fractions (- 74 microns). This may enhance the rate of acid generation because of the high quantity of exposed surface area available for oxidation. The fineness of the sulphides makes separation by gravity techniques an unlikely option.
- The sulphide minerals are well liberated from the gangue minerals making separation by flotation a viable option. With the exception of Sample 3.3, the mass of the sulphide concentrates produced will be low enough to permit separate impoundment of this material with sufficient quantities of non-reactive tailings to act as a cover material.
- Separating the sulphides from Sample 3.3 will result in the production of a pyrite concentrate containing 50% of the mass of the entire tailings stream making separate disposal difficult.

5.2 Gravity Separation

None of the gravity methods tested provided sufficient sulphur removal to produce a non-reactive tailings. The principle of gravity separation is based on differences in movement amongst minerals with different specific gravities when gravitational or mechanical forces are simultaneously applied to them. The difference in movement of the particles is dependent on a large number of factors, including the size, shape and density of the particle. In order to effect a gravity separation, it is necessary to have a specific gravity difference of at least 1.0 and to have a relatively equal size distribution. The applicability of gravity concentration and the approximate size range that can be treated is estimated by Taggart [Taggart,A.F., *Handbook of Mineral Dressing*, John Wiley and Sons, New York, 1945] using the concentration criterion based on equal settling rates of two particles:

$$\text{Concentration criterion} = (S_H - R)/(S_L - R),$$

where R is the specific gravity of the medium and S_H and S_L are the specific gravities of the heavier and lighter minerals respectively. Based on the concentration criterion, the size range of applicability can be summarized as follows:

Criterion	**Size Range of Applicability**
> 2.5	Separation easy down to -200 mesh
2.5 - 1.75	Separation effective to ~ 100 mesh
1.75 - 1.50	Separation possible to 10 mesh, but difficult
1.50 - 1.25	Separation possible to 1/4 ", but difficult
< 1.25	Relative processes not possible

The concentration criterion for the samples tested fall in the range of 1.75 - 2.5 indicating that gravity concentration would only be effective on particles larger than about 100 mesh. The distribution of the sulphides for all of the samples tested are concentrated in the fine sizes (-200 mesh) therefore the separations tend to be more of a size classification than a true gravity separation.

A comparative summary of the results for the various gravity separation tests is shown in Table 5.1.

Table 5.1 - Comparison of Gravity Devices For Recovery of Sulphides

Device		**Placer**	**Lac**	**Selbaie 3.2**	**Selbaie 3.3**
Falcon	Wt. Rec	30.5	35.6	55.3	
	% S Rec.	52.9	70.7	69.4	
	Tails % S(t)	1.6	1.8	23.8	
	% S Rec : Wt. Rec	1.7	2.0	1.3	
Knelson	Wt. Rec.	6.4	7.5	11.8	
	% S Rec.	19.2	21.4	15.6	
	Tails % S(t)	2.1	2.1	19.9	
	% S Rec : Wt. Rec	3.0	2.9	1.3	
Tray	Wt. Rec.	63.6	57.8	51.5	49.9
	% S Rec.	64.8	59.0	51.5	51.4
	Tails % S(t)	1.9	4.2	20.8	2.9
	% S Rec : Wt. Rec	1.0	1.0	1.0	1.0
Spiral	Wt. Rec.	27.9	19.1	26.3	16.1
	% S Rec.	35.5	21.3	26.8	20.8
	Tails % S(t)	1.7	3.8	19.5	3.6
	% S Rec : Wt. Rec	1.3	1.1	1.0	1.3
Table	Wt. Rec.	30.7	25.0	83.6	13.5
	% S Rec.	57.4	38.1	76.7	27.1
	Tails % S(t)	1.2	3.2	17.8	3.2
	% S Rec : Wt. Rec	1.9	1.5	.9	2.0

With any type of concentration process it is desirable to maximize the metal recovery while minimizing the mass recovery. When comparing the ratios of sulphur recovery : wt. recovery for the various separation devices it can be seen that very little upgrading is occurring and that the sulphur recovery is largely a function mass recovery. The Knelson concentrator generally produced the cleanest concentrate however sulphur recovery was very low. The Reichert trays and the spirals were the least effective at recovering and upgrading the sulphides.

The recovery of gold and silver values by the various gravity machines is shown in Table 5.2. The results, in terms of upgrading and recovery are very similar to the results obtained for the sulphides. Recovery of the precious metal values is largely dependant on the overall mass recovery. The selectivity towards the gold and silver is generally poor and may be an indication that these metals are not present as liberated particles.

Table 5.2 - Comparison of Gravity Devices For Recovery of Precious Metals

Device		Placer	Lac	Selbaie 3.2	Selbaie 3.3
Falcon	Conc. Au (gpt)	.44	.23	.18	
	Ag (gpt)				
	Recovery Au	35.8	38.9	61.2	
	Ag				
Knelson	Conc. Au (gpt)	.74	.43	.20	
	Ag (gpt)				
	Recovery Au	13.0	15.5	14.8	
	Ag				
Tray	Conc. Au (gpt)	.31	.23	.18	.13
	Ag (gpt)			23.9	7.5
	Recovery Au	70.3	59.2	50.6	55.5
	Ag			51.4	49.2
Spiral	Conc. Au (gpt)	.39	.30	.21	.30
	Ag (gpt)			23.9	11.3
	Recovery Au	42.1	28.2	31.1	30.9
	Ag			28.0	20.5
Table	Conc Au (gpt)	.39	.41	.12	.26
	Ag (gpt)			14.7	15.2
	Recovery Au	42.8	53.5	80.3	29.0
	Ag			77.2	22.8

5.3 Direct Flotation

The simple mineralogy and the fine size distribution of the sulphides within the samples makes flotation a logical choice as a method for removal of the sulphides. Although the primary objective of the study was to investigate methods of separating sulphides from the tailings for minimizing future environmental risks, the sulphide concentrates were also examined to determine if any economic benefits could be derived by further processing.

Since reagent schemes are generally ore specific, the results of the flotation tests will be discussed on an individual sample basis.

5.3.1 Placer Dome, Detour Lake

The Detour Lake concentrator was originally designed with the intent of producing a by-product copper concentrate. Flotation of the sulphides before cyanidation resulted in high gold losses to the copper concentrate and therefore a decision was made not to operate the flotation section.

During 1993, a test program was conducted at the Placer Dome Metallurgical Research Centre into the depyritization of CIP tailings by flotation. The investigation focused on the flotation of the sulphides and the subsequent production of a saleable copper concentrate. The testwork was successful at reducing sulphur values in the tailings to levels of 0.1 - 0.2 % S but failed to consistently produce a copper concentrate that met the objectives of a minimum 14 % Cu grade.

A series of 6 flotation tests were conducted on the Detour Lake tailings sample. The aim was to reproduce the sulphur rejection results from the Placer Dome testwork and to try to improve the copper concentrate grade. The initial flowsheet consisted of a cyanide destruction step followed by copper rougher and bulk sulphide flotation stages. Cyanide destruction was carried out using air and sodium metabisulphite added at a nominal rate of 4.8 g/g CN^- contained in the CIP tails. The pH during the destruction cycle ranged from 7.8 - 9.3.

The first two tests conducted were scoping tests to determine the levels of reagents required and the overall quality of the various flotation products. Aerophine 3418A was selected as the copper collector and potassium amyl xanthate was used for the bulk flotation stage. MIBC was selected as the frother. It was evident from these initial tests that the main contaminant in the copper rougher concentrate was pyrrhotite rather than pyrite. The flowsheet was modified for test 3 to incorporate a magnetic separation stage prior to the cyanide destruction. From Table 4.31 it can be seen that 52 % of the total sulphur reported to the magnetic concentrate. The distribution of sulphur to the copper rougher concentrate decreased to 6.98

% from 19.4 % for test 2. The grade of the copper rougher increased from 5.1 % Cu for test 2 to 8.3 % for test 3. Although the deportment of copper to the magnetics was almost 9 %, the copper recovery to the rougher concentrate decreased only by 3 percentage points indicating an actual improvement in the rougher stage recovery.

An analysis for cyanide contained in the feed samples indicated a level of only 40 ppm compared to the expected value of 275 - 350 ppm. The cyanide destruction stage was eliminated for test 4 with no adverse effects on the metallurgical performance. A single stage of copper cleaning was included in test 4 resulting in a cleaner concentrate grade of 15.4 % Cu. The residual sulphur content of the tailings was less than 0.1% S(t). In an attempt to improve the iron rejection in the copper rougher, the copper collector was changed from 3418A to Hostaflot LET (diethyl-dithiophoshate) for test 5. This collector was effective at lowering the iron content of the rougher concentrate from 26.7 % Fe to 21.8 % Fe but resulted in a slightly lower copper recovery. Neither the final copper concentrate or the residual level of suphur in the tailings were affected by the change in collector type.

The use of magnetic separation to eliminate the adverse effects of pyrrhotite on copper selectivity proved to be very effective. To minimize the cost of implementation on an industrial scale, test 6 investigated the possibility of applying the magnetic separation to the rougher concentrate rather than the flotation feed. This method proved effective at achieving the target copper concentrate grade of 14 % Cu and resulted in a slightly higher overall copper recovery (64.13 %). The residual sulphur level of the final tailings was higher (0.59 % S(t)) than in previous tests but it is believed that with a slight adjustment to the reagents a level of less than 0.2 % S(t) is achievable.

Based on the results of test 6, the conceptual flowsheet shown in Figure 5.1 was developed. The flowsheet makes use of equipment which already exists at the mine site. A small wet drum magnetic separator would be added to remove pyrrhotite from the copper rougher concentrate prior to cleaning. The magnetics would be combined with the bulk sulphide concentrate for final disposal.

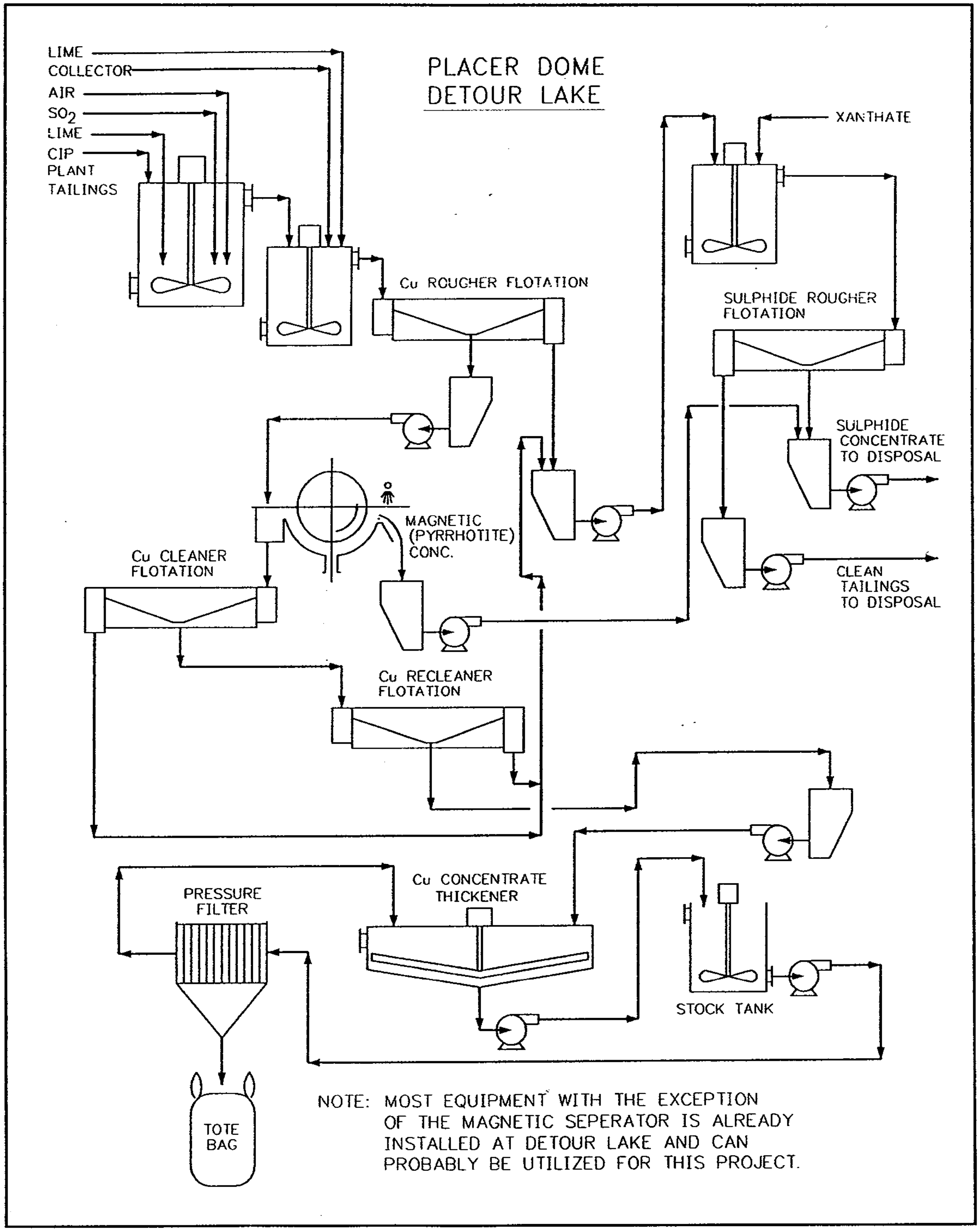
Table 5.3 provides an estimate of the mass and metallurgical balance for a nominal feed rate of 125 tph based on producing a low grade concentrate (15 % Cu). By utilizing a second stage of cleaning, as shown on the flowsheet, it may be possible to achieve a final copper grade of 18 % - 20 % Cu.

Table 5.3 - Predicted Mass and Metallurgical Balance for Detour Lake

			Assay			Dist.		
Stream	**TPH**	**Wt %**	**Au (gpt)**	**Cu %**	**S(t) %**	**% Au**	**% Cu**	**% S(t)**
CIP Tail	125.0	100.0	0.36	0.15	2.31	100.0	100.0	100.0
Cu Cleaner Tail	1.0	0.8	0.86	2.39	3.60	1.9	12.7	1.2
Cu Rougher Feed*	126.0	100.8	0.37	0.17	2.32	101.9	112.7	101.2
Cu Rougher Conc	2.4	1.9	1.56	7.19	14.17	8.4	93.2	11.9
Magnetics	0.5	0.4	1.24	2.12	32.07	1.4	5.9	5.8
Mag Sep Tail	1.9	1.5	1.64	8.58	9.26	6.9	87.3	6.1
Cu Conc	0.9	0.7	2.50	15.4	15.5	5.0	74.6	4.9
Cu Cleaner Tail	1.0	0.8	0.86	2.39	3.60	1.9	12.7	1.2
Cu Rougher Tail	123.6	98.9	0.34	0.03	2.09	93.5	19.6	89.3
Sulphide Conc	6.1	4.9	1.09	0.41	38.36	14.7	13.3	81.2
Final Tail	117.5	94.0	0.3	0.01	0.20	78.8	6.3	8.2
Combined sulphide conc and magnetics	6.6	5.3	1.09	0.54	37.84	16.1	19.2	87.0

**Cu rougher feed is a percent of CIP tails and is greater than 100% due to system recycle.*

Figure 5.1 - Conceptual Flowsheet For Treatment of Detour Lake Tailings



NOTE: MOST EQUIPMENT WITH THE EXCEPTION OF THE MAGNETIC SEPERATOR IS ALREADY INSTALLED AT DETOUR LAKE AND CAN PROBABLY BE UTILIZED FOR THIS PROJECT.

5.3.2 Lac Minerals, La Mine Doyon

The tailings sample from La Mine Doyon contained elevated levels of cyanide (250 - 300 ppm) and lime. Pulp pH values ranged from 10.8 - 11.2. A series of 6 flotation tests were conducted to evaluate a variety of reagent combinations. Cyanide destruction was conducted on all but one test (test 4) and was found to be an essential part of the process. The cyanide destruction procedure was the same as described for the Detour Lake samples.

The presence on high levels of calcium in the pulp make flotation of the pyrite difficult using xanthates alone. During the cyanide destruction step the pH is decreased to approximately 8.5 but increases as the sodium metabisulphite is consumed. The pH at end of the destruction stage was typically 9.2 - 9.5. While it normally is possible to float pyrite at this pH range, the presence of various calcium complexes hinder the adsorption of collector on the surface of the pyrite. Several reagent combinations were tested based on recommendations from the various suppliers. These included:

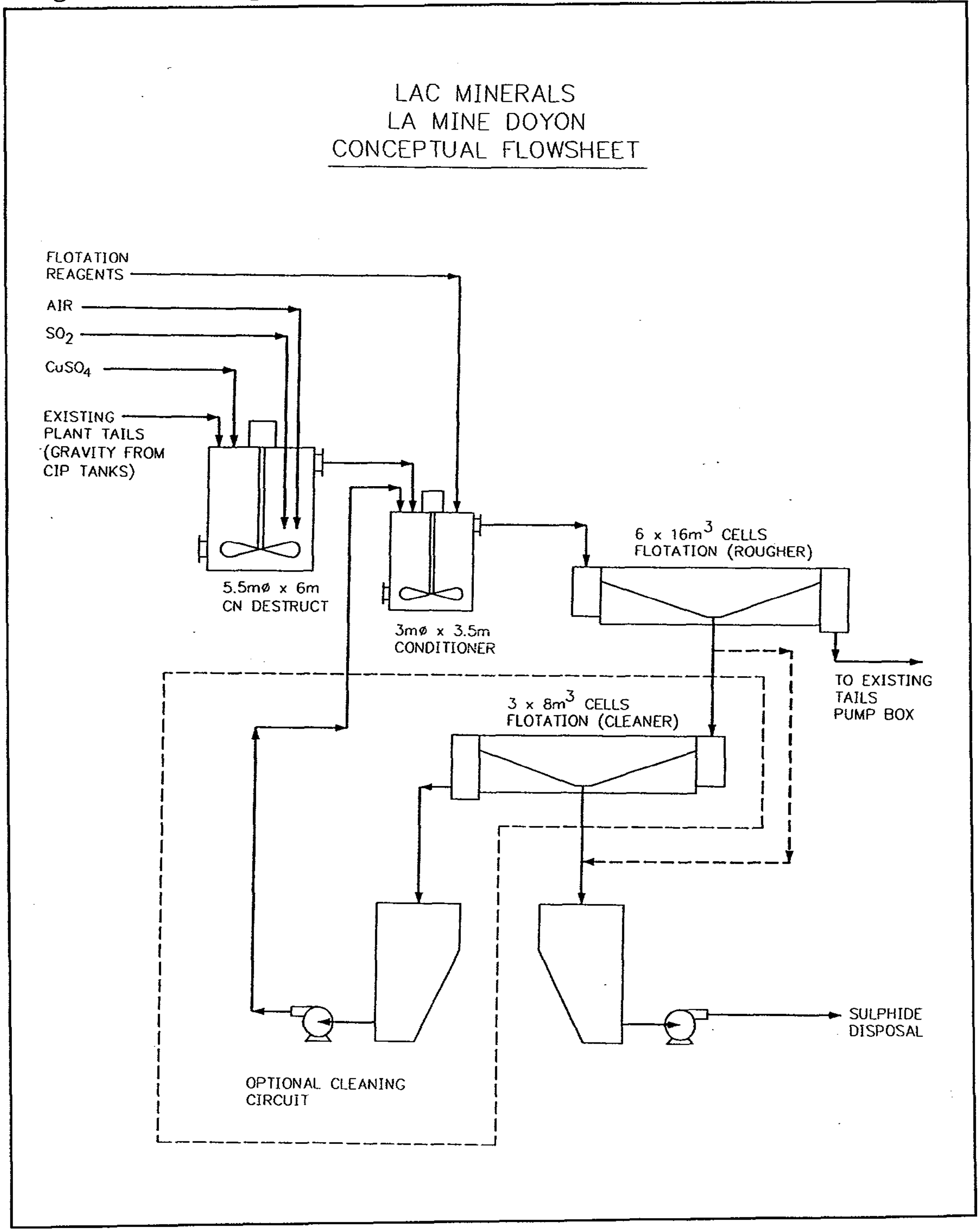
- Potassium amyl xanthate alone and in combination with Aero Promotor 404 (mixture of sodium mercaptobenzothiozal and dithiophosphate).
- Aero promoter 404 and Cyanamid S-5100 (thionocarbamate)
- Texaco SP-160 (zinc dithiophospate)

The metallurgical results for the six flotation tests are summarized in Table 4.32. The reagent combinations tested are summarized in section 4.7. Tests 1, 2, 3 and 5 produced tailings with a residual sulphur content ranging from 0.15 - 0.35 % S(t) corresponding to sulphur rejection rates of 93 % - 98 %. The associated mass recovery of the flotation concentrates ranged from 14 % - 16 %. Recovery of gold and tellurium was less than 30 % for all tests indicating very minor association with the pyrite. Test 4, conducted without cyanide destruction and using a non-selective collector (Texeco SP-160) did not produce acceptable results. Test 5 was a repeat of test 4 but included cyanide destruction. The SP-160 failed to activate the pyrite and it was necessary to add supplemental reagents (404/PAX) before flotation took place. Test 6 was conducted to evaluate potassium amyl xanthate as a collector without any additional promoters. Even at a dosage rate of 170 gpt the xanthate used alone did not achieve the same residual sulphur level as previous tests using a combination of collectors.

It is clear from the results of these tests that the pyrite can be successfully removed by flotation using a number of different commercially available collectors. Further testing would be required to optimize both the collector types and dosage rates.

A conceptual flowsheet for treating La Mine Doyon tailings is presented in Figure 5.2. The existing CIP tails would feed an agitated tank of 140 m^3 capacity. Air, sulphur dioxide and a small quantity of copper sulphate are added to tank to reduce residual cyanide from a level of 250 - 300 ppm to less than 1 ppm. The nominal residence time of the pulp in the cyanide destruction tank would be 30 minutes. The overflow from the cyanide destruction tank would feed a 3m diameter x 3.5m high agitated tank where flotation reagents would be added and conditioned for approximately 5 minutes. The overflow of the conditioner would feed a bank of 6 x 16 m^3 flotation cells for removal of the sulphides. A provision has been made for a single stage of cleaning of the sulphide concentrate in a bank of 3 x $8m^3$ flotation cells. This circuit is optional and would only be necessary if there was a need to reduce the mass of sulphide concentrate below the forecasted level of 15 wt.%. Utilizing the cleaner could reduce the mass recovery to about 10 % by weight. The final tailings and the sulphide concentrates would be pumped to the appropriate disposal sites.

Figure 5.2 - Conceptual Flowsheet For Treatment of La Mine Doyon Tailings



5.3.3 Les Mines Selbaie, Sample 3.2

This sample was more difficult to treat than any of the other samples. It is believed that due to the high sulphide content and the very fine particle size, significant oxidation of the mineral surfaces had occurred prior to testing. A series of 9 tests using various collector combinations and pre-conditioning steps were conducted. The metallurgical results for each test are reported in Table 4.33. Table 5.4 provides a brief summary of the operating conditions and selected results.

Table 5.4 - Flotation Conditions

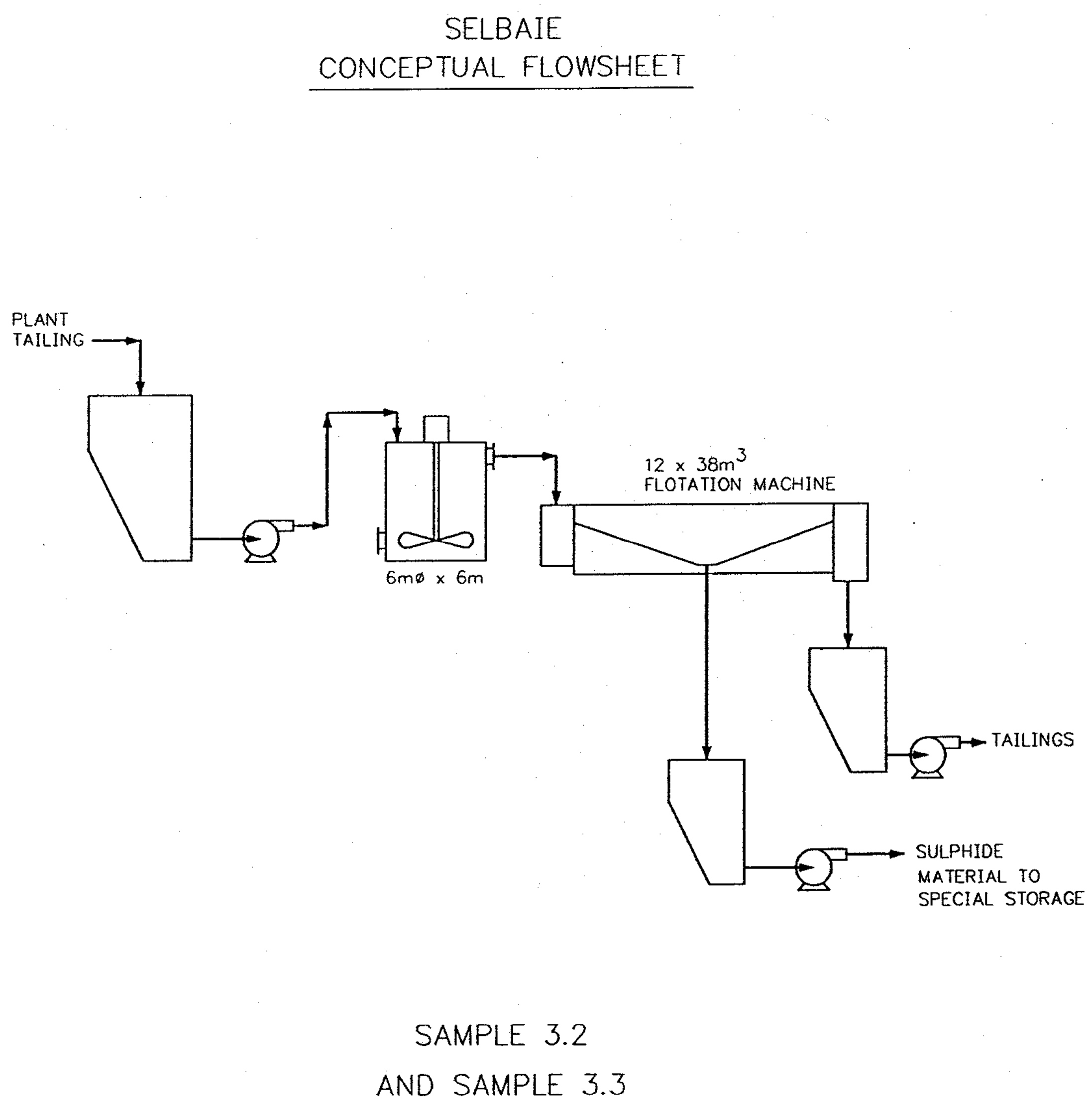
Test	Reagents	pH	Flot. Time	Wt. Rec	Tail % S(t)	Au Rec	Ag Rec
1	$CuSO_4$ 15 gpt PAX 55 gpt	9.3	15	10.8	21.1	24.9	32.6
2	PAX 30 gpt M-91 365 gpt	5.3	17	46.5	0.70	83.2	93.1
3	M-91 375 gpt	5.2	12	48.6	0.76	77.9	94.1
4	404 75 gpt PAX 60 gpt	5.4	12	50.4	1.18	71.5	89.9
5	PAX 120 gpt M-91 75 gpt	6.0	13	48.8	.97	65.5	89.9
6	SP-160 125 gpt 404 150 gpt	8.6	18	52.4	.73	81.0	90.1
7	Na_2SO_3 1000 gpt PAX 192 gpt 404 163 gpt	9.3	16	50.4	1.43	75.2	89.6
8	SP-160 168 gpt	8.7	16	46.4	1.55	60.3	91.8
9	SP-160 168 gpt aeration	9.3	15	44.7	1.19	62.9	91.5

The first test conducted using $CuSO_4$ and potassium amyl xanthate (PAX) resulted in very poor recovery of pyrite. Sulphuric acid was added to acidify the pulp for tests 2 - 5 in an attempt to clean the mineral surfaces and improve the flotation response. Several reagent combinations were tested and proved to be effective at recovering the pyrite. Sulphur recovery to the flotation concentrate ranged from 97% - 98% resulting in a tailings grade of 0.7 % S(t) - 1.18 % S(t).

Tests 6 - 9 were conducted without pH adjustment using less selective collectors. The combination of SP-160 and 404 used in test 6 produced acceptable results (0.73 % S(t) in tails). The pulp was conditioned with sodium sulphite during test 7 in an attempt to sulphidize the oxidized sulphide minerals. A combination of PAX and 404 were used as collectors but even at high dosage rates failed to achieve a final sulphur level of less than 1 %. Tests 8 and 9 were conducted using SP-160 a zinc dithiophosphate. This collector classified as a general purpose, unselective collector for sulphide minerals, did not achieve as good a sulphur rejection as some of the mixtures producing tailings of 1.55 % S(t) and 1.19 % S(t).

The reagent consumption for all of the tests on this sample were considerably higher than would normally be expected. It is possible that sample aging and oxidation of the sulphides contributed to the generally sluggish flotation response and that better results could be achieved by treating fresh tailings. Figure 5.3 shows a conceptual flowsheet for the treatment of the Selbaie tailings. Existing tailings would be pumped to a conditioning tank followed by single stage of flotation. The flotation time retention time has been based on the results of the testwork and may therefore be in excess of what may actually be required. Additional tests using fresh tailings would be required to confirm both reagent consumption and residence time.

Figure 5.3 - Conceptual Flowsheet For Treatment of Selbaie Tailings



5.3.4 Les Mines Selbaie, Sample 3.3

The low sulphur tailings from Selbaie were relatively easy to treat. A total of 5 tests were conducted using several reagent combinations. Table 5.5 summarizes the main flotation conditions. Analytical results and metal distributions are tabulated by test in Table 4.34.

Table 5.5 - Flotation Conditions

Test	Reagents	pH	Flot. Time	Wt. Rec	Tail % S(t)	Au Rec	Ag Rec
1	404 43 gpt PAX 22 gpt	9.3	6	10.6	0.49	33.4	68.1
2	PAX 30 gpt M-91 22 gpt	9.3	5	15.6	0.32	38.4	73.3
3	SP-160 74 gpt	9.3	5	19.3	0.41	32.6	98.4
4	SP-160 38 gpt	9.5	5	11.1	0.71	22.5	82.3
5	PAX 75 gpt	9.3	5	11.2	0.39	23.7	90.1

Each of the reagent schemes tested produced a tailing containing less than 1 % S(t) without any need for pH modification or special pretreatment steps. Reagent consumption was considerably lower than for Sample 3.2 as was the flotation time. The differences in gold and silver recoveries are likely a result of analytical errors rather than reagent effects due to the very low levels of these metals in the feed.

The flowsheet developed for Sample 3.2 would be suitable for treating this ore as well.

5.6 Acid Generating Capability

Static Test Results - Acid Base Accounting

The acid base accounting data on the head samples and on selected products was evaluated and graphed and is provided in Figures 5.4 to 5.7 for each of the samples tested. Although the neutralization potential (NP), acid potential (AP) and net neutralization potential (Net NP) values of a sample are important, in this analysis, more emphasis will be placed on the NP/AP ratio as an indicator of acid producing potential. A NP/AP ratio of 1 or less indicates a strong acid generating potential. A ratio $>1<3$ usually gives rise to some uncertainty in prediction, although evaluation is very site-specific. A ratio >3 is often used to indicate that a sample has sufficient neutralization capability to prevent acid generation but, again, site-specific conditions have to be considered.

Placer Dome Detour Lake Sample

Figure 5.4(a-e) provides a summary of the ABA data for this sample. The head sample has a moderately strong potential for acid production with a Net NP value of -41 and a NP/AP ratio of 0.4. The five separation processes selected indicate good weight recovery to the tails and significant reduction on sulphur content, although only two tests, flotation tests 3 and 4, provide products with positive Net NP values and NP/AP ratios greater than 3. NP values were not effected significantly in the tests. Certainly, the product from flotation test 4 can be considered to be very satisfactory for confident disposal. Flotation test 3 also produced a good product.

It should be noted that according to the CO_2 analysis of this sample, the calculated NP value representing the carbonate content is only approximately half of the NP determined by the acid base accounting test (see Table 4.36). The implication is that the available protective alkalinity of the sample may be lower than indicated, so that the calculated ratios of NP/AP would be correspondingly lower. This would be particularly significant for the product from flotation test 3. For the other three head samples, a high proportion of the NP determined by acid base accounting can be shown to be due to carbonate on the basis of the CO_2 analyses.

Lac Minerals - La Mine Doyon Sample

The ABA data for this sample are summarized in figure 5.5 (a-e). The head sample has a strong potential for acid generation with a Net NP value of -102 and a NP/AP ratio of 0.2. As with the previous sample, only flotation produced samples with positive NP values, albeit low values. For flotation test 1, a high NP/AP ratio of over 5 is satisfactory. In flotation test 2, the ratio is less than 3 although the sulphur content is low at 0.34%. This sample would need further kinetic testing to fully evaluate its AMD potential.

In all tests the NP value was not affected to any great degree.

Selbaie Sample 3.2

Figure 5.6 (a-e) summarizes the Selbaie sample 3.2 acid base accounting data. The very high sulphur content of this sample indicates a very strong AMD potential, although the significant NP value (47 kg/tonne) would suggest that acid production would be delayed. As before, only the float tests produced products with positive NP values. Only flotation test 2 produced a product with a NP/AP ratio greater than 3.

Due to the significant sulphur removal during processing, the lower weight recovery to the tailings product resulted in a significant increase in the NP values. The corollary of this, however, is that the very high sulphur contents of the concentrates produced during processing and their associated relatively high weight distribution would likely pose a disposal problem with respect to ARD at least equal to the disposal of the original head material.

Selbaie Sample 3.3

Acid base accounting data for this sample are summarized in figure 5.7 (a-e). This sample has a low NP value (15 kg/t) and despite good sulphur rejection in flotation, Net NP values and NP/AP ratios for the products are not satisfactory. Acid generation can be predicted. Confirmatory kinetic testing for the product from flotation test 2 might be advisable.

Metals Distribution in Heads and Products

From tables 4.37 to 4.40 it is apparent that for significant elements such as Cd, Cu, and Zn, significant reductions in their distribution in the products occurred, particularly in the flotation tests. These reductions will greatly reduce the contamination of any drainage from the waste due to the oxidation of their corresponding sulphides.

Kinetic Test Results - Humidity Cells

Kinetic tests performed on the four head samples have been running for 19 weeks with the exception of the Selbaie sample 3.3 which has been running 15 weeks due to its late arrival. In most cases, this is too short a time to evaluate the long term weathering potential of a sample. Analysis of leachates in these early weeks usually provide an indication of the readily mobile elements and the very early start of acid producing reactions. However, the tests do provide some indication of the behaviour of the materials. However, together with the analysis of the acid base accounting data presented above, certain observations can be made.

Placer Dome - Detour Lake Sample

The humidity cell data for the Detour Lake sample were provided in table 4.41 and are summarized for review in figure 5.8 (a-b). The humidity cell data for this sample indicates high pH, significant alkalinity, moderate specific conductance values, with low sulphate and acidity values. It is therefore indicated that carbonate dissolution is taking place and there is no evidence of the initiation of acid generation. The calcium analyses on the leachates support this observation. No significant metal mobility is indicated

Lac Minerals - La Mine Doyon Sample

The humidity cell data for the La Mine Doyon sample were provided in table 4.42 and are summarized for review in figure 5.9 (a-b). As with the previous sample, the results indicate a carbonate dissolution condition. Despite the strong acid producing potential of this sample, there is sufficient alkalinity to protect drainage for some considerable time. An initial flush of copper (43 mg/L) is indicated, but following this, there is no significant metal release.

Selbaie 3.2 Sample

The humidity cell data for the Les Mines Selbaie sample 3.2 were provided in table 4.43 and are summarized for review in figure 5.10 (a-c). An apparent lowering of pH, high specific conductance, low alkalinity, and rising acidity and sulphate indicates the early onset of AMD concomitant with the acid base accounting data and high sulphur content. The delay in AMD generation suggested by the NP content is not apparent although it is very possible that the leachate quality being seen is simply the result of the flushing of oxidation products present at the start of the test. Higher magnesium and calcium values indicate the presence of dolomite as the predominant neutralizing species. Very significant zinc values are evident in the drainage. Short term leach tests (the B.C. SWEP test or the E.P.A. 1312 test for example) could be conducted on the flotation products to indicate if metal mobility would be problem with the de-sulphurized material.

Metals concentrations are shown in figure 5.10(c) as they indicate an increase in concentration with time as more and more metals are mobilized due to the early formation of AMD. Metals concentration graphs are not provided for the other samples because there was no significant trend in the data.

Figure 5.4(a-e) - ABA Results for Detour Lake

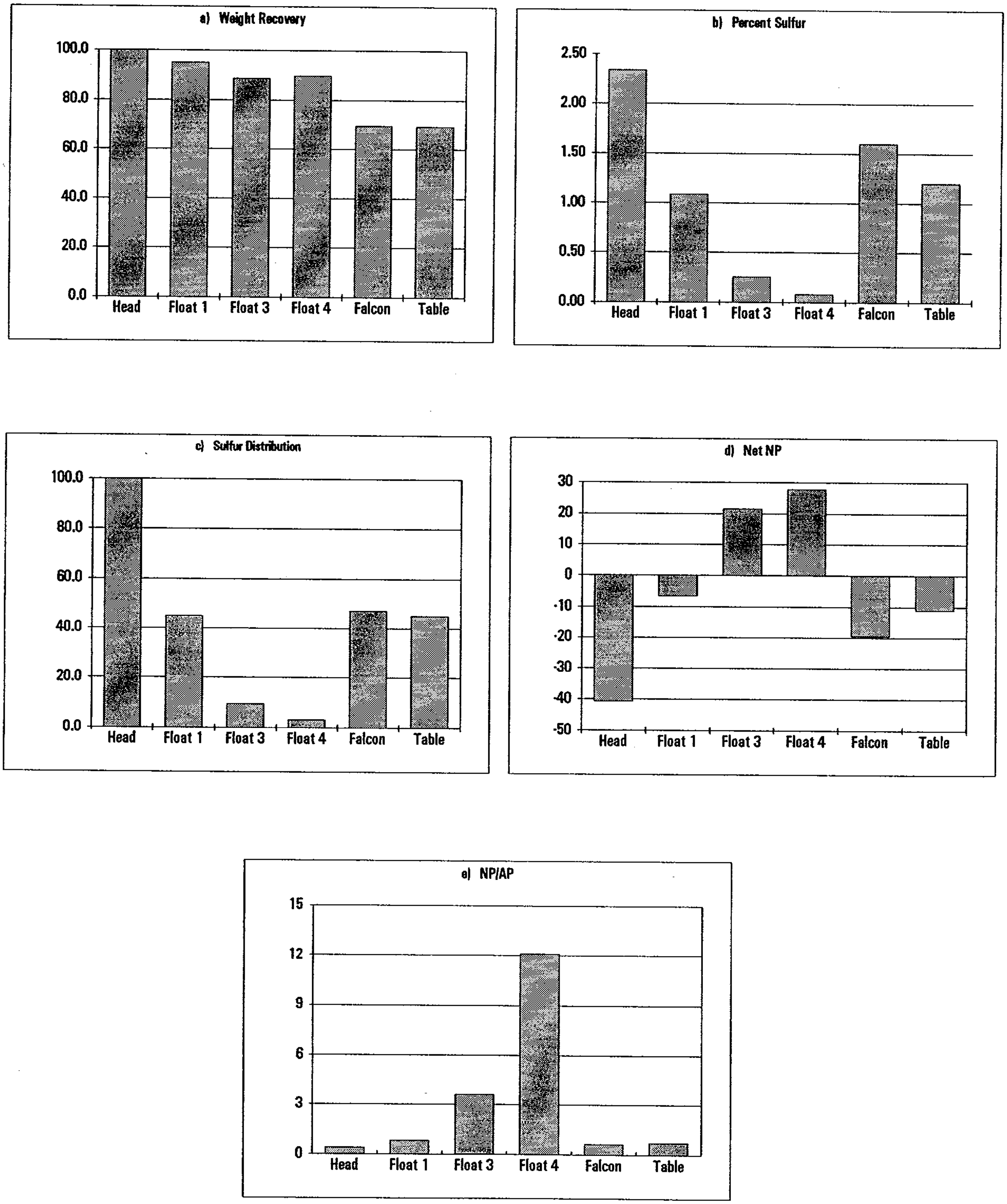


Figure 5.5(a-e) - ABA Results for La Mine Doyon

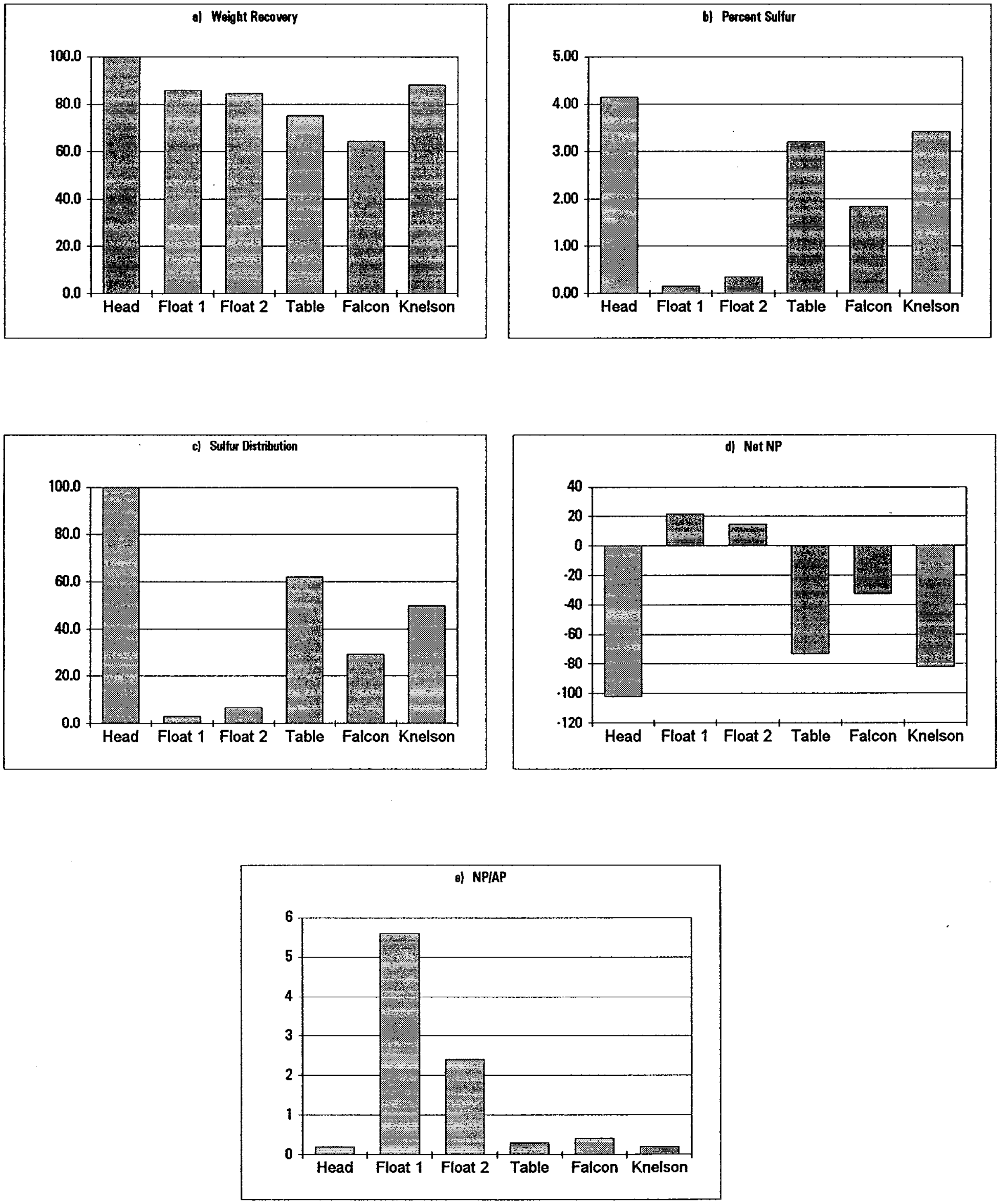


Figure 5.6(a-e) - ABA Results for Selbaie Sample 3.2

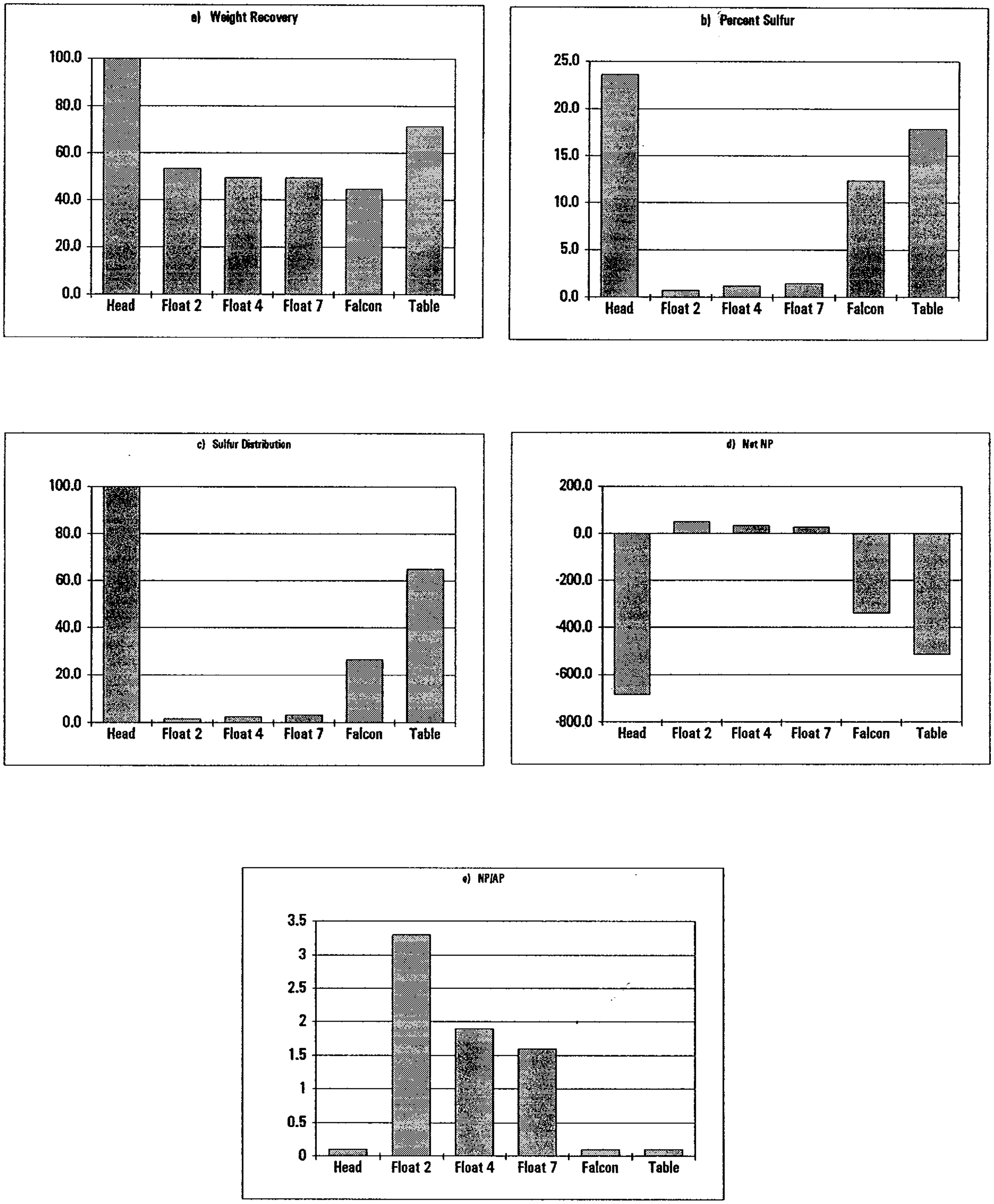
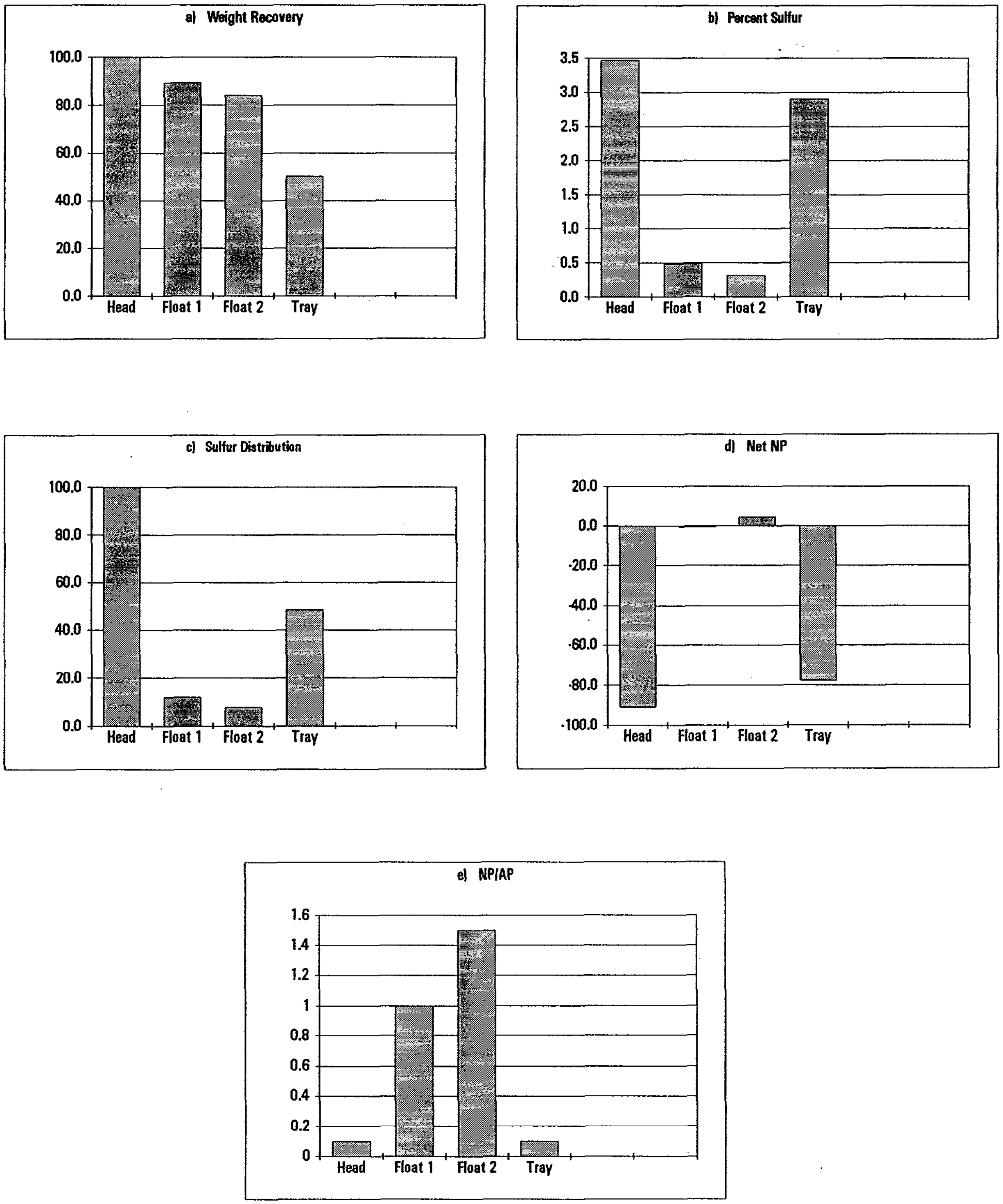


Figure 5.7(a-e) - ABA Results for Selbaie Sample 3.3



Selbaie 3.3 Sample

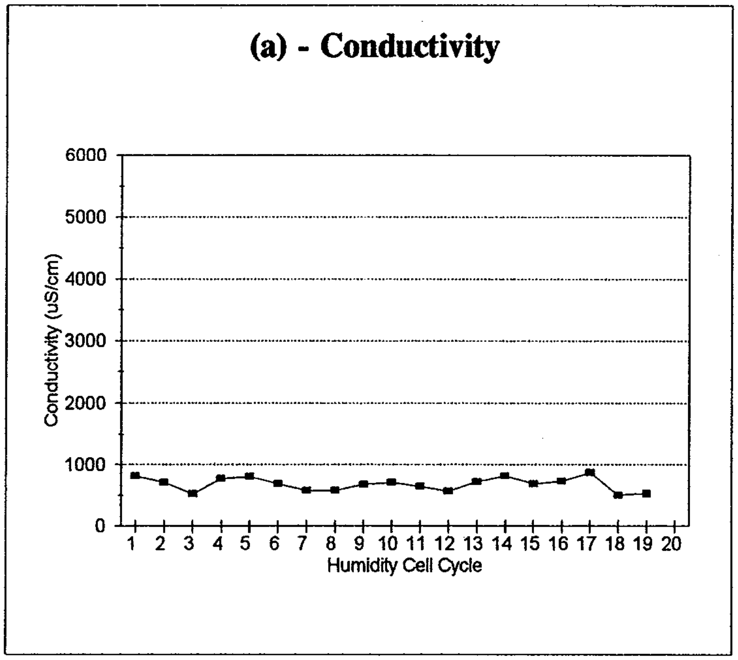
The humidity cell data for the Les Mines Selbaie sample 3.3 were provided in table 4.44 and are summarized for review in figure 5.8 (a-b). Results for the 14 cycles provided are somewhat erratic but are indicative of carbonate dissolution in the absence of sulphide oxidation. Alkalinity from calcium carbonate dissolution is evident giving rise to some moderate specific conductivity values. There is some zinc mobility (up to 11 mg/L). All other metals are in low concentration.

Conclusions

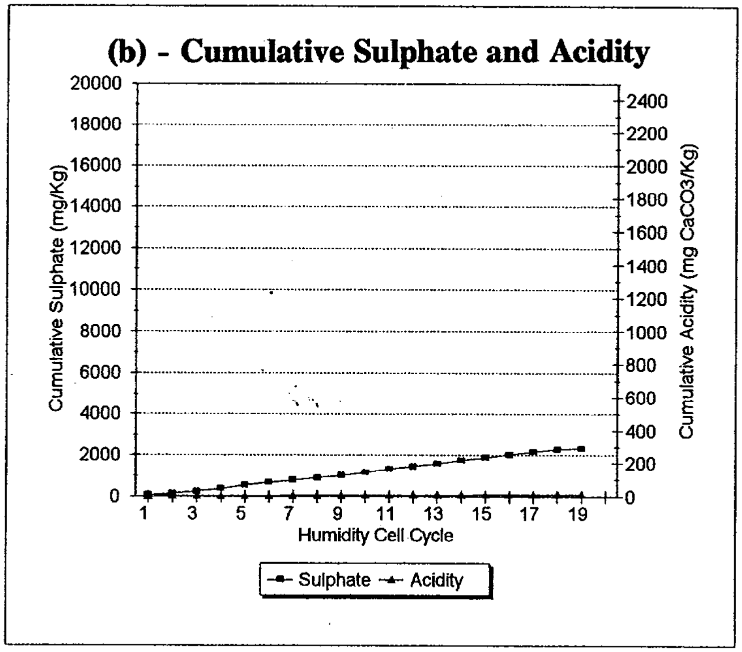
The results of the processing techniques, particularly flotation, carried out on the four potentially acid producing tailings have indicated a good possibility of producing materials with high weight recovery and low acid producing potential. For products having a NP/AP ratio less than 3, kinetic tests should be carried out to provide a proper evaluation of the long term weathering behaviour.

The kinetic tests carried out on the head samples have been operates for too short a time to make any long term conclusions concerning AMD, water quality, or the relative reactivities of the samples. Similarly, the behaviour of the materials in the tests might not relate in any significant way to the behaviour of products derived from the head samples particularly when there has been a very significant reduction in the sulphide content. After 14 to 19 weeks of testing, only the Selbaie 3.2 sample indicates the threat that this type of high-sulphur material poses to water quality and the considerable benefit that would accrue if the sulphur content could be reduced by removing it into a low-weight material for which disposal could be controlled in a satisfactory manner. The three other samples are in a carbonate-dissolution stage and acid generation might not show up for several months or longer.

Figure 5.8(a-b) - Humidity Cell Data
Placer Dome, Detour Lake



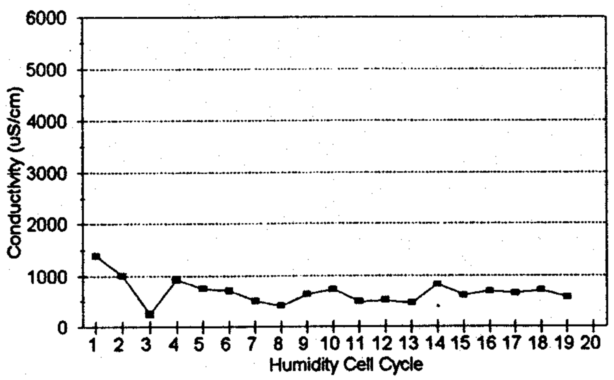
(a) - Conductivity



(b) - Cumulative Sulphate and Acidity

Figure 5.9(a-b) - Humidity Cell Data
Lac Minerals, La Mine Doyon

(a) - Conductivity



(b) - Cumulative Sulphate and Acidity

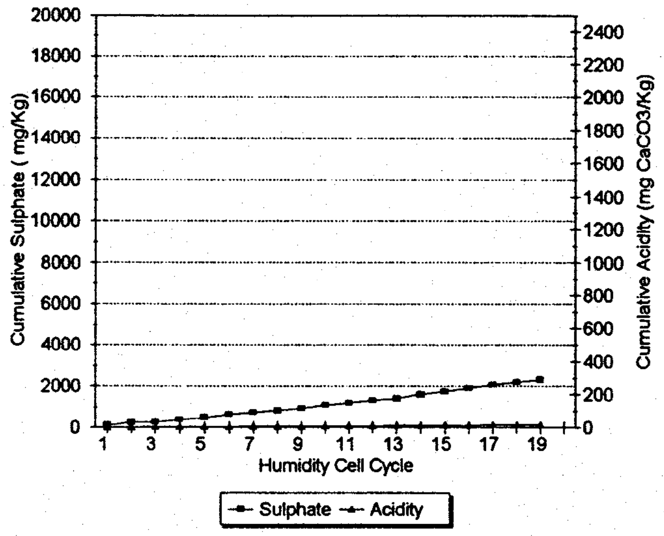
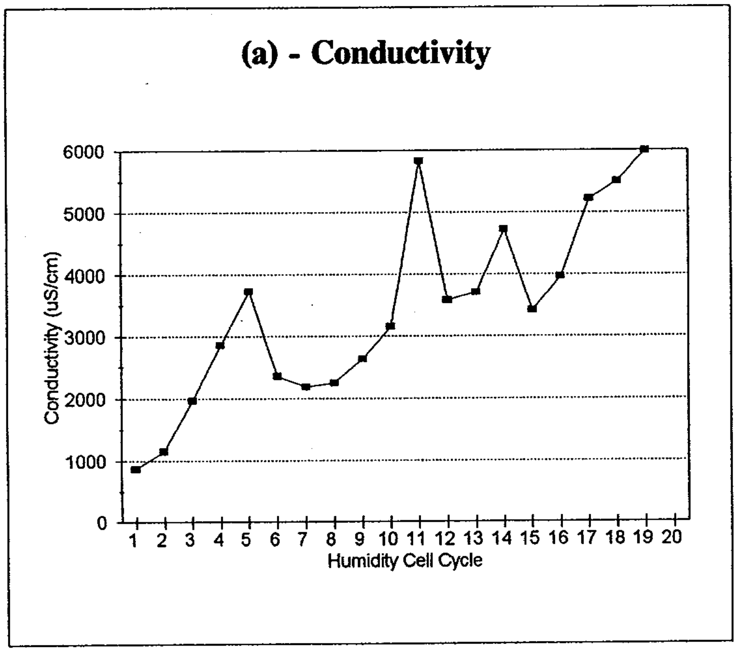
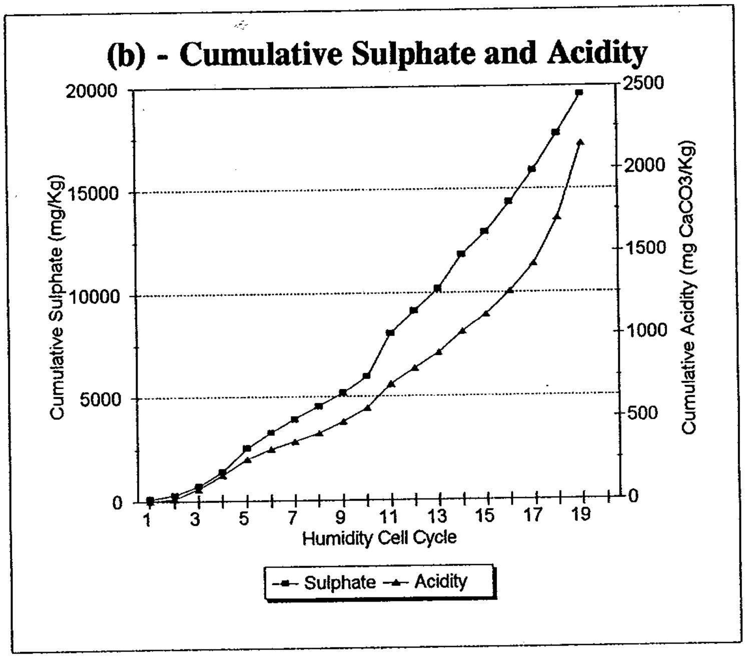


Figure 5.10(a-c) - Humidity Cell Data
Les Mines Selbaie, Sample 3.2





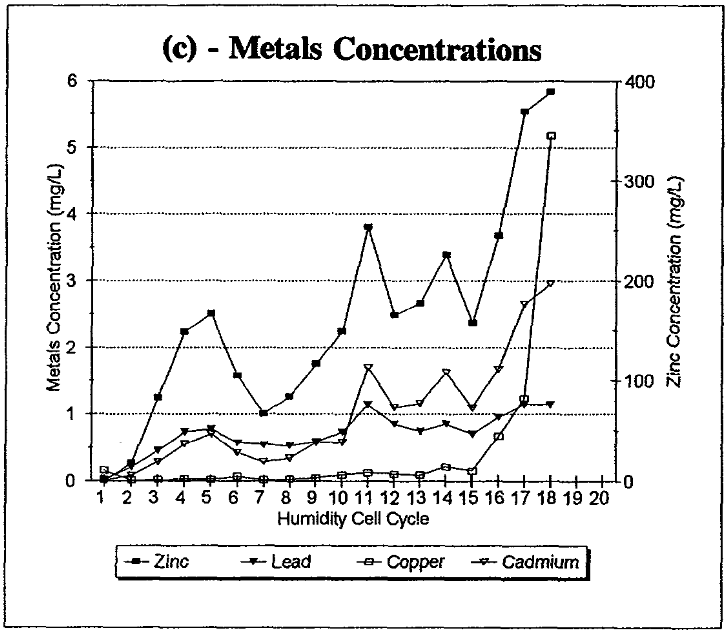
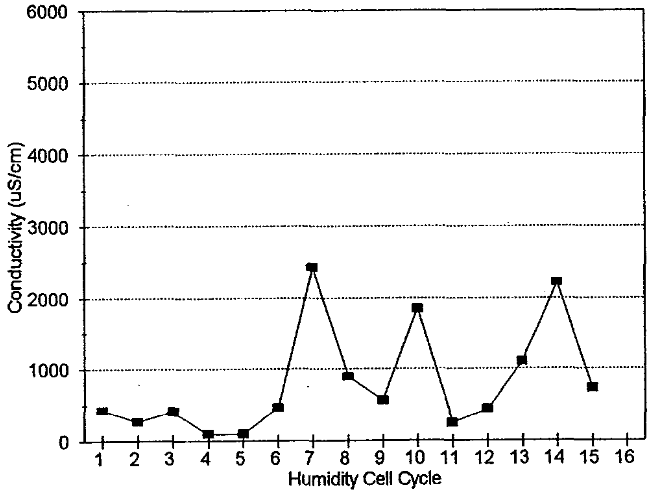
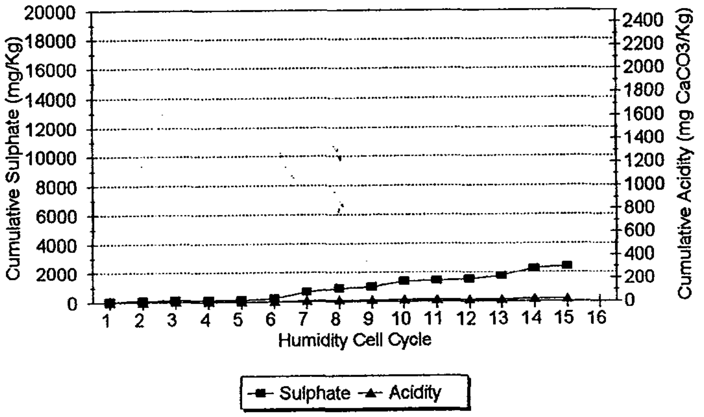

(c) - Metals Concentrations
Metals Concentration (mg/L)
0
1
2
3
4
5
6
Zinc Concentration (mg/L)
0
100
200
300
400
1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18 19 20
Humidity Cell Cycle
Zinc
Lead
Copper
Cadmium

Figure 5.11(a-b) - Humidity Cell Data
Les Mines Selbaie, Sample 3.3

(a) - Conductivity



(b) - Cumulative Sulphate and Acidity



6.0 COSTS OF IMPLEMENTATION AND OPERATION

The conceptual flowsheets in Figures 5.1 - 5.3 illustrate the general form of the proposed removal method for sulphides from the various samples. In all cases froth flotation has given good results. The estimated capital costs are based on the selection of standard commercial mechanical equipment. The main ancillary equipment such as conditioning tanks and pumps have also been costed, with allowances made for buildings, structural steel, piping, electrical, and construction and installation costs. These figures are based on recent quotes and estimates for installation of similar equipment, however, the estimate should be viewed as order of magnitude,(±30%) only as insufficient specific engineering has been done to guarantee a greater accuracy. Assumptions of equipment availability at the individual properties, and the general assumptions on the availability of services are made in each section.

6.1 Placer Dome Inc., Detour Lake Mine.

The Detour Lake Mine at one time produced a flotation copper concentrate for sale. It is understood that a flotation plant is available at Detour Lake and is of sufficient capacity to produce both the proposed saleable copper concentrate (+14% Cu) and also the pyritic concentrate from tailings for special disposal. A Larox filter is available to filter the copper concentrate. Detour Lake have plans to install and operate a cyanide destruction system for other environmental reasons. Whilst this plant may be essential to operate the currently proposed facility on CIP tailings, costs related to the installation and operation of this plant have not been included in this estimate. The grade of the copper concentrate is enhanced by the magnetic removal of pyrrhotite therefore a wet drum magnetic separator has been included in the flowsheet.

It thus appears that the only capital expenditure required at Detour Lake is the purchase and installation of magnetic separator and associated equipment for the treatment of copper concentrate. An allowance for some repiping and refurbishment of the existing flotation plant equipment has been included. Total installed costs for the system are estimated as $130 000. The cost breakdown is as follows:

• Purchase price of magnetic separators	$70 000
• Purchase of additional mechanical equipment	$15 000
• Purchase of structural and electrical components	$10 000
• Installation of magnetic separator system	$25 000
• Miscellaneous piping modifications & refurbishment	$10 000

Operating costs would be around $0.50/ tonne based upon reagent usage from the bench scale tests (about $0.25) plus an allowance for power and maintenance. It should be possible to run the proposed circuit as part of the Detour Lake operations without changing staffing levels.

6.2 Lac Minerals, La Mine Doyon

The object at La Mine Doyon is to remove sulphides from tailings by flotation. The flowsheet in Figure 5.2 shows cyanide destruction ahead of flotation (which is essential) and the cleaning of the rougher concentrate which is optional. The size of the existing building is believed to be adequate to house the new flotation circuit. It has been assumed that power, air and water are available in the amounts required without major expenditure. The minimum facility would consist of cyanide destruction, rougher flotation, and reagent mixing only accommodated within the existing plant and would have an installed cost of approximately $575 000 broken down as follows:

• Cyanide destruction tank and agitator	$100 000
• Rougher flotation equipment	$270 000
• Reagent mixing and handling (including SO_2)	$50 000
• Miscellaneous Mechanical (Pumps, conditioners)	$25 000
• Structural and Electrical	$50 000
• Installation	$100 000

Additional costs that may be worth considering are as follows:

• Cleaner flotation circuit as shown on F/S	(Installed)	$150 000
• Building to house entire facility (lean-to)		$75 000

Operating costs at La Mine Doyon are projected as $1.35/tonne based on laboratory flotation reagent consumptions ($0.50) and additional cyanide destruction reagents ($0.55) and an allowance for power and maintenance. No operations staff increase is expected.

6.3 Les Mines Selbaie.

Two samples were tested from Les Mines Selbaie. Sample 3.3 was easily treated by flotation and could be handled within the facility designed for sample 3.2. The main difference between the two samples was the speed and ease of flotation, with sample 3.3 requiring a quarter of the circuit required for sample 3.2. Thus the capital cost of the system below is based on the worst case requirement of sample 3.2. Operating costs are variable between the extremes of samples 3.2 and 3.3 depending on iron and sulphur content.

The capital cost of a flotation circuit capable of handling material similar to sample 3.2 is estimated at $1 275 000 broken down as follows:

• Conditioner Tank ahead of flotation	$50 000
• Rougher Flotation Equipment	$800 000
• Misc mechanical items (pumps, pump box, piping, etc.)	$100 000
• Electrical and structural materials	$75 000
• Installation	$150 000
• Building to house facilities (lean-to)	$100 000

Operating costs vary from a high of around $1.25 to $0.50/ tonne depending on the sulphide content. Although a number of reagents show promise optimization of the reagent schedule is required. The above higher cost is broadly based on test FS3.2-6., however it should be stressed that more work is required in establishing the optimum reagent schedule.

7.0 CONCLUSIONS AND RECOMMENDATION FOR FURTHER STUDY

Four(4) samples of tailings from three(3) operating mines were received and characterized to determine their acid generating potential. The samples were processed using standard separation techniques to remove the sulphide minerals, and then, were reanalysed to determine the effectiveness of the separation. Conceptual flowsheets were developed for each sample and the cost of implementation was estimated.

7.1 Conclusions

Based on the findings of the study, the following conclusions were reached:

- Each of the samples exhibited acid generating potential ranging in magnitude from moderate to very strong depending on the initial sulphur content of the sample.
- The sulphide minerals were well liberated from the gangue but were found to be concentrated in the minus 74 micron size fraction making gravity separation techniques ineffective.
- Direct flotation was an effective method for decreasing the sulphur content of the tailings to very low levels.
- The maximum level of sulphur that the tailings may contain before becoming a potential acid generator is site specific and depends on the neutralization potential of the ore. The neutralization potential was not affected by the flotation process.

7.2 Recommendations for Further Study

The use of the flotation process to remove sulphides has indicated a good potential for producing tailings with low acid producing potential. Further kinetic testing should be carried out to determine the long term effects of weathering.

The optimization of flotation reagents and conditions was beyond the scope of this study. Further testing is recommended to ensure that a reagent scheme is selected that will be effective for all ore types treated and for the range of process variations likely to be encountered during normal operation.

The disposal and long term storage of the sulphide concentrate will require careful consideration. For samples with a low initial sulphur content the quantity of sulphide concentrate represents a small portion of the total tailings stream. In the case of the Selbaie high sulphur tails, the sulphide concentrate represents approximately 50% of the mass of tailings and, in this concentrated form, could present an even greater environmental threat.

Certain manufacturers of gravity separation equipment claim a high potential for recovery in the 45 to 15 μm size range using centrifugal forces in the 9-10G range. This area may be worthy of further investigation if a fundamental problem exists with the use of flotation.

APPENDIX A

FLOTATION TEST DATA

FLOTATION TEST REPORT

TEST ID: FS1-1 **SAMPLE:** PLACER DOME **DATE:** DEC 7 / 93

PUPOSE: PRELIMINARY TEST ON SAMPLE. CN DESTRUCTION PRIOR TO FLOTATION

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	CU %	FE %	S(T) %	AU %	CU %	FE %	S(T) %
Copper Rougher	42.50	3.58	1.21	3.35	39.70	26.00	12.98	79.14	14.81	40.37
Bulk sulphide float	18.70	1.57	0.96	0.62	37.00	21.60	4.53	6.45	6.07	14.76
Tails	1127.00	94.85	0.29	0.02	8.00	1.09	82.49	14.41	79.12	44.88
HEAD	1188.20	100.00	0.33	0.15	9.59	2.30	100.00	100.00	100.00	100.00
(Copper Rougher Tail)	1145.70	96.42	0.30	0.03	8.47	1.42	87.02	20.86	85.19	59.63

STAGE	pH	Na2S2O5	REAGENT ADDITIONS (gpt)			AERATE	COND	FLOT		
		g/g CN	3418A	MIBC	PAX	min.	min.	min.		% Solids
	9.37									36.10
Cyanide destruction		4.90				30				
Copper rougher	7.80		20	13			3	3		
Bulk sulphide float	7.60				16		2	5		

Comments:

FLOTATION TEST REPORT

TEST ID: FS1-2 **SAMPLE:** PLACER DOME **DATE:** DEC 7 / 93

PUPOSE: Repeat FS1-1 - CN destruction at Higher Ph

DESCRIPTION	DRY WT (g)	WT %	ASSAYS AU gpt	CU %	FE %	S(T) %	DISTRIBUTION AU gpt	CU %	FE %	S(T) %
CU PREFLOAT	25.80	2.17	1.05	5.09	33.40	21.10	6.81	73.02	7.29	19.42
BULK SULPHIDE FLOAT	46.30	3.89	0.51	0.47	38.10	25.10	5.94	12.05	14.93	41.46
TAILS	1119.00	93.95	0.31	0.02	8.21	0.98	87.25	14.93	77.77	39.12
HEAD	1191.10	100.00	0.33	0.15	9.92	2.35	100.00	100.00	100.00	100.00
(Copper Rougher Tail)	1165.30	97.83	0.32	0.04	9.40	1.94	93.19	26.98	92.71	80.58

STAGE	pH	Na2S2O5 g/g CN	REAGENT ADDITIONS (gpt) 3418A	MIBC	PAX	AERATE min.	COND min.	FLOT min.		Flotation % Solids
										36.20
Cyanide destruction	9.37	4.80				30				
copper pre-float	8.20		20	13			2	2		
Bulk sulphide float	7.60			6	50		2	4		

FLOTATION TEST REPORT

TEST ID: FS1-3 **SAMPLE:** PLACER DOME **DATE:** DEC 8 / 93

PUPOSE: Remove Pyrrhotite by magnetic separation ; add some CuSo4 to CN destruction step

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	CU %	FE %	S(T) %	AU %	CU %	FE %	S(T) %
Magnetics (pyrhhotite)	80.60	6.81	0.49	0.18	32.00	18.40	9.69	8.71	22.21	52.05
Cu prefloat	14.30	1.21	1.56	8.27	19.60	13.90	5.48	70.26	2.41	6.98
Bulk sulphide float	39.60	3.35	0.78	0.36	24.50	22.60	7.58	8.56	8.35	31.41
Tails	1049.00	88.64	0.30	0.02	7.42	0.26	77.25	12.46	67.02	9.57
HEAD	1183.50	100.00	0.34	0.14	9.81	2.41	100.00	100.00	100.00	100.00
(Copper Rougher Feed)	1102.90	93.19	0.33	0.14	8.19	1.24	90.31	91.29	77.79	47.95
(Copper Rougher Tails)	1088.60	91.98	0.32	0.03	8.04	1.07	84.83	21.03	75.38	40.98

STAGE	pH	Na2S2O5	REAGENT ADDITIONS (gpt)				AERATE	COND	FLOT	Flotation
		g/g CN	CuSo4	3418A	MIBC	PAX	min.	min.	min.	% Solids
Magnetic Separation										36.10
Cyanide destruction	8.60	4.80	25				30			
copper pre-float	9.60			10	6			2	2	
Bulk sulphide float	9.20				6	125		2	5	

FLOTATION TEST REPORT

TEST ID: FS1-4 **SAMPLE:** PLACER DOME **DATE:** Jan 5/94

PUPOSE: Magnetic Separation of Pyrrhotite; No CN destruction

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	CU %	FE %	S(T) %	AU %	CU %	FE %	S(T) %
Magnetics (pyrhhotite)	128.81	5.50	0.34	0.20	40.00	19.44	13.74	7.24	25.74	47.83
Cu Conc	14.40	0.61	1.96	15.40	31.00	25.86	8.97	62.32	2.23	7.11
Cu Clnr Tail	33.92	1.45	0.68	2.05	24.80	10.31	7.34	19.54	4.20	6.68
Bulk Flot	65.97	2.82	1.15	0.27	34.00	27.91	23.99	5.01	11.21	35.17
Tail	2098.64	89.62	0.07	0.01	5.40	0.08	45.96	5.90	56.62	3.21
HEAD	2341.74	100.00	0.13	0.15	8.55	2.24	100.00	100.00	100.00	100.00

(Copper Rougher Feed	2212.93	94.50	0.12	0.15	6.72	1.23	86.26	92.76	74.26	52.17
(Copper Rougher Conc)	48.32	2.06	1.06	6.03	26.65	14.94	16.31	81.86	6.43	13.79
(Copper Rougher Tail)	2164.61	92.44	0.10	0.02	6.27	0.93	69.95	10.90	67.83	38.38

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	Flotation
			CaO	3418A	MIBC	PAX	DF250	min.	min.	% Solids
										35.70
Magnetic Separation	9.40									
copper pre-float	9.38			20	6			2	3	
Bulk sulphide float	9.20					45	4	2	5	
Copper Cleaning	11.00		34					3	2	

FLOTATION TEST REPORT

TEST ID: FS1-5 **SAMPLE:** PLACER DOME **DATE:** Jan 6 / 94

PUPOSE: Magnetic Separation of Pyrrhotite ; Try different Cu Collector

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	CU %	FE %	S(T) %	AU gpt	CU %	FE %	S(T) %
Magnetics (pyrrhotite)	213.82	7.36	0.43	0.18	34.00	15.69	6.36	8.38	32.56	53.23
Cu Conc	17.87	0.62	1.56	15.40	26.00	22.04	1.93	59.92	2.08	6.25
Cu Clnr Tail	23.10	0.80	0.26	3.05	18.50	7.32	0.41	15.34	1.91	2.68
Bulk Flot	82.15	2.83	1.03	0.29	35.00	27.15	5.87	5.19	12.88	35.39
Tail	2566.60	88.40	0.48	0.02	4.40	0.06	85.42	11.18	50.57	2.44
HEAD	2903.54	100.00	0.50	0.16	7.69	2.17	100.00	100.00	100.00	100.00
(Copper Rougher Feed)	2689.72	92.64	0.50	0.16	5.60	1.10	93.64	91.62	67.44	46.77
(Copper Rougher Conc)	40.97	1.41	0.83	8.44	21.77	13.74	2.35	75.26	3.99	8.93
(Copper Rougher Tail)	2648.75	91.22	0.50	0.03	5.35	0.90	91.29	16.36	63.45	37.83

STAGE	pH		REAGENT ADDITIONS g/tonne				AERATE	COND	FLOT	Flotation
			HF-LET	CaO	MIBC	PAX	min.	min.	min.	% Solids
										41.90
Magnetic Separation										
copper pre-float	9.57		22		5			3	3	
Bulk sulphide float					10	25		1	3	
Copper Cleaner	11.30			19				3	1	

FLOTATION TEST REPORT

TEST ID: FS1-6　　**SAMPLE:** PLACER DOME　　**DATE:** Jan 6 / 94

PUPOSE: Magnetic Separation of Pyrrhotite on Rougher Concentrate

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	CU %	FE %	S(T) %	AU gpt	CU %	FE %	S(T) %
Cu Conc	18.47	0.66	2.61	14.57	18.42	16.49	4.25	64.13	1.33	4.70
Cu Cleaner Tail	20.00	0.71	0.91	2.72	10.78	4.45	1.61	12.96	0.84	1.37
Magnetics	11.31	0.40	1.27	2.09	51.00	32.81	1.27	5.63	2.25	5.72
Bulk Flot	117.66	4.19	1.09	0.39	43.57	35.29	11.27	10.94	20.02	64.03
Tail	2657.30	94.69	0.35	0.01	7.28	0.59	81.59	6.33	75.55	24.18
HEAD	2806.27	100.00	0.40	0.15	9.12	2.31	100.00	100.00	100.00	100.00

(Copper Rougher Conc)	58.47	2.08	1.38	5.94	19.37	13.08	7.13	82.73	4.42	11.79
(Copper Cleaner Feed)	38.47	1.37	1.73	8.41	14.45	8.12	5.86	77.10	2.17	6.07
(Copper Rougher Tail)	2774.96	98.88	0.38	0.03	8.82	2.06	92.87	17.27	95.58	88.21

STAGE	pH		REAGENT ADDITIONS g/tonne				AERATE	COND	FLOT	Flotation
			HF-LET	CaO	MIBC	PAX	min.	min.	min.	% Solids
	9.63									40.90
Copper Rougher			27		10			3	5	
Bulk sulphide float	9.31					71		1	5	
Magnetic Separation										
Copper Cleaner	11.40			19				5	1	

Comments:

FLOTATION TEST REPORT

TEST ID: FS3.2-1 **SAMPLE:** Selbaie **DATE:** Dec 10 / 93

PUPOSE: Initial Test - Using CuSo4 and PAX

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	34.00	3.45	0.49	117.60	31.90	30.90	10.96	15.50	4.70	4.85
Flot 2	41.80	4.25	0.31	62.50	28.50	28.10	8.52	10.13	5.16	5.42
Flot 3	30.60	3.11	0.27	59.00	31.20	30.10	5.43	7.00	4.14	4.25
Tails	878.00	89.19	0.13	19.80	22.60	21.10	75.08	67.38	86.00	85.48
HEAD	984.40	100.00	0.15	26.21	23.44	22.02	100.00	100.00	100.00	100.00

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	Flotation
			CuSo4		MIBC	PAX		min.	min.	% Solids
Initial	9.34		15					2		30.80
Flot 1					7	35		2	5	
Flot 2					7	10		1	5	
Flot 3					7	10		1	5	

FLOTATION TEST REPORT

TEST ID: FS3.2-2 **SAMPLE:** Selbaie **DATE:** Dec 10 / 93

PUPOSE: Lower pH with H2SO4 - Add supplemental collector (M-91)

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	75.10	7.64	0.39	78.40	42.80	45.50	18.77	23.50	14.25	15.68
Flot 2	242.00	24.62	0.23	41.70	49.30	50.00	35.67	40.28	52.88	55.52
Flot 3	140.00	14.24	0.32	52.40	41.40	42.20	28.71	29.28	25.69	27.11
Tails	526.00	53.50	0.05	3.30	3.08	0.70	16.85	6.93	7.18	1.69
HEAD	983.10	100.00	0.16	25.48	22.95	22.17	100.00	100.00	100.00	100.00

(Stage 1 Tail)	908.00	92.36	0.14	21.10	21.31	20.24	81.23	76.50	85.75	84.32
(Stage 2 Tail)	666.00	67.74	0.11	13.62	11.14	9.42	45.56	36.21	32.87	28.80

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
			PAX		MIBC	M-91		min.	min.	% Solids
Initial	9.34									30.80
Condition H2SO4	5.00							5		
Flot 1	5.30		30		8	15		3	2	
					7			1	4	
Flot 2	5.30				7	200		1	5	
Flot 3	5.30					75		1	3	
						75		1	3	

FLOTATION TEST REPORT

TEST ID: FS3.2-3 **SAMPLE:** Selbaie **DATE:** Dec 10 / 93

PUPOSE: Low pH ; M-91 only

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	435.00	44.26	0.18	49.50	43.90	49.10	68.52	84.58	88.96	94.61
Flot 2	42.90	4.36	0.25	56.20	19.00	19.40	9.39	9.47	3.80	3.69
Tails	505.00	51.38	0.05	3.00	3.08	0.76	22.10	5.95	7.25	1.70
HEAD	982.90	100.00	0.12	25.90	21.84	22.97	100.00	100.00	100.00	100.00
(Stage 1 tails)	547.90	55.74	0.07	7.17	4.33	2.22	31.48	15.42	11.04	5.39

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
			H2SO4		MIBC	M-91		min.	min.	% Solids
Initial	9.29									30.80
Condition H2SO4	5.20		560					5		
Flot 1					8	150		3	2	
	5.20		20		7	150		1	3	
Flot 2	5.30		30		7	75		1	7	

FLOTATION TEST REPORT

TEST ID: FS3.2-4 **SAMPLE:** Selbaie **DATE:** Jan 4 / 94

PUPOSE: Conduct test at higher pH using mixture of 404 & PAX with pre-aeration

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	46.60	4.81	0.37	71.63	35.20	33.22	8.53	14.69	7.18	7.35
Flot 2	213.60	22.06	0.27	39.07	43.90	41.84	29.11	36.73	41.04	42.45
Flot 3	228.20	23.56	0.30	37.80	45.37	43.82	33.94	37.96	45.31	47.50
Tails	480.00	49.57	0.12	5.03	3.08	1.18	28.41	10.63	6.47	2.69
HEAD	968.40	100.00	0.21	23.47	23.59	21.74	100.00	100.00	100.00	100.00

(Stage 1 Tail)	921.80	95.19	0.20	21.03	23.01	21.16	91.47	85.31	92.82	92.65
(Stage 2 Tail)	708.20	73.13	0.18	15.59	16.71	14.92	62.36	48.59	51.78	50.19

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
			H2SO4	404	PAX	DF250	M-91	min.	min.	% Solids
Initial	8.83									30.50
Aeration								10		
Flot 1	8.60			50	40	9		5	2	
Adjust pH (H2SO4)	5.30									
Flot 2	5.45			25	20	9		1	5	
Flot 3	6.40					9	75	1	5	

FLOTATION TEST REPORT

TEST ID: FS3.2-5 **SAMPLE:** Selbaie **DATE:** Jan 4 / 94

PUPOSE: Conduct test at higher pH using mixture of M-91& PAX

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	127.60	13.00	0.41	46.00	40.00	38.56	17.46	24.81	23.61	23.08
Flot 2	351.30	35.78	0.41	43.87	41.00	45.29	48.06	65.14	66.62	74.63
Tails	503.00	51.23	0.21	4.73	4.20	0.97	34.49	10.06	9.77	2.29
HEAD	981.90	100.00	0.31	24.10	22.02	21.71	100.00	100.00	100.00	100.00
(Stage 1 Tail)	854.30	87.00	0.29	20.82	19.33	19.19	82.54	75.19	76.39	76.92

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
					PAX	DF250	M-91	min.	min.	% Solids
Initial	8.90									30.80
Flot 1	8.60				60	18	60	3	3	
Adjust pH (H2SO4)	6.00									
Flot 2					30	9	15	1	5	
	6.40				30			1	5	

FLOTATION TEST REPORT

TEST ID: FS3.2-6 **SAMPLE:** Selbaie **DATE:** Jan 12 / 94

PUPOSE: Conduct test at higher pH using mixture of Texeco SP-160

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	184.00	15.64	0.41	63.80	32.70	30.62	37.24	37.70	22.22	22.81
Flot 2	432.70	36.79	0.21	37.73	44.70	43.12	43.79	52.43	71.42	75.54
Tails	559.40	47.56	0.07	5.49	3.08	0.73	18.96	9.86	6.36	1.65
HEAD	1176.10	100.00	0.17	26.47	23.03	21.00	100.00	100.00	100.00	100.00
(Stage 1 tail)	992.10	84.36	0.13	19.55	21.23	19.22	62.76	62.30	77.78	77.19

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
			SP-160	DF-250	404			min.	min.	% Solids
Initial	9.11									35.40
Flot 1	8.60		75	18				10	3	
			50	9				5	6	
Flot 2					50			1	3	
					50			1	3	
					50			1	3	

FLOTATION TEST REPORT

TEST ID: FS3.2-7 **SAMPLE:** Selbaie **DATE:** Jan 12 / 94

PUPOSE: Condition with Sod. Sulphite

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	593.40	50.38	0.21	46.65	38.00	41.83	75.19	89.61	92.61	96.74
Tails	584.50	49.62	0.07	5.49	3.08	1.43	24.81	10.39	7.39	3.26
HEAD	1177.90	100.00	0.14	26.23	20.67	21.78	100.00	100.00	100.00	100.00

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
			Na2SO3	404	PAX	DF250		min.	min.	% Solids
Initial	9.10									35.40
			1000					10		
Flot 1	9.30			83	85	9		2	3	
				40	42	9		1	3	
Flot 2				40	35			1	5	
	6.40				30			1	5	

FLOTATION TEST REPORT

TEST ID: FS3.2-8 **SAMPLE:** Selbaie **DATE:** Feb 2 / 94

PUPOSE: Use Texaco SP-160 as collector with longer conditioning time

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	114.34	9.72	0.41	61.80	38.70	43.85	12.79	25.20	18.25	19.12
Flot 2	284.34	24.17	0.40	42.33	41.40	47.78	30.88	42.92	48.55	51.80
Flot 3	146.98	12.49	0.42	45.27	38.33	45.25	16.68	23.72	23.24	25.36
Tails	630.81	53.62	0.23	3.63	3.83	1.55	39.66	8.16	9.96	3.73
HEAD	1176.47	100.00	0.31	23.84	20.61	22.29	100.00	100.00	100.00	100.00

(Stage 1 Tail)	1062.13	90.28	0.30	19.75	18.66	19.97	87.21	74.80	81.75	80.88
(Stage 2 Tail)	777.79	66.11	0.27	11.50	10.35	9.81	56.34	31.89	33.20	29.09

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
					SP-160	DF250		min.	min.	% Solids
Initial	8.83									35.40
Flot 1					94	6		15	5	
Flot 2	8.72				44	6		5	5	
Flot 3	8.38				30			5	6	

Comments:

F1- Froth lightly loaded - not much weight
F2-Froth appearance better than F1 - more solids floating
F3 - Highly pyritic looking froth - good loading - froth near barren at end

FLOTATION TEST REPORT

TEST ID: FS3.2-9 **SAMPLE** Selbaie **DATE:** Feb 2 / 94

PUPOSE: Use Texaco SP-160 - Pre-aerate for 20 min

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Flot 1	165.20	16.90	0.37	51.20	40.64	45.54	24.10	37.64	34.71	36.25
Flot 2	239.79	24.53	0.33	42.73	41.89	48.85	31.46	45.59	51.93	56.44
Flot 3	31.94	3.27	0.58	58.27	24.16	27.42	7.37	8.28	3.99	4.22
Tails	540.58	55.30	0.17	3.53	3.35	1.19	37.07	8.49	9.36	3.10
HEAD	977.51	100.00	0.26	22.99	19.79	21.23	100.00	100.00	100.00	100.00

(Stage 1 Tail)	812.31	83.10	0.24	17.25	15.55	16.29	75.90	62.36	65.29	63.75
(Stage 2 Tail)	572.52	58.57	0.20	6.58	4.51	2.65	44.44	16.77	13.35	7.32

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
					SP-160	DF250		min.	min.	% Solids
Initial	8.99									30.70
								20		
Flot 1	8.80				94	6		15	5	
Flot 2	8.52				44			5	5	
Flot 3	8.44				30			5	5	

FLOTATION TEST REPORT

TEST ID: FS2-1 **SAMPLE:** Lac Minerals **DATE:** Jan 5 / 94

PUPOSE: Initial test with CN destruction

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	TE ppm	FE %	S(T) %	AU %	TE %	FE %	S(T) %
Bulk sulphide float	173.62	14.08	0.49	14.00		28.42	27.36	21.26		96.88
Tails	1059.30	85.92	0.22	8.50		0.15	72.64	78.74		3.12
HEAD	1232.92	100.00	0.25	9.27		4.13	100.00	100.00		100.00

STAGE	pH	Na2S2O5	REAGEN	ADDITIO	g/tonne		AERATE	COND	FLOT	
		g/g CN	CuSo4	404	DF-250	PAX	min.	min.	min.	% Solids
Initial	10.95									37.40
Cyanide destruction	8.60	4.70	25				30			
Bulk Flotation	9.18			40	6	32		3	1	
Change collector				5100						
				26				1	6	

FLOTATION TEST REPORT

TEST ID: FS2-2 **SAMPLE:** Lac Minerals **DATE:** Jan 5 / 94

PUPOSE: Test with 5100 and 404

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	TE ppm	FE %	S(T) %	AU %	TE %	FE %	S(T) %
Flot 1	147.11	11.60	0.34	17.00		20.98	16.96	21.54		57.99
Flot 2	48.79	3.85	0.68	11.00		38.36	11.30	4.62		35.16
Tails	1071.79	84.55	0.20	8.00		0.34	71.73	73.84		6.85
HEAD	1267.69	100.00	0.23	9.16		4.20	100.00	100.00		100.00
(Stage 1 Tails)	1120.58	88.40	0.22	8.13		2.00	83.04	78.46		42.01

STAGE	pH	Na2S2O5	REAGEN ADDITIO		g/tonne		AERATE	COND	FLOT	
		g/g CN	CuSo4	5100	DF-250	404	min.	min.	min.	% Solids
Initial	10.80									38.00
Cyanide destruction	8.40	4.50	24				30			
Flot 1	9.13			40	6			3	3	
				13				1	5	
Flot 2						38		1	3	

FLOTATION TEST REPORT

TEST ID: FS2-3 **SAMPLE:** Lac Minerals **DATE:** Jan 5 / 94

PUPOSE: Test with 5100 and 404

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	TE ppm	FE %	S(T) %	AU %	TE %	FE %	S(T) %
Bulk sulphide float	199.77	15.82	0.45	15.00		25.11	28.51	24.47		93.46
Tails	1063.06	84.18	0.21	8.70		0.33	71.49	75.53		6.54
HEAD	1262.83	100.00	0.25	9.70		4.25	100.00	100.00		100.00

STAGE	pH	Na2S2O5	REAGEN ADDITIO		g/tonne		AERATE	COND	FLOT	
		g/g CN	CuSo4	5100	DF-250	404	min.	min.	min.	% Solids
Initial	10.80									38.10
Cyanide destruction	8.40	4.50	25				30			
Flot 1	9.25			20	13	28		3	3	
				13		20		1	6	

COMMENTS:

FLOTATION TEST REPORT

TEST ID: FS2-4 **SAMPLE:** Lac Minerals **DATE:** Feb 3 / 94

PUPOSE: No CN destruction - Use Texaco SP-160 Collector

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	TE ppm	FE %	S(T) %	AU %	TE %	FE %	S(T) %
Bulk sulphide float	209.45	17.02	0.26	11.00		4.05	15.01	25.77		16.68
Tails	1020.85	82.98	0.30	6.50		4.15	84.99	74.23		83.32
HEAD	1230.30	100.00	0.29	7.27		4.13	100.00	100.00		100.00

STAGE	pH	Na2S2O5	REAGEN	ADDITIO	g/tonne		AERATE	COND	FLOT	
		g/g CN	CuSo4	SP160	DF-250		min.	min.	min.	% Solids
Initial	11.25									37.30
Flot 1	11.15			82	6			15	5	

COMMENTS: Unselective flotation. Froth masked by slimes
Difficult to see presence of sulphides

FLOTATION TEST REPORT

TEST ID: FS2-5 **SAMPLE:** Lac Minerals **DATE:** Feb 3 / 94

PUPOSE: CN destruction prior to flotation - Use Texaco SP-160 collector

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	TE ppm	FE %	S(T) %	AU %	TE %	FE %	S(T) %
Flot 1	19.24	1.58	0.67	26.00		4.05	3.51	5.36		1.62
Flot 2	38.17	3.13	0.24	17.00		5.35	2.49	6.96		4.24
Flot 3	131.43	10.79	0.59	9.00		32.27	20.86	12.68		87.96
Tails	1028.94	84.49	0.26	6.80		0.29	73.15	75.00		6.19
HEAD	1217.78	100.00	0.30	7.66		3.96	100.00	100.00		100.00

(Stage 1 Tail)	1198.54	98.42	0.30	7.37		3.96	96.49	94.64		98.38
(Stage 2 Tail)	1160.37	95.29	0.30	7.05		3.91	94.01	87.68		94.15

STAGE	pH	Na2S2O5	REAGEN	ADDITIO	g/tonne		AERATE	COND	FLOT	
		g/g CN	CuSo4	SP-160	DF-250	404/PAX	min.	min.	min.	% Solids
Initial	11.20									37.00
Cyanide destruction	8.80	4.50	25				30			
Flot 1				45	6			15	3	
Flot 2 add 404					6	53		1	3	
Flot 3 add PAX					6	83		2	3	

COMMENTS:

Froth was barren after the conditioning with SP-160
Added some 404 to cell. Froth characteristics were still poor
Added some PAX - getting some sulphides floating- froth darker

FLOTATION TEST REPORT

TEST ID: FS2-6 **SAMPLE:** Lac Minerals **DATE:** Feb 3 / 94

PUPOSE: Flot using PAX only after CN destruction

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	TE ppm	FE %	S(T) %	AU %	TE %	FE %	S(T) %
Flot 1	78.05	6.59	0.78	11.00		42.53	17.09	8.67		67.13
Flot 2	31.26	2.64	0.51	14.00		29.62	4.48	4.42		18.73
Tails	1075.78	90.78	0.26	8.00		0.65	78.43	86.91		14.14
HEAD	1185.09	100.00	0.30	8.36		4.17	100.00	100.00		100.00
(Stage 1 tail)	1107.04	93.41	0.27	8.17		1.47	82.91	91.33		32.87

STAGE	pH	Na2S2O5	REAGEN	ADDITIO	g/tonne		AERATE	COND	FLOT	
		g/g CN	CuSo4	PAX	DF-250		min.	min.	min.	% Solids
Initial	11.20									36.30
Cyanide destruction	8.40	4.50	25				30			
Flot 1	9.48			125	6			3	5	
Flot 2				45	6			2	2	

COMMENTS:

FLOTATION TEST REPORT

TEST ID: FS3.3-1 **SAMPLE:** Selbaie **DATE:** Jan 4 / 94

PUPOSE: Initial Test Same conditions as FS3.2- 4

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Conc	143.34	10.62	0.51	51.43		29.93	33.39	68.05		87.89
Tails	1205.83	89.38	0.12	2.87		0.49	66.61	31.95		12.11
HEAD	1349.17	100.00	0.16	8.03		3.62	100.00	100.00		100.00

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
				404	PAX	DF250		min.	min.	% Solids
Initial	9.54									40.20
Aeration								10		
Flot 1	9.30			43	22	6		5	3	
						6		1	3	
	8.90									

FLOTATION TEST REPORT

TEST ID: FS3.3-2 **SAMPLE:** Selbaie **DATE:** Jan 7 / 94

PUPOSE: Mixture of PAX and M-91

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Bulk sulphide float	205.34	15.78	0.69	40.10	22.10	20.49	38.43	73.35	53.50	92.31
Tails	1095.64	84.22	0.21	2.73	3.60	0.32	61.57	26.65	46.50	7.69
HEAD	1300.98	100.00	0.28	8.63	6.52	3.50	100.00	100.00	100.00	100.00

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
				M-91	PAX	DF250		min.	min.	% Solids
Initial	9.58									39.10
Flot 1	9.30			22	30	12		2	2	
						6		1	3	
	9.40									

FLOTATION TEST REPORT

TEST ID: FS3.3-3 **SAMPLE:** Selbaie **DATE:** Feb 2 / 04

PUPOSE: Use Texaco SP-160 as collector

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	Ag gpt	FE %	S(T) %	Au %	Ag %	FE %	S(T) %
Concentrate	261.18	19.29	0.34	33.60	16.77	16.67	32.01	98.41	57.85	90.67
Tailings	1093.07	80.71	0.17	0.13	2.92	0.41	67.99	1.59	42.15	9.33
HEAD	1354.25	100.00	0.20	6.59	5.59	3.55	100.00	100.00	100.00	100.00

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
			SP-160	DF-250				min.	min.	% Solids
Initial	9.34									40.00
Flot			74	6				15	5	

Comments: Heavily loaded froth
fast flowing

FLOTATION TEST REPORT

TEST ID: FS3.3-4 **SAMPLE:** Selbaie **DATE:** Feb 2 / 94

PUPOSE: Use Texaco SP-160 - lower addition rate

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Bulk sulphide float	155.45	11.12	0.45	48.23	24.27	25.75	22.53	82.28	41.68	81.95
Tails	1241.99	88.88	0.20	1.30	4.25	0.71	77.47	17.72	58.32	18.05
HEAD	1397.44	100.00	0.22	6.52	6.48	3.50	100.00	100.00	100.00	100.00

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
					SP-160	DF250		min.	min.	% Solids
Initial	9.50									41.20
Flot 1					38	6		15	5	

Comments: Well loaded froth - froth barren at end

FLOTATION TEST REPORT

TEST ID: FS3.3-5 **SAMPLE:** Selbaie **DATE:** Feb 2 / 94

PUPOSE: Use PAX as collector

			ASSAYS				DISTRIBUTION			
DESCRIPTION	DRY WT (g)	WT %	AU gpt	AG gpt	FE %	S(T) %	AU %	AG %	FE %	S(T) %
Bulk sulphide float	150.31	11.21	0.52	52.70	25.83	28.13	23.79	90.11	53.88	90.10
Tails	1191.01	88.79	0.21	0.73	2.79	0.39	76.21	9.89	46.12	9.90
HEAD	1341.32	100.00	0.24	6.55	5.37	3.50	100.00	100.00	100.00	100.00

STAGE	pH		REAGENT ADDITIONS g/tonne					COND	FLOT	
					PAX	DF250		min.	min.	% Solids
Initial	9.54									40.00
Flot 1	9.30				75	6		5	5	
	9.20									

Comments: Well loaded froth
Froth virtually barren after about 3 min

APPENDIX B

GRAVITY CONCENTRATION TEST DATA

DETAILS OF TESTS

LR4370-069
MEND PROJECT

Table 1 : Head Analyses

Sample No.	Sample Identification	Assay Data Au, g/t	Ag, g/t	Cu, %	S(T),%	Fe, %	Te, g/t	Size K80 micrometers	Specific Gravity
Direct Assay									
1	Detour Lake	0.24	-	0.13	1.95	8.76	-	72	2.95
2	Lac Minerals	0.19	-	0.038	4.17	5.14	5.0	63	2.84
3.2	Selbaie	0.19	26.3	-	20.9	20.4	-	57	3.31
3.3	Selbaie	0.09	8.8	-	3.63	5.23	-	53	2.75
Calculated Heads		(Average of three tests)							
1	Detour Lake	0.27	-	0.11	1.95	8.70	-	-	2.95
2	Lac Minerals	0.20	-	-	4.01	5.05	7.0	-	2.83
3.2	Selbaie	0.18	23.1	-	20.0	19.0	-	-	3.28
3.3	Selbaie	0.13	8.5	-	3.51	5.00	-	-	2.77

Table 2 : Summary of Gravity Separation Results (From Sample Period Data)

Sample #1 : Detour Lake Mine Size Analysis - 81.1 % Passing 200 mesh, K80- 72 micrometers

Gravity Method	Product	Calc. Rate Sample Per. Dry kg/h	Pulp Density g/L	Solids S.G.	Weight %	Assays, %,g/t				% Distribution			
						Au	Cu	S(T)	Fe	Au	Cu	S(T)	Fe
Sample1:Test Feed		-	-	2.95	-	0.24	0.13	1.95	8.76	100.0	100.0	100.0	100.0
Gravity	Concentrate	-	-	3.33	5.6	0.56	0.39	8.64	16.9	11.2	18.7	24.2	10.7
By	Conc+Midds	-	-	3.03	30.7	0.39	0.17	3.71	11.0	42.8	44.8	57.4	38.5
Tabling	Tailing	-	-	2.93	69.3	0.23	0.09	1.22	7.82	57.2	55.2	42.6	61.6
	Head(Calc.)	108.6	1300	2.98	100.0	0.28	0.12	1.98	8.81	100.0	100.0	100.0	100.0
Gravity	Concentrate	109	1691	3.03	8.2	0.55	0.19	4.17	11.1	17.1	13.7	17.7	10.6
By	Conc+Midds	374	-	3.03	28.0	0.39	0.12	2.44	8.81	42.1	29.7	35.5	28.7
Spiral	Tailing	965	1295	2.99	72.0	0.21	0.11	1.72	8.48	57.9	70.3	64.5	71.3
	Head(Calc.)	1340	1330	2.95	100.0	0.26	0.11	1.92	8.57	100.0	100.0	100.0	100.0
Gravity	Concentrate	405	1465	2.92	25.0	0.33	0.10	2.10	8.63	29.3	21.7	26.9	24.7
By	Conc+Cl Tl.	1030	-	2.91	63.6	0.31	0.11	1.99	8.78	70.3	62.0	64.8	64.0
Trays	Tailing	589	1295	2.94	36.4	0.23	0.12	1.89	8.62	29.7	38.0	35.2	36.0
	Head(Calc.)	1619	1390	2.95	100.0	0.28	0.11	1.95	8.72	100.0	100.0	100.0	100.0

Table 3 : Summary of Gravity Separation Results (From Sample Period Data)

Sample #2 : Lac Minerals Size Analysis - 85.3 % Passing 200 mesh, K80- 63 micrometers

Gravity Method	Product	Calc. Rate Sample Per. Dry kg/h	Pulp Density g/L	Solids S.G.	Weight %	Assays, %,g/t				% Distribution			
						Au	Te	S(T)	Fe	Au	Te	S(T)	Fe
Sample1:Test Feed		-	-	2.84	-	0.19	5.0	4.17	5.14	100.0	100.0	100.0	100.0
Gravity	Concentrate	-	-	3.12	7.2	0.65	6.0	13.7	13.2	24.1	7.6	25.3	18.6
By	Conc+Midds	-	-	3.03	24.9	0.41	5.0	5.94	6.48	53.4	20.2	38.1	31.7
Tabling	Tailing	-	-	2.83	75.1	0.12	6.0	3.20	4.62	46.6	79.8	61.9	98.3
	Head(Calc.)	110.8	1300	2.85	100.0	0.19	6.0	3.88	5.08	100.0	100.0	100.0	100.0
Gravity	Concentrate	82	1544	2.85	6.8	0.46	8.0	5.50	6.70	15.5	5.8	9.7	9.1
By	Conc+Midds	230	-	2.83	19.2	0.30	7.0	4.31	5.49	28.3	13.7	21.4	21.0
Spiral	Tailing	973	1277	2.80	80.8	0.18	10.0	3.76	4.89	71.7	86.3	78.6	79.0
	Head(Calc.)	1204	1297	2.80	100.0	0.20	9.0	3.87	5.00	100.0	100.0	100.0	100.0
Gravity	Concentrate	307	1273	2.83	14.1	0.21	6.0	4.54	5.26	13.0	16.5	15.0	14.7
By	Conc+Cl Tl.	1257	-	2.83	57.8	0.23	4.0	4.37	5.08	59.2	50.6	58.9	58.1
Trays	Tailing	916	1310	2.84	42.2	0.22	6.0	4.17	5.03	40.8	49.4	41.1	41.9
	Head(Calc.)	2173	1345	2.84	100.0	0.23	5.0	4.28	5.06	100.0	100.0	100.0	100.0

Table 4 : Summary of Gravity Separation Results (From Sample Period Data)

Sample #3.2 : Selbaie Size Analysis - 88.9 % Passing 200 mesh, K80- 57 micrometers

Gravity Method	Product	Calc. Rate Sample Per. Dry kg/h	Pulp Density g/L	Solids S.G.	Weight %	Assays, %,g/t				% Distribution			
						Au	Ag	S(T)	Fe	Au	Ag	S(T)	Fe
Sample1:Test Feed		-	-	3.31	-	0.19	26.3	20.9	20.4	100.0	100.0	100.0	100.0
Gravity By Tabling	Concentrate	-	-	4.13	13.0	0.37	37.5	40.5	35.0	27.9	21.3	26.9	24.6
	Conc+Midds	-	-	3.03	28.7	0.23	24.6	24.0	21.4	37.9	30.8	35.2	33.3
	Tailing	-	-	3.15	71.3	0.15	22.2	17.8	17.3	62.1	69.2	64.8	66.7
	Head(Calc.)	57.8	1277	3.24	100.0	0.17	22.9	19.6	18.5	100.0	100.0	100.0	100.0
Gravity By Spiral	Concentrate	108	1760	3.54	9.1	0.30	30.6	27.0	25.3	15.0	12.3	12.5	12.3
	Conc+Midds	313	-	3.30	26.3	0.21	23.9	20.0	18.9	31.1	28.0	26.8	26.7
	Tailing	876	1263	3.28	73.7	0.17	22.0	19.5	18.6	68.9	72.0	73.2	73.3
	Head(Calc.)	1189	1282	3.29	100.0	0.18	22.5	19.6	18.7	100.0	100.0	100.0	100.0
Gravity By Trays	Concentrate	307	1487	3.33	17.9	0.19	25.0	21.2	20.2	18.2	18.6	18.3	18.4
	Conc+Cl Tl.	882	-	3.31	51.5	0.18	24.0	20.7	19.7	50.7	51.4	51.5	51.5
	Tailing	830	1347	3.30	48.5	0.19	24.1	20.8	19.7	49.3	48.6	48.5	48.5
	Head(Calc.)	1712	1398	3.31	100.0	0.19	24.0	20.8	19.7	100.0	100.0	100.0	100.0

Table 5 : Summary of Gravity Separation Results (From Sample Period Data)

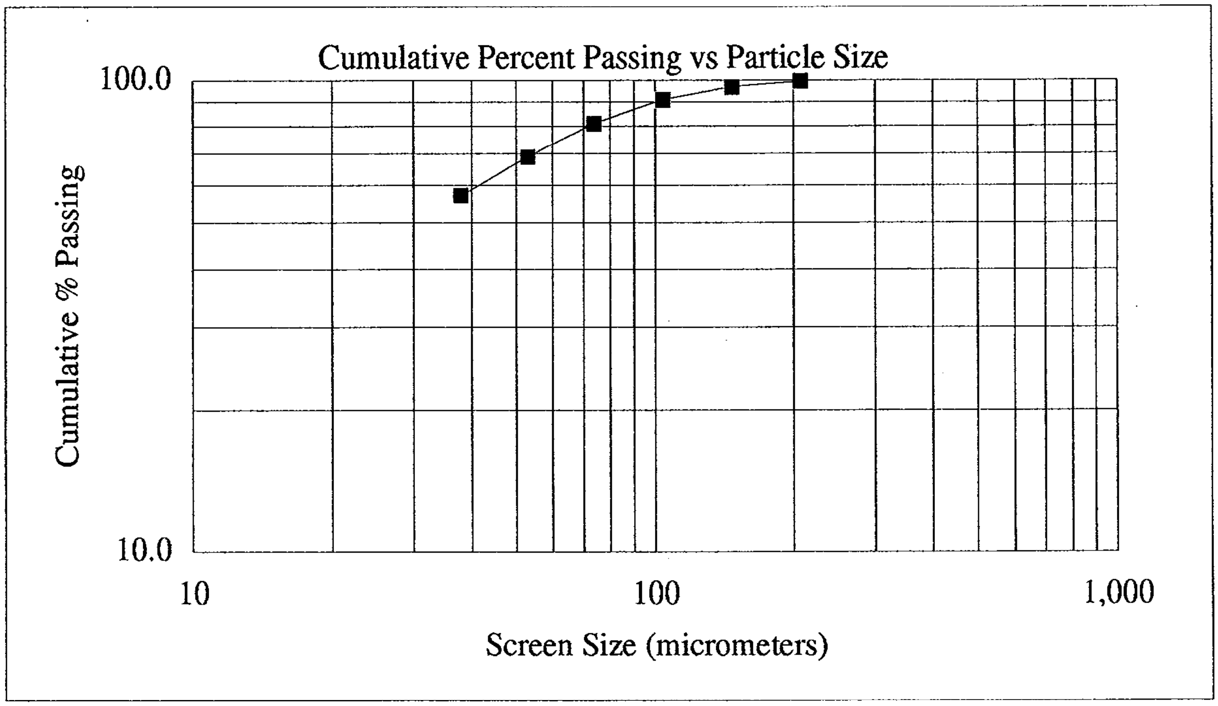
Sample #3.3 : Selbaie Size Analysis - 91.4 % Passing 200 mesh, K80- 53 micrometers

Gravity Method	Product	Calc. Rate Sample Per. Dry kg/h	Pulp Density g/L	Solids S.G.	Weight %	Assays, %,g/t				% Distribution			
						Au	Ag	S(T)	Fe	Au	Ag	S(T)	Fe
Sample1:Test Feed		-	-	2.75	-	0.09	8.8	3.6	5.23	100.0	100.0	100.0	100.0
Gravity By Tabling	Concentrate	-	-	3.18	4.7	0.47	30.5	16.8	16.3	18.0	15.9	21.0	14.5
	Conc+Midds	-	-	3.03	13.5	0.26	15.0	7.43	7.95	28.9	22.7	26.9	20.5
	Tailing	-	-	2.77	86.5	0.10	8.0	3.15	4.81	71.1	77.3	73.1	79.5
	Head(Calc.)	270	1290	2.79	100.0	0.12	8.9	3.73	5.23	100.0	100.0	100.0	100.0
Gravity By Spiral	Concentrate	94	1669	2.89	4.6	0.71	18.8	8.56	9.09	20.6	9.7	10.3	7.8
	Conc+Midds	331	-	2.80	16.2	0.30	11.3	4.89	6.07	30.9	20.5	20.8	18.5
	Tailing	1720	1284	2.77	83.8	0.13	8.4	3.58	5.16	69.1	79.5	79.2	81.5
	Head(Calc.)	2052	1297	2.77	100.0	0.16	8.9	3.79	5.31	100.0	100.0	100.0	100.0
Gravity By Trays	Concentrate	463	1476	2.76	31.8	0.14	7.6	3.13	4.18	39.5	31.8	33.1	29.7
	Conc+Cl Tl.	727	-	2.76	49.9	0.13	7.5	3.09	4.35	55.5	49.2	51.4	48.6
	Tailing	731	1440	2.73	50.1	0.10	7.7	2.91	4.58	44.5	50.8	48.6	51.4
	Head(Calc.)	1458	1460	2.74	100.0	0.11	7.6	3.00	4.47	100.0	100.0	100.0	100.0

Size Distribution Analysis

Sample: #1 Head Sample - Detour Lake Mine

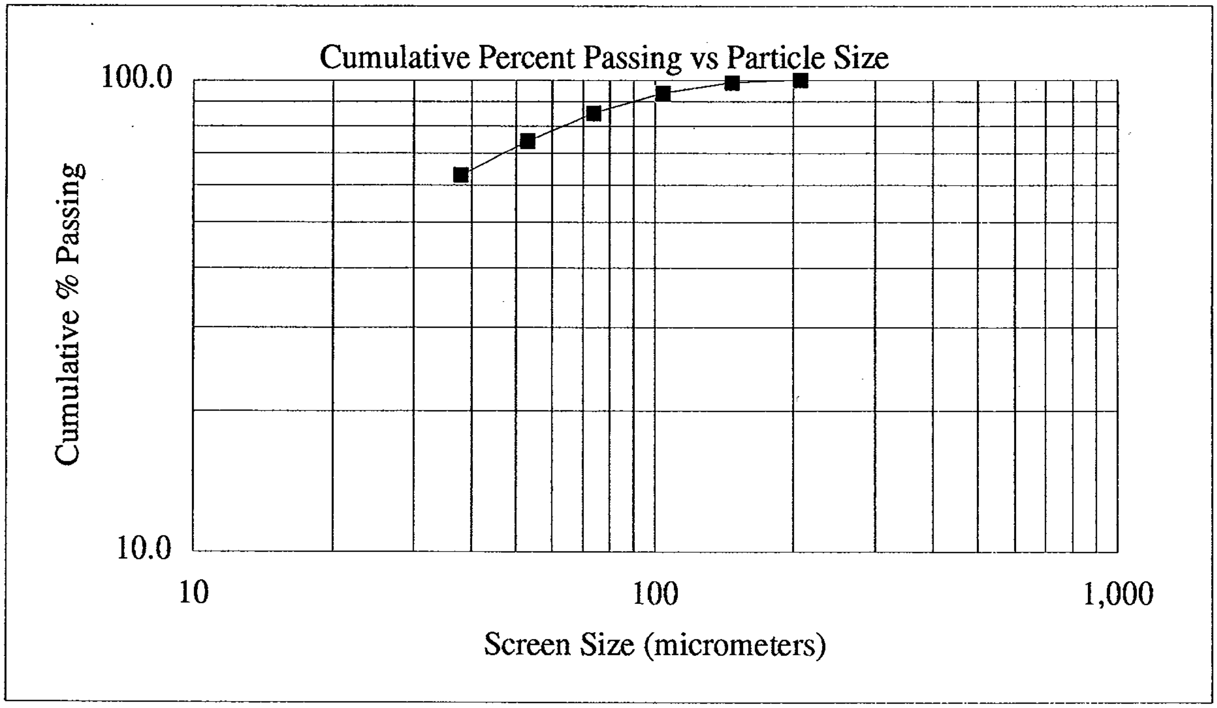
Size		Weight	% Retained		% Passing
Mesh (Tyler)	µm	grams	Individual	Cumulative	Cumulative
65	208	1.1	0.8	0.8	99.2
100	147	3.3	2.3	3.1	96.9
150	104	8.6	6.1	9.2	90.8
200	74	13.8	9.7	18.9	81.1
270	53	17.2	12.1	31.0	69.0
400	38	17.0	12.0	43.0	57.0
Pan	-38	81.0	57.0	100.0	0.0
Total	-	**142.0**	**100.0**	-	-
K80	**72**				



Size Distribution Analysis

Sample: #2 Head Sample- Lac Minerals

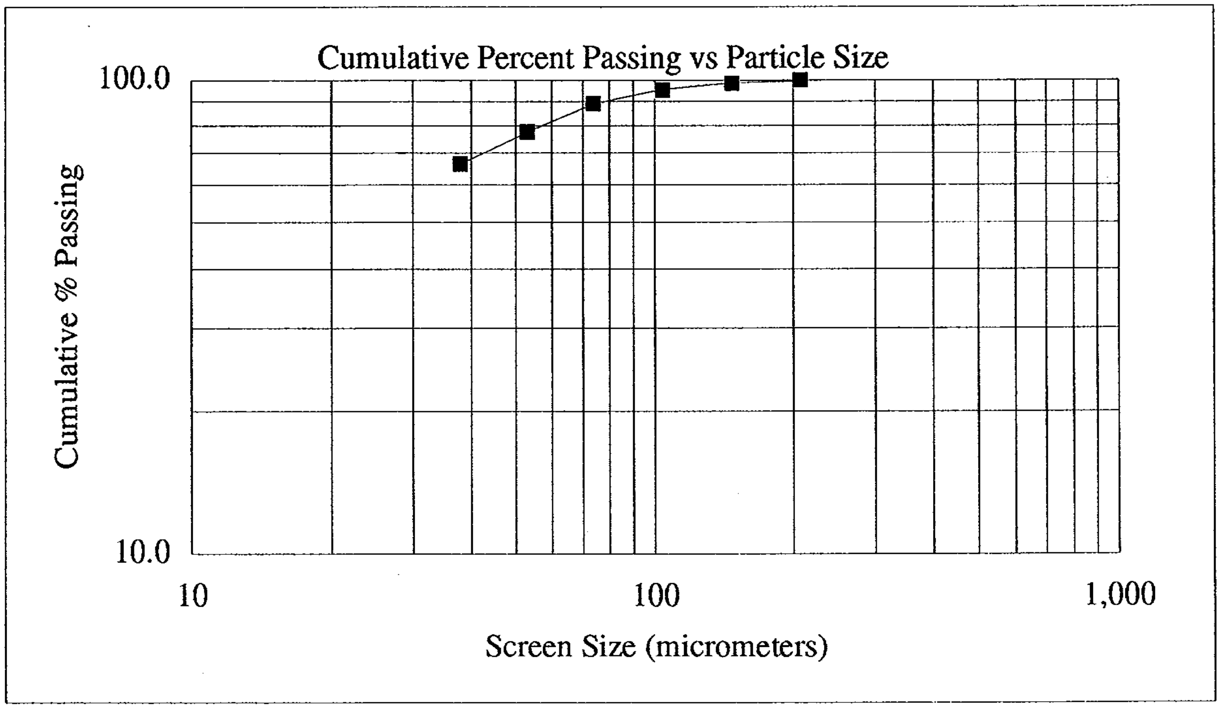
Size		Weight	% Retained		% Passing
Mesh (Tyler)	µm	grams	Individual	Cumulative	Cumulative
65	208	0.3	0.2	0.2	99.8
100	147	2.0	1.2	1.4	98.6
150	104	7.8	4.8	6.2	93.8
200	74	13.7	8.5	14.7	85.3
270	53	17.9	11.1	25.8	74.2
400	38	18.2	11.3	37.0	63.0
Pan	-38	101.8	63.0	100.0	0.0
Total	-	**161.7**	**100.0**	-	-
K80	**63**				



Size Distribution Analysis

Sample: # 3.2 Head Sample - Selbaie

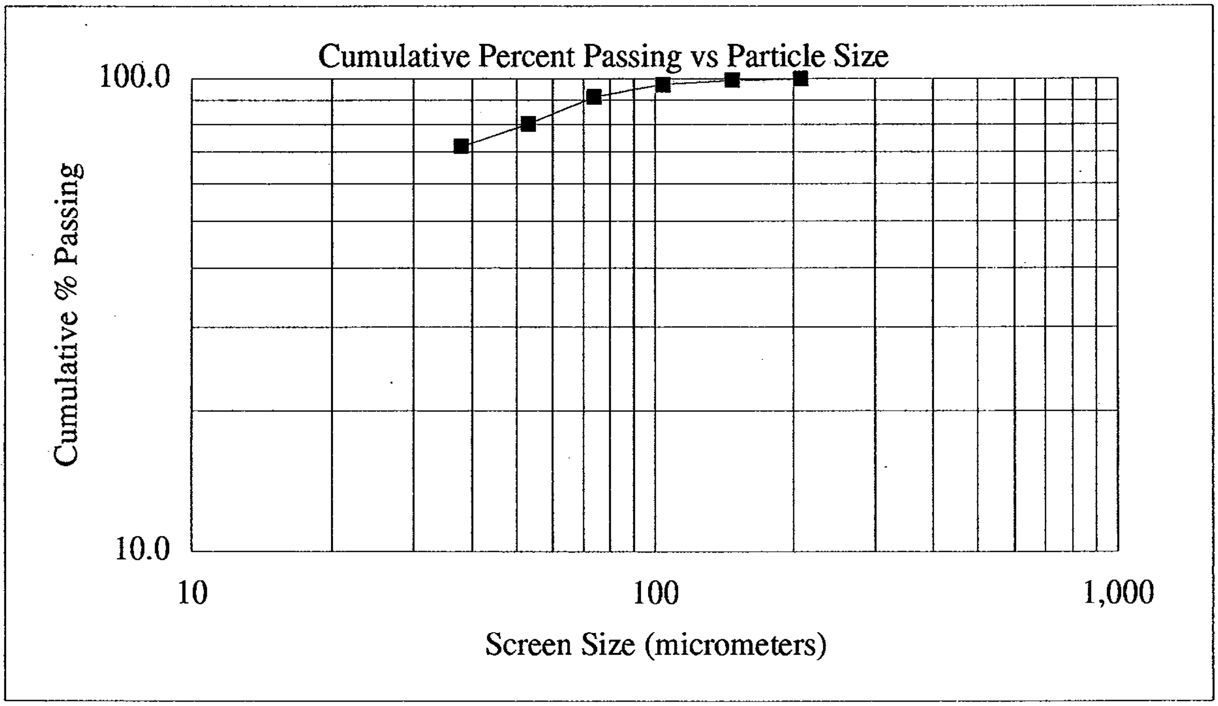
Size		Weight	% Retained		% Passing
Mesh (Tyler)	μm	grams	Individual	Cumulative	Cumulative
65	208	0.4	0.2	0.2	99.8
100	147	2.6	1.4	1.6	98.4
150	104	5.6	3.1	4.7	95.3
200	74	11.7	6.4	11.1	88.9
270	53	21.1	11.5	22.6	77.4
400	38	20.3	11.1	33.7	66.3
Pan	-38	121.4	66.3	100.0	0.0
Total	-	**183.1**	**100.0**	-	-
K80	**57**				



Size Distribution Analysis

Sample: # 3.3 Head Sample - Selbaie

Size		Weight	% Retained		% Passing
Mesh (Tyler)	µm	grams	Individual	Cumulative	Cumulative
65	208	0.4	0.2	0.2	99.8
100	147	1.2	0.7	0.9	99.1
150	104	3.5	2.0	2.9	97.1
200	74	9.9	5.7	8.6	91.4
270	53	19.8	11.4	20.0	80.0
400	38	14.2	8.1	28.1	71.9
Pan	-38	125.4	71.9	100.0	0.0
Total	-	**174.4**	**100.0**	-	-
K80	**53**				



Test No.: #1 Detour Lake -Table Project: 4370-069
MEND
Date: Dec. 15,1993

Purpose: To investigate sulphide recovery by tabling.

Procedure: Slurry as received was pulped to 1300 g/L and passed over a 1/8 Wilfley table to separate sulphides from gangue.
The test was conducted over a period of 5.2 minutes after which the table was carefully cleaned.
Products were filtered and submitted for assay.

Size Analysis: Test feed was 81.1 % passing 200 mesh, K80 : 72 micrometers.

Feed: 9400 grams of #1 -Detour Lake Sample
Repulped feed at 1300 g/L (34.8 % solids) was fed at a rate of 4 liters per minute.

Metallurgical Results:

Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Cu %	S(T) %	Fe %	Au	Cu	S(T)	Fe
Table Conc	3.33	521.5	5.6	0.56	0.39	8.64	16.9	11.2	18.7	24.2	10.7
Table Midds	3.04	2359.0	25.1	0.35	0.12	2.62	9.75	31.6	26.1	33.2	27.8
Table Tail	2.93	6504.5	69.3	0.23	0.092	1.22	7.82	57.2	55.2	42.6	61.5
Head(Calc)	2.98	9385	100.0	0.28	0.12	1.98	8.81	100.0	100.0	100.0	100.0
Head (Direct)	2.95	-	-	0.24	0.13	1.95	8.76	-	-	-	-
Comb. Products											
Conc.+Midds	3.03	2881	30.7	0.39	0.17	3.71	11.0	42.8	44.8	57.4	38.5

Comments: Table settings adjusted to suit test conditions during test.

Test No.: #1 Detour Lake -Spiral Project: 4370-069 MEND Date: Dec. 15,1993

Purpose: To investigate sulphide recovery by Spiral gravity separation.

Procedure: Slurry as received was pulped to 1300 g/L and passed over a Carpco LC3000 Spiral to separate sulphides from gangue.
The test was conducted in closed loop with a small circulating load within the pumping system.
Test products were filtered and submitted for assay.

Size Analysis: Test feed was 81.1 % passing 200 mesh, K80 : 72 micrometers.

Feed: #1 -Detour Lake Sample, Added as required to fill circuit.
Test feed density was 1300 g/L (34.8 % solids).

Sample Period Data (Measured and Calculated):

Spiral Product	Actual Measured Parameters						Calculated 10 sec. Sample Period				
	Sample Period seconds	Pulp Wet Weight grams	Weight Dry Weight grams	Data,g % Solids	Solids S.G.	Pulp Density g/L	Pulp Wet Weight grams	Weight Dry Weight grams	Data,g % Solids	Density g/L	Flow Dry kg/h
Concentrate	17.0	845	515.9	61.1	3.03	1680	497	303	61.1	1691	109
Middlings	10.3	1660	759.5	45.8	3.02	1408	1612	737	45.8	1440	265
Tailings	10.0	7875	2681.8	34.1	2.99	1270	7875	2682	34.1	1295	965
Test Feed	-	-	-	34.8	2.95	1300	9984	3723	37.3	1330	1340

Metallurgical Results: 10 second sample period

Spiral Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Cu %	S(T) %	Fe %	Au	Cu	S(T)	Fe
Concentrate	3.03	303	8.2	0.55	0.19	4.17	11.1	17.1	13.7	17.7	10.6
Middlings	3.02	737	19.8	0.33	0.091	1.73	7.87	25.0	16.0	17.8	18.2
Tailings	2.99	2682	72.0	0.21	0.11	1.72	8.48	57.9	70.3	64.5	71.3
Head(Calc)	3.00	3723	100.0	0.26	0.11	1.92	8.57	100.0	100.0	100.0	100.0
Head (Direct)	2.95	-	-	0.24	0.13	1.95	8.76	-	-	-	-
Comb. Products											
Conc.+Midds	3.03	1041	28.0	0.39	0.12	2.44	8.81	42.1	29.7	35.5	28.7

Comments: Spiral cutter setting adjusted to suit test conditions during test.

Test No.: #1 Detour Lake - Trays Project: 4370-069 Date: Dec. 15,1993
MEND

Purpose: To investigate sulphide recovery by gravity separation using Reichert Trays.

Procedure: Slurry as received was pulped to 1420 g/L and passed over a Reichert tray setup consisting of two rougher trays and a cleaner tray to treat the combined rougher concentrate. After the first sample period a volume of pulp equal to that of the sample period was was placed back into the circuit and a bulk gravity tail sample was removed for future environmental testing.
The test was conducted in closed loop with as large a circulating load as the pumping system could handle.
Test products were filtered and submitted for assay.

Size Analysis: Test feed was 81.1 % passing 200 mesh, K80 : 72 micrometers.

Feed: #1 -Detour Lake Sample, Added as required to fill circuit.
Test feed density was 1420 g/L (44.8 % solids).

Sample Period Data (Measured and Calculated):

Spiral Product	Actual Measured Parameters						Calculated 3 sec. Sample Period				
	Sample	Pulp	Weight	Data,g		Pulp	Pulp	Weight	Data,g		Flow
	Period seconds	Wet Weight grams	Dry Weight grams	% Solids	Solids S.G.	Density g/L	Wet Weight grams	Dry Weight grams	% Solids	Density g/L	Dry kg/h
Concentrate	3.0	700	337.4	48.2	2.92	1472	700	337	48.2	1465	405
Cleaner Tail	3.0	1170	521.0	44.5	2.91	1430	1170	521	44.5	1414	625
Tailings	3.0	1165	490.9	42.1	2.94	1390	1165	491	42.1	1295	589
Bulk Grav Tail	-	-	923.8	-	2.94						
Test Feed	-	-	-	44.8	2.95	1420	3035	1349	44.5	1390	1619

Metallurgical Results: 3 second sample period

Spiral Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Cu %	S(T) %	Fe %	Au	Cu	S(T)	Fe
Concentrate	2.92	337	25.0	0.33	0.10	2.10	8.63	29.3	21.7	26.9	24.7
Cleaner Tail	2.91	521	38.6	0.30	0.12	1.92	8.88	41.1	40.3	37.9	39.3
Tailings	2.94	491	36.4	0.23	0.12	1.89	8.62	29.7	38.0	35.2	36.0
Head(Calc)	2.92	1349	100.0	0.28	0.11	1.95	8.72	100.0	100.0	100.0	100.0
Head (Direct)	2.95	-	-	0.24	0.13	1.95	8.76	-	-	-	-
Bulk Grav Tl.	2.94	923.8		0.23	0.11	1.76	8.66				
Comb. Products											
Con+Clnr.Tl	2.91	858	63.6	0.31	0.11	1.99	8.78	70.3	62.0	64.8	64.0

Comments: Trays flow splitter setting was set to a setting of 5 or 6 notches for rougher and cleaner stages.

Test No.: #2 Lac -Table Project: 4370-069 MEND Date: Dec. 15,1993

Purpose: To investigate sulphide recovery by tabling.

Procedure: Slurry as received was pulped to 1300 g/L and passed over a 1/8 Wilfley table to separate sulphides from gangue.
The test was conducted over a period of 4.44 minutes after which the table was carefully cleaned.
Products were filtered and submitted for assay.

Size Analysis: Test feed was 85.3 % passing 200 mesh, K80 : 63 micrometers.

Feed: 8200 grams of #2 -Lac Minerals Sample
Repulped feed at 1300 g/L (35.5 % solids) was fed at a rate of 4 liters per minute.

Metallurgical Results:

Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Te g/t	S(T) %	Fe %	Au	Te	S(T)	Fe
Table Conc	3.12	586.5	7.2	0.65	6.0	13.7	13.2	24.1	7.6	25.3	18.6
Table Midds	2.83	1451.5	17.7	0.32	4.0	2.80	3.77	29.3	12.6	12.8	13.1
Table Tail	2.83	6152.3	75.1	0.12	6.0	3.20	4.62	46.6	79.8	61.9	68.3
Head(Calc)	2.85	8190	100.0	0.19	5.6	3.88	5.08	100.0	100.0	100.0	100.0
Head (Direct)	2.84	-	-	0.19	5.0	4.17	5.14	-	-	-	-
Comb. Products Conc.+Midds	3.03	2038	24.9	0.41	4.6	5.94	6.48	53.4	20.2	38.1	31.7

Comments: Slurry was viscous at 35.5 % solids, water added to the table feed well to dilute feed slurry.
Table settings adjusted to suit test conditions during test.

Test No.: #2 Lac Minerals -Spiral Project: 4370-069 MEND Date: Dec. 15,1993

Purpose: To investigate sulphide recovery by Spiral gravity separation.

Procedure: Slurry as received was pulped to 1300 g/L and passed over a Carpco LC3000 Spiral to separate sulphides from gangue.
The test was conducted in closed loop with a small circulating load within the pumping system.
Test products were filtered and submitted for assay.

Size Analysis: Test feed was 85.3 % passing 200 mesh, K80 : 63 micrometers.

Feed: #2 -Lac Minerals Sample, Added as required to fill circuit.
Test feed density was 1300 g/L (35.5 % solids).

Sample Period Data (Measured and Calculated):

	Actual Measured Parameters						Calculated 10 sec. Sample Period				
Spiral Product	Sample Period seconds	Pulp Weight Data,g Wet Weight grams	Dry Weight grams	% Solids	Solids S.G.	Pulp Density g/L	Pulp Weight Data,g Wet Weight grams	Dry Weight grams	% Solids	Density g/L	Flow Dry kg/h
Concentrate	14.0	590	319.8	54.2	2.85	1520	421	228	54.2	1544	82
Middlings	10.0	1005	412.3	41.0	2.81	1332	1005	412	41.0	1360	148
Tailings	10.0	8010	2704.0	33.8	2.80	1280	8010	2704	33.8	1277	973
Test Feed	-	-	-	35.5	2.84	1300	9436	3345	35.4	1297	1204

Metallurgical Results: 10 second sample period

Spiral Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays Au g/t	Te g/t	S(T) %	Fe %	% Distribution Au	Te	S(T)	Fe
Concentrate	2.85	228	6.8	0.46	8.0	5.50	6.70	15.5	5.8	9.7	9.1
Middlings	2.81	412	12.3	0.21	6.0	3.65	4.82	12.8	7.9	11.6	11.9
Tailings	2.80	2704	80.8	0.18	10.0	3.76	4.89	71.7	86.3	78.6	79.0
Head(Calc)	2.80	3345	100.0	0.20	9.4	3.87	5.00	100.0	100.0	100.0	100.0
Head (Direct)	2.84	-	-	0.19	5.0	4.17	5.14	-	-	-	-
Comb. Products											
Conc.+Midds	2.83	641	19.2	0.30	6.7	4.31	5.49	28.3	13.7	21.4	21.0

Comments: Spiral cutter setting adjusted to suit test conditions during test.

Test No.: #2 Lac Minerals - Trays Project: 4370-069 **Date:** Dec. 16,1993
MEND

Purpose: To investigate sulphide recovery by gravity separation using Reichert Trays.

Procedure: Slurry as received was pulped to 1400 g/L and passed over a Reichert tray setup consisting of two rougher trays and a cleaner tray to treat the combined rougher concentrate. After the first sample period a volume of pulp equal to that of the sample period was was placed back into the circuit and a bulk gravity tail sample was removed for future environmental testing.
The test was conducted in closed loop with as large a circulating load as the pumping system could handle.
Test products were filtered and submitted for assay.

Size Analysis: Test feed was 85.3 % passing 200 mesh, K80 : 63 micrometers.

Feed: #2 -Lac Minerals Sample, Added as required to fill circuit.
Test feed density was 1400 g/L (44 % solids).

Sample Period Data (Measured and Calculated):

	Actual Measured Parameters						Calculated 5 sec. Sample Period				
Spiral Product	Sample Period seconds	Pulp Wet Weight grams	Weight Dry Weight grams	Data,g % Solids	Solids S.G.	Pulp Density g/L	Pulp Wet Weight grams	Weight Dry Weight grams	Data,g % Solids	Density g/L	Flow Dry kg/h
Concentrate	5.0	1270	425.9	33.5	2.83	1424	1270	426	33.5	1273	307
Cleaner Tail	3.8	2180	1002.6	46.0	2.83	1410	2868	1319	46.0	1425	950
Tailings	5.0	3495	1272.8	36.4	2.84	1390	3495	1273	36.4	1310	916
Bulk Grav Tail	-	-	1206.2	-	2.83						
Test Feed	-	-	-	44.0	2.84	1400	7633	3018	39.5	1345	2173

Metallurgical Results: 5 second sample period

Spiral Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Te g/t	S(T) %	Fe %	Au	Te	S(T)	Fe
Concentrate	2.83	426	14.1	0.21	6.0	4.54	5.26	13.0	16.5	15.0	14.7
Cleaner Tail	2.83	1319	43.7	0.24	4.0	4.31	5.02	46.1	34.1	44.0	43.4
Tailings	2.84	1273	42.2	0.22	6.0	4.17	5.03	40.8	49.4	41.1	41.9
Head(Calc)	2.83	3018	100.0	0.23	5.1	4.28	5.06	100.0	100.0	100.0	100.0
Head (Direct)	2.84	-	-	0.19	5.0	4.17	5.14	-	-	-	-
Bulk Grav Tl.	2.83	1206.2		0.21	9.0	4.14	5.04				
Comb. Products											
Con+Clnr.Tl	2.83	1745	57.8	0.23	4.5	4.37	5.08	59.2	50.6	58.9	58.1

Comments: Trays flow splitter setting was set to a setting of 5 or 6 notches for rougher and cleaner stages. Feed slurry was viscous.

Test No.: #3.2 Selbaie -Table Project: 4370-069 MEND Date: Dec. 16,1993

Purpose: To investigate sulphide recovery by tabling.

Procedure: Slurry as received was pulped to 1277 g/L and passed over a 1/8 Wilfley table to separate sulphides from gangue.
The test was conducted over a period of 3.6 minutes after which the table was carefully cleaned.
Products were filtered and submitted for assay.

Size Analysis: Test feed was 88.9 % passing 200 mesh, K80 : 57 micrometers.

Feed: 3500 grams of #3.2 - Selbaie Sample
Repulped feed at 1277 g/L (30.9 % solids) was fed at a rate of 2.44 liters per minute.

Metallurgical Results:

Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Ag g/t	S(T) %	Fe %	Au	Ag	S(T)	Fe
Table Conc	4.13	455.4	13.0	0.37	37.5	40.5	35.0	27.9	21.3	26.9	24.6
Table Midds	2.94	549.4	15.7	0.11	13.9	10.4	10.2	10.0	9.5	8.3	8.7
Table Tail	3.15	2499	71.3	0.15	22.2	17.8	17.3	62.1	69.2	64.8	66.7
Head(Calc)	3.24	3503.8	100.0	0.17	22.9	19.59	18.5	100.0	100.0	100.0	100.0
Head (Direct)	3.31	-	-	0.19	26.3	20.9	20.4	-	-	-	-
Comb. Products											
Conc.+Midds	3.03	1004.8	28.7	0.23	24.6	24.0	21.4	37.9	30.8	35.2	33.3

Comments: Table settings adjusted to suit test conditions during test.

Test No.: #3.2 Selbaie -Spiral Project: 4370-069 Date: Dec. 16,1993
MEND

Purpose: To investigate sulphide recovery by Spiral gravity separation.

Procedure: Slurry as received was pulped to 1277 g/L and passed over a Carpco LC3000 Spiral to separate sulphides from gangue.
The test was conducted in closed loop with a small circulating load within the pumping system.
Test products were filtered and submitted for assay.

Size Analysis: Test feed was 88.9 % passing 200 mesh, K80 : 57 micrometers.

Feed: #3.2 - Selbaie Sample, Added as required to fill circuit.
Test feed density was 1277 g/L (31.1 % solids).

Sample Period Data (Measured and Calculated):

Spiral Product	Actual Measured Parameters						Calculated 5 sec. Sample Period				
	Sample Period seconds	Pulp Wet Weight grams	Weight Data,g Dry Weight grams	% Solids	Solids S.G.	Pulp Density g/L	Pulp Wet Weight grams	Weight Data,g Dry Weight grams	% Solids	Density g/L	Flow Dry kg/h
Concentrate	5.0	250	150	60.0	3.54	1803	250	150	60.0	1760	108
Middlings	5.0	710	284.6	40.1	3.18	1348	710	285	40.1	1386	205
Tailings	5.0	4220	1217.3	28.8	3.28	1220	4220	1217	28.8	1263	876
Test Feed	-	-	-	31.1	3.31	1277	5180	1652	31.9	1282	1189

Metallurgical Results: 5 second sample period

Spiral Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Ag g/t	S(T) %	Fe %	Au	Ag	S(T)	Fe
Concentrate	3.54	150	9.1	0.30	30.6	27.0	25.3	15.0	12.3	12.5	12.3
Middlings	3.18	285	17.2	0.17	20.4	16.3	15.6	16.1	15.6	14.3	14.4
Tailings	3.28	1217	73.7	0.17	22.0	19.5	18.6	68.9	72.0	73.2	73.3
Head(Calc)	3.29	1652	100.0	0.18	22.5	19.6	18.7	100.0	100.0	100.0	100.0
Head (Direct)	3.31	-	-	0.19	26.3	20.9	20.4	-	-	-	-
Comb. Products											
Conc.+Midds	3.30	435	26.3	0.21	23.9	20.0	18.9	31.1	28.0	26.8	26.7

Comments: Spiral cutter setting adjusted to suit test conditions during test.

Test No.: #3.2 Selbaie - Trays **Project:** 4370-069 MEND **Date:** Dec. 16,1993

Purpose: To investigate sulphide recovery by gravity separation using Reichert Trays.

Procedure: Slurry as received was pulped to 1430 g/L and passed over a Reichert tray setup consisting of two rougher trays and a cleaner tray to treat the combined rougher concentrate. After the first sample period a volume of pulp equal to that of the sample period was was placed back into the circuit and a bulk gravity tail sample was removed for future environmental testing.
The test was conducted in closed loop with as large a circulating load as the pumping system could handle.
Test products were filtered and submitted for assay.

Size Analysis: Test feed was 88.9 % passing 200 mesh, K80 : 57 micrometers.

Feed: #3.2 -Selbaie Sample, Added as required to fill circuit.
Test feed density was 1430 g/L (42.9 % solids).

Sample Period Data (Measured and Calculated):

Spiral Product	Actual Measured Parameters						Calculated 5.38 sec. Sample Period				
	Sample Period seconds	Pulp Wet Weight grams	Weight Dry Weight grams	Data,g % Solids	Solids S.G.	Pulp Density g/L	Pulp Wet Weight grams	Weight Dry Weight grams	Data,g % Solids	Density g/L	Flow Dry kg/h
Concentrate	5.38	990	458.5	46.3	3.33	1510	990	459	46.3	1487	307
Cleaner Tail	5.38	1980	859.7	43.4	3.30	1450	1980	860	43.4	1441	575
Tailings	5.38	3380	1240.2	36.7	3.30	1415	3380	1240	36.7	1347	830
Bulk Grav Tai	-	-	1712.0	-	3.27						
Test Feed	-	-	-	42.9	3.31	1430	6350	2558	40.3	1398	1712

Metallurgical Results: 5.38 second sample period

Spiral Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays Au g/t	Ag g/t	S(T) %	Fe %	% Distribution Au	Ag	S(T)	Fe
Concentrate	3.33	459	17.9	0.19	25.0	21.2	20.2	18.2	18.6	18.3	18.4
Cleaner Tail	3.30	860	33.6	0.18	23.4	20.5	19.4	32.4	32.7	33.2	33.1
Tailings	3.30	1240	48.5	0.19	24.1	20.8	19.7	49.3	48.6	48.5	48.5
Head(Calc)	3.31	2558	100.0	0.19	24.0	20.8	19.7	100.0	100.0	100.0	100.0
Head (Direct)	3.31	-	-	0.19	26.3	20.9	20.4	-	-	-	-
Bulk Grav Tl.	3.27	1712		0.19	24.1	20.6	19.7				
Comb. Products											
Con+Clnr.Tl	3.31	1318	51.5	0.18	24.0	20.7	19.7	50.7	51.4	51.5	51.5

Comments: Trays flow splitter setting was set to a setting of 5 or 6 notches for rougher and cleaner stages.

Test No.: #3.3 Selbaie -Table Project: 4370-069 MEND Date: Dec. 15,1993

Purpose: To investigate sulphide recovery by tabling.

Procedure: Slurry as received was pulped to 1290 g/L and passed over a 1/8 Wilfley table to separate sulphides from gangue.
The test was conducted over a period of 4.81 minutes after which the table was carefully cleaned.
Products were filtered and submitted for assay.

Size Analysis: Test feed was 91.4 % passing 200 mesh, K80 : 53 micrometers.

Feed: 23800 grams of #3.3 - Selbaie Sample
Repulped feed at 1290 g/L (35.2 % solids) was fed at a rate of 10.9 liters per minute.

Metallurgical Results:

Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Ag g/t	S(T) %	Fe %	Au	Ag	S(T)	Fe
Table Conc	3.18	1107.8	4.7	0.47	30.5	16.8	16.3	18.0	15.9	21.0	14.5
Table Midds	2.76	2113.7	8.9	0.15	6.9	2.52	3.57	10.9	6.8	6.0	6.1
Table Tail	2.77	20600	86.5	0.10	8.0	3.15	4.81	71.1	77.3	73.1	79.5
Head(Calc)	2.79	23822	100.0	0.12	8.9	3.73	5.23	100.0	100.0	100.0	100.0
Head (Direct)	2.75	-	-	0.09	8.8	3.63	5.23	-	-	-	-
Comb. Products Conc.+Midds	3.03	3221.5	13.5	0.26	15.0	7.43	7.95	28.9	22.7	26.9	20.5

Comments: Table settings adjusted to suit test conditions during test.

Test No.: #3.3 Selbaie -Spiral Project: 4370-069 Date: Dec. 14,1993
MEND

Purpose: To investigate sulphide recovery by Spiral gravity separation.

Procedure: Slurry as received was pulped to 1290 g/L and passed over a Carpco LC3000 Spiral to separate sulphides from gangue.
The test was conducted in closed loop with a small circulating load within the pumping system.
Test products were filtered and submitted for assay.

Size Analysis: Test feed was 91.4 % passing 200 mesh, K80 : 53 micrometers.

Feed: #3.3 -Selbaie Sample, Added as required to fill circuit.
Test feed density was 1290 g/L (35.2 % solids).

Sample Period Data (Measured and Calculated):

Spiral Product	Actual Measured Parameters						Calculated 10 sec. Sample Period				
	Sample Period seconds	Pulp Weight Data,g Wet Weight grams	Dry Weight grams	% Solids	Solids S.G.	Pulp Density g/L	Pulp Weight Data,g Wet Weight grams	Dry Weight grams	% Solids	Density g/L	Flow Dry kg/h
Concentrate	20.0	860	522.2	60.7	2.89	1640	430	261	60.7	1669	94
Middlings	11.4	1860	749.1	40.3	2.75	1340	1637	659	40.3	1348	237
Tailings	9.0	12400	4300.7	34.7	2.77	1260	13778	4779	34.7	1284	1720
Test Feed	-	-	-	35.2	2.75	1290	15845	5699	36.0	1297	2052

Metallurgical Results: 10 second sample period

Spiral Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays				% Distribution			
				Au g/t	Ag g/t	S(T) %	Fe %	Au	Ag	S(T)	Fe
Concentrate	2.89	261	4.6	0.71	18.8	8.56	9.09	20.6	9.7	10.3	7.8
Middlings	2.75	659	11.6	0.14	8.3	3.44	4.88	10.3	10.8	10.5	10.6
Tailings	2.77	4779	83.8	0.13	8.4	3.58	5.16	69.1	79.5	79.2	81.5
Head(Calc)	2.77	5699	100.0	0.16	8.9	3.79	5.31	100.0	100.0	100.0	100.0
Head (Direct)	2.75	-	-	0.09	8.8	3.63	5.23	-	-	-	-
Comb. Products											
Conc.+Midds	2.80	921	16.2	0.30	11.3	4.89	6.07	30.9	20.5	20.8	18.5

Comments: Spiral cutter setting adjusted to suit test conditions during test.

Test No.: #3.3 Selbaie - Trays Project: 4370-069 MEND Date: Dec. 14,1993

Purpose: To investigate sulphide recovery by gravity separation using Reichert Trays.

Procedure: Slurry as received was pulped to 1450 g/L and passed over a Reichert tray setup consisting of two rougher trays and a cleaner tray to treat the combined rougher concentrate. After the first sample period a volume of pulp equal to that of the sample period was was placed back into the circuit and a bulk gravity tail sample was removed for future environmental testing.
The test was conducted in closed loop with as large a circulating load as the pumping system could handle.
Test products were filtered and submitted for assay.

Size Analysis: Test feed was 91.4 % passing 200 mesh, K80 : 53 micrometers.

Feed: #3.3 - Selbaie Sample, Added as required to fill circuit.
Test feed density was 1450 g/L (48.5 % solids).

Sample Period Data (Measured and Calculated):

Spiral Product	Actual Measured Parameters						Calculated 4.0 sec. Sample Period				
	Sample Period seconds	Pulp Wet Weight grams	Weight Dry Weight grams	Data,g % Solids	Solids S.G.	Pulp Density g/L	Pulp Wet Weight grams	Weight Dry Weight grams	Data,g % Solids	Density g/L	Flow Dry kg/h
Concentrate	5.00	1247	643.1	51.6	2.76	-	998	514	51.6	1476	463
Cleaner Tail	5.31	797	389.5	48.9	2.76	-	600	293	48.9	1453	264
Tailings	4.00	1680	811.7	48.3	2.73	-	1680	812	48.3	1440	731
Bulk Grav Tail	-	-	1021.0	-	2.72						
Test Feed	-	-	-	48.5	2.75	1450	3278	1620	49.4	1460	1458

Metallurgical Results: 4.0 second sample period

Spiral Products	Specific Gravity g/cc	Wt. grams	Wt. %	Assays Au g/t	Ag g/t	S(T) %	Fe %	% Distribution Au	Ag	S(T)	Fe
Concentrate	2.76	514	31.8	0.14	7.6	3.13	4.18	39.5	31.8	33.1	29.7
Cleaner Tail	2.76	293	18.1	0.10	7.3	3.03	4.65	16.1	17.4	18.3	18.9
Tailings	2.73	812	50.1	0.10	7.7	2.91	4.58	44.5	50.8	48.6	51.4
Head(Calc)	2.74	1620	100.0	0.11	7.6	3.00	4.47	100.0	100.0	100.0	100.0
Head (Direct)	2.75	-	-	0.09	8.8	3.63	5.23	-	-	-	-
Bulk Grav Tl.	2.72	1021.2		0.10	7.5	3.08	4.73				
Comb. Products											
Con+Clnr.Tl	2.76	808	49.9	0.13	7.5	3.09	4.35	55.5	49.2	51.4	48.6

Comments: Trays flow splitter setting was set to a setting of 5 or 6 notches for rougher and cleaner stages.